N.I. 43-101 TECHNICAL REPORT & PRELIMINARY ECONOMIC ASSESSMENT OF THE MARBAN PROJECT

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Important Notice

This report was prepared as National Instrument 43-101 Technical Report for O3 Mining Inc. (O3 Mining) by Ausenco Engineering Canada Inc. (Ausenco), Golder Associates Ltd. (Golder), Moose Mountain Technical Services (MMTS), WSP Canada Inc. (WSP), and Kenneth Williamson (Williamson), 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 subject to terms and conditions of the Report Authors' respective contracts with O3 Mining. Except for the purposed legislated under Canadian provincial and territorial securities law, any other uses of this report by any third party is at that party's sole risk.

1 Summary

1.1 Introduction

Ausenco Engineering Canada Inc. (Ausenco) has prepared a preliminary economic assessment (PEA) report for O3 Mining Inc. (O3 Mining) on the Marban Project located in Val-d'Or, Québec. The report was prepared in compliance with the Canadian disclosure requirements of National Instrument 43-101 (NI 43-101) and in accordance with the requirements of Form 43-101 F1. O3 Mining owns 100% of the Malartic property, which hosts the Marban Project.

The NI 43-101 responsibilities of the engineering consultants are as follows:

- Ausenco was commissioned by O3 Mining to manage and coordinate the work related to the NI 43-101. Ausenco also developed the PEA level design and cost estimating of the process plant and surface infrastructure.
- WSP Canada Inc. (WSP) was commissioned to support environmental planning, assessment, licensing, and permitting.
- Golder Associates Ltd. (Golder) was commissioned to complete the PEA design of the tailings management facility (TSF) and mining geotechnical for the pits and waste rock stockpiles. Ausenco's qualified person reviewed and approved Golder's design.
- Moose Mountain Technical Services (MMTS) was commissioned to complete the quality assurance and quality control (QAQC) analyses, mineral resource estimates and to design the open pit mine plan, mine production schedule, and mine capital and operating costs.
- Kenneth Williamson (Williamson) was commissioned to complete a geological model of the Marban resource.

1.2 Terms of Reference

The Report supports disclosures by O3 Mining in a news release dated 8 September, 2020 entitled "O3 Mining Delivers Positive PEA For Marban Project".

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

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 Project area contains three past-producing mines (Marban, Norlartic and Kierens), which collectively produced 585,000 ounces of gold between 1959 and 1992. O3 Mining owns 100% of the Malartic property, which hosts the Marban Project.

1.3 Property Description & Location

The Marban Project is located in the western portion of the province of Québec, Canada, about midway between the towns of Val-d'Or and Malartic and is comprised of 30 mining claims and 3 mining concessions covering 1,023 hectares. The project lies at the junction of Dubuisson, Fournière, Malartic, and Vassan townships, at and south of Latitude 48.15497°N, Longitude 78.00544°W, on NTS sheets 32D/01 and 32C/04.

1.4 Accessibility, Climate, Local Resources, Infrastructure & Physiography

The Marban Project property is located on the north side of Highway 117, approximately 15 kilometres northwest of Val-d'Or, Québec and immediately east of Monarch Gold's Camflo mill. The Camflo road provides access to the western part of the project area. A well-maintained all-weather gravel road running north from Highway 117 provides access to the historic Norlartic, Kierens, and Marban mines. Winter access for snowmobiles and all-terrain vehicles is provided by trails and winter roads.

The mean elevation of the property is about 300 metres above sea level. The property is relatively flat, with local relief of 15 to 30 metres. The surface is dominated by swamps and glacial deposits; a north-trending esker lies on the west side of the Norlartic Mine road. The overburden is generally between 10 and 40 metres thick, but locally it is very shallow and there are some outcrops (Ducharme et al., 2009).

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

1.5 History

Exploration conducted at the Marban Project property apparently 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.

Since first gaining interests in the property in early 2006, NioGold Mining Corporation (NioGold) which was acquired by Osisko Mining Inc. (Osisko) in 2016 and subsequently span-out to O3 Mining in 2019, including the work of the Aurizon- NioGold joint venture, has carried out the following activities at the Marban Project 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
- Structural study using LANDSAT imagery

- Drilling of 722 holes on the four claim groups (representing 220,929 metres of drilling), as well
- Metallurgical testwork

1.6 Geological Setting & Mineralisation

The Marban Project lies within the Archean Abitibi greenstone belt of the Superior Province, Québec, which consists of alternating east-trending metavolcanic-plutonic and sedimentary belts that are bounded by crustal-scale faults. The metavolcanic rocks within the Malartic mining district are composed of allochthonous komatiitic to tholeiitic volcanic suites that are interpreted to have been derived from volcanic plateaus and volcanic islands juxtaposed during the late Archean.

The metavolcanic rocks within the Marban Project are cut by three major northwest to west-northwest-striking shear zones of regional extent – the North, Norbenite, and Marbanite shears. The Marbanite shear hosts the Marban resource described in this report, while the Norbenite shear hosts both the Kierens and Norlartic gold resources. The historic Kierens, Norlartic, and Marban gold mines all exploited zones lying within the basalts and ultramafic rocks of the Jacola Formation, which have been intruded by numerous diorites and felsic porphyry dikes and sills.

Gold mineralization at the Marban Project occurs primarily as mesothermal deposits that lie within the north-dipping shear zones and associated felsic to intermediate sills and dikes. In addition to the resource areas, O3 Mining has been actively exploring sub-parallel gold-mineralized horizons within the Marban Project at the North-North, North, and Gold Hawk zones.

The Marban resource is located on the northern flank of a west-northwest-trending syncline. From north to south, the units include:

- An alternating sequence of sheared ultramafic volcanic rocks (now talc schist) and granodiorite dikes
- The “Mine Sequence” of pillowed, sheared, aphanitic and fine-grained basalts
- Sheared and massive or pillowed ultramafic volcanic rocks

These units dip 45 to 70° to the north. The mineralized zones occur almost exclusively within the Mine Sequence basalts, which frequently have been converted to chloritic schists within the Marbanite shear. Gold is associated with quartz-carbonate veinlets, pyrite, and pyrrhotite. There is also disseminated replacement-type gold mineralization in tholeiitic-komatiitic volcanic rocks of the Jacola Formation, and locally there is gold in granodioritic dikes.

The Kierens and Norlartic resources are localized in the Norbenite shear, an important northwest trending deformation zone that dips moderately to steeply northeast. Mafic volcanic rocks, talc-chlorite schists, and felsic dikes are included within the shear zone. The gold mineralization is closely associated with altered and pyritized intermediate intrusions that cut sheared mafic and ultramafic volcanic rocks.

The Norlartic resource includes mineralization within the Main zone, which lies west of the Norlartic shaft and was the focus of historic mining. The extension of the Main zone east of the shaft, as well as the Actinolite zone that lies in the hanging wall of the Main zone, are also

included in the resources. The Main zone mineralization occurs within and adjacent to a sill or dike that occurs within a wide envelope of sheared mafic rocks in the core of the Norbenite shear. Gold is usually directly associated with the intrusion in quartz-calcite-pyrite veinlet stockworks. In contrast to the Main zone, the Actinolite zone is not lithologically controlled, but instead it appears to be an actinolite-bearing alteration zone that overlaps various units of mafic or ultramafic volcanic rocks. Gold is disseminated without regards to lithology in the Actinolite zone.

The Kierens resource occurs in a strongly silicified and carbonatized alteration zone that occupies the central portion of the Norbenite shear, which dips steeply to the north in this area. The alteration and mineralization transect ultramafic rocks, more massive mafic units, and felsic intrusions. A westerly plunging drag fold may exert the main control on gold mineralization at Kierens. In addition to this type of mineralization, there are native-gold-bearing quartz veins at Kierens.

1.7 Deposit Types

The Marban Project property lies within the Malartic mining camp and is in close proximity to the Val-d'Or camp. These mining areas have a combined historic production of 25 million ounces of gold and continue to produce gold today; historic production has been primarily from underground mines, although significant open-pit production is evident at the present time. The Malartic mining area is noted for its variety of gold occurrences, including such major gold deposits such as the East-Malartic, Camflo-Malartic Hygrade, Malartic Gold Fields, Barnat-Sladen, and Canadian Malartic. These mines yielded 8.9 million ounces of gold from 1935 to 1984.

Gold mineralization from these mining districts have been classified as mesothermal or late-orogenic lode deposits. The mineralization is associated with regional structural zones (e.g. the Cadillac-Larder Lake tectonic zone), regional drag folds, and structural splays, as well as with syn to late-tectonic intrusive rocks. The different styles of mineralization range from disseminated-sulfide deposits to quartz tourmaline gold-bearing veins and vein stockwork zones. Deposits range from early to late-tectonic in age.

The Marban Project is similar in many respects to mines and deposits in the surrounding area. The Norbenite and Marbanite shears are major structures that range from tens to hundreds of metres in width. The presence of lamprophyre intrusions within or adjacent to these structural corridors is suggestive of a deeply rooted deformation system. These shear zones are affected by at least two major deformation episodes that caused both open and very tight folding. The Marbanite shear hosts mineralization at Marban, which is a multi-folded mesothermal-type deposit. The high-grade Gold Hawk veins also occur within this shear corridor. There are numerous gold occurrences encountered between Marban and Gold Hawk, as well as along strike, including some at the Malartic Hygrade mine.

The Kierens and Norlartic mineralization is confined to the Norbenite shear and is associated with intermediate dikes that host auriferous stockworks. This shear may also host the adjacent Kiena and Goldex mines. An additional style of mineralization occurs as gold-bearing quartz-tourmaline stockworks within larger intrusions, such as the North-North granodiorite.

1.8 Exploration

The two Marban Project drill-hole databases were provided to MMTS by O3 Mining, who acquired the historic database, and incorporated Osisko's drill data. The total O3 Mining

database contains collar coordinates and geologic and assay information for a total of 5,809 drill holes; the database for the Marban deposit also includes 1,356 channel samples.

1.9 Drilling

O3 Mining drilling 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 drill holes 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 drill holes 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 modeling technique that constrains all intercepts within interpreted domains that respect the true mineralized widths.

The drill-hole data were reviewed in the context of the geology of the Marban, Kierens, and Norlartic mineralization, and MMTS believes that the sample lengths are generally appropriate for the style of mineralization in each target area.

1.10 Sample Preparation, Analyses & Security

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. The drillhole data subsequent to the 1960s drilling that is lacking QAQC has been validated through analysis and comparison of re-assays as discussed below.

1.11 Data Verification

The data has been verified through a site visit, certificate checks, and numerous statistical analyses, particularly to validate the pre-NioGold assays. 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 modeling because it is shown to be biased and unreliable in a range of significance to the modelling.
- The Kierens-Norlartic historic data earlier than 1986 is shown to be biased and unreliable and has not been included in the data set for resource modeling.
- The Marban data since the 1980s and the Kierens-Norlartic data from 1986 forward is considered of acceptable quality for resource modeling.
- All assay data subsequent to the dates above for both areas, including re-assays of the historic drilling has been included.

1.12 Mineral Processing & Metallurgical Testing

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

The following sources of technical and project information were referenced in developing the process plant design for the preliminary economic assessment:

- 2012 SGS Revised Preliminary Metallurgy Study Marban report 13333-001
- 2015 FLSmidth Gravity Circuit Modelling and scale-up report. NioGold Mining Corp. KRT20951
- 2015 Report – Marban Gold Mineralogy NG15-01
- 2016 SGS The Recovery of Gold from Marban Project Samples report 14947-002
- 2016 SGS The Grinding Circuit Design Based on Ten Bench-Scale Grindability Test results from the Marban Project report 149647-003
- 2016 SGS The Recovery of Gold from Norlartic Project Samples report 14947-004

Preliminary metallurgical testing was carried out in 2012 at SGS located in Lakefield, Canada, on a high grade and low-grade composite using fifty drill core samples for each composite, collected from the Marban deposit.

The testwork program included head analysis, mineralogy, gravity separation and cyanidation and flotation testing on gravity tailings and whole ore cyanidation on the two composites. The direct gold head grades were 1.24 g/t to 4.59 g/t for the 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.

The whole ore leach tests on the two composites displayed gold extractions, which ranged from 95.4 to 97.6% for the low-grade head grade composite, and for the high grade composite was 95.7 to 97.3%.

Cyanide and, lime consumptions were derived from the testwork and from leach tests with pre-aeration, which reduces reagent consumption. Average cyanide consumption was 0.77 kg/t and lime consumption was 1.12 kg/t.

1.13 Mineral Resource Estimate

The Mineral Resource estimate for O3’s Marban and Kierens-Norlartic deposits have been updated since the previous estimate in 2016 due to updated geologic modelling, updated QAQC and updated gold prices. The Mineral Resource with an effective date of 12 August, 2020 is listed in Table 1-1. Canadian Institute of Mining, Metallurgy and Petroleum (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 87%, Processing + G&A costs of CDN$15/tonnes and a US$1400/oz Au price, with smelter terms as detailed in the

notes below. The underground resource is within a 3.5 g/t gradeshell. At these cut-offs, the total Measured and Indicated Mineral Resource is estimated at 54.1 Mt at a grade of 1.10 g/t Au for a total of 1.9 Moz.

Table 1-1: Mineral Resource Estimate (effective date: 12 August, 2020)

Class	Source	Tonnage (kt)	Au (g/t)	Au Metal Ounces
Measured	Marban - Pit	65	1.32	2,762
	Kierens-Norlartic - Pit	450	1.03	14,900
Indicated	Marban - Pit	46,260	1.03	1,536,671
	KN-Pit	6,646	1.15	246,430
	Marban - UG	220	7.77	54,982
	Kierens-Norlartic - UG	510	3.57	58,504
Meas. + Ind.	**All**	**54,151**	**1.10**	**1,914,249**
Inferred	Marban - Pit	6,465	1.09	227,226
	Kierens-Norlartic - Pit	6,299	1.42	286,724
	Marban - UG	304	8.73	85,317
	Kierens-Norlartic - UG	119	3.02	11,560
	All	**13,187**	**1.44**	**610,827**

Notes:

1. *The Mineral Resource estimate has been prepared by Sue Bird, P.Eng., an independent Qualified Person.*
2. *Resources are reported using the 2014 CIM Definition Standards and were estimated in accordance with the CIM 2019 Best Practices Guidelines.*
3. *Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.*
4. *The open pit Mineral Resource has been confined by a "reasonable prospects of eventual economic extraction" pit shell generated using the following assumptions: US$1,800/oz. Au at a currency exchange rate of 0.75 US$ per CDN$; 99.95% payable Au; CDN$4.30/oz Au offsite costs (refining, transport and insurance); a 3% NSR royalty; $16/t process and G&A costs; $2.60/t mining costs; grade dependent mill process recoveries; and pit slopes varying from 25 to 50 degree overall depending on geotechnical characteristics.*
5. *The underground Mineral Resource reports all material within a continuous 3.5g/t Au gradeshell*
6. *Metallurgical recovery is based on the formula: ln(Au)*0.0372+0.9017, maximum 96.7%*
7. *The specific gravity of the deposit has been determined by lithology as being between 2.67 and 2.81.*
8. *Numbers may not add due to rounding.*

The QP 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.

1.14 Mining Methods

Open pit mine designs, mine production schedules and mine capital and operating costs have been developed for the Marban and Kierens-Norlartic (KN) deposits at a scoping level of engineering. The Mineral Resources form the basis of the mine planning.

The open pit operations are designed for approximately fifteen years of operation. Mine planning is based on conventional open pit methods suited for the project location and local site requirements. The subset of Mineral Resources contained within the designed open pits are summarized in Table 1-2, with a 0.35 g/t gold cut-off, and forms the basis of the mine plan and production schedule.

Table 1-2: PEA Mine Plan Production Summary

Description	Value
PEA Mill Feed	60,356 kt
Mill Feed Gold Grade	0.97 g/t
Waste Overburden and Rock	355,627 kt
Strip Ratio	5.9

Notes:

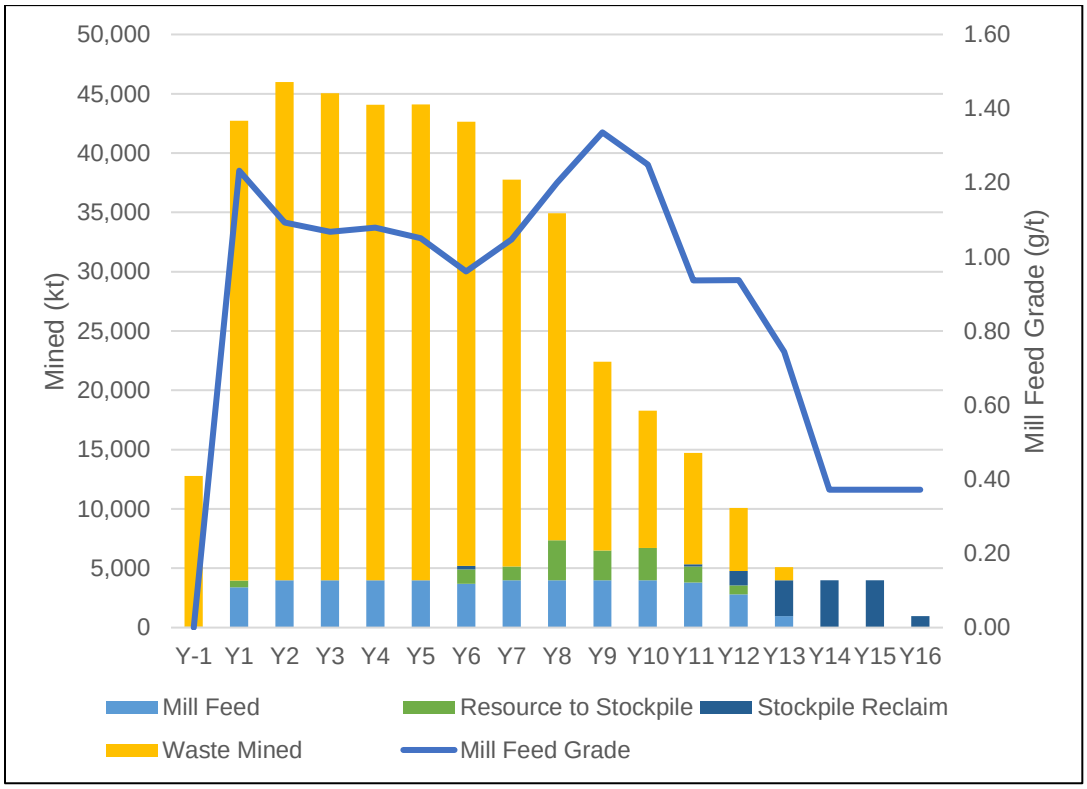
1. *The PEA Mine Plan and Mill Feed estimates are a subset of the August 12, 2020 Mineral Resource estimates and are based on open pit mine engineering and technical information developed at a Scoping level for the Marban and Kierens-Norlartic deposit.*
2. *PEA Mine Plan and Mill Feed estimates are mined tonnes and grade, the reference point is the primary crusher.*
3. *Waste/mineralization contact edge dilution of 27% at 0.14 g/t for Marban 20% at 0.15 g/t for KN is applied to the insitu Mineral Resources. Mining Recovery of 100% of diluted tonnages is assumed.*
4. *Cut-off grade of 0.35 g/t assumes US$1,400/oz. Au at a currency exchange rate of 0.75 US$ per C$; 99.95% payable gold; $4.30/oz offsite costs (refining, transport and insurance); a 3.0% NSR royalty; and a variable metallurgical recovery for gold that is anticipated to be 88% at cut-off.*
5. *The cut-off grade above covers processing costs of $14.00/t, administrative (G&A) costs of $2.00/t, and low-grade stockpile Rehandle costs of $1.50/t.*
6. *Estimates have been rounded and may result in summation differences.*

Ultimate pit limits are split up into phases or pushbacks to target higher economic margin material earlier in the mine life. The Marban deposit is split into four phases, and the KN deposit is split into six phases, two for Kierens, two for Norlartic, one for North Zone and one for Gold Hawk. Pit designs are configured on 5 m bench heights, with 8 to 9 m wide berms placed every four benches, to create quadruple benching. Bench face angles, and subsequent inter-ramp angles, are varied based on prescribed geotechnical zones.

The mill will be fed with material from the pits at an average rate of 4.0 Mtpa (11 ktpd). Cut-off grade optimization is employed that results in a low-grade stockpile south of the Marban deposit and is planned for reclamation to the mill in the later years of the mine life. Overburden will be placed in a stockpile directly north of the Marban pit. Waste rock will be placed in stockpiles adjacent to all pits. Waste rock will also be used for construction of the tailings and water management pond northwest of the deposits.

The mine production schedule is summarized in Figure 1-1 below.

Figure 1-1: Mine Production Schedule Summary



Source: Moose Mountain, 2020

Mining operations will be based on 365 operating days per year with two 12-hour shifts per day. An allowance of 10 days of no mine production has been built into the mine schedule to allow for adverse weather conditions.

The mining fleet will include diesel-powered down-the-hole (DTH) drills with 165 mm bit size for production drilling, diesel-powered reverse circulation (RC) drills for bench-scale grade control drilling, 12 m^3 bucket size diesel hydraulic excavators and 13 m^3 bucket sized wheel loaders for production loading, and 91 t payload rigid-frame haul trucks and 36 t articulated trucks for production hauling, plus ancillary and service equipment to support the mining operations. In-pit dewatering systems will be established for each pit. All surface water and precipitation in the pits will be handled by submersible pumps.

The mine equipment fleet is planned to be purchased via a lease financing arrangement. Maintenance on mine equipment will be performed in the field with major repairs to mobile equipment in the shops located near the plant facilities.

1.15 Recovery Methods

The unit operations were selected for the respective processing routes based on the outcome of the preliminary testwork and preliminary financial evaluations. The unit options for the options are also are very common to the industry.

The project focused on capital cost-effective mill design, with a target grind of 80% passing (P_{80}) 100 µm based on the preliminary testwork reviewed in Section 13.

Key process design criteria listed in Table 1-3 were derived from the preliminary comminution testwork.

Table 1-3: Key Process Design Criteria

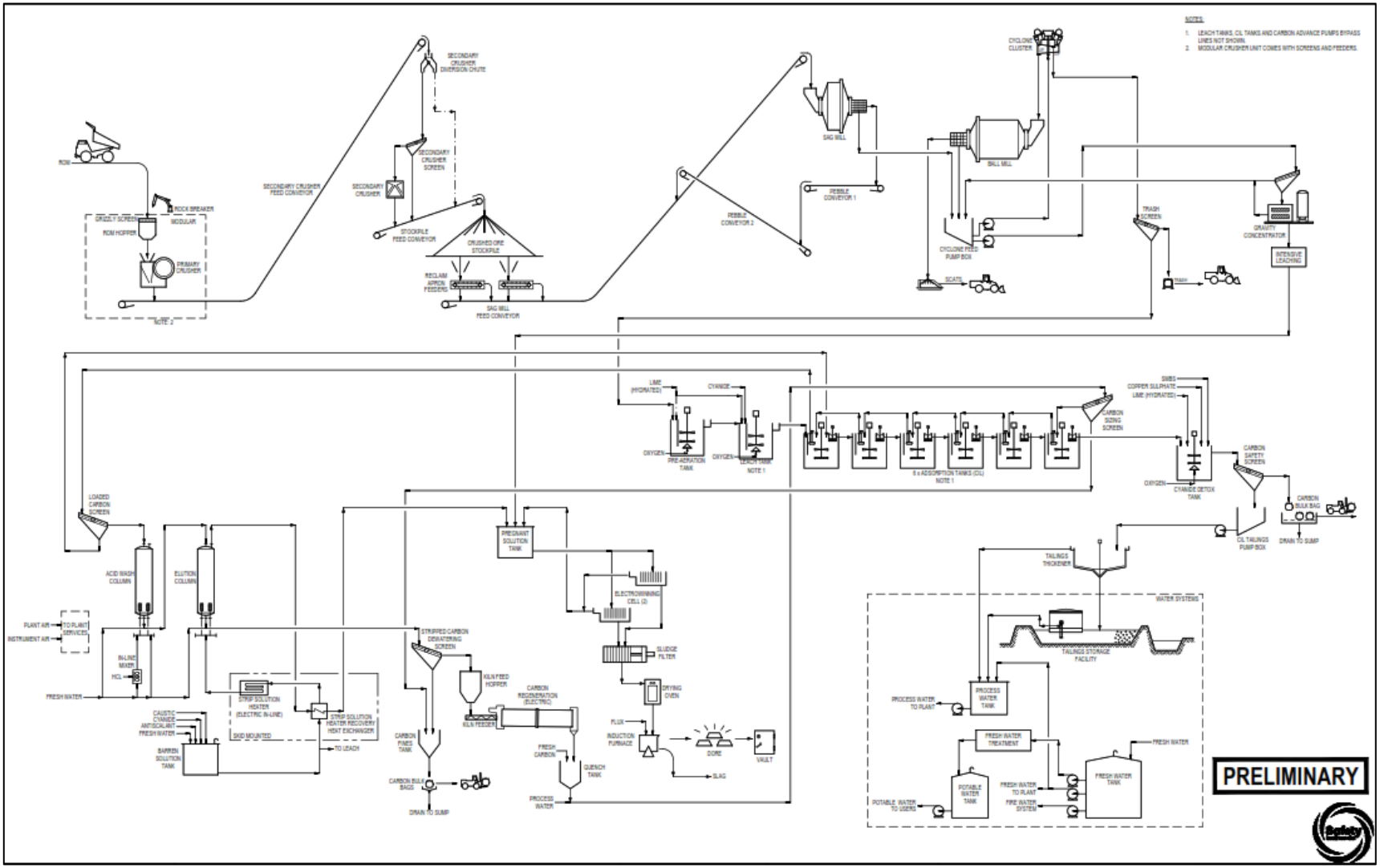
Design Parameter	Units	Value
Throughput	Mt/a	4.0
	t/d	10,960
Material Specific Gravity	t/m³	2.78
Primary Grind size (P80)	µm	100
Moisture Content	%	3.0
Crushing Area Availability	%	67
Crushing Operating Throughput	t/h	685
Process Plant Availability	%	92
Process Plant Operating Throughput	t/h	496
Bond Crusher Work Index (CWi)	kWh/t	16.5
Bond Ball Mill Work Index (BWi)	kWh/t	11.0
Gravity Circuit Feed Source		Ball mill discharge
Gravity Circuit Feed Rate	%cyclone recirc	25
L-CIL Residence Time	h	28
L-CIL Extraction	Au (%)	92.5
L-CIL Operating Density	% w/w	42.5
L-CIL DO Target	mg/L	20
L-CIL pH Target		10.5 – 11.0
L-CIL Carbon Concentration	g/L	10
L-CIL Sodium Cyanide Addition	kg/t	0.3
L-CIL Hydrated Lime Addition	Ca(OH)2 kg /t	1.0
Pre-aeration, Leach & CIL Tanks	#	1 + 1 + 6
Overall Recovery	Au (%)	93.7

1.16 Process Plant Description

The process design is comprised of the following as illustrated in the flow diagram below :

- primary and secondary crushing of run-of-mine (ROM) material
- a covered crushed material stockpile to provide buffer capacity for the process plant
- SAG mill with trommel screen followed by a ball mill with cyclone classification
- gravity recovery of ball mill discharge by one semi-batch centrifugal gravity concentrator, followed by intensive cyanidation of the gravity concentrate and electrowinning of the pregnant leach solution.
- trash screening

Figure 1-2: Process Flowsheet



Source: Ausenco, 2020

- pre-aeration, leach and carbon in leach (CIL) adsorption
- acid washing of loaded carbon and Anglo-American Research Laboratory (AARL) type elution followed by electrowinning and smelting to produce doré
- carbon regeneration
- cyanide destruction of tailings using SO_2 air process
- carbon screening, tailings thickening and disposal

1.17 Project Infrastructure

The Marban project is located in the Abititi-Témiscamingue region, between Val-d'Or and Malartic in the province of Québec in Eastern Canada. The site is accessible by a short well-maintained gravel road from Highway 117.

Infrastructure to support the Marban project will consist of site civil work, site facilities/building, a water system, and site electrical. These are indicated in Figure 18-1.

Site civil work includes designs for the following infrastructure:

- Light vehicle and heavy equipment roads;
- Access road;
- Overburden stripping and stockpiling;
- Mine facility platforms and process facility platforms;
- Water management ponds;
- Tailings storage facility (TSF);
- Waste storage facilities;
- High voltage substation platform.

Site facilities will include both mine facilities and process facilities:

- The mine facilities will include the administration offices, truck-shop and warehouse, tire repair shop, mine workshop, mine dry, and miscellaneous facilities;
- The process facilities will include the process plant, crusher facility, process plant workshop and assay laboratory;
- Both the mine facilities and process facilities will be serviced with potable water, fire water, compressed air, power, diesel, communication, and sanitary systems.

The Marban Project site location is located east of De Montigny Lake. The Kierens and Norlartic (KN) open pits require a diversion of Kierens Creek water with two dams either side of the planned excavations. The tailings storage facility and water treatment pond are located to the north-west, outside of the Marban Project Property, but within the Malartic property. Waste rock is stockpiled in a central location south of the KN pits and west of the Marban pit. The processing plant is located south of the waste rock stockpile. The arrangement of the project site is illustrated in Figure 1-3.

The selection of the site was based on the following observations and factors:

- Within the claim boundary
- Limit building on wetlands
- Locate the waste rock stockpile close to the mine pits to reduce haul distance
- Locate the process plant in an area safe from flooding
- Existing roads south and east of the site (Highway 117 and Chemin Gervais)
- Arrange administration, processing plant and offices in close proximity to reduce to limit walking distances (important during extreme cold weather)
- Proximity of the processing plant to the existing power line (Hydro-Québec)
- Maintain distance from CN railway

1.18 Environmental Studies, Permitting & Social or Community Impact

Several baseline studies have been completed on the Marban Project site. Additional field inventories and surveys will have to be completed within the project area, as well as environmental assessments to support the permitting process.

With its average extraction rate of 12,702 t/d and a process plant with a capacity of 10,959 t/d, the preliminary analysis of the project indicates that it is subject to both provincial and federal environmental impact assessment (EIA) processes.

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.

Based on characterization results for the waste rock and ore samples, it is anticipated that ore and mine wastes will be non-PAG but are metal leaching. Therefore, management facilities will have to be designed to respect a daily percolation rate < 3.3 L/m^2 at the bottom of the storage area and runoff water will have to be collected and treated before being released to the environment.

No information and consultation activities were initiated with First Nations, local communities, or stakeholders.

Figure 1-3: Overall Site Plan

Source: Ausenco, 2020

Total guarantee for the Marban Project evaluated by Golder is at approximately $12.2 M, including the direct and indirect costs, and a 20% contingency. Under the *Mining Act*, O3 Mining 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 the mining operations to begin.

1.19 Capital Costs

1.19.1 Summary

The capital cost estimate was developed in Q2CY2020 from Ausenco's in-house database of projects and studies and experience from similar operations to a level of accuracy of ±50% in accordance with the Association for the Advancement of Cost Engineering International (AACE International). All costs are expressed in Canadian Dollars (CAD), unless otherwise noted. The capital cost estimate is presented in Table 1-4 with an exchange rate assumption of US$0.74:C$1.00.

Table 1-4: Capital Cost Summary - $CM

WBS	Cost Centre	Initial Capital	Sustaining	Total Capital
100	Mining	48.8	153.9	202.7
200	Crushing	19.2		19.2
300	Processing	75.5		75.5
400	Utilities	10.5		10.5
500	Tailings Storage Facility	17.9	35.0	52.8
600	Infrastructure	14.1		14.1
700	Offsite Infrastructure	4.3		4.3
	Total Directs	**190.3**	**188.9**	**379.2**
800	Indirects	24.5	10.2	34.7
900	Owner's Cost	10.6		10.6
	Contingency	30.6		30.6
	Total Project Cost	**256.0**	**199.1**	**455.1**

1.19.2 Capital Cost Estimate (Mine)

Mine capital costs have been derived from historic data collected by MMTS at other Canadian open pit mining operations, combined with the Marban PEA mine production schedule.

Mine capital costs include consideration for the mobile equipment fleet, fixed mine operations infrastructure, as well as open pit development and operating costs incurred prior to mill start-up.

The mine equipment fleet is planned to be purchased via a lease arrangement with the vendors, therefore most of the fleet payments are deferred beyond the initial project capital period.

1.19.3 Capital Cost Estimate (Process and Infrastructure)

Process equipment costs were derived using recent similar projects, recent and historical budget quotes on file from vendors. Delivery and installation of process equipment is a factored cost relative to the total purchase price of equipment. A detailed estimate of man-hours was not completed for this study. The factors developed were based on installation by local contractors.

Bulk earthworks for the plant site, mine ancillary buildings, tailings storage facility and water management infrastructure were developed based on semi-detailed cut and fill volumes based on site layout and site topographical information. Unit rates were benchmarked based on recent projects within the region.

1.20 Operating Costs

1.20.1 Summary

The operating cost estimate was developed in Q2CY2020 from Ausenco's in-house database of projects and studies and experience from similar operations to a level of accuracy of ±50% and an exchange rate assumption of US$0.74:C$1.00. The estimate includes mining, processing, general and administration (G&A), mobile equipment and TSF.

The overall life-of-mine operating cost is $1,702M over 15.2 years, or $28.20/t of ore milled.

1.20.2 Operating Cost Estimate (Mine)

Mine operating costs are built up from first principles and applied to the Marban PEA mine production schedule. Productivity and cost inputs are derived from historical data collected by MMTS. Estimated life-of-mine unit mining costs are shown in Table 1-5.

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 PEA.

Table 1-5: Marban Mine Operating Cost Summary

Item	$/t Mined	$/t Milled	$M
Drilling	$0.29	$1.95	$117.8
Blasting	$0.59	$3.97	$239.6
Loading	$0.29	$1.92	$116.1
Hauling	$0.98	$6.56	$395.9
Support	$0.32	$2.11	$127.6
Site	$0.03	$0.18	$10.6
Unallocated Labour	$0.05	$0.34	$20.2
DIRECT COSTS - Subtotals	$2.55	$17.03	$1,027.8
General Mining Expenses (GME)	$0.14	$0.91	$55.0
TOTAL MINE OPERATING COST	**$2.69**	**$17.94**	**$1,082.9**

1.20.3 Operating Costs – Process & Infrastructure & G&A

The operating cost estimates are presented in CAD and summarised in Table 1-6. These are derived from benchmarking, against existing gold processing plants located in eastern Canada as well as in-house data and quotations.

The tailings management facility was jointly developed by Golder and Ausenco correspond to the 4.0 Mt/a plant throughput. The processing and infrastructure operating costs total $576.6M and the G&A and mobile equipment costs total $42.8M over the LOM.

Table 1-6: Marban Mill and G&A Operating Cost Summary

Cost Centre	C$M per Year	C$/t
Processing	36.57	9.14
Labour	7.16	1.79
Power	4.87	1.22
Reagents	14.61	3.65
Consumables	8.08	2.02
Maintenance	1.53	0.38
Lab Services	0.31	0.08
Effluent Treatment	1.09	0.27
General & Administration	2.35	0.59
Mobile Equipment	0.36	0.09
Total OPEX Costs	**40.38**	**10.09**

Common to all operating cost estimates are the following assumptions:

- For material sourced or benchmarked in US dollars, an exchange rate of 0.74 US dollar per Canadian dollar was assumed.
- Diesel costs used are C$1.00/L and gasoline costs are C$1.045/L.
- The annual power costs were calculated using a unit price of C$0.053/kWh applying Rate L.
- Labour is assumed to come mostly from Abitibi region, and locally from places such as Rouyn-Noranda, Val-D'or, Kirkland Lake and Amos and salaries are based on the region and benchmarked against similar gold projects
- The processing unit operations were benchmarked against similar or comparable processing plants. For example, the 2C_SAB crushing and grinding circuit was compared to circuits of the same configuration.
- Equipment and materials will be purchased as new
- Process plant operating costs are calculated based on labour, power consumption, and process and maintenance consumables.

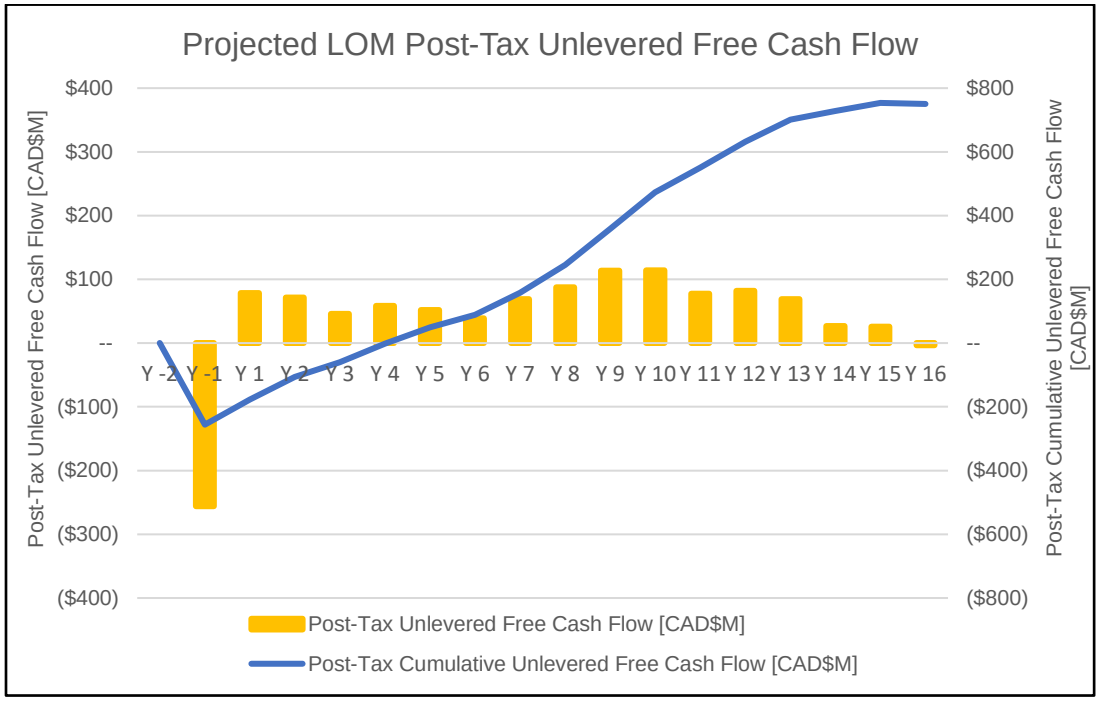
- General and administration (G&A) costs were baselined against previous project experience.
- Grinding media consumption rates have been estimated based on the ore characteristics. Note: Abrasion Index is currently unavailable so was assigned
- Reagent consumption rates have been estimated based on the metallurgical testwork and typical reagent consumptions for similar projects where no testwork data was available
- Mobile equipment cost provides for fuel and maintenance.

1.21 Economic Analysis

The economic analysis was performed assuming a 5% discount rate. The pre-tax net present value discounted at 5% (NPV5%) is C$715M, the internal rate of return IRR is 31.5%, and payback is 3.7 years. On an after-tax basis, the NPV5% is C$423M, the IRR is 25.2%, and the payback period is 4.0 years.

A summary of the Project economics is included in Table 1-7 and shown graphically in Figure 1-4.

Figure 1-4: Projected Post-Tax Cash Flow



Source: Ausenco, 2020

Table 1-7: Summary, Project LOM Cashflow Assumptions & Results

General	LOM Total / Avg.
Gold Price (US$/oz)	$1,450
Exchange Rate ($US:$CAD)	0.74
Mine Life (years)	15.2
Total Waste Tonnes Mined (kt)	355,627
Total Mill Feed Tonnes (kt)	60,356
Strip Ratio (waste: mineralization)	5.89
Production	**LOM Total / Avg.**
Mill Head Grade (g/t)	0.97
Mill Recovery Rate (%)	93.7%
Total Mill Ounces Recovered (koz)	1,757
Total Average Annual Production (koz)	115
Operating Costs	**LOM Total / Avg.**
Mining Cost (CAD$/t Mined)	$2.7
Mining Cost (CAD$/t Milled)	$17.9
Processing Cost (CAD$/t Milled)	$9.6
G&A Cost (CAD$/t Milled)	$0.7
Total Operating Costs (CAD$/t Milled)	$28.2
Refining Cost (CAD$/oz)	$2.5
Royalty NSR	1.5%
Cash Costs (US$/oz Au)	$741
AISC (US$/oz Au)	$822
Capital Costs	**LOM Total / Avg.**
Initial Capital (CAD$M)	$256
Sustaining Capital (CAD$M)	$189
Closure Costs (CAD$M)	$10
Salvage Costs (CAD$M)	$7
Financials - Pre Tax	**LOM Total / Avg.**
NPV (5%) (CAD$M)	$715
IRR (%)	31.5%
Payback (years)	3.7
Financials - Post Tax	**LOM Total / Avg.**
NPV (5%) (CAD$M)	$423
IRR (%)	25.2%
Payback (years)	4.0

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.

1.22 Sensitivity Analysis

A sensitivity analysis was conducted on the base case pre-tax and after-tax NPV, IRR and Payback of the Project, using the following variables: metal price, foreign exchange rate, discount rate, grade, recovery, total capital costs, and operating costs. The table below summarizes the sensitivity analysis results.

Analysis revealed that the Project is most sensitive to changes in metal prices, foreign exchange rate, head grade and recovery, then, to a lesser extent, to operating costs and capital costs.

Table 1-8: Gold Price Sensitivity Summary

	Gold Price US$/oz	$1,200	$1,350	$1,450	$1,750	$1,950
Pre-Tax	**NPV (5%)**	C$318 M	C$556 M	C$715 M	C$1,191 M	C$1,508 M
	IRR	16.9%	25.6%	31.5%	49.9%	62.6%
	Payback	7.3 yrs	4.8 yrs	3.7 yrs	2.0 yrs	1.6 yrs
Post-Tax	**NPV (5%)**	C$174 M	C$327 M	C$423 M	C$707 M	C$892 M
	IRR	13.1%	20.5%	25.2%	38.5%	47.2%
	Payback	7.5 yrs	5.1 yrs	4.0 yrs	2.5 yrs	1.9 yrs

1.23 Risks and Opportunities

1.23.1 Risks

1.23.1.1 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
- Historic underground openings location

1.23.1.2 Mining

Risks to the PEA defined are mill feed quantities gold grades and associated open pit strip ratio include changes to the following factors and assumptions:

- Metal Prices
- Interpretations of Mineralization geometry and continuity in mineralized zones
- Exact dimensions of underground voids created by historic mining
- Geotechnical and hydrogeological assumptions

- Ability of the mining operation to meet the annual production rate and anticipated grade control standards
- Operating cost assumptions
- Mine operation and process plant recoveries
- Ability to meet and maintain permitting and environmental licence conditions, and the ability to maintain the social license to operate. Specific concern exists regarding the timing to permit the diversion of existing water flow above the KN deposits.

1.23.1.3 Processing

Risks to the PEA defined include changes to the following factors and assumptions:

- The blend and/or grade of ore processed from the Marban pits as a function of the mine planning
- The future metallurgical testwork yielding results that vary from the historical testwork
- Fixed inputs into operating costs such as power, reagents, steel or labour varying from current industry typical costs

1.23.1.4 Environment and Permitting

Table 1-9: Environmental Risk Assessment

Risk Description and Potential Impact		Mitigation Approach
1.	Project delayed due to changes in regulations/government representatives as a result of elections: Project delayed.	Stay alert to the changes in regulatory regimes.
2.	Inadequate design prompts request for major modifications from one or both levels of government: Project delayed, increased design costs.	Include technologies and practices widely accepted in the mining industry. Share project information with governments early during the project development. Include technologies and practices well accepted in the mining industry.
3.	Project cannot proceed if not accepted by local communities due to lack of communication: Project delayed.	Keep regular contact with stakeholders and develop and implement a consultation plan.
4.	Authorizations and permits from both levels of government could take longer than expected: Project delayed.	Develop and maintain a realistic schedule, keep close contact with the authorities throughout the process.
5.	Encroachment in fish habitat by mine waste disposal: Alternative assessment (Schedule 2) – Federal requirement leading to major delay (1-year minimum).	Proceed with fish habitat validation in the TSF and WMP sector. Confirm with federal authorities that using the pits for waste rock disposal after the Kierens creek diversion will not trigger Schedule 2.

6.	Only static tests were conducted on waste rock and ore samples, which may underestimate/overestimate the metal leaching and acid generation potential: Impact on management and closure approaches.	Proceed with kinetic tests. Reassess geochemical behaviour at every stage of the Project.
7.	At this stage, a limited number of ore samples (4) have been analyzed for geochemical characterization, and these are not sufficient to be representative of the overall mineralized zone: Impact on management.	Continue the geochemical characterization on additional ore samples to ensure representativeness.
8.	No tailings geochemical properties available: Impact on management.	Carry out a geochemical characterization on representative tailings samples.
9.	Weathering and metal leaching rates may be underestimated during geochemical characterization; metals may also behave in unexpected ways: Impact on planned water treatment approaches.	Reassess geochemical behaviour at every stage of the Project.
10.	Delay of mining lease issue due to delays in getting the closure plan approval: Project delayed.	Develop a detailed closure plan during feasibility stage with proper mine layout and sufficient geochemistry data on ore, waste rock and tailings.
11.	Some soils within the proposed layout might be contaminated. Increased costs.	Evaluate the need for additional soil characterization on potentially contaminated lands to evaluate the environmental liability.

1.23.2 Opportunities

1.23.2.1 Exploration

Exploration activities are planned in two stages of drilling in the next year. The first phase targets areas of high-grade mineralization with a $5.8M budget. The second phase includes infill drilling to upgrade Inferred resources. The initial brownfields exploration program is expected to result in identifying additional mineralized zones, information on the style of mineralization to that currently identified, on the scale of the Project, and on the potentially deleterious elemental issues identified.

1.23.2.2 Resource Modelling

Re-assaying of pre-NioGold era drilling will significantly increase the usable assay database and therefore will help in upgrading the Classification from Inferred to Indicated, particularly at Kierens-Norlartic. There is additional upside to the Project from the planned drill program both on extending the mineralization in areas still open with a brownfields drill program and in further upgrading the Classification to Indicated through an infill drill program. Additional geologic modelling as was done at Marban to better delineate the geologic controls is expected to increase the extent and confidence of the mineralized zones.

1.23.2.3 Mining

With detailed metallurgical testwork information on lithologies and zones, the mining sequence may be altered to provide higher value initially. There is potential for improved slope design when additional geotechnical data such as waste rock strength are available from drill testing.

1.23.2.4 Environment and Permitting

Table 1-10: Environmental Opportunity Assessment

	Opportunity Explanation	Benefit
1.	Initiating geochemical tests on tailings and pursuing kinetic tests on waste rock and ore would provide a better understanding of metal leaching and acid generation potential.	Understanding of mine waste behaviour under conditions that are relevant to natural environments would help define appropriate management and restoration options and lower the closure costs.
2.	Development of a water quality model will improve the understanding of constituents that need to be treated, as well as their expected concentrations.	The model outcome would support adequate water treatment requirements.
3.	Making more land available, may prevent encroaching fish habitat and avoid or minimize impact on wetlands.	Reduction in compensation costs and permitting effort.
4.	Multiplying engagement activities and consultation effort to keep the stakeholders, First Nations and local community involved in the Project development.	Facilitate social acceptability.

1.24 Conclusions

The total Measured and Indicated Mineral Resource is estimated at 54.1 Mt at a grade of 1.10 g/t Au for a total of 1.9 Moz. Additional Inferred Mineral Resources represent 24.2% of the total ounces when combined with the Measured and Indicated Mineral Resource.

Based on the assumptions and parameters presented in this Report, the PEA shows positive economics (i.e. C$423 M post-tax NPV(5%) and 25.2% post-tax IRR). The PEA supports that additional detailed studies are warranted.

1.25 Recommendations

1.25.1 Introduction

The recommended work is divided into two main initiatives. The first initiative will consist of re-assaying pulps, drilling; additional metallurgical testwork; materials handling tests; mine geotechnical data collection, data reviews in support of geotechnical and hydrological assumptions; additional hydrological data gathering; water treatment testwork; review of cost assumptions for grade control; additional mine studies, reviews of available climate data; collection of additional climate-related information, and geotechnical data collection in support of infrastructure locations and designs, and data collection on potential borrow pit sources.

The second initiative will consist of project environmental, permitting, and social de-risking activities.

1.25.2 Drilling

Future drilling at Marban should focus on converting the inferred resource to indicated category and on expanding the resource base. The extensions of the Marban, Norlartic, Kierens, North-North and Gold Hawk deposits as well as Orion 8, MK, Malartic Hygrade, Malartic H and Malartic NE zones provide substantial opportunities to expand the resource of the Marban Project.

1.25.3 Sampling and QAQC

MMTS considers the available data to be of acceptable quality at this level of resource reporting. Prior to future reporting of higher significance, it is recommended that:

- Commercially available and certified blank material be sourced and used for QAQC in future drilling.
- Re-assays of the 10 samples before and after the failed blanks be run with newly acquired blank material inserted in the stream.
- All future drilling and QAQC to include CRMs of lower grades, such as 0.15 g/t, 0.25 g/t and 0.35 g/t Au.
- All future QAQC employ the definition of a failed CRM at the +/- 3 SD level, or two consecutive samples in the warning zone of +/-2 SD.
- Re-assays are accomplished on failed samples at the +/- 3 SD level, to include the 10 samples before and after the fail, with particular attention to Marban assays in 2006 and 2007. Insertion of newly acquired CRM material is advisable.
- Duplicate core samples in Kierens-Norlartic be resampled to 5% so that the possibility that the original core assays were selectively sampled can be excluded.
- The Kierens-Norlartic QAQC database be augmented with external laboratory duplicates from drilling after 2007.

1.25.4 Mining

The following recommendations are made with regards to advancing the mine engineering of the Marban project to a Pre-Feasibility Study:

- Targeted open pit geotechnical drilling and detailed geotechnical logging of some exploration drillholes, review of the geological models and review of historical geotechnical and underground geological mapping. Holes should be drilled both parallel and perpendicular to foliation to eliminate blind zones in the structural data. The following is recommended:
    - Marban deposit: 6 drillholes, approximately 300 m – 400 m length.

- Norlartic deposit: 4 drillholes, approximately 150 m – 200 m length.
- Kierens deposit: 4 drillholes, approximately 100 m length.

- Gather structural fabric data (foliation and major joint sets orientation) to assess variability. Using the structural fabric data, limit equilibrium analyses of potential planar, wedge and toppling failures should be conducted to determine bench, inter-ramp and overall slope design.
- Lithological and/or alteration interpretation to identify schistosic zones (geotechnically poorer ground). Develop a 3D geotechnical interpretation of the Talc-Chlorite-Schist (TCS) bands and develop a more precise pit slope domaining relative to the TCS unit.
- Laboratory testing for intact rock strength (unconfined compressive strength tests and indirect tensile strength tests) and for discontinuity strength (direct shear tests). It is recommended that the ISRM intact rock strength be logged in exploration holes.
- Crown pillar analysis for open pit mining over historic underground openings. Specific stability assessments should be done where historical openings are planned to be located behind interim or final pit walls or below pit floor.
- Overburden investigation for pit slope assessment.
- Packer testing should be conducted to determine pit hydrogeology, hydraulic conductivity and refine pit water inflow estimates.
- Further hydrogeological and hydrological characterization are required in the Kierens-Norlartic area where the pits will be mined beneath the Kierens stream.
- Condemnation drilling of the footprints identified for the waste rock and overburden storage facilities, as well as site infrastructure.
- Updates to designs of open pits, stockpiles, and mine haul roads incorporating results from all other recommended work programs.
- Mine operational trade-off studies examining contractor vs. owner equipment fleet, lease vs. purchase equipment fleet, cost comparisons of various equipment class sizes, and utilization of electrical driven mine equipment over diesel driven units.

A budget of $1.5M is estimated for the above work programs and studies.

1.25.5 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. crushing work index, rod work index, SMC 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.

An extended gravity-recoverable gold test should be conducted on a master composite sample to confirm the PEA flowsheet.

Tailings cyanide detoxification testwork is recommended. Tailings dewatering and concentrate dewatering testwork is also recommended. Rheology of the tailings slurry stream should be conducted on the combined composite sample, without any prior treatment that would affect the rheology. Testwork should be designed to include all representative composites.

1.25.6 Material Handling

Material handling test work is recommended for design of bins, chutes, conveyors and stockpile drawdown.

1.25.7 Hydrological

Hydrogeological testing (packer testing, profile tracer testing) and instrumentation (i.e. piezometers) should be installed in select holes to provide basic data for groundwater modelling and excavation dewatering/depressurization simulations.

1.25.8 Water Treatment

After the site-wide water balanced has been further evaluated and ML/ARD parameters are better understood, water treatment testwork should be conducted to confirm that impurities can be removed from water prior to discharge to the environment.

1.25.9 Infrastructure Geotechnical, Construction Borrow Materials, and Hydrological

Field mapping, geotechnical sample collection (boreholes, tests pits) and laboratory studies should be conducted to identify borrow material sources for construction activities, and provide information for support of the WRS, OBS, plant site, ancillary facilities locations and the TSF design.

1.25.10 Tailings Storage Facility

The following activities are recommended to support the design of the TSF for the PFS:

- Fish sampling in the proposed tailings and water management areas should be conducted to confirm fish presence/absence in the waterbodies of interest that may be impacted by the proposed development.
- Condemnation drilling for the selected tailings management site to verify the absence of mineralization.
- Geotechnical site investigations at the preferred site to characterize the foundation conditions and potential borrow sources.
- In-situ permeability tests of the overburden soils and bedrock.

- Tailings assessment to help characterize the geotechnical properties in order to understand the deposition characteristics.
- Refinement and further study of the proposed TSF embankment and water management pond dam alignment and configuration.
- Further evaluation of the water management pond dam requirements such as allowable seepage gradients and management of seepage water.
- Formal dam breach analysis along with dam hazard classification in accordance with Canada Dam association (CDA, 2013) Dam Safety Guidelines

1.25.11 Environment and Permitting

In the next stages of the project, it is recommended to continue the environmental baseline studies and environmental assessment as required to better define the environmental and social impacts. Mainly, the following activities should be undertaken:

- Undertake environmental baseline studies to cover the TSF and WMP sector
- Complete environmental baseline studies of the main sector
- Initiate public and First Nations engagement activities to ensure that their expectations and concerns are addressed throughout the Project development
- Initiate government involvement to ensure that their expectations and concerns are addressed throughout the Project development
- Initiate Kierens Creek diversion conceptual engineering
- Initiate provincial and federal environmental assessment processes
- Proceed with kinetic testing on sufficient samples of waste rock and ore
- Initiate an exhaustive geochemical characterization on tailings as soon as samples are made available

2 Introduction

2.1 Introduction

This report was prepared by Ausenco Engineering Canada Inc. (Ausenco) for O3 Mining Inc. (O3 Mining) to summarise the results of a preliminary economic assessment (PEA) of the Marban Project. The report was prepared in compliance with the Canadian disclosure requirements of National Instrument 43-101 (NI 43-101) and in accordance with the requirements of Form 43-101 F1.

The PEA was prepared in accordance with NI 43-101 Standards of Disclosure for Mineral Projects. Readers are cautioned that the PEA report is preliminary in nature.

Golder Associates Ltd. (Golder), Moose Mountain Technical Services (MMTS), WSP Canada Inc. (WSP), and Kenneth Williamson (Williamson) provided input to the report. The individuals presented in Section 2.3, by virtue of their education, experience, and professional association, are considered Qualified Persons (QPs) as defined by NI 43-101 (CIM, 2014). The QPs meet the requirement of independence defined in NI 43-101.

2.2 Terms of Reference

The Report supports disclosures by O3 Mining in a news release dated 8 September, 2020 entitled “O3 Mining Delivers Positive PEA For Marban Project”.

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

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 Project area contains three past-producing mines (Marban, Norlartic and Kierens), which collectively produced 585,000 ounces of gold between 1959 and 1992. O3 Mining owns 100% of the Malartic property, which hosts the Marban Project.

2.3 Qualified Persons

This report was prepared by the following qualified persons

- Mr. Tommaso Roberto Raponi, P. Eng., Principal Metallurgist, Ausenco
- Mr. Scott Elfen, P.E., Global Lead Geotechnical Service, Ausenco;
- Mr. Mike Petrina, P.Eng, Project Manager, Mining Engineering, MMTS
- Ms. Sue Bird, P.Eng, Principal – Resource Modelling, MMTS
- Mr. René Fontaine, P. Eng., Director, Environment, Geotechnical and Materials testing, WSP

2.4 Site Visits & Scope of Professional Inspection

Mr. Petrina visited the Marban property on 12 August 2020. During that visit he viewed the general topography, independently located and surveyed surface drill hole collars and the existing infrastructure.

Ms. Bird visited the Marban property on 12 August 2020. The core storage area, site and infrastructure were viewed. The geology and mineralization were discussed with site geologists providing representative core of mineralized zones.

2.5 Effective Dates

The Report has of the following effective dates:

- Mineral Resource estimate: 12 August, 2020;
- Date of PEA financial analysis: 03 September, 2020.

The overall effective date of this Report is the effective date of the financial analysis which is 03 September, 2020.

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.

2.7 Previous Technical Reports

The following technical report has previously been filed on the Marban property:

Belzile, Elzéar, 2016: Updated Mineral Resource Technical Report on the Marban Block Property, Québec, 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, Québec, Canada: report prepared by Mine Development Associates for NioGold Mining Corporation.

3 Reliance on Other Experts

The authors of this report have assumed and relied on the fact that all the information and technical documents listed in Chapter 27, References, are accurate and complete in all material aspects. 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 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.

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 from Deloitte LLP. The calculations are based on the tax regime as of the date of the PEA and include estimates for O3's expenditures, and related impacts to various tax pool balances, between the PEA and the assumed construction start date.

4 Property Description & Location

The author is not an expert in land, legal, environmental, and permitting matters. The information presented in this and Section 5 is based entirely on information provided to Ausenco by O3 Mining, including title opinions prepared by Valiquette (2006), Lécuyer (2006), Morin (2007), and Miller Thomson Pouliot LLP (2011) with respect to O3 Mining's interests in the Marban Project within the Malartic property. Ausenco presents this information to fulfill reporting requirements of NI 43-101, but expresses no opinion regarding the legal or environmental status of the Marban Project or any of the agreements and encumbrances related to the property.

4.1 Description & Location

4.1.1 Location

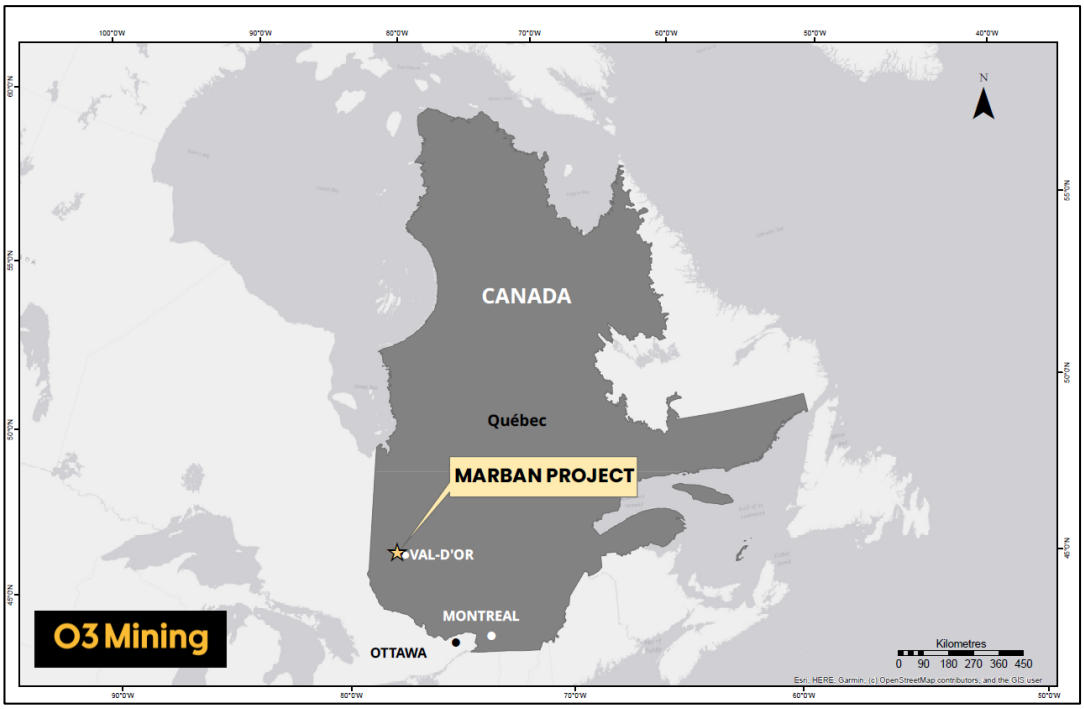
The Marban Project is located in the western portion of the province of Québec, Canada, about midway between the towns of Val-d'Or and Malartic (Figure 4-1) and is comprised of 30 mining claims and 3 mining concessions covering 1,023 hectares. The project lies at the junction of Dubuisson, Fournière, Malartic, and Vassan townships, at and south of Latitude 48.15497°N, Longitude 78.00544°W, on NTS sheets 32D/01 and 32C/04.

4.1.2 Land Area

The Marban Project 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 Malartic property, which hosts the Marban Project (Figure 4-2). The remainder of the Malartic property is not the subject of this report.

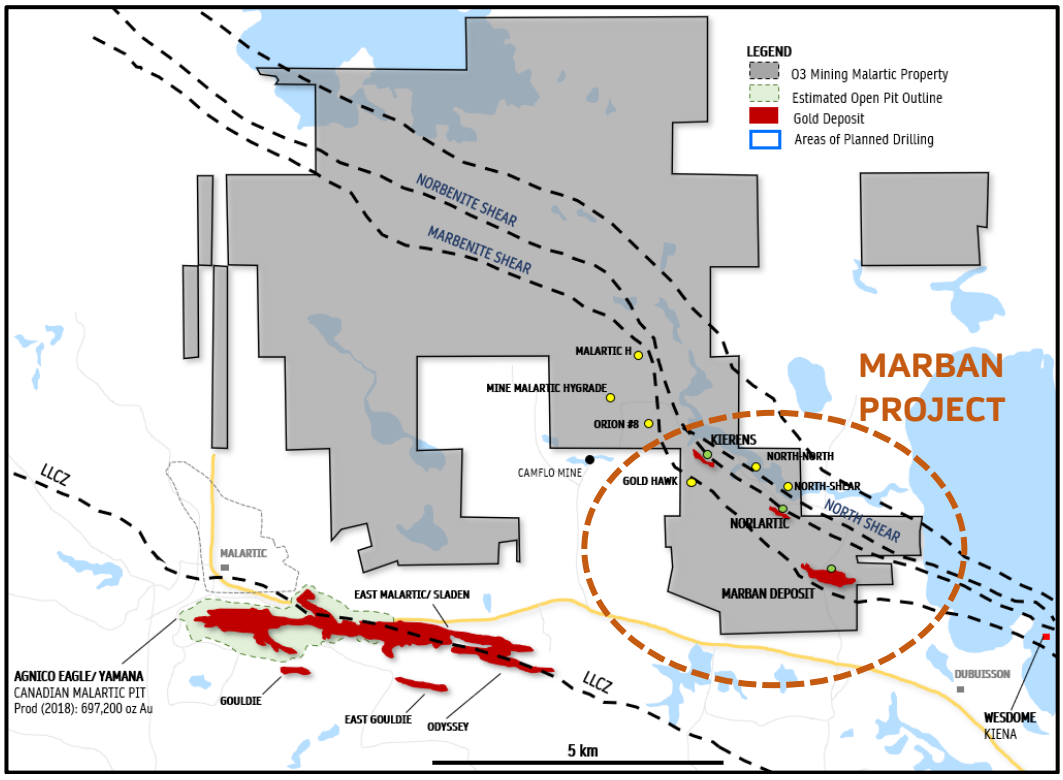
On March 16, 2015, the claims of the Marban Project were converted in new CDC claims. Thus the property surface was slightly increased so now, it covers 1023.13 hectares and comprises 30 mining claims and 3 mining concessions (Figure 4-3 and Table 4-1). Consequently, the Gold Hawk, First Canadian, Norlartic, and Marban original contours do not fit the actual claims boundaries. The Marban historical property, which includes the gold resources of the Marban deposit, covers 718.16 hectares and includes three mining concessions; it was formerly jointly owned by Aur Resources Inc. ("Aur") 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 cover 196.77 hectares. The Gold Hawk historical property covers 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 Project



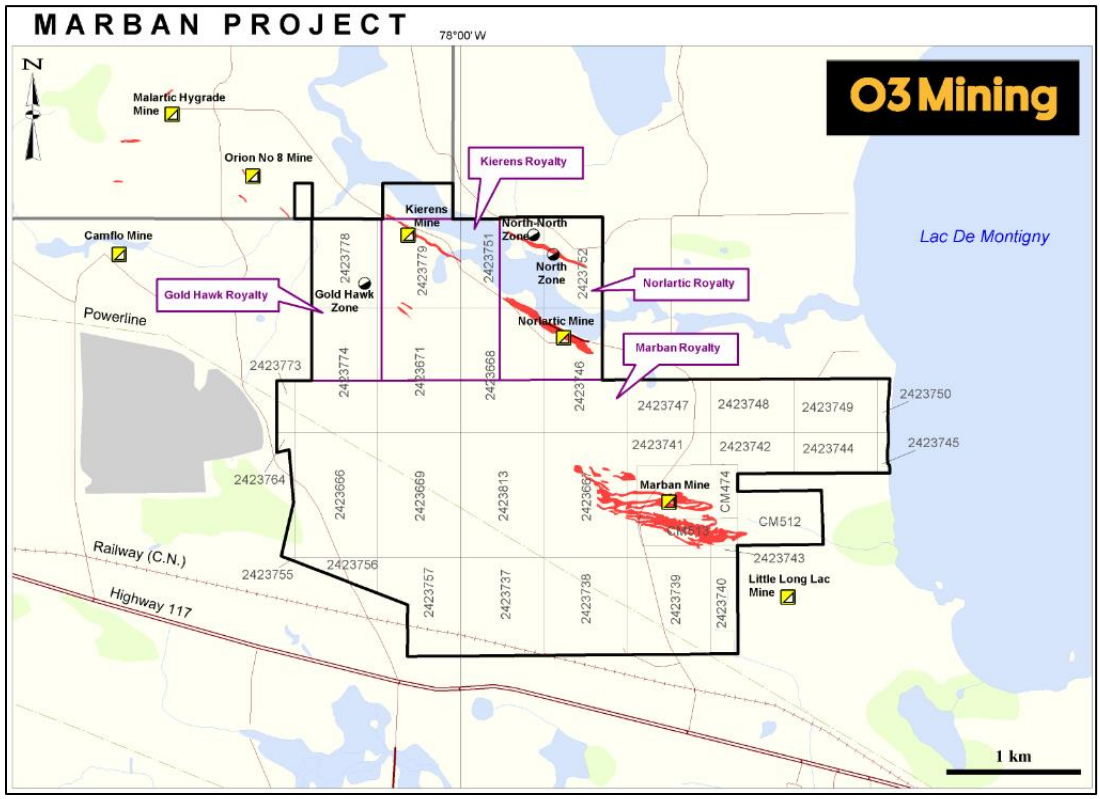
Source: O3 Mining, 2020

Figure 4-2: O3 Mining Malartic Property



Source: O3 Mining, 2020

Figure 4-3: Marban Project Mining Claims and Concessions



Source: O3 Mining, 2020

Table 4-1: Marban Mining Concessions & Claims

Title No.	Type	Area hectares	Expiry Date	Range	Lot	Township
CDC 2423666	Mining Claim	57.49	2022-02-12	17	59	Fournière
CDC 2423667	Mining Claim	57.49	2022-02-12	17	2	Dubuisson
CDC 2423668	Mining Claim	57.49	2022-02-12	18	1	Dubuisson, Vassan, Malartic
CDC 2423669	Mining Claim	57.49	2022-02-12	17	60	Dubuisson, Fournière
CDC 2423671	Mining Claim	57.49	2022-02-12	18	60	Dubuisson, Fournière, Malartic
CDC 2423737	Mining Claim	45.46	2022-02-12	16	1	Dubuisson
CDC 2423738	Mining Claim	45.11	2022-02-12	16	2	Dubuisson
CDC 2423739	Mining Claim	44.74	2022-02-12	16	3	Dubuisson
CDC 2423740	Mining Claim	14.44	2022-02-12	16	4	Dubuisson
CDC 2423741	Mining Claim	24.90	2022-02-12	17	3	Dubuisson
CDC 2423742	Mining Claim	17.56	2022-02-12	17	4	Dubuisson
CDC 2423743	Mining Claim	1.76	2022-02-12	17	4	Dubuisson
CDC 2423744	Mining Claim	18.77	2022-02-12	17	5	Dubuisson
CDC 2423745	Mining Claim	2.48	2022-02-12	17	6	Dubuisson
CDC 2423746	Mining Claim	47.45	2022-02-12	18	2	Dubuisson, Vassan
CDC 2423747	Mining Claim	24.41	2022-02-12	18	3	Dubuisson
CDC 2423748	Mining Claim	24.59	2022-02-12	18	4	Dubuisson
CDC 2423749	Mining Claim	24.76	2022-02-12	18	5	Dubuisson
CDC 2423750	Mining Claim	3.38	2022-02-12	18	6	Dubuisson
CDC 2423751	Mining Claim	41.38	2022-02-12	19	1	Malartic, Vassan
CDC 2423752	Mining Claim	29.27	2022-02-12	19	2	Vassan
CDC 2423755	Mining Claim	0.11	2022-02-12	16	58	Fournière
CDC 2423756	Mining Claim	9.21	2022-02-12	16	59	Fournière
CDC 2423757	Mining Claim	36.17	2022-02-12	16	60	Dubuisson, Fournière
CDC 2423764	Mining Claim	4.84	2022-02-12	17	58	Fournière
CDC 2423773	Mining Claim	5.09	2022-02-12	18	58	Fournière
CDC 2423774	Mining Claim	50.53	2022-02-12	18	59	Fournière, Malartic
CDC 2423778	Mining Claim	36.05	2022-02-12	19	59	Malartic
CDC 2423779	Mining Claim	54.89	2022-02-12	19	60	Malartic
CDC 2423813	Mining Claim	57.49	2022-02-12	17	1	Dubuisson
CM 474	Concession	31.55		10	18	Dubuisson
CM 512	Concession	24.52		10	18	Dubuisson
CM 513	Concession	14.77		10	19	Dubuisson

Title No.	Type	Area hectares	Expiry Date	Range	Lot	Township
	Total	**1,023.13**				

In Québec, a claim is a mineral right that gives its holder a two-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 Malartic Project area, including the three Marban mining concessions and the 30 mining claims. The mining concessions on the Marban claim group are renewed each year by submittal of the required work credits accrued during the previous on or before January 31, while the Marban mining claims are renewed every two years and also require the completion of assessment work. The mining claims and concessions that comprise the Marban Project property are listed as being in good standing on the MERN 'Gestim' claim management website.

4.2 Project Ownership Agreements

4.2.1 Gold Hawk, Kierens, Norlartic, & Marban Claims & Concessions

The status of the Gold Hawk, Kierens, Norlartic, and Marban historical properties has changed considerably since the 2007 technical report (Gustin, 2007); readers interested in a more detailed discussion of the history of the properties are referred to that report. The following information has been provided by Osisko.

NioGold purchased the McWatters' 50% interest in the Marban historical property in 2004. 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.

Mining Lease BM 781, which covered a portion of the First Canadian and Norlartic historical properties, expired in April 2009 and could not be renewed, as there had been no commercial production on the lease in the last two years. As owner of the mining lease, NioGold had the first right to stake mining claims on the ground, which it did, thereby adding five mining claims to their holdings at the time.

O3 Mining now holds 100% of the rights, title, and interest in the Gold Hawk, First Canadian, Norlartic, and Marban historical properties. On March 11th, 2016, Oban Mining Corporation acquired all shares of NioGold in a friendly takeover. On June 14th, 2016, Oban Mining Corporation changed officially its name for Osisko Mining Inc.

On July 5th, 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"). Under the Arrangement, Osisko Mining transferred to O3 Mining the Marban project, the Garrison project, 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 Option & Joint Venture Agreement with Aurizon Mines Ltd.

NioGold signed an option and joint venture agreement for the Marban Project with Aurizon on July 5th, 2010. Under the terms of the agreement, Aurizon spent $11.6M in exploration expenditures and technical studies. On June 3rd, 2013, Hecla Mining Company completed acquisition of Aurizon and walked out of the agreement on August 20th, 2013. Aurizon (Hecla) did not earn any interest in the Marban Project.

4.3 Surface Rights

According to O3 Mining, Aur sold the Norlartic mine infrastructure and the “dry land” portion of the surface rights of the First Canadian and Norlartic claim groups (which were then part of mining lease BM 781) to Installations Minière Promocan, a mining contractor (“Promocan”). Camroc Inc. purchased the surface rights to Lots 62 and 63, Range 1, Malartic Township, which cover 2/3 of the First Canadian claim group (including the Kierens mine site), in 1997 for the purpose of storing mining equipment, but never actually used the site. Camroc Inc. sold their rights to these two lots to NioGold. Promocan later went bankrupt, and the surface on the remaining four lots, in addition to the Norlartic mine installations, were sold to Legault Métal Inc., who in turn sold these interests to mining contractor A.B.F. Mines. According to Osisko, A.B.F. Mines has granted Osisko access to all parts of the properties for exploration. The title opinions of Valiquette (2006), Lécuyer (2006), and Miller Thomson Pouliot LLP (2011) do not mention these transfers of ownership of the surface rights of the Marban Project.

4.4 Mining Concessions

O3 Mining holds three mining concessions within the Marban Project, all of which are located on the Marban historical property (Table 4-1). Lécuyer (2006) stated that NioGold “…possesses good and valid title on mining concessions number 474, 512, and 513”, but pointed out errors in the documents for mining concessions 474 and 512 in the Register. Morin (2007) stated that there is no error in concession 474, and recommended that NioGold record a deed of correction in order to amend the description of concession 512; MDA reviewed documentation of a correction filed by NioGold with the MRNQ on June 15, 2007.

The three mining concessions are subject to the royalties shown in Table 4-2 for the Marban historical property.

4.5 Royalties

Table 4-2 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 CAD$5,000,000 to Aur (now Teck Resources Ltd). On December 18th, Osisko Gold Royalties (Osisko GR) bought the Teck NSR and payment rights.

Table 4-2: Royalties Payable on the Marban Property

Historical Property	Royalty	Mineral Resources Subject to Royalty
Gold Hawk	**2% NSR** payable to Thundermin Resources Inc. pursuant to an agreement dated February 16, 2006 between Thundermin Resources Inc. and NioGold Mining Corporation; buy-back clause of one-half of the NSR for $1.0M at any time.	
Norlartic	**1.7% NSR** payable to Teck Resources Ltd. (to Osisko GR now) pursuant to an agreement dated February 3, 2006 between Aur Resources Inc. and NioGold Mining Corporation with buy-back clause for one-half of the NSR owned by Osisko GR in consideration of $1.0M at any time.	Norlartic
	0.3% NSR payable to Sodémex pursuant to an agreement dated May 20, 2016 between Osisko GR and Sodémex.	Norlartic
	See Section 4.61 for information on possible additional royalties affecting claims 382411A, 382411B, 382416A, 382416B, 382428A, and 382428B.	
Kierens	**0.85% NSR** payable to Teck Resources Ltd. (to Osisko GR now) pursuant to an agreement dated February 3, 2006 between Aur and NioGold Mining Corporation with buy-back clause for one-half of the NSR owned by Osisko GR in consideration of $1.0M at any time..	Kierens
	0.15% NSR payable to Sodémex pursuant to an agreement dated May 20, 2016 between Osisko GR and Sodémex.	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 $0.5M (source Osisko Mining annual information form Dec 31, 2017).	Marban

	0.85% NSR on 50% of any production payable to Teck Resources Ltd (to Osisko GR now). pursuant to an agreement dated February 3, 2006 between Aur Resources Inc. and NioGold Mining Corporation, with buy-back clause for one-half of the NSR owned by Osisko GR in consideration of $1.0M at any time.	Marban
	0.15% NSR payable to Sodémex pursuant to an agreement dated May 20, 2016 between Osisko GR and Sodémex	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

4.5.1 Norlartic Historical Property

The 2011 title report by Miller Thomson Pouliot LLP noted the following information from the Public Register of Real and Immovable Mining Rights ("PRRIMR"):

"Letter agreement dated May 21, 1985 between Aur Resources Inc. ("Aur") and Cogesco Resources Inc. ("Cogesco"), registered in the PRRIMR on August 26, 1986 under the number 43182, providing that if the undivided interest of each party is diluted (10% for Cogesco and 20% for Aur), it will be converted to a net profit interests (10% for Cogesco and 20% for Aur) from any production from the M. 15 and M. 16 Claims (as identified in Schedule "D-1"). Cogesco also acknowledged that Misseguay Mines Inc. holds a 1.5% net smelter return royalty from any production from the M. 15 and M. 16 Claims."

However, Schedule E of the same title report, which is a "Client List of Recorded and Non-Recorded Royalties" on the Marban Project, does not include the above referenced royalties. The M. 15 and M. 16 claims are those numbered 382411A, 382411B, 382416A, 382416B, 382428A, and 382428B which are part of the Norlartic historical property (Figure 4-4).

Schedule D-1 of the title report (Miller Thomson Pouliot LLP, 2011) describes the chain of title of the Marban claims. The schedule notes that on July 26, 1988, Les Ressources Aur Inc. transferred all of its interest in the M. 15 and M.16 claims to Ressources Nova-Cogesco Inc. and on March 1, 1992, Ressources Nova-Cogesco Inc. transferred all of its interest in the same claims to Les Ressources Aur Inc. The author is not an expert in land and legal matters. Ausenco has no information about whether there was any dilution in the interests of Cogesco or Aur, as referenced above, and has no information about the validity of the above-referenced NSR to Misseguay Mines Inc.

Figure 4-4: Historical Mining Claims & Concessions of the Marban Property

Source: O3 Mining, 2020

4.6 Environmental Permits & Liabilities

O3 Mining holds a Forest Intervention Licence for mining activities delivered by the Minister of Forests, Wildlife and Parks (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. 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, 2018c). 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.

5 Accessibility, Climate, Local Resources, Infrastructure & Physiography

5.1 Accessibility

The Marban Project property is located on the north side of Highway 117, approximately 15 kilometres northwest of Val-d'Or, Québec and immediately east of Monarch Gold's Camflo mill. The Camflo road provides access to the western part of the project area. A well-maintained all-weather gravel road running north from Highway 117 provides access to the historic Norlartic, Kierens, and Marban mines. Winter access for snowmobiles and all-terrain vehicles is provided by trails and winter roads.

5.2 Physiography

The Marban Project 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. The property is relatively flat, with local relief of 15 to 30 metres. The surface is dominated by swamps and glacial deposits; a north-trending esker lies on the west side of the Norlartic Mine road. The overburden is generally between 10 and 40 metres 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 Project property, while the Kierens Creek trends northwest across the northern portion of the First Canadian and Norlartic claim groups.

5.3 Climate & Vegetation

The climate is continental and is characterized by relatively dry, cold winters and hot, damp summers. The average daily temperature is 12 ºC, with January temperature averaging -17 ºC and July averaging 17 ºC (MELCC, 2020). The average number of days per year at or below freezing is 209. The low temperature record was -43.9 ºC in 1962 and the high temperature record was 36.1 ºC set in 1975 (ECC Canada, 2019). The annual rain precipitation is approximately 900 mm and the annual snow precipitation is approximately 240 cm (MELCC, 2020). Snow falls from November to March and usually remains on the ground until mid-April. Mining activities can proceed year-round.

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 Project property is located midway between the towns of Malartic and Val-d'Or, Québec. The town of Val-d'Or (population 32,500) 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 Project property. The communities of Malartic and Val-d'Or, each within 15 kilometres of the property, provide full services for exploration and mining, including a labor force experienced in mining. The Camflo mill can be accessed by a gravel road that connects Highway 117 to the Kierens, Norlartic, and Marban mine sites.

5.4.2 Local Labour

According to the 2016 census prepared by Statistics Canada, the population of the Regional County Municipality (RMC) of La Vallée-de-l'Or was 43,226, including 28,580 residents aged 15-64, and a median age of 42.2 years old. Male population accounts for 51% of the population, 49% is female, and 8.5% is Aboriginal. In 2016, 64.4% of the population participated in the labour force, with 14.2% of the labour force employed in mining, quarrying, and oil and gas extraction. 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 is, however, located on land that is subject to a comprehensive land claims agreement or a self-government agreement.

No information and consultation activities were initiated with First Nations, local communities or stakeholders.

6 History

6.1 Introduction

This section describes exploration and production history of the Marban property prior to consolidation of the properties by NioGold (now O3 Mining) in 2006.

Except where otherwise noted, the information in this section is adapted from the original technical report (Carrier, 2006), which referenced a due-diligence review by NioGold (Lefrançois, 2005).

The Marban property is located in the Malartic gold district in the southern Abitibi greenstone belt, a district that has produced an estimated 9+ million ounces of gold. There are three past-producing mines within the Marban property – the Norlartic, Marban, and Kierens mines.

Exploration conducted at the Marban property apparently 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 the Marban property was during the 1960s, primarily from the Marban and Norlartic mines. The second phase of production was during the late 1980s and early 1990s from the Norlartic and Kierens mines. Production from the Marban operations during these two periods totaled almost 600,000 ounces of gold from approximately 3.7 million tonnes of ore. Table 6-1 lists estimated gold production from the Marban deposits from 1959 to 1992 based on data provided to Ausenco; the North Zone deposit was accessed through the Norlartic mine workings, and the Gold Hawk zone through the Kierens mine workings.

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

Table 6-1: Estimated Historical Gold Production from the Marban Property (1959-1992) (adapted from Ducharme et al., 2009, with changes by MDA)

Mine	Company	Years	Tonnes	g Au/t	Au Ounces
Marban	Marban Gold Mines	1961-74	1,983,000	5.27	330,000
Norlartic	Norlartic Mines	1959-66	1,435,000	4.07	188,000
	Aur Resources	1990-92			
Kierens	Norlartic Mines	1965-66	251,000	6.30	52,000
	Aur Resources	1988-92			
North Zone	Norlartic Mines	1965-66	81,000	4.35	11,000
Gold Hawk	Aur Resources	1989-91	8,300	13.95	3,700
Totals			**3,758,300**		**584,700**

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 6-2lists pre-NioGold exploration chronologically through 1994 and references activity at the four claim groups that comprise the Marban property, i.e., the Marban, Norlartic, First Canadian, and Gold Hawk groups. According to O3 Mining, 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 20th, 2013.

Table 6-2: History of Marban Property Exploration (Revised from Carrier, 2006)

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 ninety-six 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 drill holes 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 metres.	Stuart and Martin (1986)
1943-1946	Norbenite Malartic Mines – Norlartic	Sixty-six drill holes totaling 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 metres.	Stuart and Martin (1986)
1946-1948	Norbenite Malartic Mines – Norlartic	Three-compartment shaft sunk at the Norlartic mine to 165 metres. A total of 1508 metres of drifting on 38, 76, 114, and 152-metre levels, cross-cuts totaling 542 metres, and raises totaling 160 metres. Underground drilling totaling 4,208 metres.	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-tonne 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 metre and -152-metre levels. By the time all activities on the Norlartic property closed down during the year, 572 metres of drifting, 225 metres of cross-cutting, and 3,943 metres 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)

Year	Company - Property	Work Description & Results	Reference
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 metres, drifting on -61, -107, -152, -198, and -244-metre 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 metres. Development of the 190, 228, 266, and 305-metre levels. Production started in mid-December 1959 for a 7-year period. Total mine production of 1,076,823 tonnes 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-metre level. Exploration was conducted below this level but was limited to nine holes directly below the workings to a depth of 195 metres. Only one of these holes hit ore-grade mineralization. An additional nine holes were drilled east of the workings above the 152-metres 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 totaled 5,055 metres in 18 holes. The zone was accessed from the Norlartic -230-metre level drift. Limited mining undertaken and 37 exploration holes, for a total of 2,280 metres, were drilled from underground.	Stuart and Martin (1986)
1963	Marban Gold Mines Ltd. – Marban and First Canadian	Magnetic survey, geological survey, and a few drill holes.	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-drill hole (427 metres).	Martin and Bubar (1988)

Year	Company - Property	Work Description & Results	Reference
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)
1973-1975	First Canadian Gold Corp. – Norlartic	Acquisition of Norlartic by First Canadian. Magnetometer survey and 33 surface holes totaling 4,166 metres at the North-North Zone.	Mannard (1986)
1975	First Canadian Gold Corp. – First Canadian	Diamond drilling of two holes for 415 metres. 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 metres.	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 E-W 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 metres, including 36 holes (3,757 metres) at the Main Zone and 26 holes (2,720 metres) at the North-North Zone.	Mannard (1986)
1980	Thunderwood – Gold Hawk	VLF-EM and magnetometer surveys. Diamond drilling of seven holes (1,422 metres). 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 metres; hole 80-5 returned 16.35 g/t Au over 0.9 metres; and hole 80-6 returned 14.40 g/t Au over 1.5 metres.	Martin and Bubar (1988)
1981	Gold Hawk Resources Ltd. – Gold Hawk	Diamond drilling of seven holes (874 metres).	Martin and Bubar (1988)
1981	SOQUEM – First Canadian	Compilation, mapping, magnetic survey, pole-dipole I.P. survey, and diamond drilling of two holes for 306 metres. Detected two strong I.P. 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 metres).	Martin and Bubar (1988)

Year	Company - Property	Work Description & Results	Reference
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 metres). 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 metres, including 5.42 g/t Au over 1.8 metres.	Stuart and Martin (1986)
1984	Brominco Inc. – Norlartic	Line cutting, property-scale surface magnetic and VLF-EM surveys, and three diamond drill holes. Compilation of existing geotechnical data.	Stuart and Martin (1986)
1984-1985	Lac Minerals Ltd - Marban	Work completed includes line cutting (122-metre 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, I.P., and H.E.M. Supplementary line cutting at 61-metre 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 totaling 5,675 metres 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-drill holes for 602 metres at the Norlartic Main Zone. All three holes intersected gold grades assaying greater than 6.86 g/t Au over widths greater than 3 metres. Hole 2001-48 cored a 6-metre interval that returned 16.35 g/t Au.	Mannard (1986)
1985-1986	Aur Resources - Norlartic	A 4,452-metre 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 3.8m, 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 metres. The best assay from this hole was 0.41 g/t over 1.5 metres, which occurred in strongly sheared talc-chlorite schists interpreted to be the Marban Shear.	Stuart and Martin (1988)

Year	Company - Property	Work Description & Results	Reference
1986	AurResources – 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 behavior of the hanging wall rocks during dewatering. Six holes totaling 356 metres were drilled into the North Zone stopes; one of these (2001- 75) returned 19.41 g/t Au over 14.9 metres (7.6-metre 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 metres and -213 metres. These holes returned intersections ranging from 3.12 g/t Au over 1.5 metres to 7.2 g/t Au over 3.05 metres.	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 metres in 12 holes completed. Regional compilation map at 1”=1,000’. Definition magnetic contouring at 100 gamma intervals. D.D.H. lithogeochemical sampling for major elements and gold. Major element lithogeochemical interpretation report. Au in ppb distribution summarized in interpretation report. Partial I.P. survey over Kewagama – Malartic contact. Construction of a 3-D model of the Marban mine.	Desbiens (1992)
1986-1987	Aur Resources – First Canadian	Sinking of 222-metre two-compartment exploration shaft at Kierens. Surface diamond drilling of 20 holes for 4,582 metres. Underground diamond drilling of 122 holes for 11,417 metres. Drifting (7’ x 8’) in ore, with East and West drifts totaling 206 metres. Raising in ore for a total of 37 metres. Sludge sampling program for 132 holes that totaled 322 metres. 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 metres of drilling to test the Gold Hawk Horizon. The highly encouraging results (Martin and Bubar, 1988) prompted Aur to test the strike extension of the horizon onto the Norlartic property.	Stuart and Martin (1988)
1987	Aur Resources - Norlartic	Hole 2001-106 was drilled on section 20+00W, approximately 365 metres 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 metres, but neither the mafic volcanic package nor the North Zone-style of mineralization was encountered.	Mannard (1986)

Year	Company - Property	Work Description & Results	Reference
1987-1988	Lac Minerals Ltd – Marban	Phase I drill program: 12,268 feet in 18 holes. Phase II drill program: drilling below -244-metre level to test depth extensions with 18 holes (3,658 metres). 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 sixty-two holes totaling 16,000 metres, 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 drill-hole program was carried out to test the favorable Héva-Kewagama contact. Hole 22 returned five clast-supported till samples with high visible pristine gold counts. Diamond drilling program of four holes totaling 1061 metres.	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 totaling 4,224 metres (10 holes) consisting six holes to test the potential of the Marban Tonalite, two holes (501-09 and 501-11) to continue a section across the Héva Formation, and two holes (501-13 and 501- 14) to test the area around the magnetic inflection of the interpreted Heva- 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 metres, 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)

6.2.1 Marban Mine

The Marban mine is located on the Marban claim group, south of the Kierens-Norlartic mineralized zones. The Marban resources described in Section 14 are approximately centered on the shaft of the Marban mine. This Subsection 6.1.1 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 Marbanite shear and the Norlartic zone hosted within the Norbenite shear. Definition drilling on the Marban zone delineated a gold-bearing structure 370 metres long in an east-west direction that dips 40° to 60° to the north, with depths of the drilling ranging from 152 to 275 metres 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 metres. In addition, drifts were initiated on levels at 61, 107, 152, 198, and 244 metres 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 Au/t and 0.50 g Ag/t. 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 (Québec) 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 26th, 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 (Québec) 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 totaling 1,877 metres into the Marban deposit in 1986 and 26 holes totaling 7,179 metres 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 four holes (1,061 metres), three to test the stratigraphy in a northerly (up-ice in the opposite direction of glacial transport) direction from the overburden holes drilled by Lac and one to test a magnetic inflection in the Heva/Kewagama contact. Another campaign was conducted in 1994 that included 10 drill holes (4,220 metres) - six of these holes tested the Marban Tonalite, two holes completed a stratigraphic section across the Heva Formation, and two holes followed up the magnetic inflection of the interpreted Heva-Kewagama contact.

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

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 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 to 2003. In 2003, McWatters purchased Barrick's 50% interest in the property subject to a 2 to 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).

6.2.2 Other Marbanite 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 five holes for a total of 931 metres in the Marbanite shear; the best intersection from this drilling was 5.42 g Au/t over 1.8 metres (Ducharme et al., 2009). Aur drilled one hole in 1985-1986 to test the extent of the Marbanite 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 metres) in 1987 to test the Marbanite 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 Au/t over 0.9 metres and 4.66 g Au/t over 5.2 metres. The drilling over this area remains widely spaced. The zone is interpreted to plunge moderately into the Marban historical property.

Table 6-3: Historical Drilling on the Marban Property

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-1 to 8-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-93-01 to 22	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: Historical Channel Sample Data Included in the Marban Project 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**	

6.2.3 Kierens, Norlartic, & 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 "Marbanite shear" to be consistent with more recent usage. The Kierens zone was part of the Norlartic mine in the pre-1985 period.

The Marbanite 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 drill holes, for 5,664 metres, were drilled in this program. The company followed up on the discovery with 66 holes (12,071 metres), and in 1946 through 1948 sank a 165-metre, three-compartment vertical shaft, completed drifts and crosscuts on the 38, 76, 114, and 152-metre levels, and conducted 4,207 metres 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 metres 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 metres of underground drilling. In 1959, Norlartic Mines dewatered the mine once again, deepened the shaft to 320 metres, developed new levels at 190, 229, 267, and 305 metres, 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 drill holes; 5,054 metres) completed in 1963 to 1964. The Kierens zone was reached by drifting 1,190 metres on the 229-metre level from the Norlartic mine in 1964-65, and underground development and drilling followed (37 drill holes; 2,281 metres).

In the period from 1959 to 1966, Norlartic Mines mined 1,076,846 tonnes at 4.46 g Au/t. Production came mainly from eight levels on the Main (Norlartic) zone at depths of 30 to 305 metres and distances of 60 to 365 metres west of the shaft. Stopes averaged six metres 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-metre levels, while the A- and B- zones at the North Zone were mined from two levels above 152 metres. This North Zone production amounted to approximately 90,000 tonnes at 16.46 g Au/t. 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 Au/t. 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 to 1975, First Canadian Gold Corp. acquired the property and conducted drilling on the North-North Zone (33 drill holes for 4,166 metres). In 1979, the three Norlartic claims lapsed and were staked by Brominco. In 1980 to 1981, Brominco conducted drilling on the Main zone (36 drill holes; 3,757 metres), primarily over untested areas east of the shaft and above the 152-metre level,

as well as on the North-North Zone (26 drill holes; 2,720 metres). During the same period, SOQUEM completed two holes (306 metres) 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 (Post-1985)

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 drill holes from 1985 to 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 totaling 603 metres tested the Main zone east of the Norlartic shaft above the 152-metre level. Nine other holes served mainly as a widely spaced test of the depth potential of the Main zone below the 549-metre level. Above this level, tightly spaced underground drilling was completed by Aur from 1986 to 1990. The Main zone remained sparsely tested below the 549-metre 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 drill-hole 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-metre level.

Table 6-5: Norlartic Zone: Aur Underground Drilling (1986-90) (From Carrier, 2006)

Hole #	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 to 305-metre 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 to 549-metre 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 to 1992 and produced a total of 56,000 ounces of gold.

6.2.3.3 Kierens Mine (Post-1985)

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 drill holes on the First Canadian historical property from 1985 to 1989. The up-dip extensions of the Kierens zone (three drill holes; 611 metres) were tested in 1985-86. An intersection of 12.00 g Au/t over 4.6 metres prompted a second program (18 drill holes for 5,063 metres) that tested the Kierens zone over a strike length of 425 metres and to a depth of 275 metres. 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-metre, vertical, two-compartment shaft, 20 surface holes (4,580 metres), and 122 underground holes (11,414 metres), mainly on the 223-metre 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 to 1992.

Table 6-6: Kierens: Aur Underground Drilling 1986-90 (From Carrier, 2006)

Hole#	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 metres 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-metre level was mined from 1959 through 1966. In 1985 and 1986, Aur completed 16 holes (4,451 metres); ten holes were drilled below the 152-metre level, and five were drilled along the eastern extent above the 152-metre level. In 1998, Aur drilled seven additional holes (1,579 metres) on two sections 30 metres apart in order to test the western extent of the North Zone. This was a 60-metre 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 Au/t over 2.0 metres, 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-metre centers and up to 305 metres west of the old workings. The holes proved

the continuation of the zone, with a notable intersection of 32.00 g Au/t over 1.5 metres on the westernmost section at a vertical depth of 380 metres. Two holes tested the eastern extent of the North Zone, and an intersection of 10.32 g Au/t over 0.8 metres was obtained.

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

6.2.3.5 North-North Zone

The following information is taken from Lefrançois (2005).

The North-North Zone is located 450 metres north of the Norlartic Main zone and 700 metres 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 (six drill holes) and subsequently by First Canadian Gold Corp. in 1974 (33 drill holes for 4,166 metres), Brominco in 1980 (26 drill holes for 2,720 metres), and Aur in 1985-86 (32 holes for 5,753 metres). 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 metres and to a vertical depth of 300 metres. Limited drilling has been done on strike to the east, and only three holes were drilled below a depth of 230 metres.

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 metres) between 1980 and 1983, Thunderwood completed an additional 62 holes (16,000 metres) in 1987 using 30-metre drill-section spacings to a vertical depth of 400 metres. Veins/vein systems #1, #2, and #3 were identified by this program. Visible gold in veins was observed in 40% of the 1987 drill holes. Vein #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 #1 in 1988 via a 500-metre crosscut driven from the 230-metre level of the Kierens mine. From 1989 to 1991, Aur conducted small-scale mining of Vein #1 within the Gold Hawk historical property on the 230-metre level and 218 and 187-metre sub-levels. At the end of October 1991, 8,273 tonnes with an average head grade of 13.95 g Au/t (3,711 oz Au) were mined from Vein #1. Pillars were recovered in November 1991, but production numbers are not available. Stopes averaged 1.2 to 1.5 metres in width. The mined-out section of Vein #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 #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 kilometres. The drilling was mainly completed on 60-metre section spacings. Limited underground drilling was also completed from a cross¬cut driven from the 400-metre 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 Marbanite shear to the south. The 20 holes tested the Gold Hawk horizon over a strike of 550 metres and to a vertical depth of 300 metres. The zone was further tested by 24 underground holes from the 395-metre level of the Kierens mine.

6.3 Historical Mineral Inventory Estimates

A number of estimates of mineralization at the Marban property were completed prior to NioGold's involvement in the project. There are insufficient details available on the procedures used in these estimates to permit Ausenco to determine if any of the estimates meet NI 43-101 standards.

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. The author has not done sufficient work to classify these historic estimates as current mineral resources or mineral reserves, and Osisko is not treating these historic estimates 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.

The information in Sections 6.3.1 through 6.3.4 is taken from Carrier (2006). Ausenco is not aware of any historic estimates of the Marban zone mineralization.

6.3.1 Norlartic & North Zones

Discovered in 1943 to 1944, the Norlartic Main zone was developed in 1946 through 1948, when "reserves" were estimated at 544,320 tonnes grading 5.83 g Au/t (102,000 ounces) in the Main zone above the 152-metre level. In the early 1950s, the North Zone was discovered 365 metres north of the Norlartic shaft.

Additional underground drilling led to the calculation of "reserves" totaling 566,093 tonnes grading 6.51 g Au/t (118,560 ounces) in the Main zone west of the shaft, in addition to 90,000 tonnes grading 25.37 g Au/t in the North Zone. Drilling west of the shaft and below the 305-metre level in 1959 added 108,864 tonnes at 3.77 g Au/t (13,200 ounces) to the "reserves."

Following drilling by Brominco in 1980-1981 on the Main zone, primarily in an untested area east of the shaft and above the 152-metre level, "reserves" were estimated in the Main zone at 780,192 tonnes grading 4.46 g Au/t (111,800 ounces) above the 152-metre level.

6.3.2 Kierens Zone

The Kierens zone on the First Canadian historical property was discovered in 1963-1964 and was reached by drifting from the Norlartic mine in 1964 and 1965. Underground development and drilling led to an estimate of "reserves" of 90,720 tonnes at an average grade of 5.83 g Au/t (17,000 ounces) in the Kierens zone.

6.3.3 Gold Hawk Horizon

Following the drilling of 84 holes between 1980 and 1987, Thunderwood calculated a "resource" of 110,000 tonnes at a grade of 13 g Au/t for Veins #1 and #2 on the Gold Hawk claims.

A "reserve" based on widely spaced drill holes of 254,016 tonnes grading 8.57 g Au/t (70,000 ounces) was reported by Aur in 1989 for the Gold Hawk horizon. This mineralization lies on the First Canadian historical property at a depth of 250 to 400 metres.

6.3.4 Aur Mineral Inventory Estimates

Aur calculated "possible reserves" at the North-North Zone in October 1986 (Table 6-7). These were defined over a 215-metre strike length and to a depth of 150 metres.

Table 6-7: North-North Zone 1986 Historical "Possible Reserves" (not NI 43-101 compliant) (From Carrier, 2006)

Location	Cut-off (g Au/t)	Tonnes	g Au/t	Oz Au
No Crown Pillar 30-metre Crown Pillar	2.6	441,975 385,735	3.57 3.63	50,673 45,075
No Crown Pillar 30-metre Crown Pillar	1.7	754,315 652,615	2.85 2.88	69,020 60,443

Following surface drilling and most of the underground drilling discussed in Section 6.1.2, diluted "reserves" totaling 450,000 ounces of gold were calculated for the Kierens, Norlartic, Gold Hawk, North, and North-North Zones by the Kierens Mining Division of Aur in October 1989 (Table 6-8).

Table 6-8: 1989 Pre-Production Historical "Reserves" (not NI 43-101 compliant) (From Carrier, 2006, citing Lefrançois, 2005)

	"Proven" & "Probable"			"Possible"			Total		
	Tonnes	g Au/t	Oz Au	Tonnes	g Au/t	Oz Au	Tonnes	g Au/t	Oz Au
Kierens	112,375	8.09	29,278				112,375	8.09	29,278
	100,619	3.81	12,285				100,619	3.81	12,285
				675,042	4.90	104,173	675,042	4.90	104,173
Actinolite	212,994	6.07	41,563	675,042	4.90	104,173	888,036	5.11	145,736
L-Lens	101,807	6.99	22,839	87,574	4.29	12,042	189,381	5.73	34,881

	"Proven" & "Probable"			"Possible"			Total		
	Tonnes	g Au/t	Oz Au	Tonnes	g Au/t	Oz Au	Tonnes	g Au/t	Oz Au
Total Kierens	136,080	3.43	15,000	226,800	3.43	25,000	362,880	3.43	40,000
Actinolite				299,376	4.46	42,900	299,376	4.46	42,900
L-Lens				252,202	5.11	41,422	252,202	5.11	41,422
H-Lens	237,887	4.94	37,839	865,952	4.35	121,364	1,103,839	4.49	159,203
Main East				254,016	8.57	70,000	254,016	8.57	70,000
Total Norlartic				136,080	6.85	30,000	136,080	6.85	30,000
Gold Hawk				385,560	3.63	45,050	385,560	3.63	45,050
Total									**449,989**

Table 6-9 summarizes historical "resources" 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 totaled 1,354,249 tonnes averaging 5.33 g Au/t.

Table 6-9: 1993 Remaining Historic "Resources" (not NI 43-101 compliant) (From Carrier (2006): Aur internal report, remaining resources in 1993 by P. Pelz, P.Geo.)

Zone	Tonnes	g/t Au	Ounces	Description
Kierens	261,269	4.80	40,320	undeveloped drill-defined ore below 1300' level (395 m)
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 Horizon	254,016	8.57	70,000	defined by widely-spaced drilling

6.4 Previous NI 43-101 Compliant Resource Estimates

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. The interested reader is referred to those reports, which can be found on SEDAR. These previous estimates are superseded by the one described in Section 14, which includes all drilling at the Marban deposit as of the Effective Date of this report.

7 Geological Setting & Mineralisation

Information on the geology, mineralization and deposit types of the Marban Project 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 company. The information provided herein is modified from data provided by O3 Mining and from those reports.

7.1 Geological Setting

7.1.1 Regional Geology

The Marban Project is located in the Malartic mining camp, which is 15 kilometres northwest of the Val-d'Or mining camp, within the Precambrian Canadian Shield in western Québec.

Rocks of the Malartic gold district belong to the Archean Abitibi greenstone belt of the Superior Province, Québec. The Abitibi greenstone belt consists of east-trending alternating volcanic-plutonic and sedimentary belts that are bounded by crustal-scale faults (Figure 7-1). The volcanic rocks yield dates ranging from 2750 to 2698 Ma. On the basis of 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") (Ludden and Hubert, 1986). The two zones 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). The SVZ is composed of several fault-bounded blocks that include komatiitic-tholeiitic volcanic assemblages derived from a mantle plume and tholeiitic to calc-alkaline arc/back-arc assemblages related to a subduction zone (Wyman, 1999). One of the blocks, the Malartic Composite Block ("MCB"), hosts the Malartic and Val-d'Or gold districts. The MCB is composed of allochthonous, komatiitic to tholeiitic volcanic suites that are interpreted to be derived from volcanic plateaus and volcanic islands that were juxtaposed during the Late Archean (Jackson and Fyon, 1991; Ludden et al., 1986). Rocks of intermediate composition are rare. To the south, the MCB is juxtaposed with turbidite sediments of the Pontiac sub-province along the Cadillac Tectonic Zone, thought to mark a second accretion zone (Feng et al., 1992).

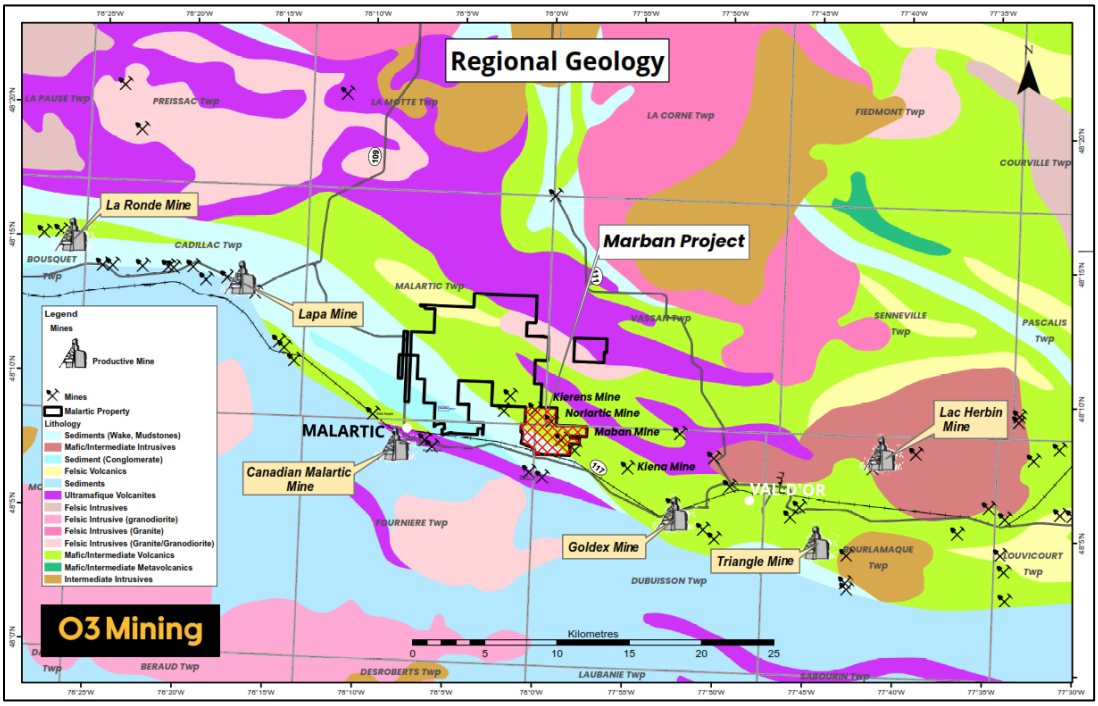
A controversy still persists as to the origin of these blocks. They have long been regarded as a collage of volcanic arcs (Jackson and Fyon, 1991; Thurston and Chivers, 1990; Card and Ciesielski, 1986), but it has been suggested that they consist of accreted allochthonous fragments of oceanic plateaus (Desrochers et al., 1993). Each of these tectonic panels has its own lithologic assemblage, structure, and geochemical signature. In this scenario, calc-alkaline magmatism is triggered by extension following the period of convergence and is superimposed upon the amalgamation of oceanic plateau fragments. Narrow east and northwest-trending metasedimentary troughs subsequently developed along the wrench faults responsible for the juxtaposition of the volcanic panels (Ludden and Hubert, 1986).

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

The Abitibi is one of the most gold-rich of the worldwide occurrences of Archean greenstone belts, with a high concentration of lode gold deposits found in the SVZ. Historic production for this segment totals over 170 million ounces of gold. Major gold camps within the SVZ are

primarily localized at flexures along the Destor-Porcupine-Manneville and Cadillac structural zones.

Figure 7-1: Regional Geologic Map of the Southern Abitibi Belt



Note: "Marban Project" refers to Kierens, Gold Hawk, Norlartic, and Marban claim groups. (source: O3 Mining, 2020)

7.1.2 Local Geological Setting

The geology of the Malartic-Val-d'Or region was defined by Latulippe (1976), Imreh (1984), and Rocheleau et al. (1987), but more recent work by the MRNQ (reports MB 98-01, MB 98-05, and DV 99-03) has led to a better division of the local volcanic stratigraphy. The MCB is subdivided into a lower Malartic Group and an upper Louvicourt Group. The Malartic Group consists of plume-derived komatiitic-tholeiitic marine-plain volcanic assemblages, divided into the La Motte-Vassan, Dubuisson, and Jacola formations, although several lithotectonic domains (North, Central, Vassan, de Montigny), possibly representing distinct accreted blocks, have been established (Desrochers and Hubert, 1996). In contrast, the overlying Louvicourt Group represents an arc-type complex. Recent dating by the MRNQ suggests that the volcanic arc was constructed between 2706±1 and 2702±1 Ma. The Louvicourt Groupis subdivided into the Val-d'Or Formation, a transitional to calc-alkaline volcanic complex, and the Heva Formation, characterized by geochemically distinct iron tholeiites.

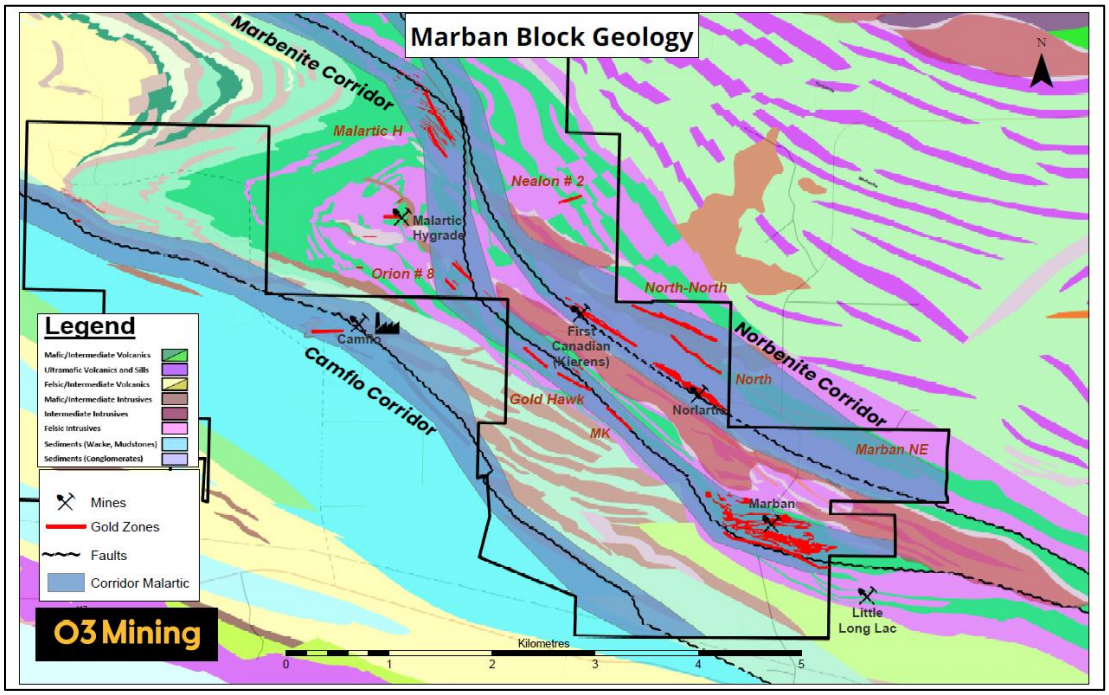
The East Malartic (2.9 million oz), Barnat-Sladen (1.2 million oz), Canadian Malartic (2.4 million oz), and Malartic Goldfields (1.7 million oz) gold deposits occur within a 900-metre-wide band of highly sheared and faulted rocks named the Malartic Tectonic Zone ("MTZ"), part of the 300-kilometre-long Cadillac Tectonic Zone. Two other northwest-striking fault systems are recognized to the north of the MTZ, namely the Norbenite and Marbanite 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 (First Canadian), Norlartic, Marban, Kiena, Shawkey, Wissik, Mine Ecole, and

Goldex. Stratigraphic units in the area of the Malartic district generally strike at 300° azimuth, and their contacts are usually sheared and faulted.

7.1.3 Geological Setting of the Property

The geologic setting of the Marban Project property is illustrated in Figure 7-2. The property is dominated in its northern portion by volcanic and intrusive rocks of the Jacola Formation in the Lower Malartic Group. The Kierens, Norlartic, and Marban gold mines are all hosted within the basalts and ultramafic rocks of the Jacola Formation. Two major granodioritic stocks sub-parallel to the stratigraphy occur within the Jacola Formation in this area: the North-North granodiorite and the Central granodiorite. These intrusions are locally altered but do not appear to be deformed. The northern part of the property also hosts three major deformation corridors: the North, Norbenite, and Marbanite (sometimes called the Marban) shears.

Figure 7-2: Geologic Map of the Marban Project Property



Source: O3 Mining, 2020

To the south, the central part of the Marban claim group consists of mafic volcanic and volcaniclastic rocks of the Heva Formation in the Upper Malartic Group. Further south, these rocks are in contact with greywacke and volcano-sedimentary rocks of the Kewagama Group in the southern part of the Marban claim group. Dikes and sills of various compositions intrude all of these units.

The Marban resource is located on the northern flank of a west-northwest-trending syncline. From north to south, the units include:

- An alternating sequence of sheared ultramafic volcanic rocks (now talc schist) and granodiorite dikes;

- The “Mine Sequence” of sheared, pillowed, aphanitic and fine-grained basalts that host the bulk of the Marban gold mineralization; and

- Sheared and massive to pillowed ultramafic volcanic rocks.

These units dip 45-70° to the north. Dikes cut the ultramafic rocks and the wide granodioritic intrusions. The Norbenite shear is an important northwest-trending deformation zone that dips to the northeast at moderate to steep angles. The shear zone is composed of mafic volcanic rocks, talc-chlorite schists, and felsic dikes. It is particularly well developed on the First Canadian and Norlartic claim groups, where the shear zone localizes the Kierens and Norlartic gold resources. The gold mineralization is closely associated with altered and pyritized intermediate intrusions that cut sheared mafic and ultramafic volcanic rocks within the shear.

7.1.3.1 Litho-Structural Model of the Marban Deposit

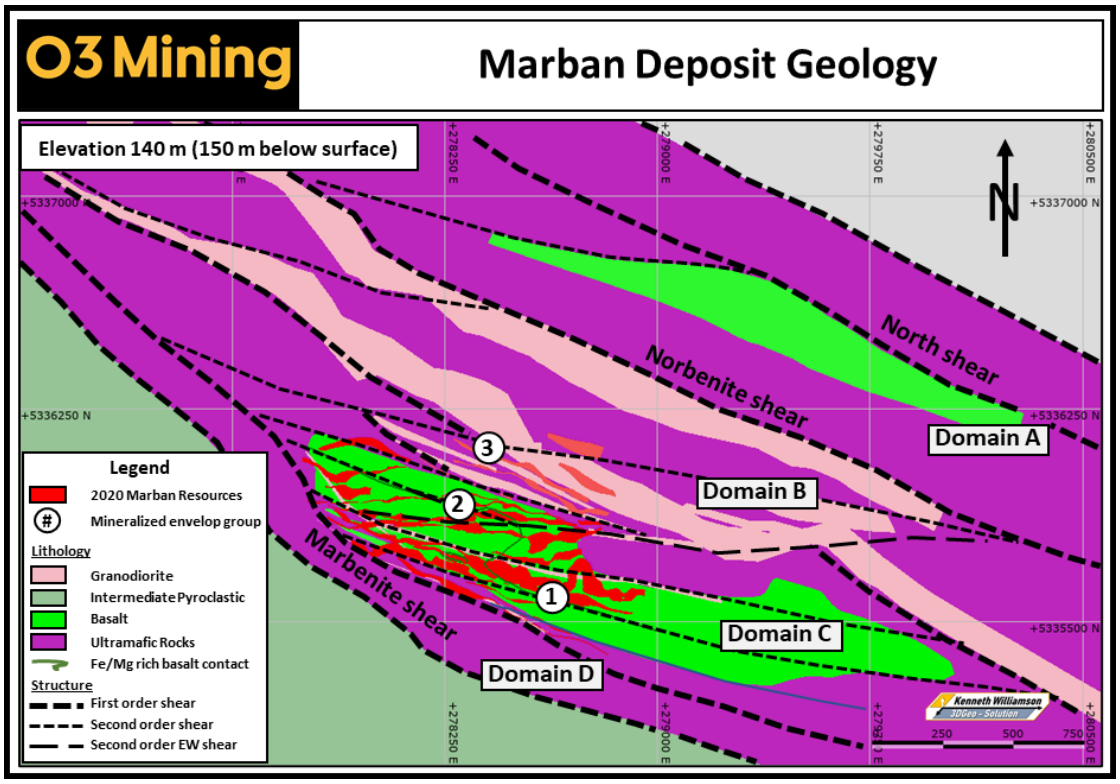
O3 Mining has recently asked Kenneth Williamson 3DGeo-Solution to elaborate a 3D litho-structural model covering the Marban Project property. Figure 7-3 presents a plan view cut at 150 m below surface through this 3D litho-structural model and provides complementary information on the geology and structural setting of the Marban Project property.

From a structural standpoint, four (4) structural domains are recognized. These structural domains are separated one from the other by the major NW-trending regional deformation corridors (i.e. North, Norbenite and Marbenite shears). From north to south, Domain A: consists primarily of highly deformed ultramafic and mafic rocks; Domain B is characterized by highly deformed ultramafic rocks intruded by numerous granodiorite dikes; Domain C: “the mine sequence”, is dominate by a sequence of Mg-rich and Fe-rich basalts, and hosts most the Marban Deposit gold mineralization; and Domain D: an highly sheared ultramafic unit at the contact with the intermediate pyroclastic rocks of the Heva Formation.

The 3D modeling work suggests the presence of numerous secondary structures. These structures are interpretated from drillholes structural data, the existence of planar lithological contacts, presence thin tabular felsic intrusions and from geological discontinuities. They are WNW-trending and appear to be preferentially developed within both Domain B and C. Following the current interpretation, the regional NW-trending thrust faults are thus considered as first order structures, and the secondary structures are considered splays of the latter.

The development of WNW-trending structures appears to be responsible for the formation of asymmetrical drag-folds. The overprinting of the first generation of tight NW-trending folds by such later oblique drag-folds has likely created fold interference patterns, explaining the doubly-plunging fold axis observed at the mine scale (see section 7.2.1 below).

Figure 7-3: Geologic Map of the Marban Project Area



Source: Belzile, 2016

7.2 Mineralization

The Marban Project property is the site of three former gold producers – the Kierens, Norlartic, and Marban mines – that collectively produced almost 600,000 ounces of gold (Norlartic Mines and Aur reports, as well as MRNQ assessment files). Three major shear zones have been traced across the property; from north to south, these are the North, Norbenite, and Marbanite shears. Six sub-parallel gold-mineralized horizons are recognized; from northeast to southwest these include the North-North Zone, the North Zone (along the North shear), the Norlartic Main and Kierens zones (within the Norbenite shear), and the Gold Hawk horizon and the Marban zone (within the Marbanite shear). This report focuses on resource estimates of the Marban, Kierens, and Norlartic deposits.

7.2.1 Marban Resource Area

The Marban mine is located on the Marban claim group, lies to the south and east of the Kierens and Norlartic mines, and is situated within the Marbanite shear zone. The Marban deposit was discovered after a drilling program that followed-up on the discovery of glacially transported gold-mineralized boulders. Marbenor Malartic Mines Ltd. drilled 96 holes that resulted in the discovery of two gold-bearing zones, referred to as the Marban North and Marban South zones.

Gold mineralization is hosted primarily in the Mine Sequence basalts within east-west-oriented shear zones that dip 45 to 70° to the north and are from 10 to over 70 metres wide.

The deposit is tightly folded, but the shear zones and the lithologies are affected differently. The plunge of folds defined by the shear zones is approximately sub-horizontal, while the folds defined by lithologic contacts plunge westerly in the eastern part of the deposit and easterly in the western part of the deposit. The changes in the plunge of fold axes defined by lithologies create a dome and basin, or egg-carton-like, configuration. The lithologic contacts are oblique to the mineralized zones.

The mineralization consists of gold-bearing pyrite and pyrrhotite, with free gold occurring locally. The timing of the mineralization seems to be syn- to post-tectonic, based on the morphology of the pyrite crystals, and was probably a continuous event (Renou, 2007). The pyrite and pyrrhotite occur as finegrained euhedral clusters elongated along shear planes, but pyrite also can be coarse-grained and disseminated in the host rocks.

The Mine Sequence basaltic flows within the Marbanite shear host gold associated with quartzcarbonate veins accompanied by pyrite and pyrrhotite. Gold may also be disseminated within the mafic volcanic rocks. Although free gold is observed in diamond-drill core (“core”), the gold generally occurs as inclusions or fillings in cracks in pyrite (Renou, 2007). Locally, gold mineralization is also hosted in the granodiorite and ultramafic flows relatively close to their contacts with the Mine Sequence basalts. Ducharme et al. (2009) reported that the mineralization exhibits a pinch and swell pattern, with great variation in both width and grade.

The Marban mineralization has been the focus of O3 Mining’s drilling since 2006. To date, O3 Mining has drilled 337 holes at the deposit, which combined with historic drilling, brings the total to over 1,700 holes drilled in the Marban area. The following summarizes O3 Mining observations relating to the Marban area:

- The depth of mining and exploration is shallow compared to other gold deposits in the district
- The prospective Mine Sequence is thickened by tight folding of the mine stratigraphy
- Mined ore shoots are suggestive of tight S-shaped geometries
- Large mined stopes occur at fold noses
- The hinges of folds, as defined by mineralized shear horizons, are all approximately at the same elevation, which suggests the potential to find another series of such folds at depth
- Gold is hosted in sheared mafic volcanic rocks that are encompassed within a predominantly ultramafic sequence
- Disseminated gold mineralization is associated with pyrite-chlorite-carbonate-silica-albite
- High-grade values in O3 Mining holes are associated primarily with free gold, although some mineralization occurs with strong silicification and increased concentrations of pyrite and pyrrhotite
- The down-dip extent of the Mine Sequence to the north is not fully investigated
- There was only minor production west of the shaft

7.2.1.1 Marban Mineralized Envelopes

Following the creation of the first iteration of the Marban litho-structural 3D, O3 Mining has mandated Kenneth Williamson 3DGeo-Solution 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 envelops are designed based on gold grade continuity using a 0.30 g/t Au cut-off grade, and larger size envelopes containing internal dilution are preferred to an "individual narrow vein" design. The interpretation of the mineralized envelopes takes into consideration both the structural controls imposed by the different generations of folds and faults, and also the geometry of the existing stopes at the former underground Marban mine. The current interpretation recognizes a total of 27 mineralized envelopes. The 27 mineralized zones are grouped into three (3) distinct groups based on their geometry and relative location. The resulting vein network geometry is illustrated on Figure 7-3.

The first two (2) groups of zones appear to coincide spatially with both limbs of an asymmetrical WNW-trending antiform affecting the "mine sequence". Zones of group 1, along the south limb, are generally NW-trending and shallowly dipping. They show the greatest vertical extent; likely consequence of being located along the long limb of the asymmetrical fold. Zones of group 2, along the northern short limb, show a lesser vertical extent and appear to be more discrete and more widely spaced. The intersection of both sets of zones in the hinge area leads to the development of tubular shallowly plunging ore shoots that follow the fold axis.

The third group of mineralized envelopes is spatially associated to the felsic intrusions found in the southern portion of Domain B. Individual zones of this group are co-planar with the NW-trending felsic intrusions. Another fold hinge (synform) is interpreted at the intersection of zones from group 2 and group 3. The intersection lineation between zones of group 2 and group 3 is suggesting a shallowly plunging fold axis.

7.2.2 Kierens & Norlartic Resource Areas

Gold mineralization at the First Canadian and Norlartic claim groups is related to west-northweststriking, north-dipping shear zones related to the Norbenite shear, as well as swarms of felsic to intermediate sills and dikes that intrude the shear zones (Lefrançois, 2005). The Norbenite shear is comprised of highly altered and deformed units of mafic volcanic to talc-chlorite schists and felsic dikes. These units tend to be discontinuous due to the effects of superimposed shearing and drag folding. The shear zone ranges from 45 to 90 metres in width and is approximately conformable with encompassing hanging wall and footwall ultramafic units.

Mafic volcanic rocks (Stuart and Martin, 1986): Mafic volcanic rocks make up 20 to 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 silicification or carbonatization.

Relatively unaltered mafic volcanic rocks are medium to dark green, non-magnetic, fine-grained, and massive to moderately foliated. Abundant fine-grained chlorite and carbonate are common. Locally these rocks may also contain brown chlorite, biotite, and talc and, near the contacts of the mineralized horizons, up to 50% post-kinematic amphibole and/or

actinolite needles up to one centimetre long. Up to 5% fine- to coarse-grained pyrite is very common, and anomalous gold values occur locally in these rocks.

The altered mafic volcanic rocks are a distinct sub-unit representing up to 30% of the mafic volcanic unit. The widest zone of altered mafic volcanic rocks identified to date is 20 metres (true width; hole 1902-18). These rocks are characterized by intense silicification, carbonatization, fracture-controlled quartz, calcite flooding, up to 8% medium to coarse-grained pyrite mineralization, and almost always contain economic gold mineralization. The matrix or host rock within these altered zones is dark grey-green, fine grained, and probably recrystallized. Contacts with the relatively unaltered mafic volcanic rocks are gradational. The name “mafic volcanic rock” is a field term for these rocks that may be, in part, incorrect, as in some areas they resemble highly chloritized ultramafic rocks.

Talc-chlorite schist (Stuart and Martin, 1986): Talc-chlorite schist makes up approximately 50% of the mineralized horizons, occurring in units varying from less than one metre to over 30 metres 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 to 70% fine-grained talc and chlorite, with 30 to 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 schists typically contain 1 to 2% medium to coarse-grained disseminated euhedral pyrite. Gold appears to be erratically distributed within this unit. The blue-grey talc-chlorite schists are moderately to strongly magnetic and are interpreted to be altered and deformed equivalents of the hanging wall and footwall komatiites. The green talc-chlorite schists are non-magnetic, but their origin is probably similar to that of the blue-grey units, the green color being due to the presence of abundant magnesian chlorite.

Felsic dikes (Stuart and Martin, 1986): Felsic dikes 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 metres, and are oriented parallel to the principal schistosity.

The felsic dikes are typically medium to pale grey or reddish grey (hematite staining), fine-grained to aphanitic, and have a massive to weakly foliated texture. They are usually strongly silicified 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 dikes also commonly contain 5 to 15% randomly oriented carbonate and quartz-carbonate veins.

The felsic dikes 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 dikes are relatively homogenous, weakly or moderately altered, and contain fine disseminated pyrite, but the dikes host more important mineralization where they are strongly altered and are cut by numerous quartzcarbonate- pyrite veins.

Most of the felsic dikes that occur within the mineralized horizon probably represent altered counterparts of the granodiorite dikes encountered in the hanging wall and footwall ultramafic units. Ghost "granodioritic" texture was observed in a few of the dikes, and in hole 1902-24 one of the dikes within the footwall ultramafic unit shows a zonation from a fresh granodioritic core to "ghost granodioritic" texture to typical silicified felsic dike material at the edges of the dike. The dikes do not seem to be chemically associated with their volcanic host rocks (basalts and komatiites) of the Jacola Formation (Carrier, 2006).

7.2.2.1 Norlartic Deposit

The Norlartic mineralization occurs primarily at the Main zone (Figure 7-4). Additional mineralization has been defined below and to the east of the old workings (Stuart and Martin, 1988). The Main zone consists of a one-metre-wide felsic sill or dike that occurs within a large, sheared envelope of mafic rocks in the core of the Norbenite shear. Above the 500 level of the Norlartic mine, the main sill seems to change into a swarm of smaller dikes in “horse tail” style. Gold is usually directly associated with the intrusion(s). The intermediate intrusive and mafic volcanic rocks in the Main zone have been strongly silicified and pyritized and have undergone intense brittle fracturing and quartz-carbonate-pyrite vein injection. Economic gold concentrations in the Main zone are found in quartz-calcite-pyrite vein stockworks that occur in closely spaced lenticular intermediate intrusions.

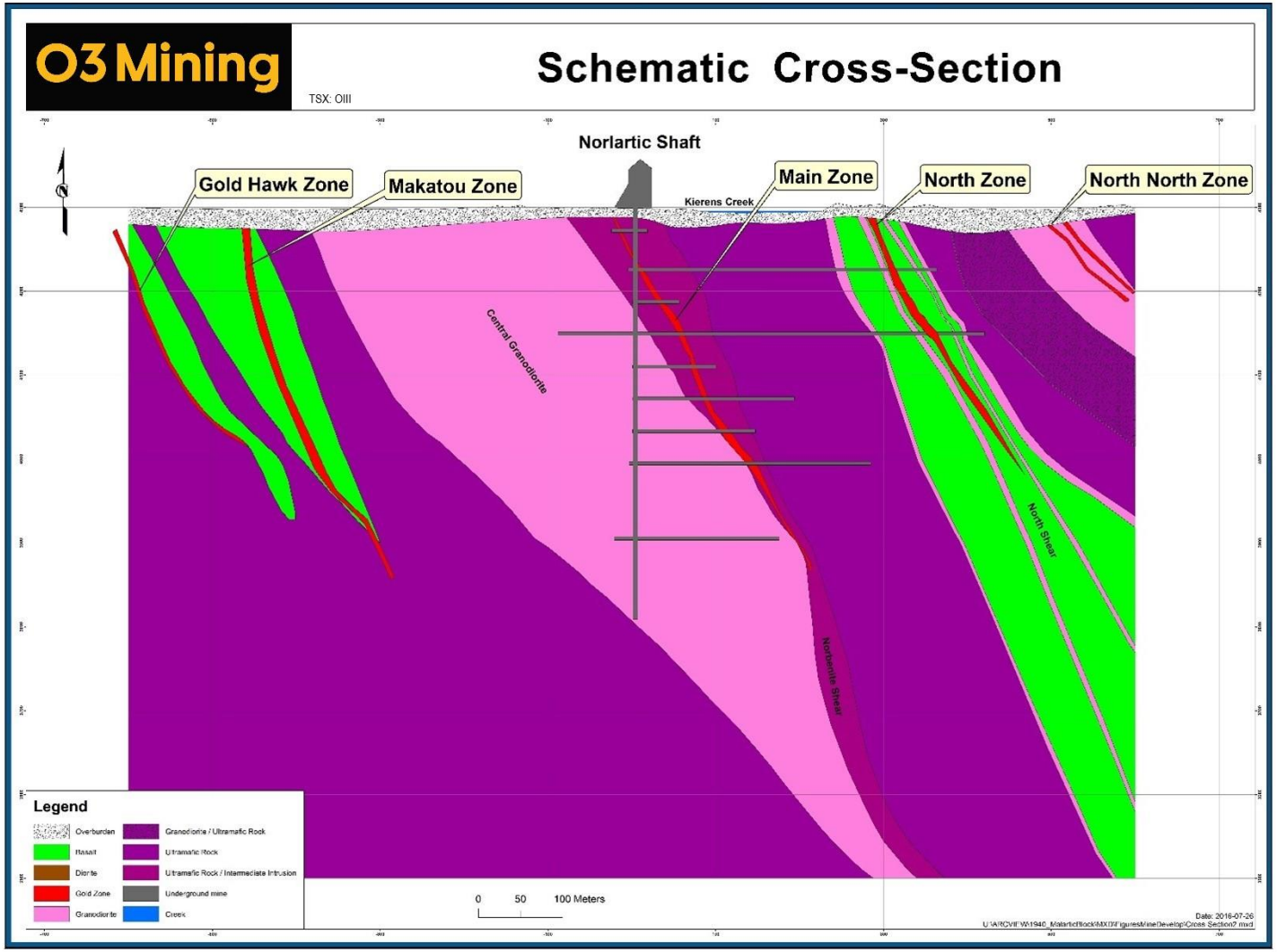
The Actinolite zone was discovered in the late 1980s. It lies to the east of the Norlartic shaft above the 152-metre level. In contrast to the Main zone, the Actinolite zone is not lithologically controlled, but instead disseminated gold occurs within an actinolite-bearing alteration zone that overlaps various units of mafic or ultramafic volcanic rocks. A swarm of felsic dikes appears to be related to the zone, and a large felsic dike caps the zone in the hanging wall.

7.2.2.2 Kierens Deposit

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 silicified and carbonatized zone that ranges in thickness from three to 30 metres (Figure 7-5). 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, more massive mafic units, and felsic 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).

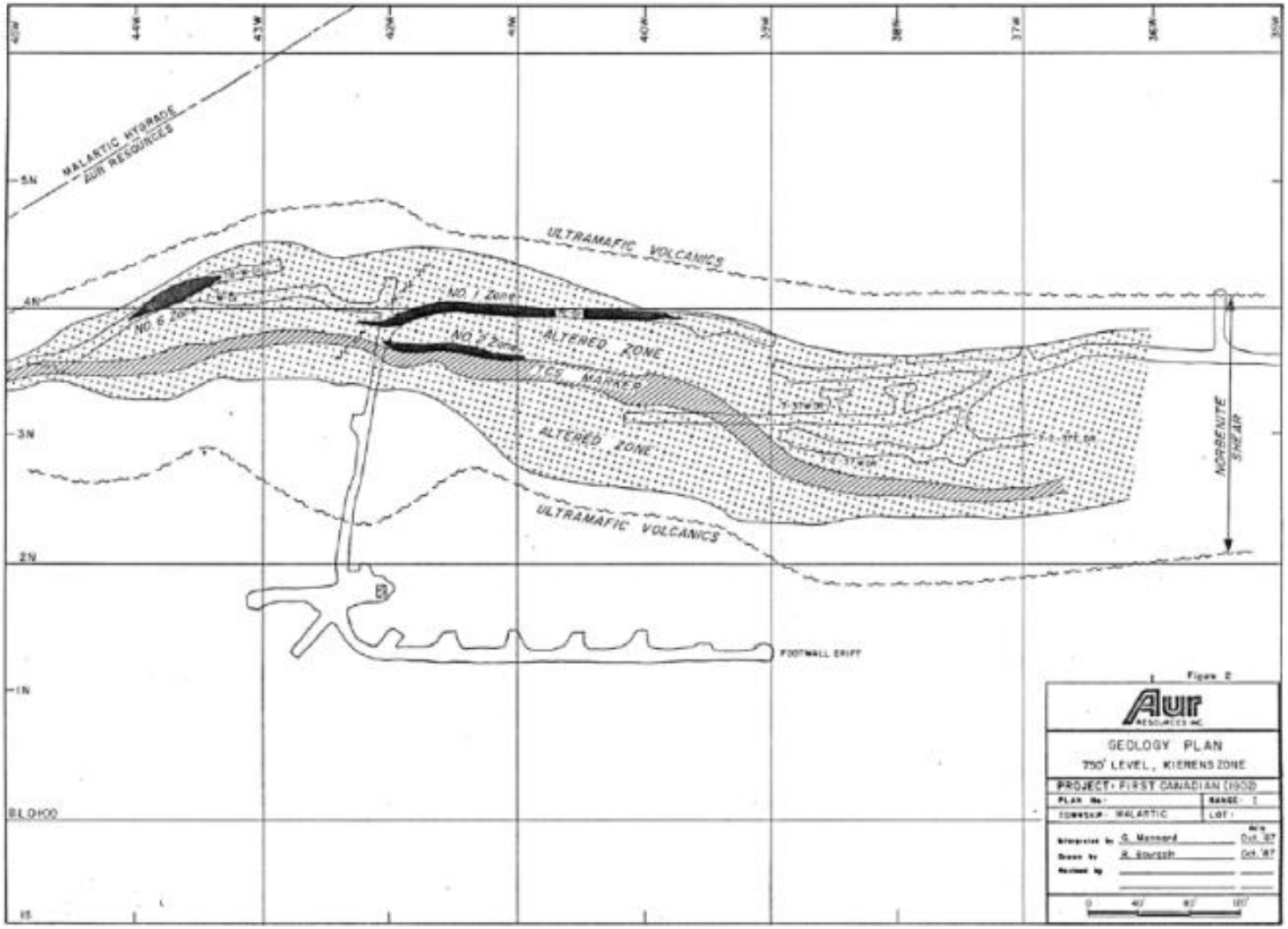
The boundaries of the gold zone at Kierens define the area within which continuously anomalous gold concentrations occur, and outside of which anomalous gold values are rare and discontinuous. The gold zone is approximately conformable with the Norbenite shear zone as a whole, but it clearly transects the individual lithologic units within the shear zone. Gold does not seem to be preferentially associated with any one of the felsic intrusions, mafic units, or ultramafic rocks, but is instead associated with the carbonate alteration. Toward the west and with depth, the zone splits into two distinct sub-zones (Figure 7-5). Mineralization may be continuous across the entire gold zone or may occur as discontinuous subzones separated by relatively unaltered intervals.

Figure 7-4: Geologic Cross-section of the Norlartic Mine



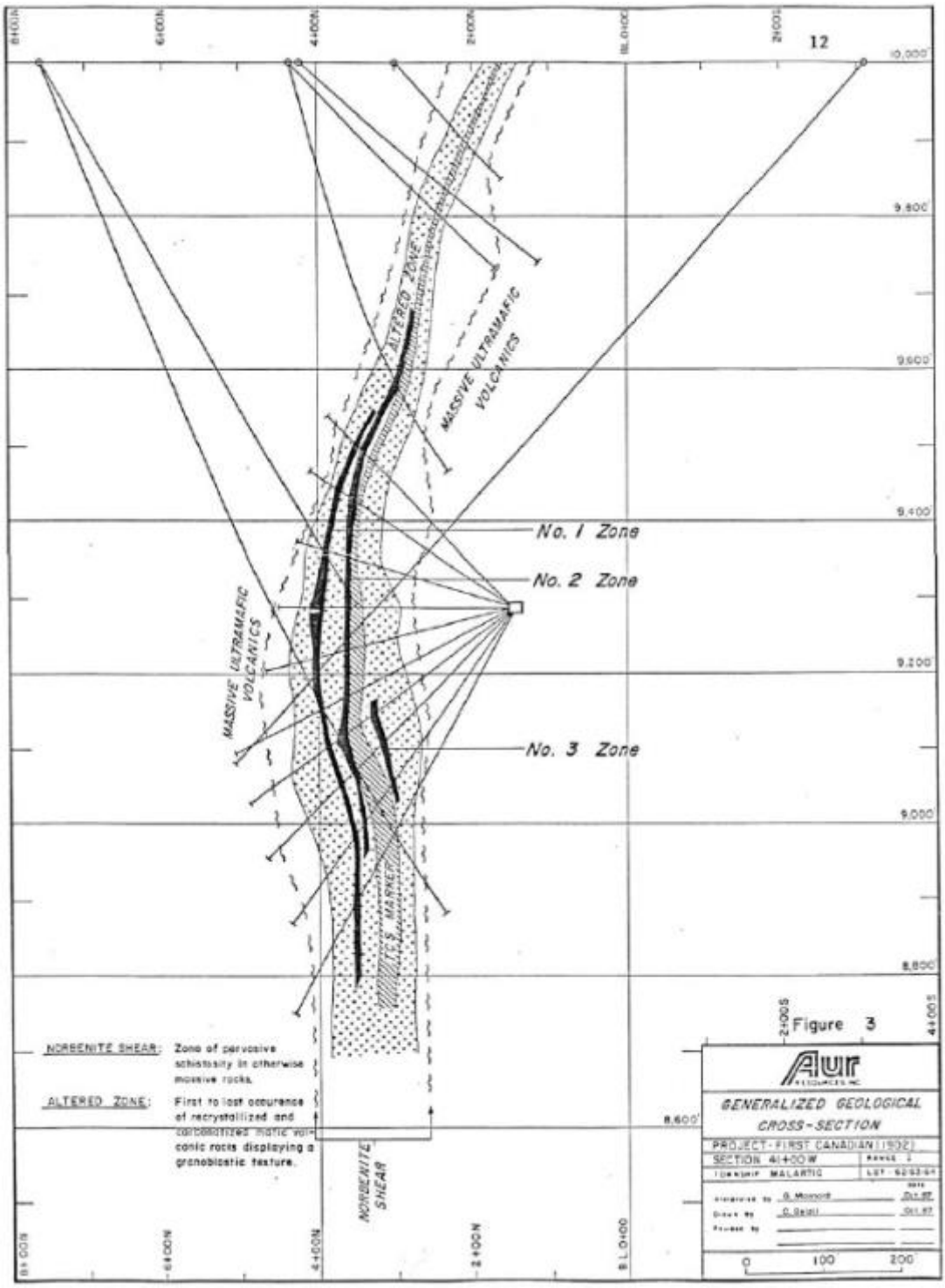
(source: O3 Mining, 2020)

Figure 7-5: Kierens Underground Geologic Plan of 229-Metre Level



Source: Mannard, 1987

Figure 7-6: Kierens Generalized Geological Cross-section



Source: Mannard, 1987

The alteration is characterized by intense silicification, carbonatization, and pyritization with up to 8% fine- to coarse-grained, disseminated euhedral pyrite. The pyrite may also occur in small, randomly oriented quartz and quartz-carbonate veins. In general, the best gold values occur where the pyrite is coarse (two to five millimetres) and the veins are most abundant.

The mineralization/alteration assemblage is most commonly hosted by the mafic volcanic rocks and felsic dikes and is only rarely hosted by the talc-chlorite schists. In some cases, the intensity of alteration is such that the host rock has been totally replaced.

A second style of gold mineralization at Kierens consists of native-gold-bearing quartz veins intersected in four holes (1902-15, -16, -18, and -20). Gold occurs as coarse visible grains in white quartz veins that are oriented parallel to the primary schistosity. The veins do not exhibit a lithologic control but appear to be confined to the hanging-wall side of the gold zone. Individual veins are four centimetres to 0.6 metres wide, bull-white in color, and massive. Many of the veins are barren, but those in which visible gold was observed are often very rich, containing up to 30 grains of visible gold.

The gold occurs as isolated particles within the quartz or may be concentrated in and around tiny, chlorite-filled fractures. Similar quartz veins were observed on the hanging-wall side of the gold zone. In contrast to Stuart and Martin's (1986) description of the mineralization, Mannard (1987) identified three styles of epigenetic gold mineralization at Kierens: (1) high-grade single-vein structures; (2) sill stockworks; and (3) laminated veins in recrystallized mafic-volcanic rocks.

The high-grade single-vein structures are vertically dipping, white quartz veins varying from 15 centimetres to 0.9 metres in width, thicker than the veins described by Stuart and Martin (1986). They contain numerous chlorite inclusions and native-gold occurrences. The wall rocks are commonly strongly chloritized and contain quartz stringers up to eight centimetres in width. The veins are generally tabular in form, locally boudinaged, and transect their host lithologies at an oblique angle.

Sill stockworks are characterized by a white quartz-vein stockwork system that comprises 10 to 30% of the intermediate sill host rocks. These sills display strong fracture-controlled silicification and carry 1 to 5% disseminated pyrite and locally coarse native gold.

The laminated veins in recrystallized mafic-volcanic rocks are characterized by a system of parallel, concordant, vertically dipping quartz-carbonate veins and dislocated lenses. The veins are blue-grey to white in color, comprise up to 30% of the rock, and average one centimetre in width. The carbonatized volcanic host rocks contain 1 to 5% disseminated, medium- to coarse-grained euhedral pyrite cubes. Localized zones of strong patchy silicification commonly return high-grade assays. Free gold has been observed to occur erratically within the quartz veins.

Mineralogical associations within the gold zones include leafy native gold, molybdenite, chalcopyrite, pyrite, pyrrhotite, sphalerite, platy ilmenite, axinite, quartz, calcite, and chlorite. Molybdenite and/or chalcopyrite are found intimately associated with coarse clusters of native gold within the high-grade single-vein structures.

7.2.3 Other Mineralization at the Marban Project Property

The drill intersections discussed below are only examples of some significantly mineralized core, 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.

7.2.3.1 North-North Zone

The North-North Zone is located 700 metres northwest of the Norlartic mine shaft (Figure 7-3) and is a near-surface intrusive-hosted deposit with mineralized quartz-tourmaline stockwork. The North-North mineralization has strong similarities to Agnico-Eagle's Goldex deposit, located 10 kilometres on trend to the southeast.

Gold mineralization is confined to a conformable quartz-albite-carbonate-pyrite alteration envelope with quartz-tourmaline-carbonate vein stockwork localized in the central to lower portions of a 60-metrewide 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 40º to 55º to the northeast. Based on very limited drilling below 200 metres, the dip of the granodiorite apparently abruptly steepens at depth, and the alteration envelope becomes thinner and less developed. The zone is somewhat irregular, with an average thickness of 4.5 to 6 metres. It can be followed on strike for about 800 metres and to a depth of about 150 metres, with deeper exceptions (Ducharme, 2007).

The best grades and thicknesses of North-North Zone mineralization occur in a near-surface (45 to 105- metres depth) northeast-dipping zone of gold-bearing quartz-tourmaline stockwork mineralization hosted within the North-North granodiorite intrusive sill. As indicated by Osisko drilling in 2006, this mineralized zone dips 40 to 55º to the northeast, plunges shallowly in a westerly direction, is up to 15 metres in width, and shows continuity in both strike and dip. 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, which is consistent with the shallow plunge of the zone.

Sulfide mineralization – mainly pyrite with minor chalcopyrite and pyrrhotite – occurs as disseminations and coarse clusters. Gold typically occurs as isolated particles averaging two millimetres 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, 2008). Proximity of sheared and carbonatized mafic dikes may also characterize the North-North Zone, although they have not always been sampled, even though some are gold bearing (Ducharme, 2008). The presence of coarse gold creates a nugget effect, which complicates the collection of representative assays.

Five additional holes were drilled to test the strike and depth extent of the North-North Zone gold mineralization in 2006. Holes NL-06-22, NL-06-23, and NL-06-31 were drilled to investigate the westerly down-plunge extent, and holes NL-06-24 and NL-06-25 investigated the easterly strike and depth extensions of the North-North Zone. Thick well-mineralized alteration and quartz-tourmaline stockwork zones, typical of the North-North Zone, were intersected in holes NL-06-22 and NL-06-24.

Osisko's initial drilling at the North-North Zone confirmed the continuity of the gold-bearing lenses. Initial step-out holes have indicated that the North-North Zone lies within a large continuous gold system currently defined over a strike length of 600 metres and to a vertical depth below 200 metres. The gold grades appear to be affected by a coarse free-gold component, causing a nugget effect. This characteristic of the mineralization is supported by

the fact that total metallic-sieve assay results for higher-grade values obtained from the 2006 drilling indicate that gold is mainly contained within the coarse fraction (+150 mesh). Coarse native gold was noted in a petrographic study of five thin sections of North-North Zone mineralization.

7.2.3.2 North Zone

The North Zone mineralization is associated with the North shear zone, which is located 350 metres north of the Norlartic shaft and strikes northwesterly across the First Canadian and Norlartic properties (Figure 7-3). During the 1960s, the North Zone was accessed through the Norlartic mine workings, and some mining of the North Zone mineralization was undertaken.

Gold mineralization at the North Zone is similar to that at the Marban mine, located 1,600 metres to the southeast. Three sub-parallel gold-mineralized zones dipping 60º to the northeast are recognized within the North shear (the A, B, and Footwall zones; the Footwall zone is also called the “C” zone in some Osisko news releases) and are confined to zones of quartz-carbonate veining and pyrite alteration within sheared mafic volcanic rocks (Stuart and Martin, 1988). The A zone is erratically high grade, probably due to coarse gold. Highest grades occur where the A and B zones appear to converge in the near surface. According to Mannard (1986), the A and B zones are structurally controlled, and they appear to plunge at shallow angles toward the east. The A and B zones were mined above the 500 level, while the Footwall zone was drifted on two levels but was considered to be too discontinuous to mine.

The zones typically occur as clusters of foliation-parallel, milky white to blue-grey quartz veins, each less than 30 centimetres wide. The zones range between 0.6 metres and 3.0 metres wide, and quartz veins may comprise up to 50% of the strongly sheared, silicified, and carbonatized mafic volcanic rock. Both the host rocks and quartz veins carry up to 5% pyrite, which occurs as fine to medium-grained disseminations or as fine-grained wisps and streaks oriented parallel to schistosity.

Historic surface drilling by Aur from 1985 to 1989 returned significant intersections along strike and at depth from the A and B Zones underground workings. The drilling extended gold mineralization within the North Zone Shear over a strike length of 700 metres and to a vertical depth of -400 metres.

Osisko completed six holes (NL-06-26 to NL-06-30 and NL-06-33) on the North Zone in 2006 in order to confirm the historic intersections obtained by Aur. Five holes (NL-06-26 to NL-06-30) were completed on sections 900 W and 1100 W, located 60 metres and 120 metres west of the North B zone stopes.

Hole NL-06-33 targeted an area 100 metres below the North B zone stopes at a vertical depth of 250 metres and returned a high-grade intersection over 1.3 metres core length (true width is uncertain). This intersection occurs within a previously unrecognized structure located in the hanging wall of the A zone.

Osisko completed 17 new holes and extended one of its 2006 holes in 2012-2013, testing the A, B, and C zones of the North Zone (Osisko news release, April 4, 2013).

7.2.3.3 Gold Hawk Zone

The Gold Hawk zone lies within the Marbanite shear and is exposed at the Gold Hawk claim group of the Marban Project property. The Marbanite shear extends beyond the Gold Hawk

property across the First Canadian and Marban claim groups on the surface and is present at depth beneath the Norlartic claim group.

High-grade gold veins at Gold Hawk were outlined by the drilling programs of Thunderwood Explorations Ltd., Gold Hawk Resources Ltd., and Villebon Resources Ltd. in 1980 to 1983 and 1987. Drilling at 30-metre section spacings and to a vertical depth of 400 metres identified three veins or vein systems named Vein #1, #2, and #3.

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 to 90-metre-thick mafic volcanic and intrusive sequence. The volcanic rocks are relatively massive, although they lie directly adjacent to the Marbanite 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 horizon, and these most commonly occur on the hanging-wall side of the mafic volcanic package. Individual gold-bearing quartz veins range from four centimetres to 50 centimetres wide. The only exception to this was encountered in hole 2001-112, where a zone approximately six metres wide of almost continuous quartz was intersected. In this case, economic gold grades were only obtained within the first 0.3 metres 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. Veins with visible gold were observed in 40% of the 1987 drill holes.

Concurrent with its mining of Vein #1 (see Section 6.1.2.6), Aur tested the Gold Hawk horizon by surface drilling on the adjoining First Canadian and Norlartic claim groups over a strike of 1.2 kilometres. The drilling was mainly done on 60-metre section spacings, although limited underground drilling was also conducted from a crosscut driven from the 400-metre level of the Kierens mine.

Osisko drilled 15 short holes in November 2006 for a total of 1,379.3 metres to further test the potential of Vein #2. The drilling concentrated on testing the vein system at shallow levels along a southeast-plunging trend of high-grade gold outlined by historic drill-hole intersections. The vein was tested over a distance of 120 metres and a vertical depth of 100 metres. Nine of the holes intersected numerous quartz tension veins, while the other six holes intersected both quartz tension veins and quartz stockwork zones. The veins have variable orientations.

The Osisko work indicates that the Vein #2 system is hosted within a massive non-foliated intermediate intrusion. Three holes (GH-06-001, 002 and 005) yielded high-grade gold values associated with specks of visible gold (“VG”) and four holes (GH-06-003, 010, 013, and 015) yielded gold values above 2 g/t.

Osisko believes that the 2006 drilling of the Gold Hawk Vein #2 system proved the difficulty of evaluating vein systems characterized by coarse native gold.

7.2.3.4 Marban Northeast

Geophysical data suggest a 1.1 kilometre stretch of the North Shear, which hosts the North Zone mineralization, exists in the northeast corner of the Marban claim group. Norbenite Malartic Gold Mines first tested the southeast extent of the North Shear structural zone at Marban Northeast in 1946 with a single drill hole (MBS-51) that returned a value of 3.09 g Au/t over 0.3 metres within a “syenite” dike that crosscuts mafic volcanic rocks. No gold was found associated with the sheared rocks encountered. According to the historic data available, no other work was completed in the area until 1987, when Lac completed nine drill holes that returned some high-grade gold values.

During October 2007, Osisko created drill-hole cross sections at 100-metre spacings at Marban Northeast using the historic drill data. This study led to the following observations:

- Mafic and ultramafic volcanic units are intruded by numerous “diorite” and “syenite” dikes;
- Units dip approximately 60° to the northeast
- A 100-metre wide magnetic low feature associated with the North Shear is suggestive of a Z-fold pattern, which could explain the geological repetitions
- Two sub-parallel, steeply north-dipping gold zones are outlined
- Gold mineralization is associated with intensely altered ‘bleached’ intrusions (albite, silica, and pyrite) close to mafic/ultramafic contacts

In 1987, Lac completed 10 drill holes at Marban Northeast. Significant results from this drilling are summarized in Table 7-1

Osisko drilled two holes in 2007 (MB-07-021 and MB-07-022) as an initial investigation of Marban Northeast. The 2007 holes were located on the same section as Lac holes MBS87-01 and MBS87-02. Follow-up drilling was completed in early 2010 with holes MB-10-100 and MB-10-101, which were drilled 100 metres more to the southeast. Significant results from the four Osisko holes are summarized in Table 7-2.

Hole MB-07-021, drilled between holes MBS87-01 and MBS87-02, encountered variably altered and mineralized rocks with quartz-tourmaline veins (North-North Zone type). The intervening volcanic rocks were also found to be mineralized. Mineralization consists of disseminated pyrite and chalcopyrite. Hole MB-07-022, drilled beneath MBS87-02, encountered three similar intersections. Hole MB-10-101 intersected low-grade mineralization hosted by sheared basalt that is intruded by several dikes, and it confirmed the presence of a high-grade shoot within the low-grade envelope.

Table 7-1: Results of Drilling at Marban Northeast by Lac

Hole #	Line (m)	Station (m)	Az	Dip	Depth (m)	Geological Unit	From (m)	To (m)	Core Length (m)	Grade (g/t)
MBS-51	17+00E	5+91N	210°	-45°	276.8	Siliceous Diorite dykes	94.2	94.6	0.4	3.09
							186.6	187.3	0.7	1.37
MBS-87-01	20+00E	5+53N	210°	-43°	197.2	Altered diorite	69.7	70.0	0.3	27.50
						Altered diorite	73.3	75.8	2.5	7.99
						Including	74.1	75.0	0.9	21.20
MBS-87-02	20+00E	7+62N	210°	-53°	331.3	Altered diorite	145.7	146.9	1.2	9.17
						Altered diorite	166.1	166.7	0.6	2.57
						Altered diorite	292.2	292.9	0.7	2.06
MBS-87-03	22+00E	5+86N	210°	-54°	197.2	Altered diorite	144.5	145.5	1.0	13.54
MBS87-04	22+00E	7+39N	210.5°	-54°	267.3	Altered diorite	118.9	122.4	3.5	1.15
						Including	119.3	119.8	0.5	5.31
						Altered diorite	143.3	146.9	3.6	1.65
						Including	145.4	146.9	1.5	2.91
MBS87-05	17+00E	3+10N	210°	-45°	199.9	No significant results				
MBS87-14	19+00E	5+43N	210°	-45°	214.9	Carbonatized zone	106.2	107.8	1.6	9.26
							111.9	112.8	0.9	1.03
MBS-87-15	19+00E	6+29N	210°	-45°	306.3	Altered dyke	68.6	70.1	1.5	1.37
						Altered dyke	178.9	180.4	1.5	1.71
						Altered dyke	189.0	189.8	0.8	1.71
MBS87-16	19+00E	5+91N	210°	-80°	193.6	Altered diorite	19.1	20.5	1.4	1.37
						Altered diorite	26.0	27.3	1.3	2.40
						Including	26.0	26.6	0.6	3.94
						Altered diorite	39.1	40.1	1.0	1.89
MBS87-17	23+00E	6+57N	210°	-47°	187.5	Altered diorite	22.2	23.1	0.9	3.43

Table 7-2: Results of Drilling at Marban Northeast by Osisko

Hole #	Line (m)	Station (m)	Az	Dip	Depth (m)	Geological Unit	From (m)	To (m)	Core Length (m)	Grade (g Au/t)
MB-07-021	20+00E	6+62N	213°	-45°	285.0	Altered felsic dyke	31.7	33.0	1.3	2.48
						Altered felsic dyke	62.2	66.0	3.8	2.38
						Including	64.8	66.0	1.1	3.12
						Altered felsic dyke	74.9	77.3	2.4	1.93
						Including	74.9	76.0	1.1	3.12
						Pillowed basalt	107.0	108.0	1.0	1.09
						Basalt-Qz vein-Py	251.0	253.0	2.0	3.24
						Including	251.0	252.0	1.0	5.48
MB-07-022	20+00E	7+52N	213°	-66°	405	Altered felsic dyke	158.0	159.2	1.2	1.86
						Altered felsic dyke	167.7	168.7	1.0	1.90
						Altered felsic dyke	194.0	195.0	1.0	20.70
MB-10-100	21+21E	6+55N	217°	-45°	44	No significant results				
MB-10-101	21+21E	6+53N	217°	-45°	286.3	Altered felsic dyke	54.6	75.0	20.4	0.41
						Basalt-Qz vein-Py	218.9	220.2	1.3	31.20
						Basalt-Qz vein-Py	226.9	228.1	1.2	5.22

8 Deposit Types

The Marban Project property lies within the Malartic mining camp and is in close proximity to the Val-d'Or camp. These mining areas have a combined historic production of over 25 million ounces of gold and continue to produce gold today; historic production has been primarily from underground mines, although significant open-pit production is evident at the present time. The Malartic mining area is noted for its variety of gold occurrences, including such major gold deposits as the East-Malartic, Camflo-Malartic Hygrade, Malartic Gold Fields, Barnat-Sladen, and Canadian Malartic. These mines yielded 8.9 million ounces of gold from 1935 to 1984.

Gold mineralization from these mining districts has been classified as mesothermal or late-orogenic lode deposits. The mineralization is associated with regional structural zones (e.g. the Cadillac-Larder Lake tectonic zone), regional drag folds, and structural splays, as well as with syn- to late-tectonic intrusive rocks. The different styles of mineralization range from disseminated-sulfide deposits to quartz tourmaline gold-bearing veins and vein stockwork zones. Deposits range from early- to late-tectonic in age.

The Marban property is similar in many respects to mines and deposits in the surrounding area. The Norbenite and Marbanite shears are major structures that range from tens to hundreds of metres in width. The presence of lamprophyre intrusions within or adjacent to these structural corridors is suggestive of a deeply rooted deformation system. These shear zones are affected by at least two major deformation episodes that caused both open and very tight folding. The Marbanite shear hosts mineralization at Marban, which is a multi-folded mesothermal-type deposit. The high-grade Gold Hawk veins also occur within this shear corridor. There are numerous gold occurrences encountered between Marban and Gold Hawk, as well as along strike, including some at the Malartic Hygrade mine.

The Kierens and Norlartic mineralization is confined to the Norbenite shear and is associated with intermediate dikes that host auriferous stockworks. This shear may also host the adjacent Kiena and Goldex mines. An additional style of mineralization occurs as gold-bearing quartz-tourmaline stockworks within larger intrusions, such as the North-North granodiorite. There are a number of styles of mineralization identified at the Marban property, including gold associated with: (1) sheared basalts; (2) quartz-pyrite veinlets within or adjacent to dioritic dikes; (3) quartz-tourmaline stockwork veinlets within granodiortic intrusions; (4) quartz + chalcopyrite veins; and (5) strongly altered felsic dikes. Gold mineralization hosted within shear zones in basaltic rocks is the most important type, and it is found at the Marban and North Zone deposits as well as at Marban Northeast and in the northwest-trending portion of the Marbenite shear zone. This style of mineralization is accompanied in various degrees by silicification, albitization, quartz-carbonate veinlets, and/or disseminated pyrite and pyrrhotite.

Mineralization associated with dioritic dikes dominates the Kierens and Norlartic deposits, although this style also occurs at the North Zone and locally in other deposit areas. Quartz-tourmaline mineralization is best exemplified at the North-North Zone, with similar mineralization also intersected in granodiorite that occurs in the hanging wall portion of the Marban deposit. The Gold Hawk zone is unique in that its gold mineralization, which can be quite high grade, is associated with quartz veins (as opposed to veinlets). The final style of gold mineralization at the Marban Project is found in the hanging wall of the Marban deposit and also at Marban Northeast; it is characteristically associated with highly altered (albitized and silicified) felsic dikes and disseminated pyrite with or without quartz-pyrite veins.

9 Exploration

The following information is taken from the original technical report (Carrier, 2006) with updated information added from Lefrançois (2006, 2007), Ducharme et al. (2009), Gustin (2013) and additional references as cited. Exploration described in this section includes only work done by Osisko. Exploration work done by previous companies is presented in Item 6.

Osisko consolidated and acquired the Marban Project property early in 2006. The company's first efforts on the property consisted of compilations of historical reports, maps, cross-sections, longitudinal views, prior underground work, a database of over 3,800 surface and underground drill holes, and geophysical results with the intention of building a three-dimensional model of the known gold zones.

MRB & Associates Mining Consultants of Val-d'Or completed three-dimensional computer models of the mine workings at the Norlartic mine, North Zone, Kierens mine, and Gold Hawk zone Vein #1 workings. Osisko personnel completed an in-house three-dimensional computer model of the Marban mine workings.

In September 2006, a high-resolution airborne magnetic survey was flown by Prospectair Inc. over the Marban Project property, as well as claims on the adjacent Osisko Malartic Hygrade property. The survey, which consisted of a total of 602 linear kilometres, was flown at 50-metre line spacings.

J. L. Coriveau and Associates Inc. surveyed the Norlartic, Kierens, and Marban shaft collars, as well as several control points, using a real-time high-precision GPS. In addition, they surveyed 26 Aur drillhole casings identified on the North-North and North Zones. The survey points were used to merge the drill-hole databases from the various properties into a project-wide coordinate system.

Abitibi Geophysics of Val-d'Or conducted an orientation induced polarization ("IP") survey for Osisko designed to detect near-surface gold-bearing disseminated pyrite mineralization intersected in Osisko holes MB-06-001 and MB-06-002 in the Marban mine area (Lefrançois, 2007). One 1,250-metre 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).

In 2006, a petrographic study of the gold mineralization in the North-North Zone was undertaken (Ducharme et al., 2009, citing Renou, 2006), and in 2007 a similar study was done on the mineralization and alteration from hole MB-06-001 on the Marban zone (Ducharme et al., 2009, citing Renou, 2007). 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.

In 2008, Technologies Earthmetrix conducted a structural study of the Marban Project property using LANDSAT imagery (Ducharme et al., 2009). This study has been used to generate exploration targets and to interpret structures.

Review of all project data, including pre-Osisko core and results from resampling of the older drill holes, led to development of multiple drill targets on the property. Osisko initiated a drill campaign in the summer of 2006, and by late November 2006 had drilled approximately 10,000 metres in 64 drill holes. This drilling was mainly intended to assess historic "resources"

previously outlined in the undeveloped North-North, North, and Gold Hawk zones, as well as the Kierens deposit (Davy, 2008).

From April 2007 to December 2008, Osisko drilled an additional 94 holes and two wedges for a total of over 33,000 metres. Little exploration work was conducted in 2009 on the Marban Project property. Osisko completed a fence of four core holes (1,736 metres) one kilometre east of the Marban shaft, about 450 metres east of any previous Osisko holes, late in the year.

Osisko drilled another three holes (905.6 metres) 900 metres west of the shaft at the Marban deposit early in 2010, with another two holes (330.3 metres) drilled to test the Marban Northeast zone. Drilling by the Osisko-Aurizon joint venture began in August 2010 with a Phase I program that included 146 new holes and extensions of eight holes previously drilled at and just north of the Marban deposit. The Phase I program was completed in August 2011, and the holes were included in the 2012 SGS resource update of the Marban deposit (Camus et al., 2012). The joint venture completed an additional 90 new holes and extensions of another nine previous holes at the Marban deposit in the 2011-2012 Phase II exploration program, bringing the total number of holes drilled by Osisko and the joint venture at the Marban deposit to 337, plus extensions of 17 previously drilled holes, for a total of 116,142 metres.

The joint venture also completed 16 holes at the Kierens deposit in February and March 2013, 24 holes at the Norlartic deposit in 2011 as part of the joint venture's Phase I program, plus one hole in 2013, and

17 new holes plus one extension of a previously drilled hole at the North Zone in 2012 and 2013.

In 2014, four holes were drilled at Kierens and six at Norlartic (representing 1,646 m and 1,368 m respectively) by Osisko. Also, in 2014-2015, 246 supplementary holes representing 72,489 metres were drilled at Marban to complete the drilling pattern to about 25 m by 50 m.

O3 Mining's exploration efforts focused primarily on the Marban mine, Norlartic Main, North-North, North, Kierens, and Gold Hawk zones. Additional details on O3 Mining's drilling program can be found in Section 10.0.

10 Drilling

10.1 Summary

O3 Mining provided MMTS with the drill-hole databases for the Marban Project as csv files from their GEMS database. The files were received in two sets: one that covers the northern portion of the Marban Project property, including the North-North, North, Kierens, Norlartic, and Gold Hawk zones, and a second that includes the Marban zone. These project databases include collar location, down-hole survey, assay, lithologic, alteration, mineralization, geotechnical, and structural data.

Table 10-1: Summary of the Marban the Databases

Year (approx)	Owner	Type	Hole Series	Driller	Core Size	Number of holes	Length (m)	Assayed Length (m)	Re-assayed length (m)
1945	Little Long Lac	Surface	CL			5	1,424.3	393.5	-
1959	Little Long Lac	Underground	CU			24	2,243.9	663.0	-
1959	Little Long Lac	UG_TH	CU			3	153.9	33.0	-
1960	Marban Mines	Surface	MB			3	100.3		-
1960	Marban Mines	Surface	MBS			82	22,647.6	7,036.4	1,028.2
1960	Marban Mines	UG	MB			1,103	55,362.0	23,521.5	-
1960	Marban Mines	UG_TH	MB			66	3,242.7	1,424.4	-
? 1960s	Marban Mines	UG	8-D			6	152.7	130.8	-
1986	Lac Minerals	Surface	MBS		BQ	8	1,248.6	906.5	364.1
1987	Lac Minerals	Surface	MBS		BQ	6	2,207.4	2,060.4	1,157.7
1988	Lac Minerals	Surface	MBS		BQ	3	1,353.0	1,273.1	493.3
1993	Aur Resources	Surface	501-			3	1,168.4	294.2	-
1993	Aur Resources	Surface	M-93			1	10.0	8.0	-
2006	Osisko	Surface	MB	Forage	NQ	2	1,116.0	1,005.3	-
2007	Osisko	Surface	MB	Forage	NQ	27	11,818.9	10,493.0	-
2008	Osisko	Surface	MB	Forage	NQ	61	23,364.2	18,356.7	-
2010	Osisko	Surface	MB	Forage	NQ/BQ	45	13,253.4	11,743.7	-
2011	Osisko	Surface	MB	Forage	NQ/BQ	94	28,956.7	25,403.3	-
2012	Osisko	Surface	MB	Forage	NQ/BQ	79	26,946.0	24,375.4	-
2013	Osisko	Surface	MB	Forage		1	438.0	406.0	-

Year (approx)	Owner	Type	Hole Series	Driller	Core Size	Number of holes	Length (m)	Assayed Length (m)	Re-assayed length (m)
2014	Osisko	Surface	MB	Forage		36	12,665.8	11,312.5	-
2015	Osisko	Surface	MB			219	61,614.4	51,147.0	-
2016	Osisko	Surface	MB			1	558.0	288.6	-
2017	Osisko	Surface	MB			4	729.0	557.4	-
(blank)		UG_Chips	(blank)			1,356	9,183.4		-
Total						**3,238**	**281,958.6**	**192,833.5**	**3,043.3**

Table 10-2: Summary of the Kierens-Norlartic the Databases

Year (approx)	Owner	Type	Hole Series	Driller	Core Size	Number of holes	Length (m)	Assayed Length (m)	Re-assayed length (m)
(blank)		Surface	FC-1 to FC-33			34	4,580.5	1,076.6	-
1944	Norbenite Malartic	Surface	NM	Walter Hargrave		34	5,267.6	824.4	-
1945	Norbenite Malartic	Surface	NM	Walter Hargrave		32	6,371.7	1,610.2	-
1958	Malartic Goldfields	Underground	N			272	16,153.0	10,828.6	115.9
1958	Malartic Goldfields	UG_TH	N			194	6,461.8	3,516.8	-
1960		Surface	MBS			15	2,574.5	415.6	-
1964	Norlartic Mines	Surface	NS			47	10,995.7	2,509.8	458.4
1980	Brominco Inc	Surface	BV-80	Forage Dominik		32	3,325.4	1,547.0	126.4
1980		Surface	NS-80			7	1,492.0	193.4	1.5
1981	Brominco Inc	Surface	BV-81	Forage Dominik		30	3,150.8	1,823.3	-
1981		Surface	THD/1981			9	1,179.6	423.1	96.0
1983		Surface	VM			8	610.8	97.8	6.1
1985		Surface	85-			5	1,073.5	160.0	-
1986-1990 (?)	Aur	Underground				558	12,322.8	4,457.6	-
1986	Aur	Surface	1902-	Forage Benoit - Orbit	BQ, NQ	144	35,854.5	11,775.4	1,592.4
1986	Aur	Surface	2001-	Forage Benoit	BQ	180	43,379.7	12,486.2	1,024.9
1987	Aur	Surface	1950-	Forage Benoit	BQ, AW34	63	16,186.0	2,854.0	307.9
1987		Surface	MBS			4	880.9	743.2	-
1987	Aur	Underground	GU	Forage Orbit	AGM	21	2,558.8	1,202.2	210.7
1987	Aur	UG_TH	GU	Forage Orbit	AGM	6	672.8	578.2	-
1988	Aur	Underground	NU	Dominik - Orbit	BQ	292	38,780.1	13,251.2	3,817.5
1988	Aur	UG_TH	NU	Dominik - Orbit	BQ	34	3,879.7	1,661.0	440.1

Year (approx)	Owner	Type	Hole Series	Driller	Core Size	Number of holes	Length (m)	Assayed Length (m)	Re-assayed length (m)
1989		Surface	199-			1	454.2	26.9	-
1990	Aur	Surface	OVB	Forage Benoit	NQ	42	724.6	37.4	-
1990	Aur	Underground	FU	Morissette Drilling - Orbit		282	38,804.2	15,173.9	3,466.5
1990		UG_TH	FU			68	4,584.7	2,174.5	545.3
1993		Surface	501-			1	365.2	80.5	-
2006	Osisko	Surface	FC			10	1,465.2	874.2	-
2006	Osisko	Surface	GH/NL			53	8,355.9	3,814.7	-
2007	Osisko	Surface	NL			5	1,104.0	434.9	-
2011	Osisko	Surface	MB/NL			31	6,871.6	5,755.4	-
2012	Osisko	Surface	MB/NL			9	3,902.0	3,642.5	-
2013	Osisko	Surface	FC			16	7,717.4	7,202.5	-
2013	Osisko	Surface	MB/NL			16	3,859.6	3,430.1	-
2014	Osisko	Surface	FC			4	1,646.0	941.6	-
2014	Osisko	Surface	NL			6	1,368.0	1,179.0	-
2016	Osisko	Surface	MB/NL			6	4,673.7	299.4	-
Total						**2,571**	**303,648.5**	**119,102.9**	**12,209.8**

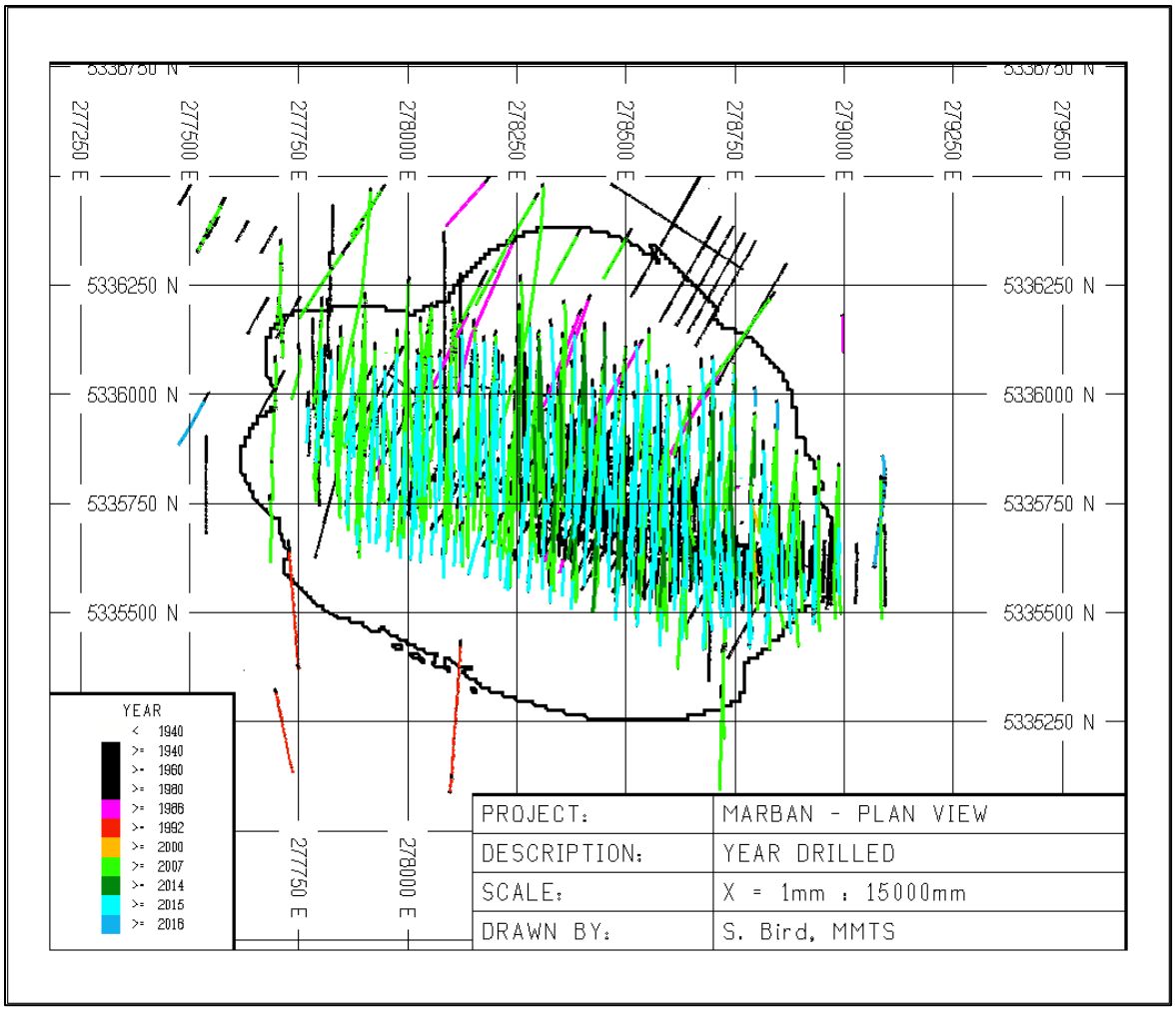
O3 Mining drilling is mainly oriented 180° to 220° with dip generally between 45° and 65°. This orientation is more or less perpendicular to the main trend of mineralisation. The historic underground drill holes have highly variable azimuths and dips, as they were drilled in series of fans from individual drill stations, which creates a multitude of angles between drill holes and mineralization. Some of them may then be more or less parallel to the mineralization and be not representative of true mineralized width. However, this impact is mitigated by the modeling technique that constrains all intercepts within interpreted domains that respect the true mineralized widths.

The drill-hole data were reviewed in the context of the geology of the Marban, Kierens, and Norlartic mineralization, and the QP believes that the sample lengths are generally appropriate for the style of mineralization in each target area.

Figure 10-1 illustrates all the drillholes within the Marban database by year drilled, also showing the Marban resource pit outline for reference. It is noted that drilling prior to 1986 (shown in black) has not been used in the resource estimate due to detection limit issues for these historic dates. It is seen that there remains good drill density throughout the Marban deposit with drilling from 1986 to the present.

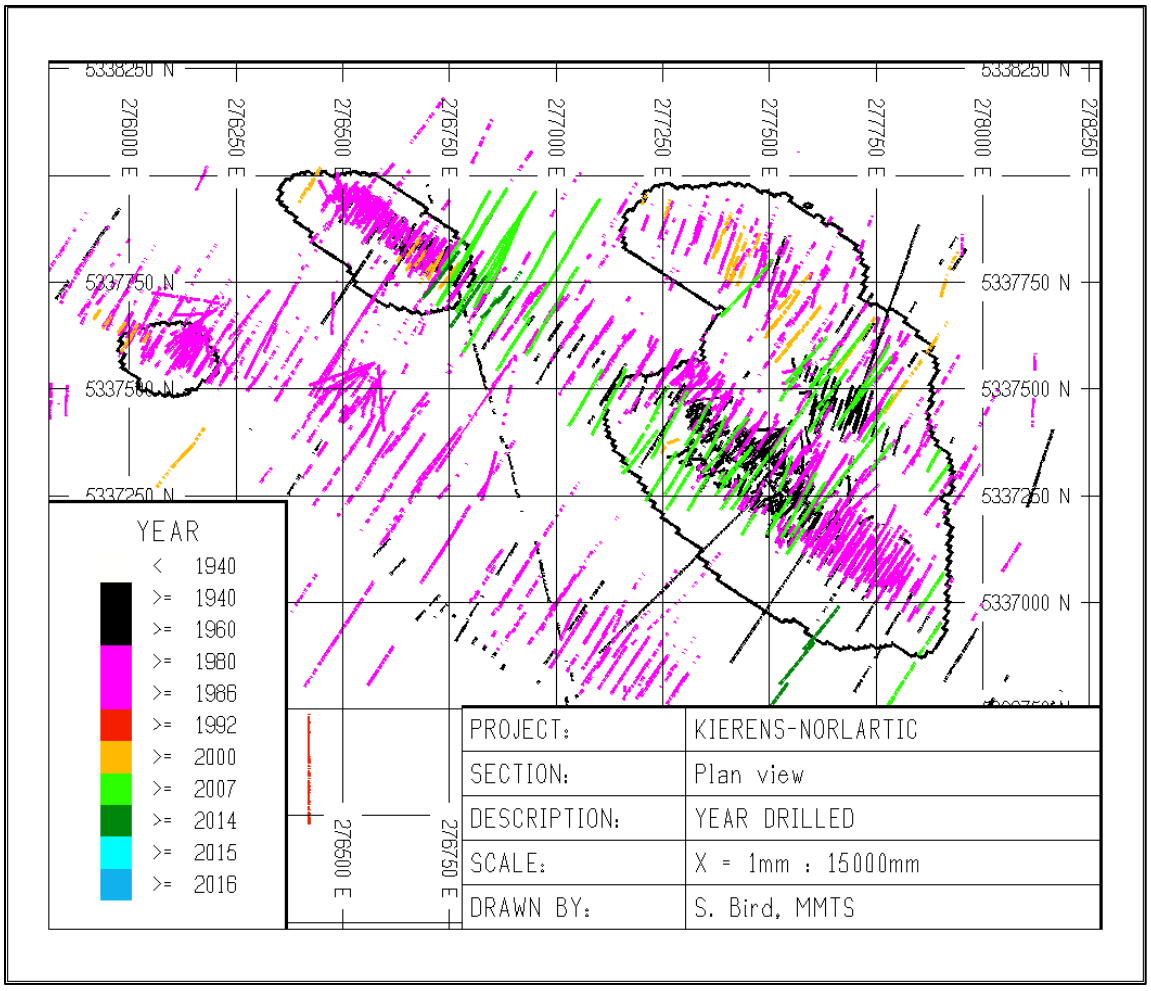
Figure 10-2 illustrates all the drillholes within the Kierens-Norlartic database by year drilled, also showing the Kierens, Norlartic, gold Hawk and North zone resource pit outlines for reference. It is noted that drilling prior to 1980 (shown in black) has not been used in the Kierens-Norlartic area resource estimate due to detection limit issues for these historic dates.

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



Source: MMTS, 2020

Figure 10-2: Plan View of Drillhole Database by Year - Kierens-Norlartic



Source: MMTS, 2020

10.2 Drilling – Historic and Osisko

Core size, dates and drill contractors for the pre-Osisko holes are derived primarily from copies of the original geologic logs of drill holes. The summary for historic holes is based on the audit done for the 2013 report (MDA, 2013).

Osisko's first drilling program at the Marban Project 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 metres, that tested Marban (two holes), Kierens (10 holes), Norlartic (two holes), Gold Hawk (15 holes), and North and North-North (35 holes) zones.

All 2006 holes were drilled with NQ-size (47.75 millimeters) 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 drill-hole casings as control points, as well as re-established field-grid baselines.

Down-hole azimuth, inclination, and magnetic-field measurements were taken using the FLEXIT Survey System every 30 metres (50 metres for longer holes) while drilling using a down-hole single-

shot instrument, and subsequently every three metres after completion of the hole with an up-hole multi-shot instrument, both operated by the drilling contractor (Davy, 2006, 2008; Lefrançois, 2007). The drill-hole collars were surveyed by J. L. Corriveau and Associates Inc. using a real-time high-precision GPS unit. Drill-hole casings were left in place when the sites were abandoned.

The drill core was taken by Osisko 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 Osisko geologists using GeoticLog® software before sampling. Various drilling parameters, including down-hole surveys, were also compiled into the database.

Osisko'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, three holes and two wedges at Norlartic, and one hole at Gold Hawk. Eighty-eight of the Marban deposit holes were drilled in a 50 metre by 50-metre 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 Osisko 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 drill contractor for Osisko'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. Osisko rented and operated the equipment to survey the collars from J. L. Corriveau and Associates Inc. The precision of the surveys is reported by Osisko to be about three centimeters for the easting and northing and about 30 to 50 centimeters for the elevation. As in the 2006 program, down-hole surveying was conducted by the drillers using the FLEXIT Survey System. Single-shot measurements were taken every 30 metres during drilling of the hole, and up-hole multi-shot measurements were taken every three metres when the hole was finished. In addition to the second Osisko phase of drilling, some intervals not previously sampled in 2006 drill holes MD-06-001 and MD-06-002 were sampled in 2007.

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

Osisko drilled 50 holes at the Marban deposit in 2010, with another 103 new holes and eight extensions of previous holes completed in 2011. In addition, they drilled 24 new holes at the Norlartic deposit in 2011 and one additional hole in 2013. Osisko completed 88 new holes and nine 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 drill 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. Osisko continued completing in-house drill-hole 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 metres. Since July 2011, they have used a Trimble Geo XH with a Zephyr antenna and with real-time correction; accuracy is 0.1 metre.

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 metres and multi-shot at every three metres. Osisko personnel enter the down-hole survey data into the project databases and then carefully review the results. Abrupt changes in azimuth and, less commonly, dip are flagged in the database as "invalid."

In 2014, four holes were drilled at Kierens and also four at Norlartic (representing 1,646m and 1,368m respectively). Also, in 2014-2015, 246 supplementary holes and one extension representing 72,672 metres were drilled at Marban to complete the drilling pattern to about 25m by 50m. Forage Spektra of Val-d'Or was the drill contractor for this program. Procedures for surveying and down-hole measurements remained the same than in 2012-2013.

10.3 Sampling Methods

10.3.1 Pre-Osisko Sampling

The following subsection on pre-Osisko sampling procedures is taken from the original technical report (Carrier, 2006). MMTS is unaware of any further information on pre-Osisko sampling at the Kierens, Norlartic, and Marban deposits.

There has been no systematic reviews of the sampling method and approach in the historical assessment work, but sampling techniques have varied through time and the different stages of activity on the Property. Sampling methods used during underground operations and exploration programs have included core with different size (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 (all deposits) and underground channels (Marban); no chip, muck, or sludge samples are in the project databases.

10.3.2 Kierens Sampling

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

Diamond drill 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. Diamond drill 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 diamond drill 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 in 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 two to four feet 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 cubic yard bucket of muck dumped on surface. This resulted in a sample for each round that weighed approximately one hundred 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 foot 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 eight feet into the wall of the drift and three samples were procured from each hole (two 2-foot samples followed by a 4-foot sample).

10.3.3 Osisko Sampling

The following summary is taken from the 2016 Ni43-101 report. Details on sampling methods during Osisko's drilling were provided by Osisko personnel (Yan Ducharme, 2016) and various news releases.

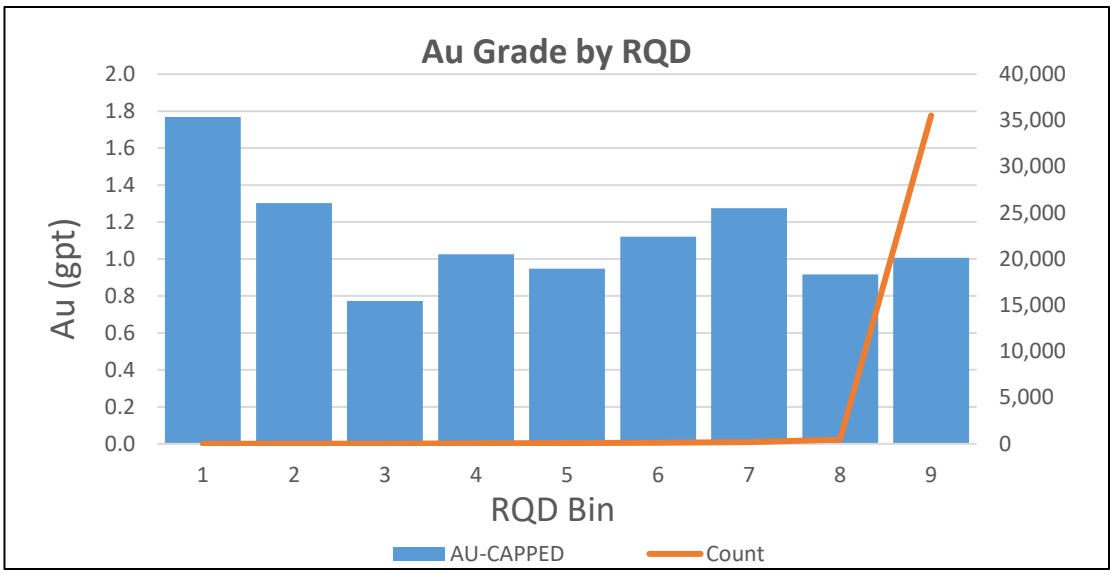
The NQ sized core from the drill was taken to either the Osisko office in Val-d'Or (2006-2007) or to Osisko's core facility located at the Norlartic property (2007-2015). The boxes were then opened by Osisko geologists and technicians. The core was photographed, measured and geologic and geotechnical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

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 is done by Osisko personnel or, when possible, by laboratory personnel.

10.4 Osisko Drill Core Sample Quality

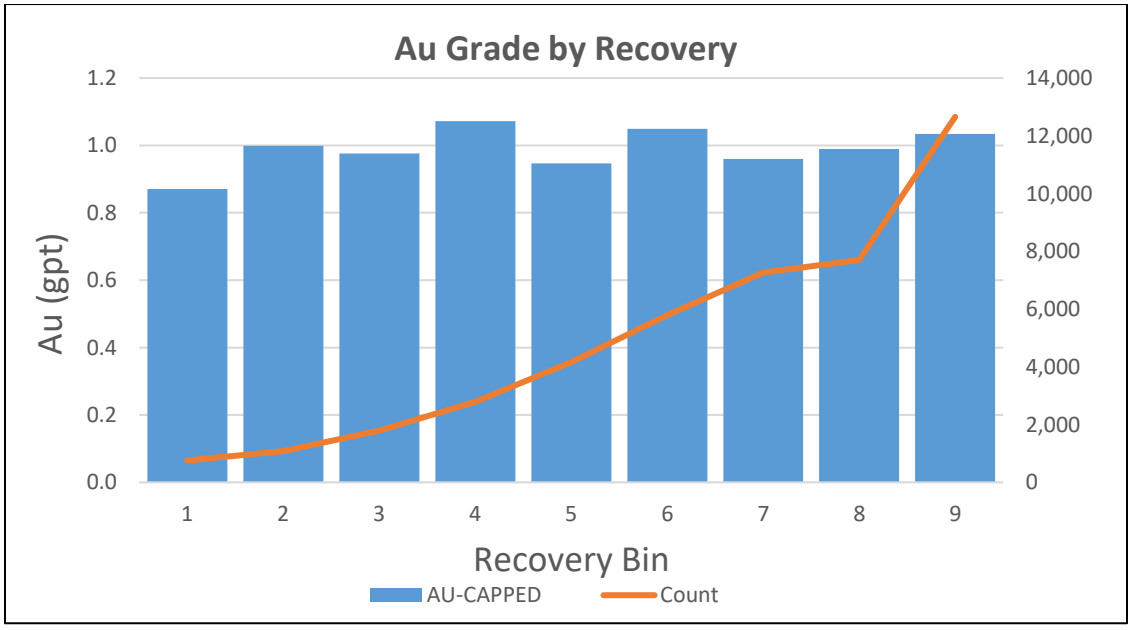
In Figure 10-3, the assayed gold grade is compared with their associated RQD values. Figure 10-4 compares gold grade with core recoveries. The bins are in 10% increments from 0%-100% in each case. As expected, the grades increase slightly with Recovery and lower RQDs generally have higher grades. The change in grade with recovery is not considered to be material and may be conservative for the lowest recovery bin of core with some apparent Au loss.

Figure 10-3: Gold Grade vs. RQD



Source: MMTS, 2020

Figure 10-4: Gold Grade vs. Core Recovery



Source: MMTS, 2020

10.5 Summary

MMTS believes that the drill-sampling procedures implemented at the Marban Project provide samples that are sufficiently representative and of sufficient quality for use in the resource estimations for drilling in the years that also contain sufficient QAQC or can be validated with re-assays as discussed in Section 11, 12 and 14 of this report.

11 Sample Preparation, Analyses & 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 quality assurance/quality control ("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 0.001 oz Au/ton 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 of 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, using standard fire assays. Assays were also done at Chimitec in Québec City, with some overflow 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 aliquots of one-assay-ton 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 0.01 oz Au/ton (0.343 g Au/t). Examination of the gold analytical data of Little Long Lac and Marban Mines from Marban drill holes, and Malartic Gold Fields, Norbenite Malartic, Norlartic Mines, and SOQUEM from Kierens-Norlartic holes, leads to the inference that the detection limit for these assays was also 0.343 g Au/t. The precision above the detection limit appears to have been 0.01 oz Au/ton as well, which led to a very large number of assays with grades that are multiples of 0.343 g Au/t.

11.1.1 NioGold Sample Preparation & Analysis

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 ("ALS")) 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; 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 finish ("FA-AA") on 30-gram or 50-gram pulp splits of 1,000-gram pulps.

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

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 two-millimetre sieve. After homogenization, an approximately 250-gram split was crushed to 85% passing a 75-micron sieve. Cleaner sand was run between all samples. Samples were analyzed by 50-gram FA-AA. For every sample with a result greater than 2 g Au/t, a second pulp from 250-gram 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 grams 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 two-millimetre sieve. After homogenization, an approximately 250-gram split was crushed to 85% passing a 75-micron sieve. Samples were analyzed by FA-AA on 50-gram pulp splits, with re-assaying of samples exceeding 2 g Au/t 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 Au/t were sent to Bourlamaque for check assaying.

Specific-gravity measurements were obtained on selected samples of core from mineralized intervals.

11.1.2 NioGold Sample Security

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.

The 2007-2013 core was cut and sampled at NioGold's core shack located at the Marban 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.

11.2 NioGold Quality Assurance / Quality Control (QAQC) Programs

According to Lefrançois (2006; 2007), both NioGold and the laboratories they used for assaying implemented quality assurance and quality control (QA/QC) programs to monitor the precision, accuracy, and reproducibility of the analytical method and results. For the 2006 drilling, standards were inserted every 20 to 25 samples in the sample batches sent to the laboratories, with a minimum of one standard per batch. At the beginning of the project, three different standards were used, with two other standards 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-gram FA-AA for samples that originally assayed from 2 to 5 g Au/t, and 50-gram fire assay with a gravimetric finish for samples originally assaying over 5 g Au/t.

If the laboratory returned more than a 15% deviation for the inserted standard 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, standard, blank, and duplicate-core samples were inserted into the stream at every 10-15 samples. Starting in February 2008, standard samples were inserted alternately at every 20th sample, with a minimum of one standard per batch; all sample numbers ending in 20, 40, 60, 80, and 00 were a standard. When assay results were received, the values obtained for every standard 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 standard. If the assay results were not acceptable, the lab re-assayed the whole batch of samples in which the standard 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 to NioGold approximately 5% of the drill-sample pulps chosen randomly by the labs. 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 as part of all subsequent drilling programs, protocols for choosing the sample pulps to be checked have varied. Results of the NioGold QA/QC programs follow.

11.3 NioGold / Osisko Era QAQC 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 modern era and is being considered with respect to contemporary QAQC practices of blanks, certified reference materials (CRM) and duplicates, blindly inserted with respect to the analyzing laboratories. All CRMs were obtained from Rocklabs Ltd of New Zealand.

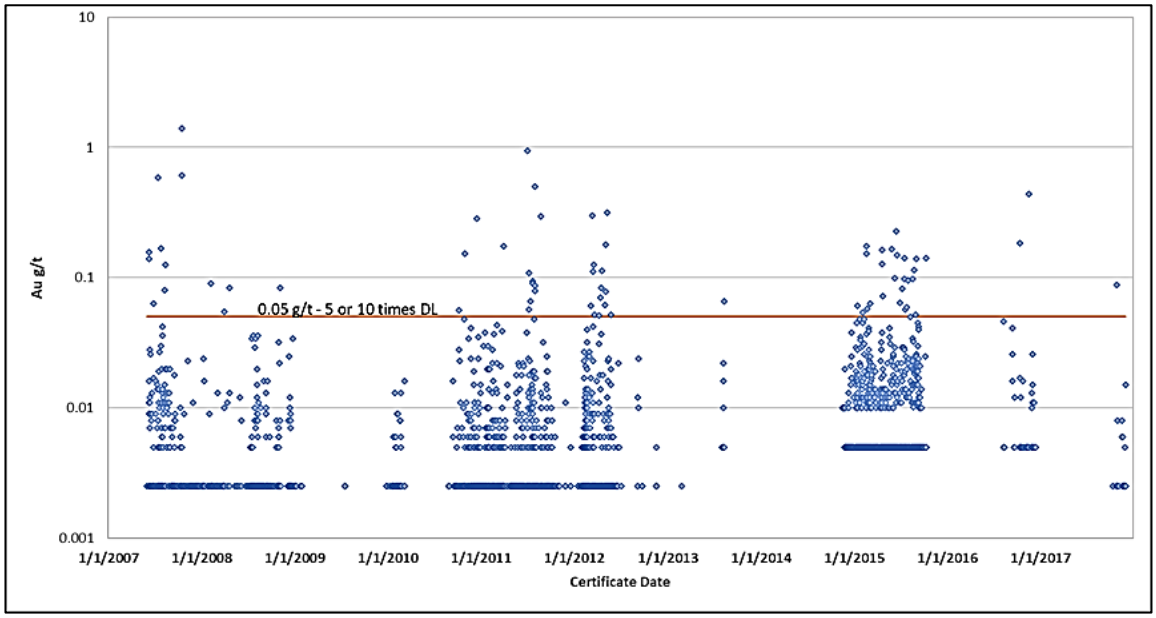
11.3.1 Marban Blanks

The database of QAQC samples 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 compositions (2017 Activity Report) Internal and previous reports indicate that failed blank samples are investigated in with the contracted lab and indicate that re-assays may have been requested on occasion, however, the QAQC database contains numerous failures and does not show a record of re-assay and replacement.

Figure 11-1 shows the assay values of the blanks plotted in order of the certificate date. Assays by ActLabs in 2016 and 2016 had a detection limit of 0.01 g/t compared with all other years at 0.05 g/t. 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 culprit in 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

11.3.2 Marban CRMs

The provided database of assays from 8558 blindly inserted CRMs indicates the targeted rate of 1 per group of 20 assays had been met. Analysis results of CRMs with more than 20 insertions, in order of increasing grade is presented in Table 11-1 below. It needs to be noted that the lowest grade CRM is 0.583g/t, and normally CRMs with lower grade would be included. It is observed that

the % error, coefficient of variation, and % of failed is 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 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 4 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-assay 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.

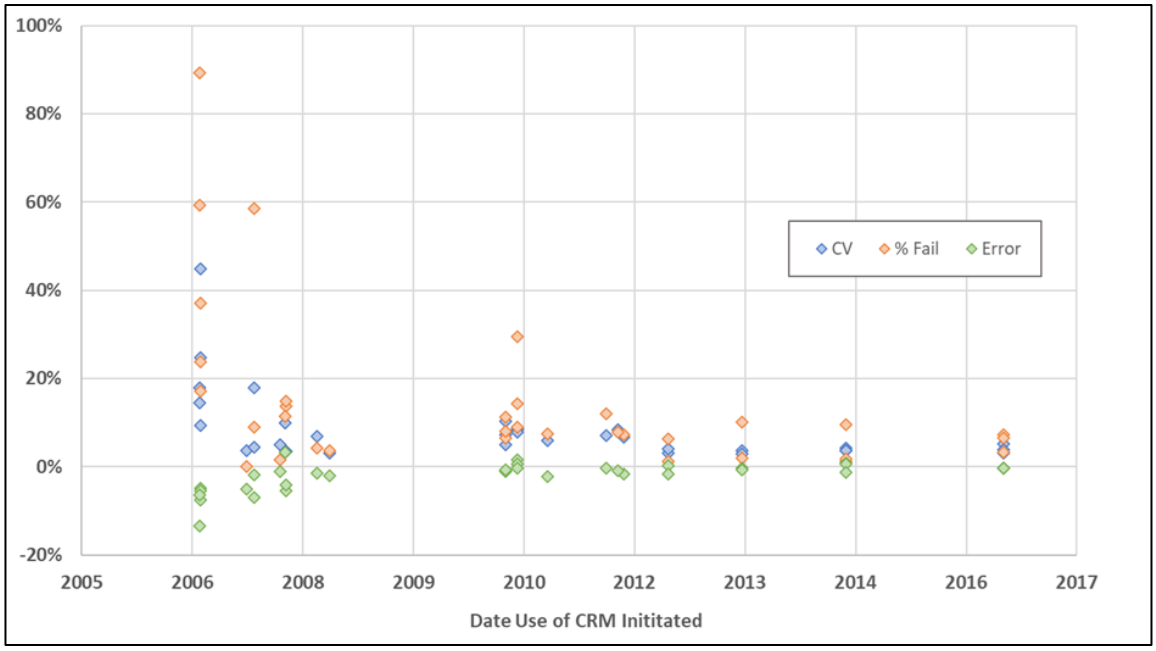
The statistics describing the accuracy of the assays are plotted in order of year that use of the CRM began, in Figure 11-2, and by grade, in Figure 11-3. It is observed that the greatest deviations occurred in years 2006 and 2007 and that accuracy improves over time. It is also seen that two of the 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	Std Dev Au Assay g/t	% Error	CV	Total Fails (+- 3 SD)	% Fail	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

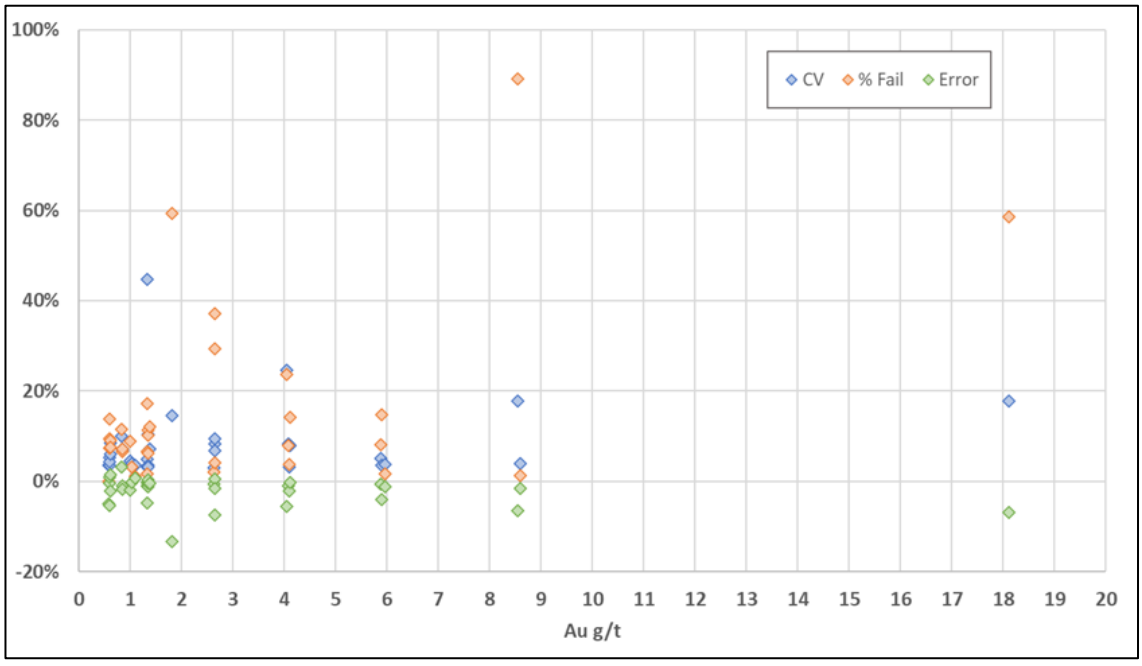
CRM	Total Insertions	Au Expected Value g/t	Au Average Assay g/t	Std Dev Au Assay g/t	% Error	CV	Total Fails (+- 3 SD)	% Fail	Used From	Used To
1006/SH24	29	1.326	1.265	0.594	-4.8%	44.8%	5	17.2%	2006	2007
1037/SH82	61	1.333	1.329	0.042	-0.3%	3.1%	4	6.6%	2016	2017
1019/SH41	486	1.344	1.329	0.139	-1.2%	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
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

Figure 11-2: Marban CRM Statistics by Year



Source: MMTS, 2020

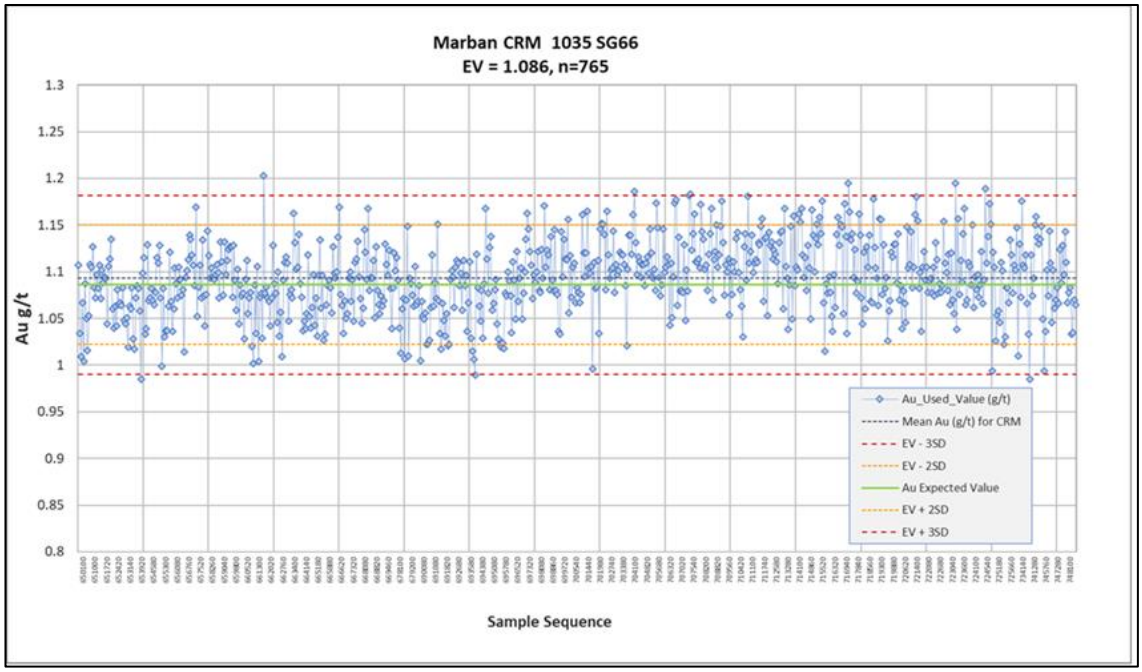
Figure 11-3: Marban CRM Statistics by Grade



Source: MMTS, 2020

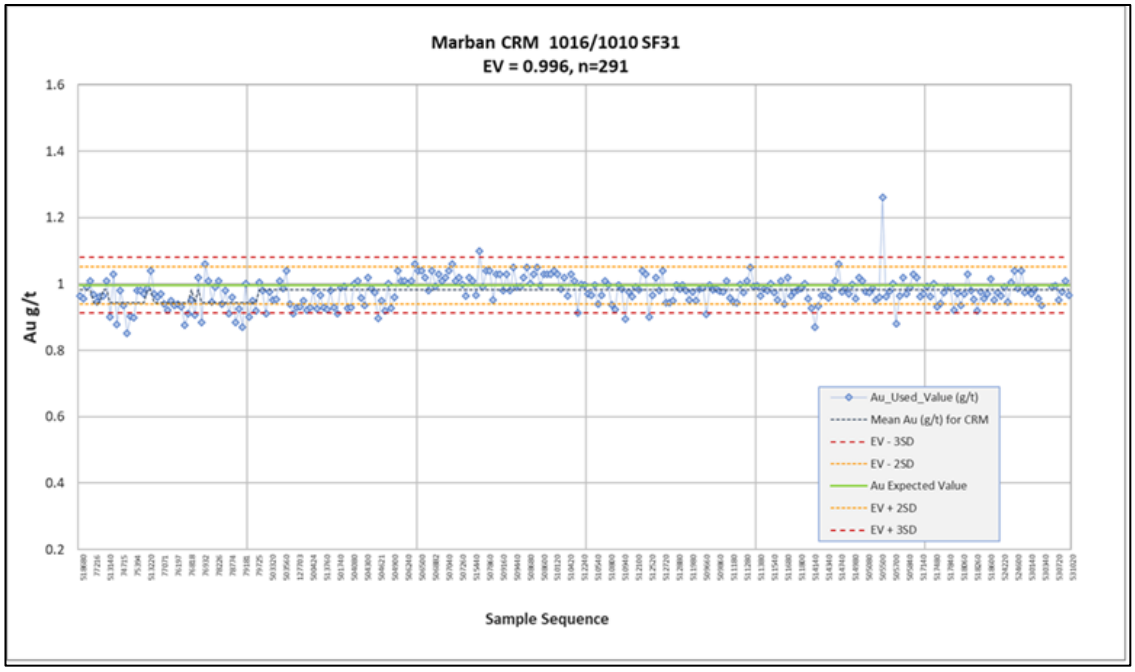
An example of a process control chart is presented in Figure 11-4 below for CRM 1035/SG66 with expected grade of 1.086 g/t and used between 2014 and 2017. This standard performed relatively well over 765 insertions with only 9 fails.

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



Another example is given in Figure 11-5 for CRM 1016/1010/SF31 used from 2007 to 2011. This CRM has almost 9% fails at the +- 3 SD level. It is possible some are mislabeled 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

11.3.3 Marban Duplicates

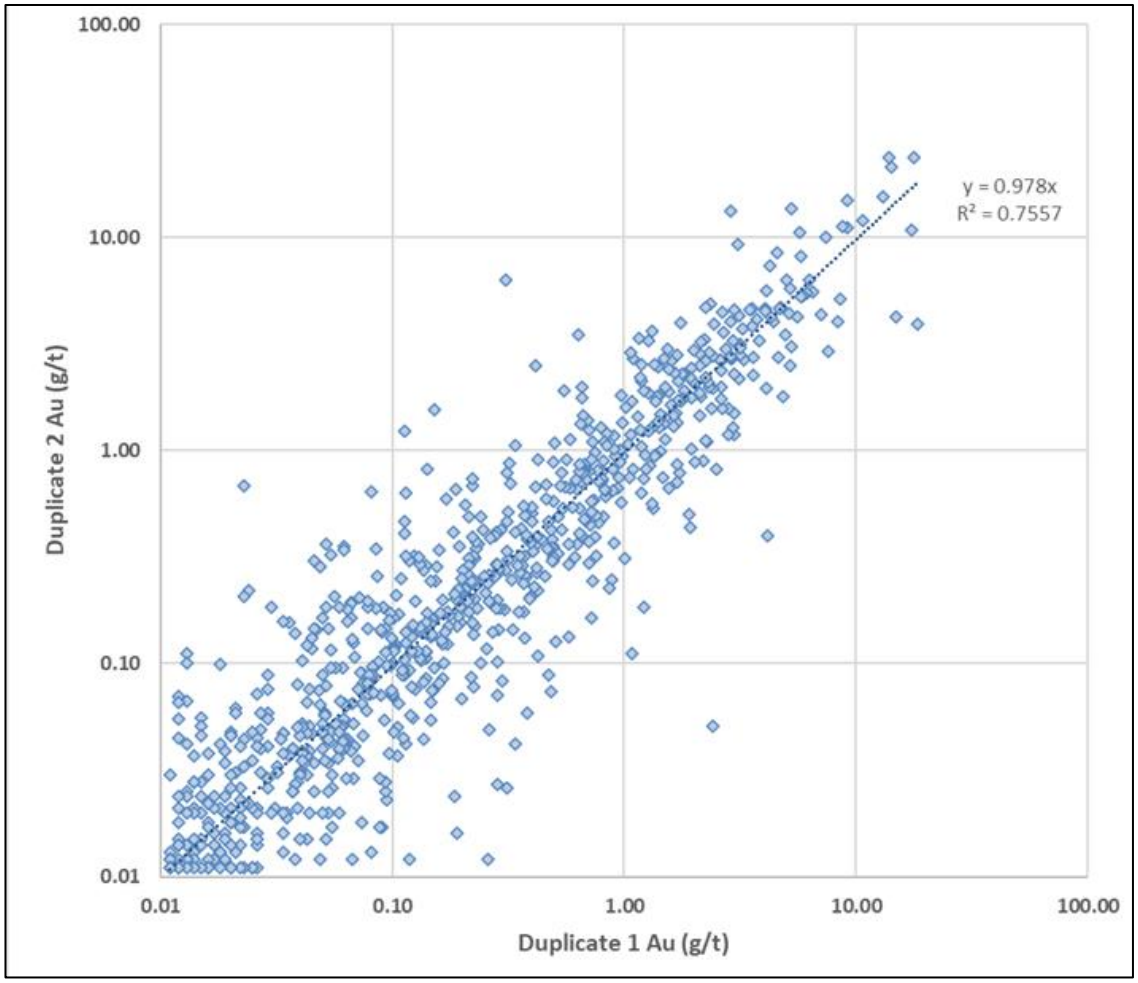
Duplicate assays of the Marban samples are available are available as Field Duplicates and Pulp Duplicates by external and internal laboratories. Analysis of the Field Duplicates show strong heterogeneity in samples of split core typical of “nuggety” gold deposits. Analysis of external duplicates give little indication of bias at a level that would affect the resource estimate. Analysis of internal duplicates show good repeatability at the primary laboratories.

11.3.3.1 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 years 2006 and 2017.

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

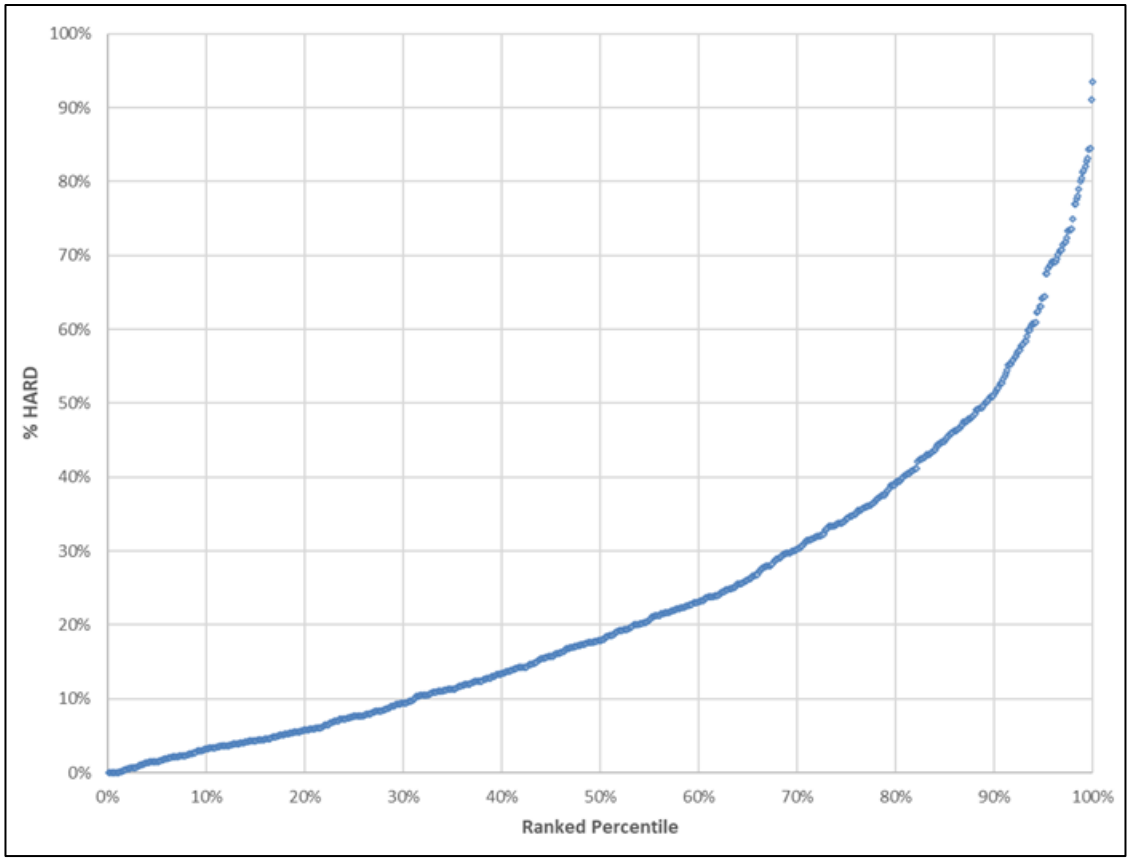
Figure 11-6: Marban Field Duplicates 2006-2017



Source: MMTS, 2020

Figure 11-7 gives the ranked half absolute relative difference (HARD) plot which gives approximately 70% with greater than 10% HARD. Field duplicates for disseminated deposits not typical of gold would give approximately 30% greater than 10%. This implies that the nature of the deposit is highly variable, or “nuggety” and this is of importance to the resource estimate.

Figure 11-7: Marban Field Duplicates Ranked HARD



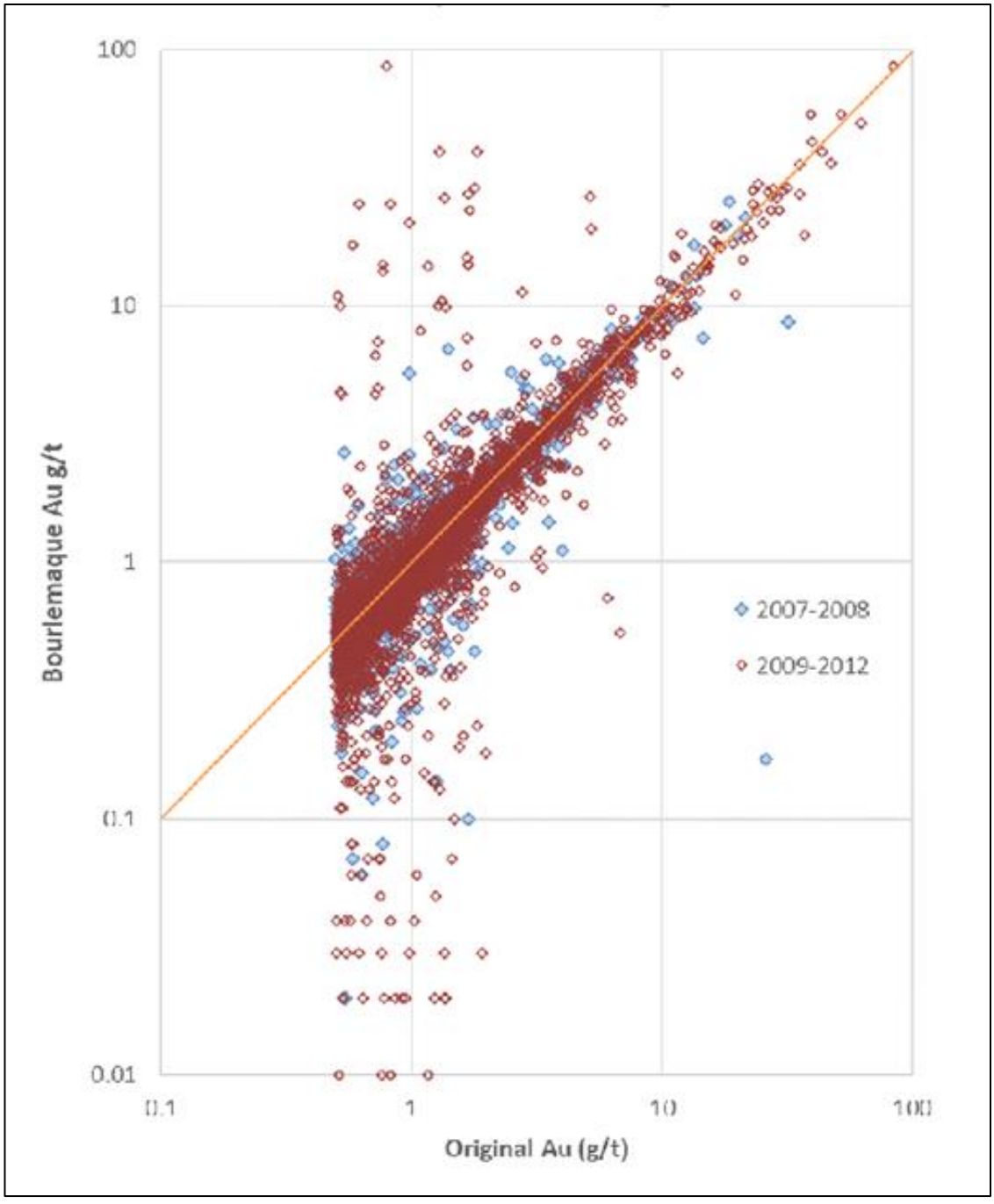
Source: MMTS, 2020

11.3.3.2 Marban External Duplicates

Bourlamaque was selected to perform additional checks on Actlabs pulps from 2007 and 2008 and ALS pulps from 2009 through 2012 that had gold grades above 0.5g/t. An additional 573 samples below 0.5g/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 0.5g/t level, it is not appropriate to analyze this data as a single set.

The pairs with original assay at 0.5g/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 1.98g/t is slightly lower than the mean of the re-assays at 2.04g/t, indicating an error of almost three percent. The median of both sets is 1.03g/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

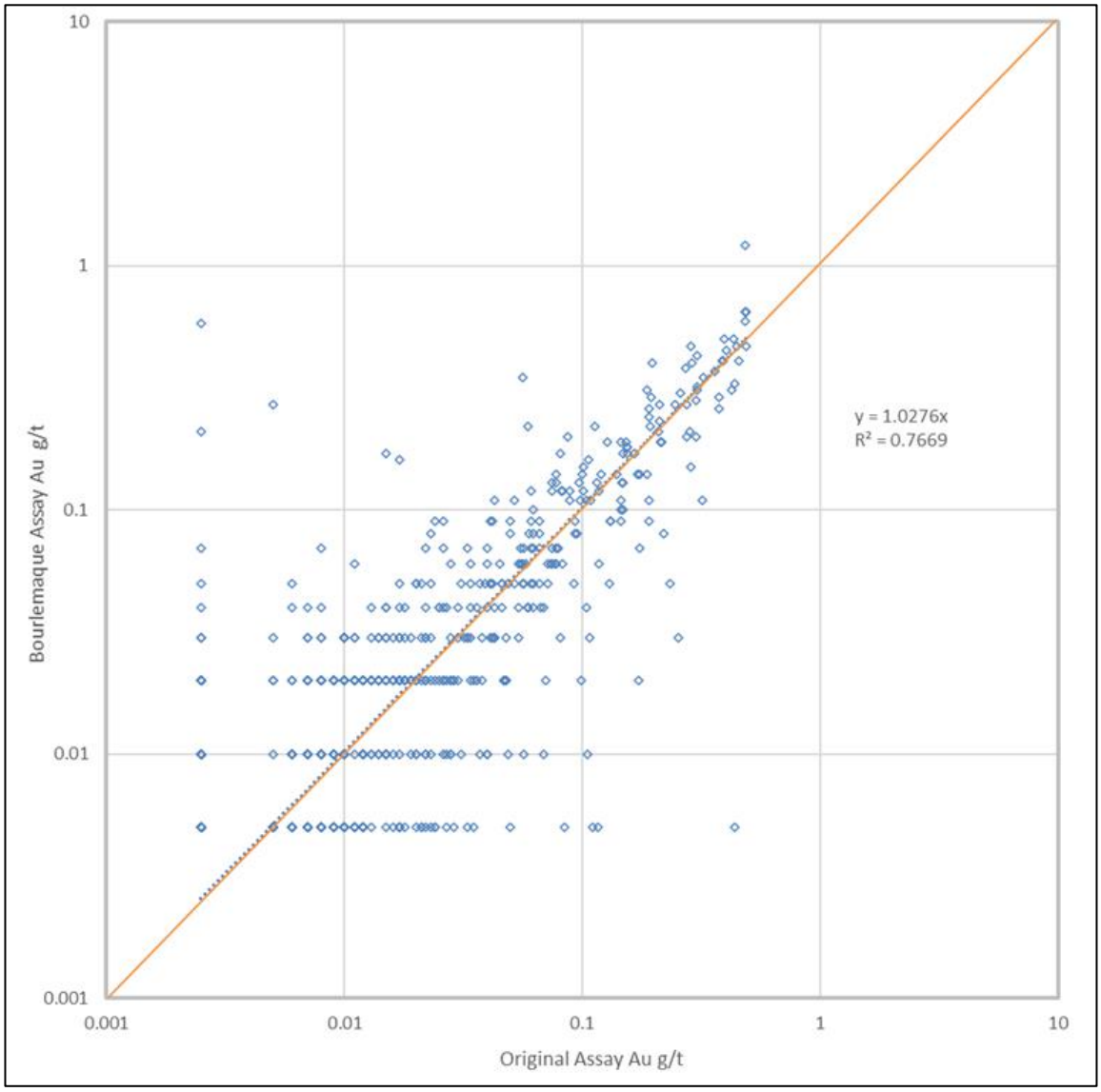


Source: MMTS, 2020

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 0.01g/t or 0.02g/t compared to the original detection limit of 0.05g/t. Of the data with mean assay >0.02g/t, the average of the original assay is 0.11g/t and the

Bourlemaque average is 0.12g/t, confirming again that the Bourlemaque re-assay data is slightly higher.

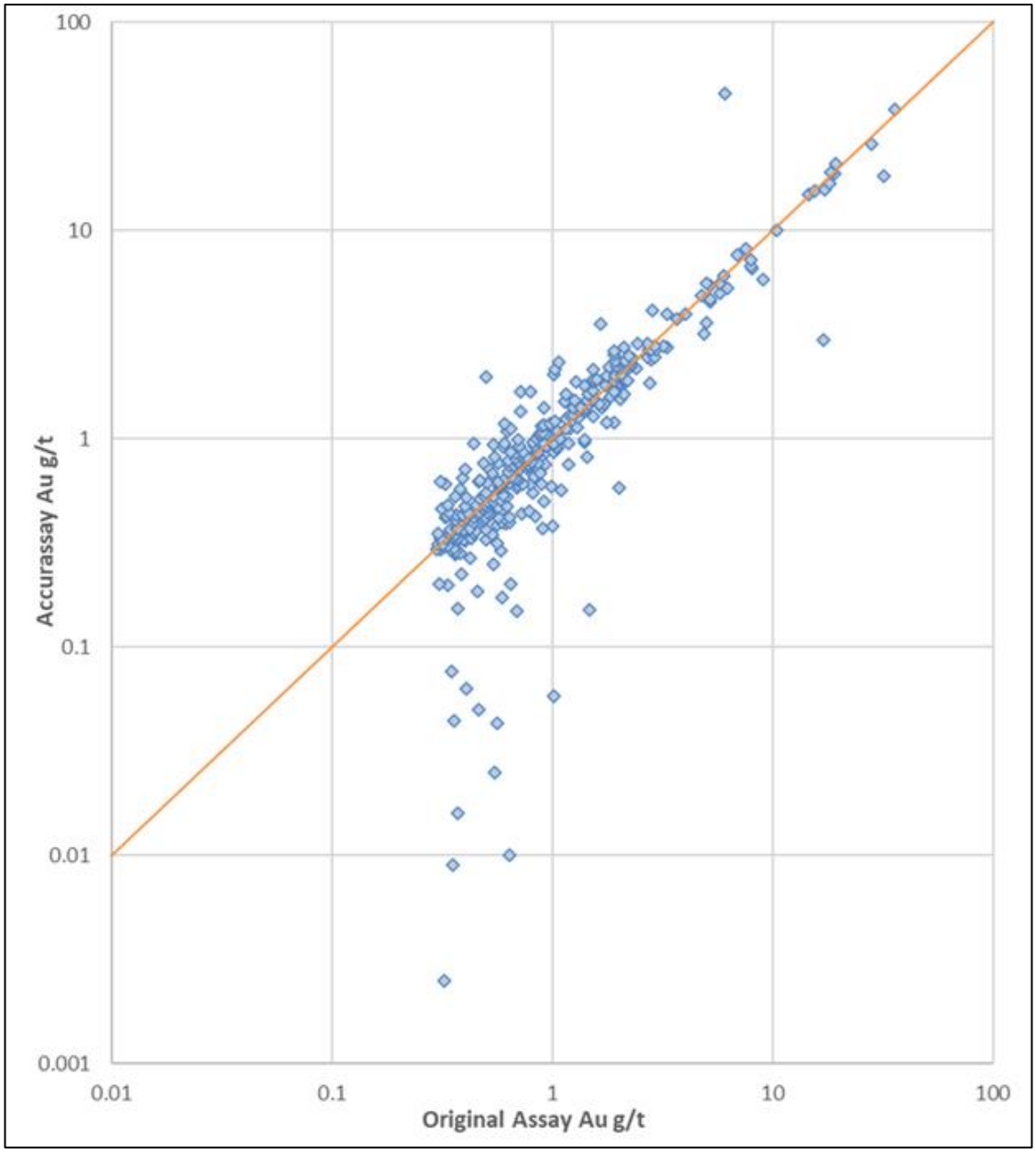
Figure 11-9: Marban External Duplicates <0.5 g/t, 2007-2009



Source: MMTS, 2020

A set of 345 pulps with greater than 0.3g/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 1.94g/t for the original and 1.95g/t for the re-assay, the medians are 0.79g/t and 0.80g/t.

Figure 11-10: Marban External Duplicates, 2014-1015 => 0.3 g/t



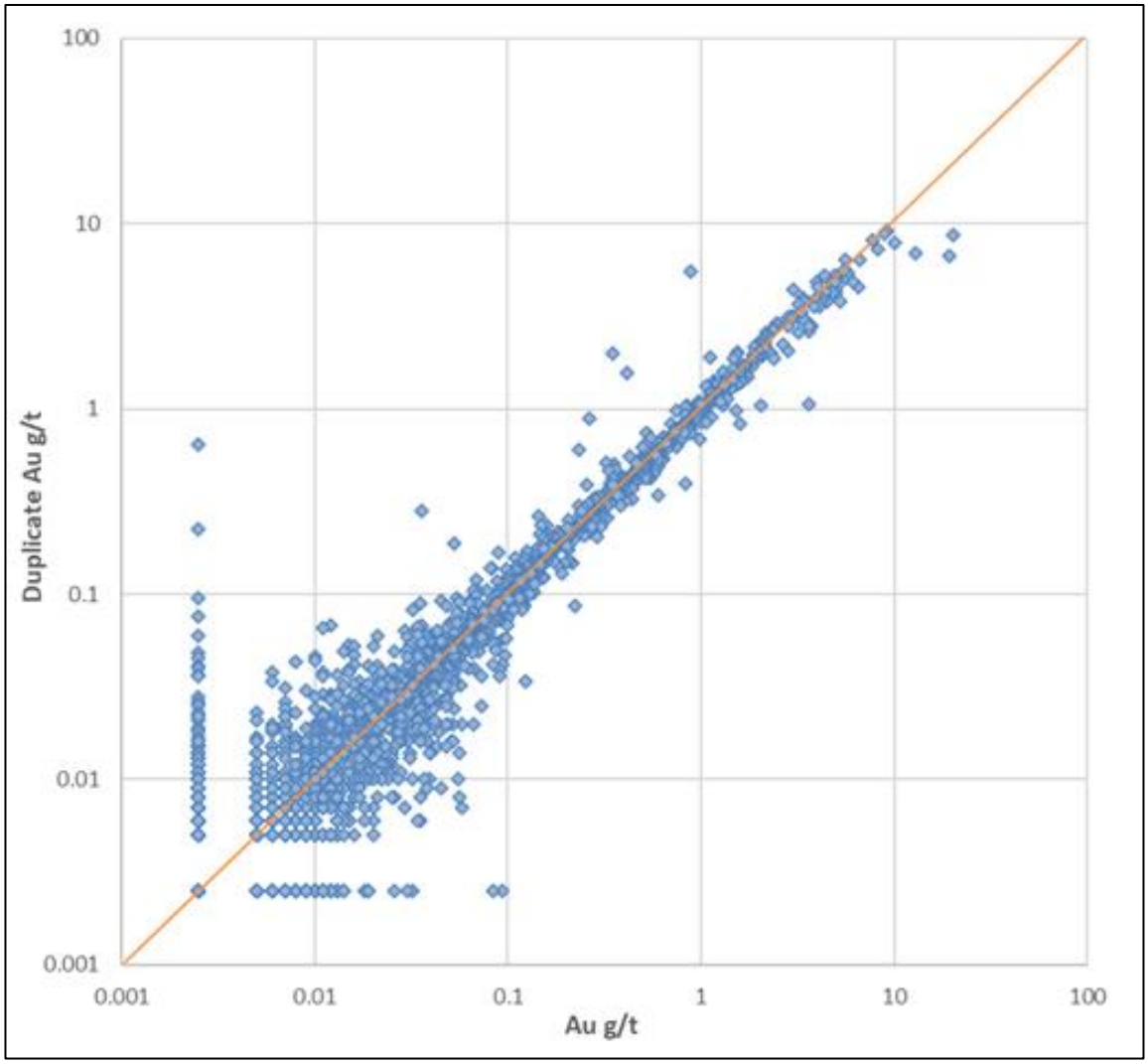
Source: MMTS, 2020

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.

11.3.3.3 Marban Internal Laboratory Duplicates

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

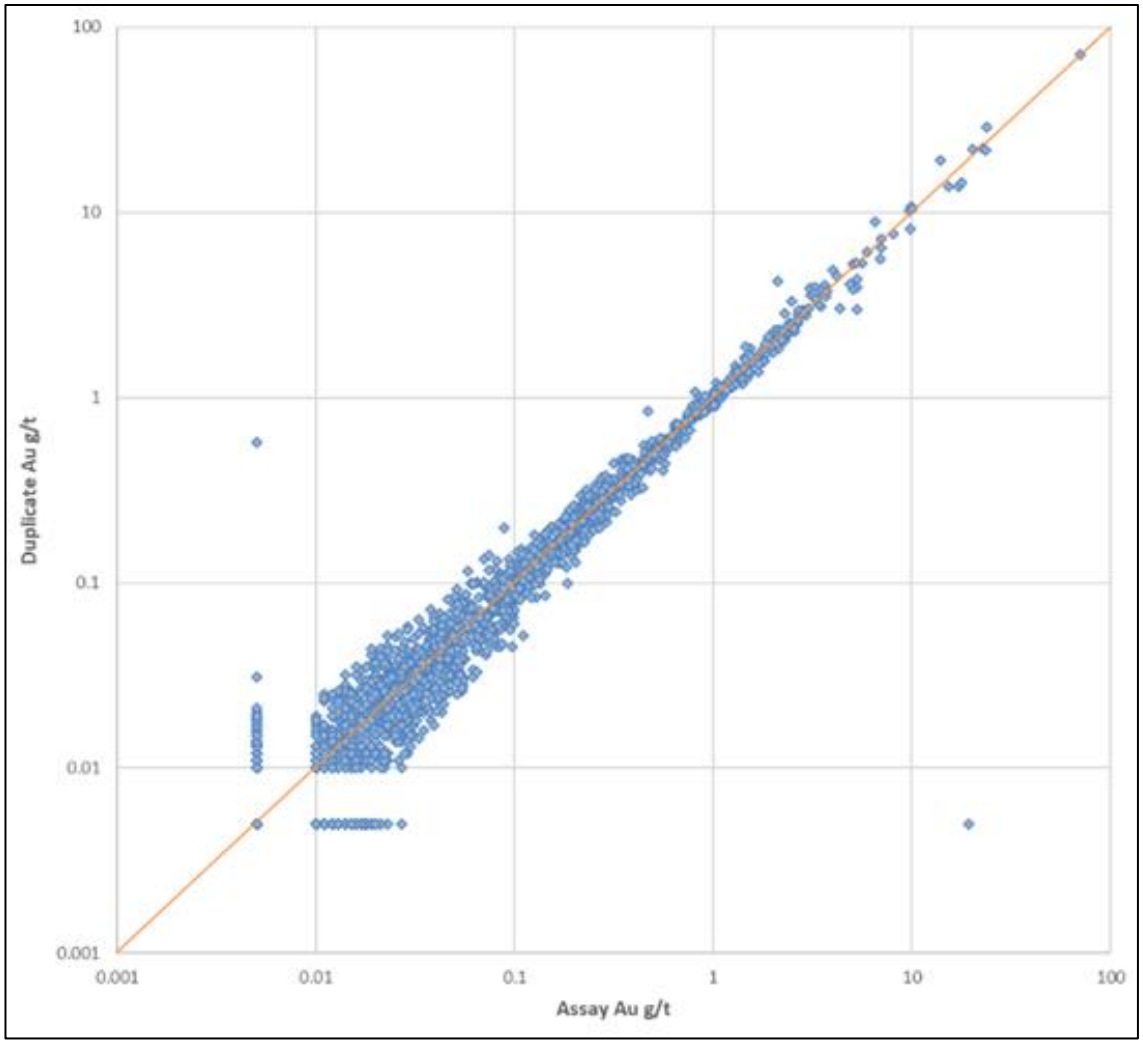
Figure 11-11: Marban Internal Duplicates, 2006-2012



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

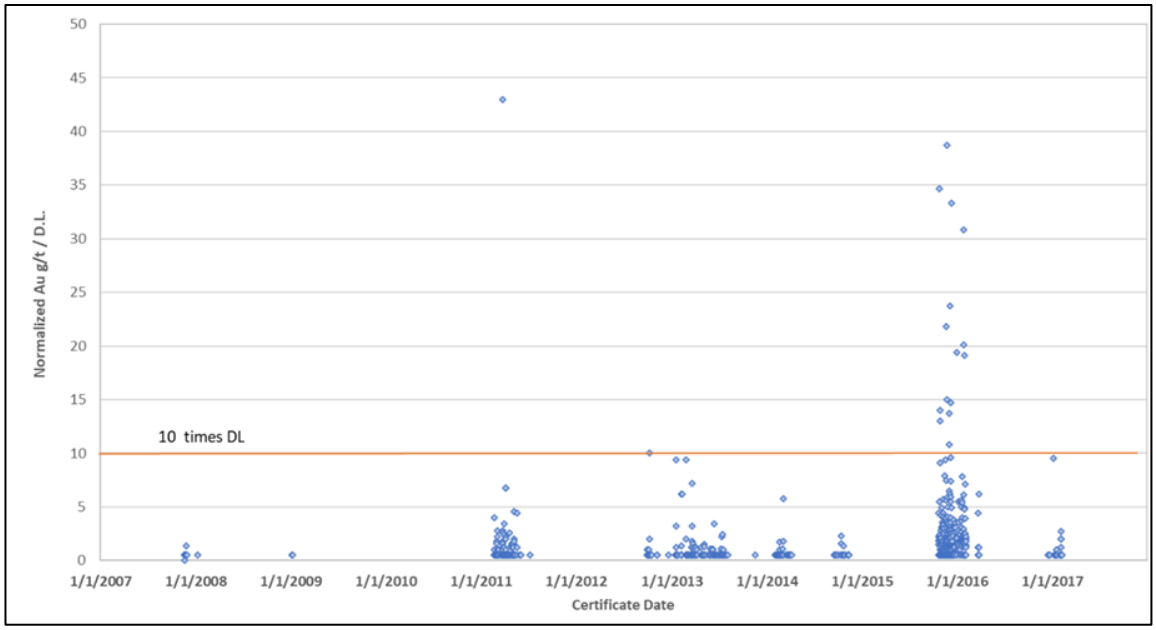


Source: MMTS, 2020

11.3.4 Kierens-Norlartic Blanks

The QAQC 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 generous criteria of 10* DL. The percent of samples failing this criteria is 2.3%, while 6.8% fail against a more stringent criteria of 5* 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.

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



Source: MMTS, 2020

11.3.5 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 had been 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 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 mislabeled, 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.

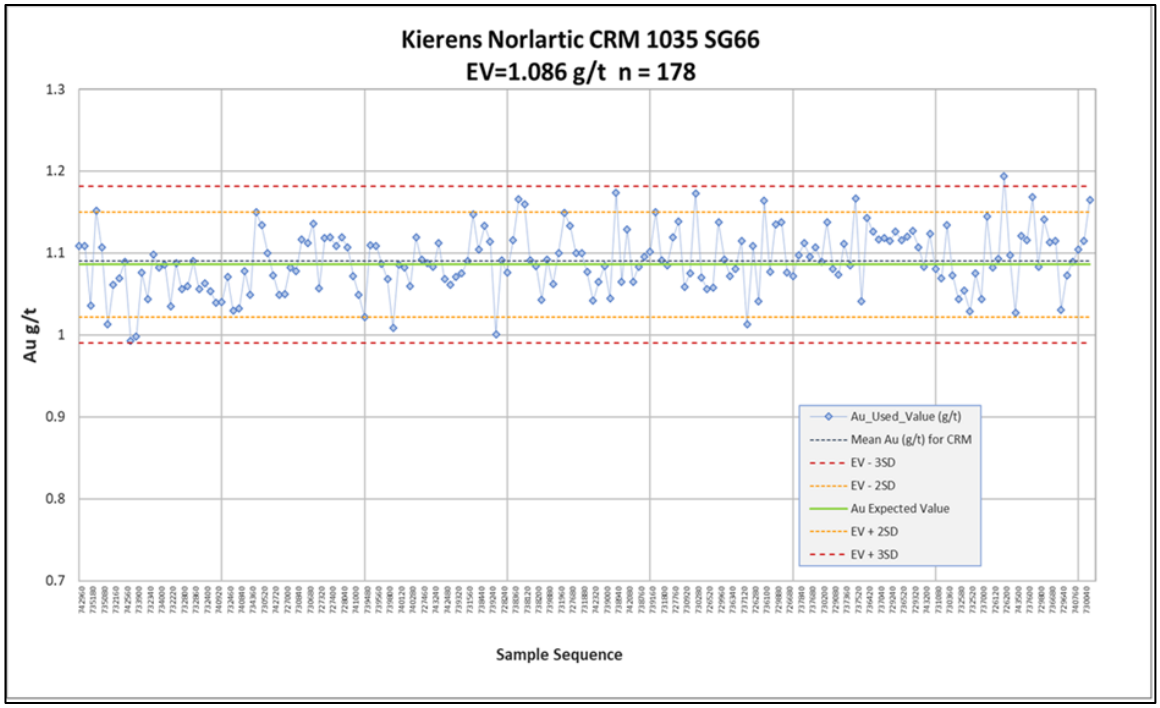
Table 11-2: Kierens-Norlartic CRM Results

CRM	Total Insertions	Au Expected Value g/t	Au Average Assay g/t	Std Dev of Au Assay g/t	% error	CV	Total fail (+- 3 SD)	% 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

CRM	Total Insertions	Au Expected Value g/t	Au Average Assay g/t	Std Dev of Au Assay g/t	% error	CV	Total fail (+- 3 SD)	% Fail	Used From	Used To
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

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.

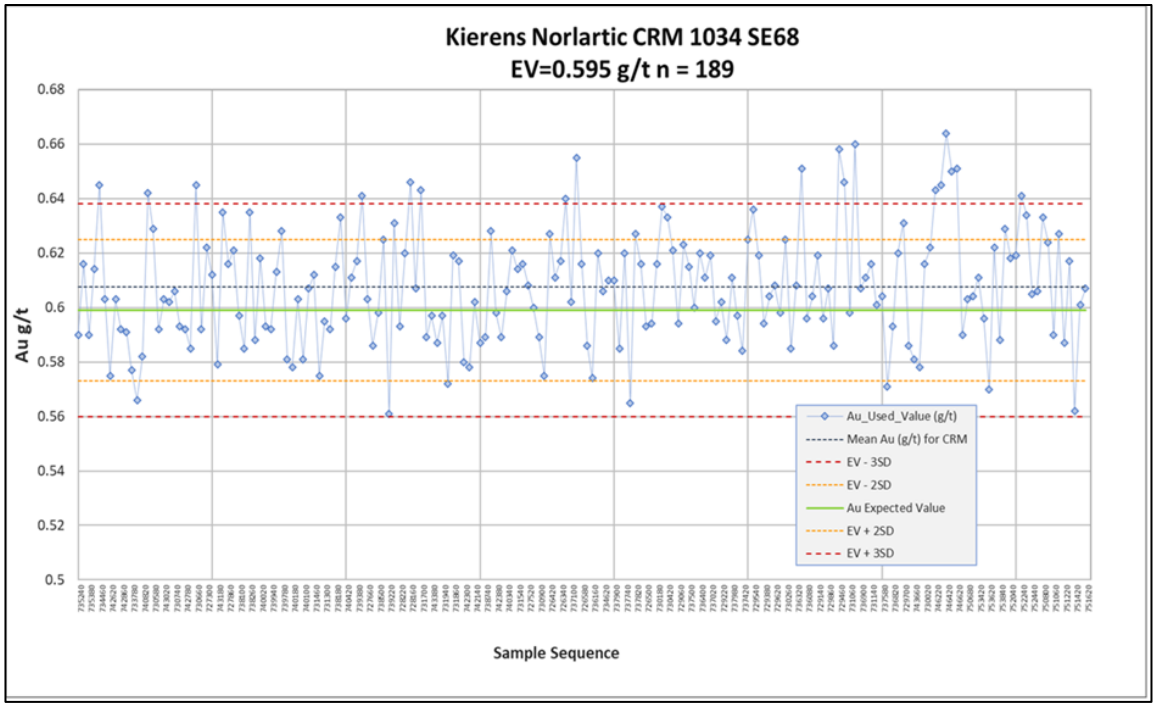
Figure 11-14: Kierens-Norlartic Process Control Chart for CRM 1035/SG66, EV = 1.086 g/t



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

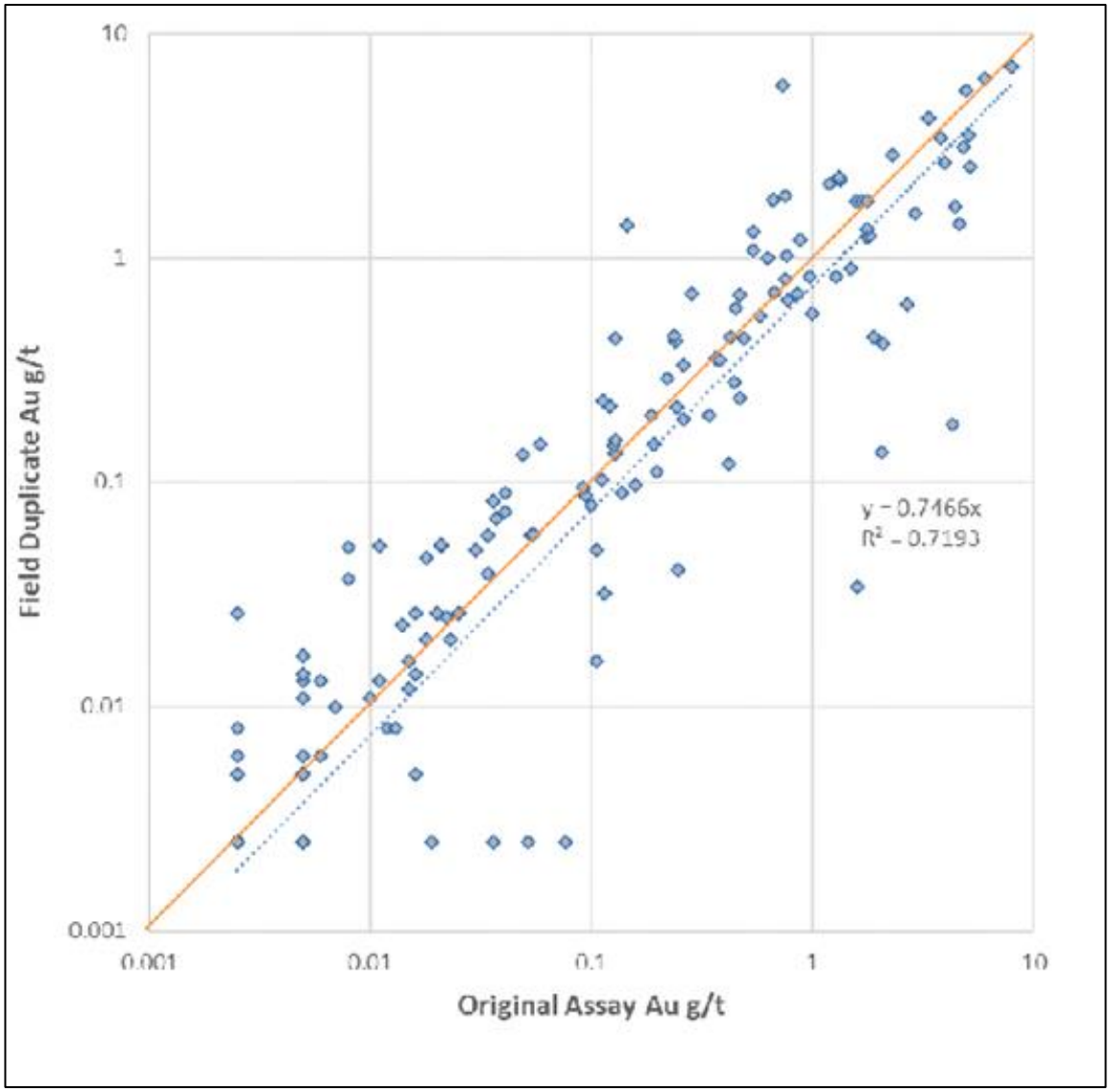
11.3.6 Kierens-Norlartic Duplicates

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

11.3.6.1 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 considerably less than 1.0 indicating that the assays of the field duplicates are less than the original assay. Systematic removal of outliers did not improve the slope.

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 in this era, MMTS prefers to withhold judgement at this stage and recommend future studies consider this in depth.

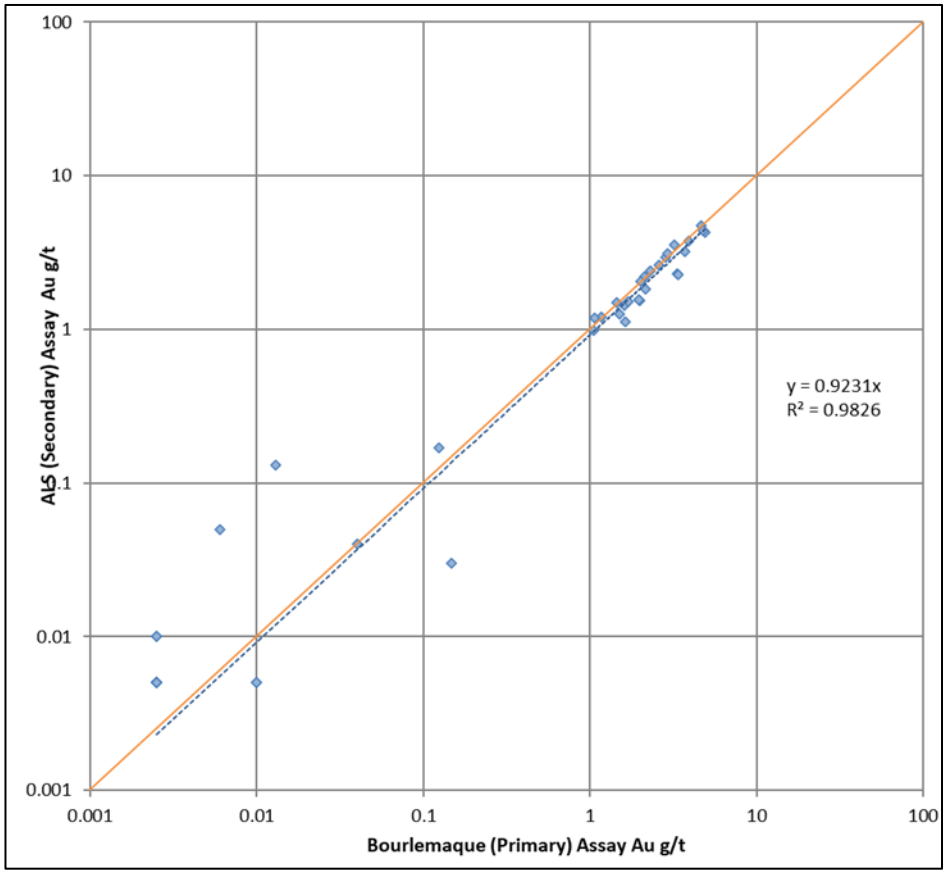
Table 11-3: Kierens-Norlartic Field Duplicates Subset Statistics 2007-2014

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	

11.3.6.2 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. And 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.

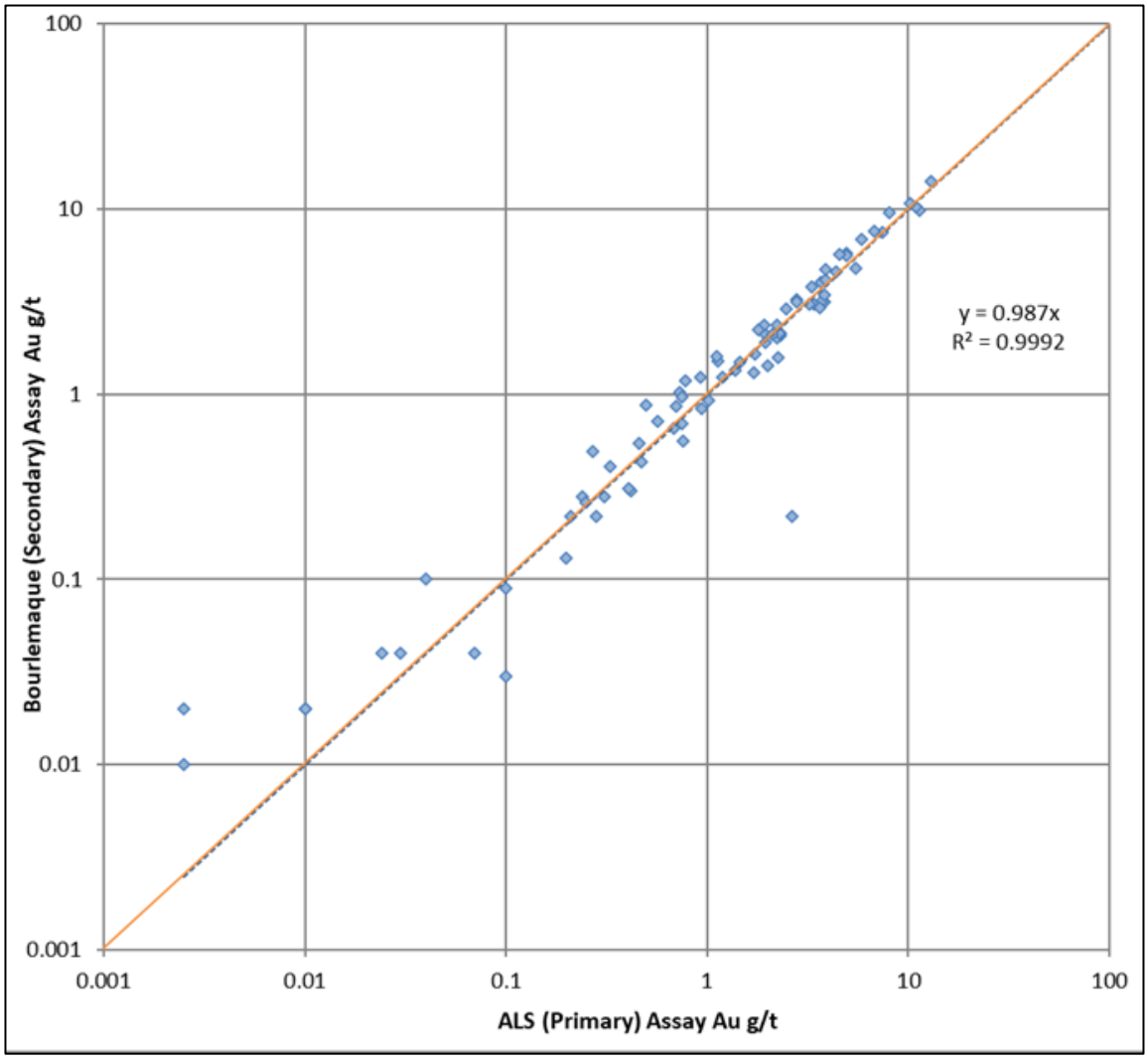
Figure 11-17: Kierens-Norlartic External Duplicates, 2006



Source: MMTS, 2020

Eighty-one duplicates pairs of 30g 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

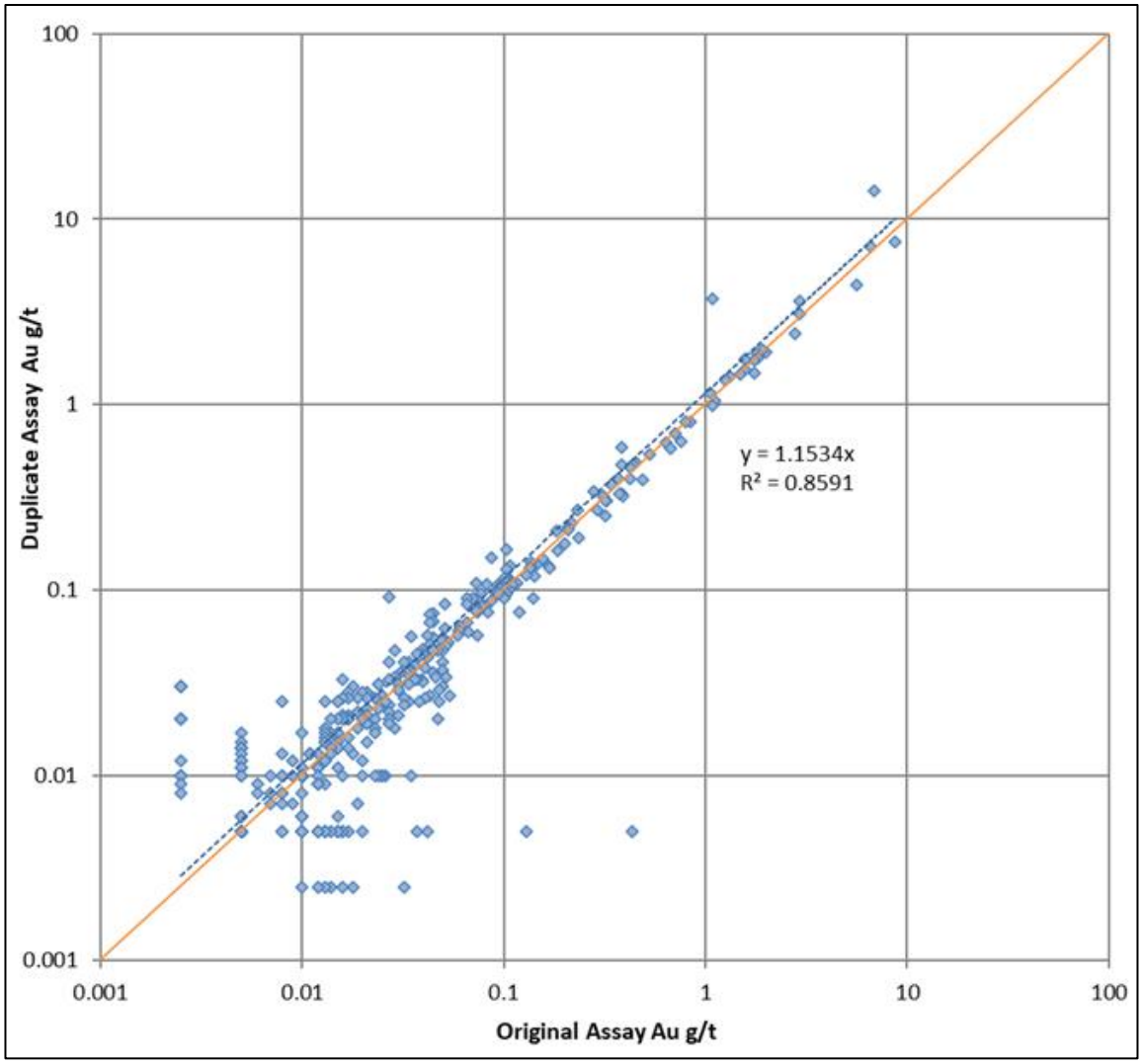


Source: MMTS, 2020

11.3.6.3 Kierens-Norlartic Internal Duplicates

The QAQC 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 11-1 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 0.1g/t and 10g/t mean assay	average	0.944	0.954	1.1%
	median	0.386	0.381	
Between 0.01g/t and 10g/t mean assay	average	0.275	0.278	1.0%
	median	0.041	0.037	

11.3.7 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 QAQC has been validated through re-assays. Validation of the pre-NioGold data is presented in the next section.

MMTS considers the data to be of acceptable quality at this level of resource reporting. Prior to future reporting of higher significance, it is recommended that:

- Commercially available and certified blank material be sourced and used for QAQC in future drilling.
- Re-assays of the 10 samples before and after the failed blanks be run with newly acquired blank material inserted in the stream.
- All future drilling and QAQC to include CRMs of lower grades, such as 0.15 g/t, 0.25 and 0.35 g/t Au.
- All future QAQC employ the definition of a failed CRM at the +/- 3 SD level, or two consecutive samples in the warning zone of +/-2 SD.
- Re-assays are accomplished on failed samples at the +/- 3 SD level, to include the 10 samples before and after the fail, with particular attention to Marban assays in 2006 and 2007. Insertion of newly acquired CRM material is advisable.
- Duplicate core samples in Kierens-Norlartic be resampled to 5% so that the possibility that the original core assays were selectively sampled can be excluded.
- The Kierens-Norlartic QAQC database be augmented with external laboratory duplicates from drilling after 2007.

12 Data Verification

12.1 Site Visit

Sue Bird, Principal of MMTS and an Independent Qualified Person engaged by O3 Mining, visited the site on August 12, 2020. The core storage facility, general site layout, and offices were toured. Samples of core from the storage area was examined with core sampling, tagging and storage method verified. Representative split core samples of the mineralization were examined and discussed with the site geologists.

12.2 Drillhole Database Verification

MMTS checked certificates for 1% of the assay database for accuracy and no errors were found.

12.3 Other Data Verification

Verification of metallurgical, hydrologic, environmental and geotechnical data is discussed in the relevant sections of this Report. The data are concluded to be adequate to support the study.

12.4 Historic Database Verification

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 here.

12.4.1 Marban Historic Verification

Previous analysis of the 3,360 core duplicates from historic drill hole sample by Gustin (2013) concluded that the re-assays did not correlate well with the historic data and indicated the data was biased high at grades below 0.7g/t. The author concludes that there is inherent risk in resource modeling in zones where grade estimations rely principally on the historic assays.

Table 12-1 gives the total number of assay samples available in the Marban area by year and length which indicates the impact of the historic portion of the database is of significant size.

Table 12-1: Marban Assay Database Samples by Year and Length

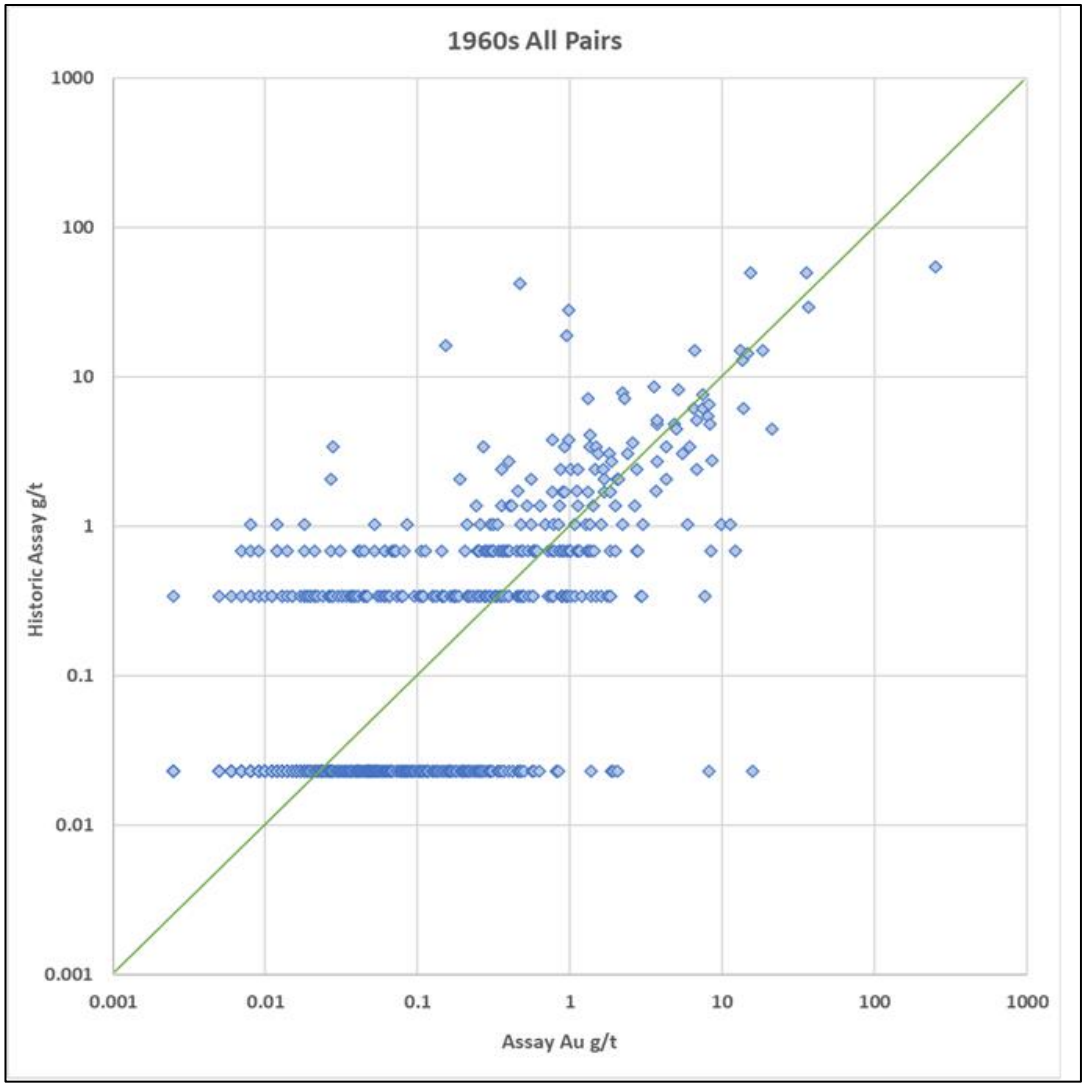
Year	Assay Samples	Total length (m)	% of Total Length
unknown	460	793	2.65%
1960s	9,710	12,362	41.29%
1986	97	139	0.46%
1987	36	48	0.16%
1988	69	89	0.30%
2006	60	48	0.16%
2007	727	699	2.33%
2008	2,088	1,981	6.61%
2010	1,985	1,747	5.83%
2011	2,076	1,802	6.02%
2012	2,401	2,183	7.29%
2014	1,336	1,163	3.88%
2015	7,630	6,889	23.01%
Total	**28,675**	**29,942**	**100.00%**

12.4.1.1 Marban Historic 1960s Data

A scatter plot of the 758 duplicate pairs of the 1960s data is given in Figure 12-1. The paired analysis of the 1960s data is overshadowed by the impact of the historic detection limit (DL) of 0.01 opt (0.34 g/t), a range of critical interest to current resource estimation and an insignificant grade at the time of historic underground operation. The granularity of the 1960s data is apparent in plots of re-assay to historic values as shown in Figure 12-1. Note that the historic values reported as “tr” are in the database both at 0.17 g/t, one-half the DL, and 0.023, 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 4 times DL.

The 1960s data is considered at the level of each increment of detection level and the re-assay values are binned according to these historic increments. Statistics of the binned data is given in Table 12 2 which does not instil confidence in the data at and below 5*historic Detection Limit (DL).

Figure 12-1: Marban Scatter Plot of 1960 vs Re-Assay



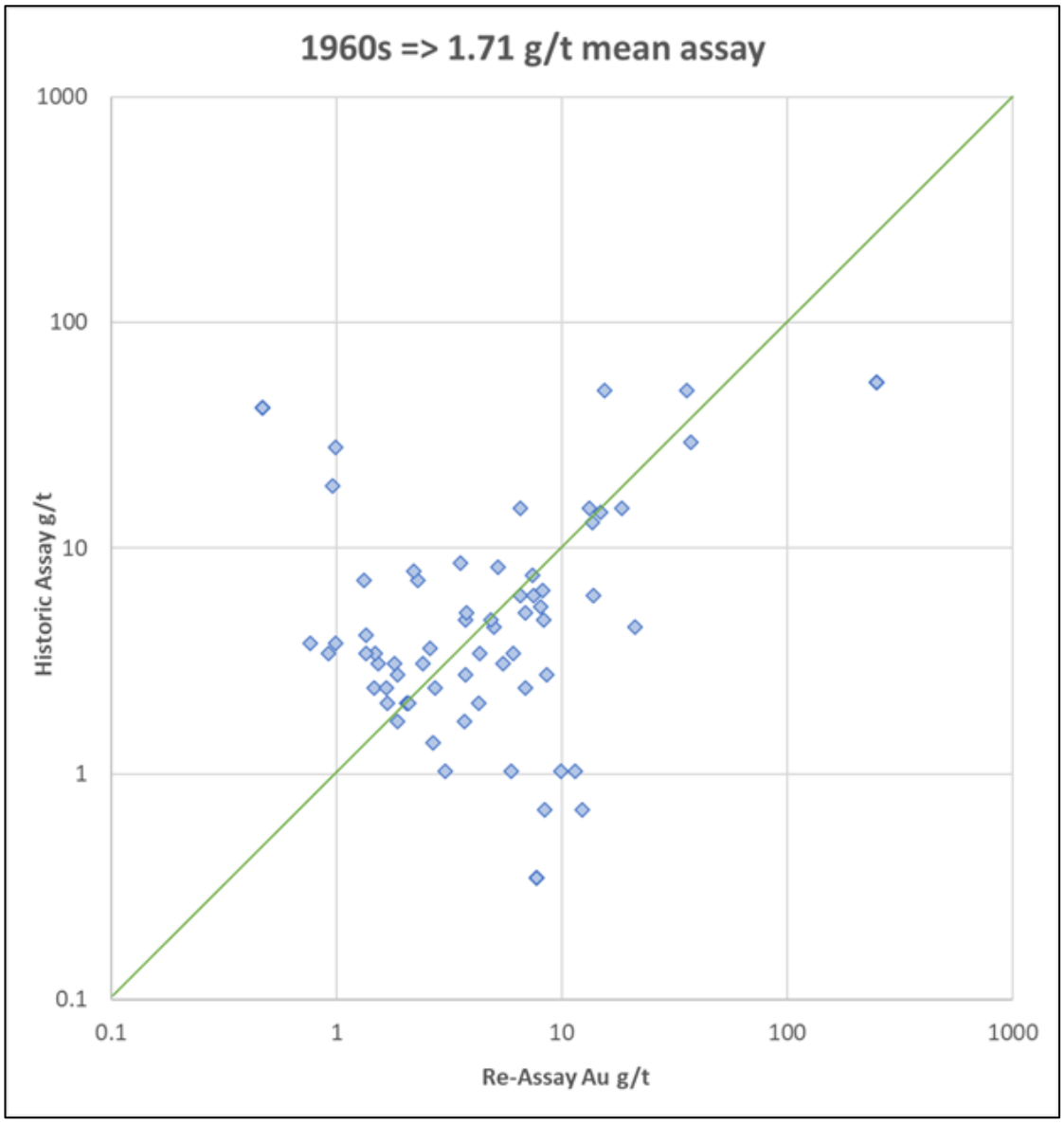
Source: MMTS, 2020

Table 12-2: Marban Statistics of 1960s Binned Re-Assays

1960s Historic g/t	Re-assays		
	Mean (g/t)	Median (g/t)	Samples
tr	0.138	0.024	433
0.34	0.384	0.146	134
0.69	0.804	0.384	81
1.03	1.782	0.627	24
1.37	0.968	0.635	11
1.71	1.415	1.110	9

A scatter plot of 1960s data above 5*DL or 1.71 g/t in mean assay values is presented in Figure 12 2. The data does not appear clustered around a 1:1 line on visual inspection and linear regression confirms the dataset of 69 pairs is not well correlated with an R2 value of 0.392. Although reducing the data set by eliminating outliers until an acceptable statistical relationship appears. In the opinion of the QP, because the data at and below 5*Historic DL is shown to be inherently unreliable and should not be used, there can be no justification for cherry picking data above 5*Historic DL, even if it were to be clearly correlated, and that no original 1960s data should be included in the resource model.

Figure 12-2: Marban Scatter Plot -1960s Re-Assays with mean assay > 1.71 g/t

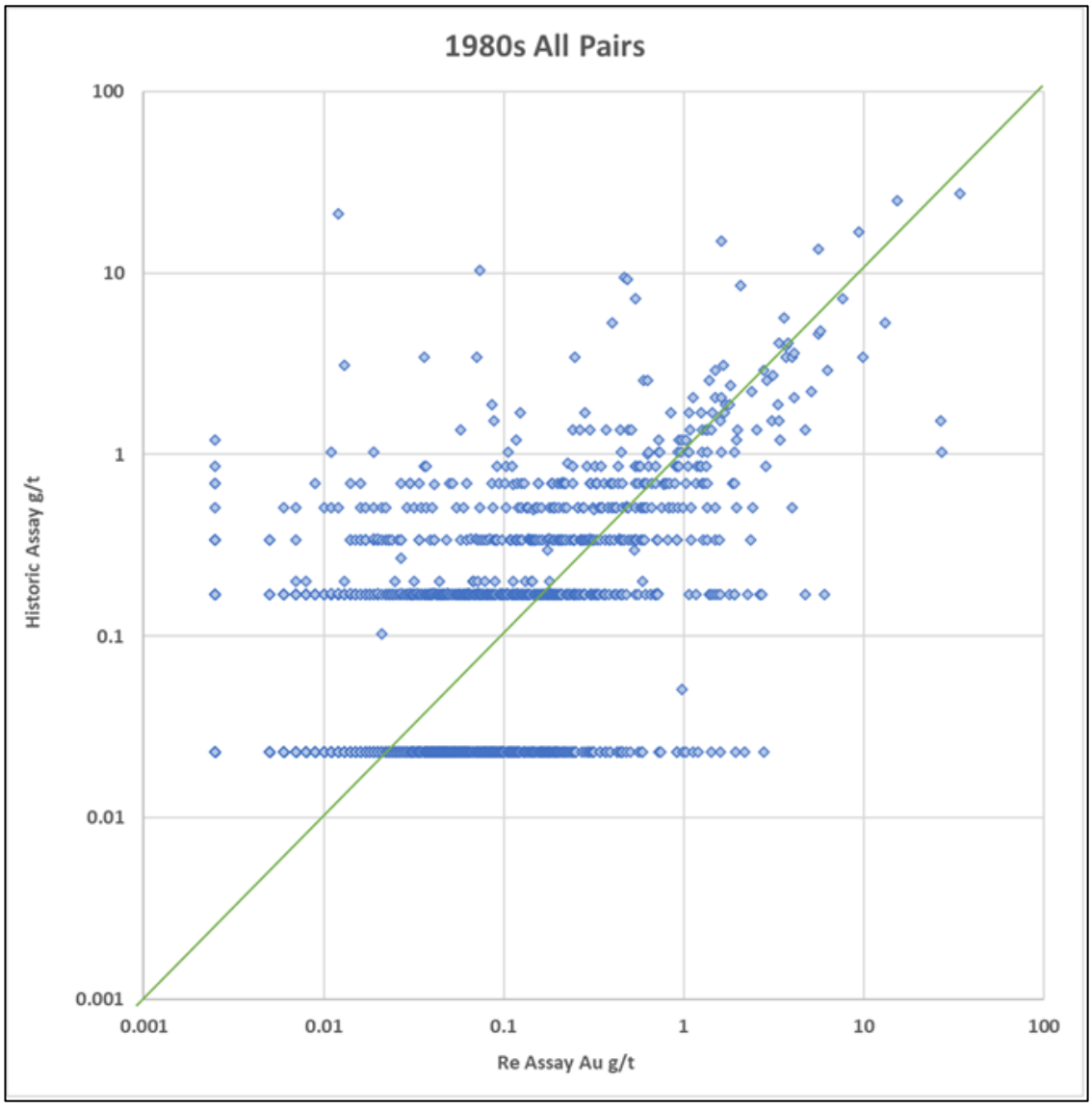


Source: MMTS, 2020

12.4.1.2 Marban Historic 1980s Data

The 2,691 pairs of 1980s Marban re-assay data is presented in a scatter plot in Figure 12 3. This historic data has a detection limit of 0.005 opt, or 0.17g/t, which can clearly be seen in the horizontal bands at increments up to 5*DL or 0.86 g/t. Because of the problems already discussed with the incremented data in critical grade range, and the fact that the 1980s data is less than 1% of the entire database, the historic 1980s data is not included in the resource model.

Figure 12-3: Marban Scatter Plot 1980 vs Re-Assays



Source: MMTS, 2020

12.4.2 Kierens-Norlartic Historic Verification

Lack of Certificates and QAQC for assay data prior to 2000 resulted in 13,849 re-assays being done 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 at Kierens-Norlartic prior to 1986 is significantly biased both at grades near detection limit and at high grades above about 10gpt. It is therefore concluded that this data should not be used going forward. All re-assay data and drillhole data after 1986 is acceptable and is included in the assay database for the purpose of resource reporting

Comparison of mean and median values for all relevant years of re-assays to historic assays is presented in Table 12 3. 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 12-3: Comparison of Kierens-Norlartic Re-assay and Historic Means and Medians by Year

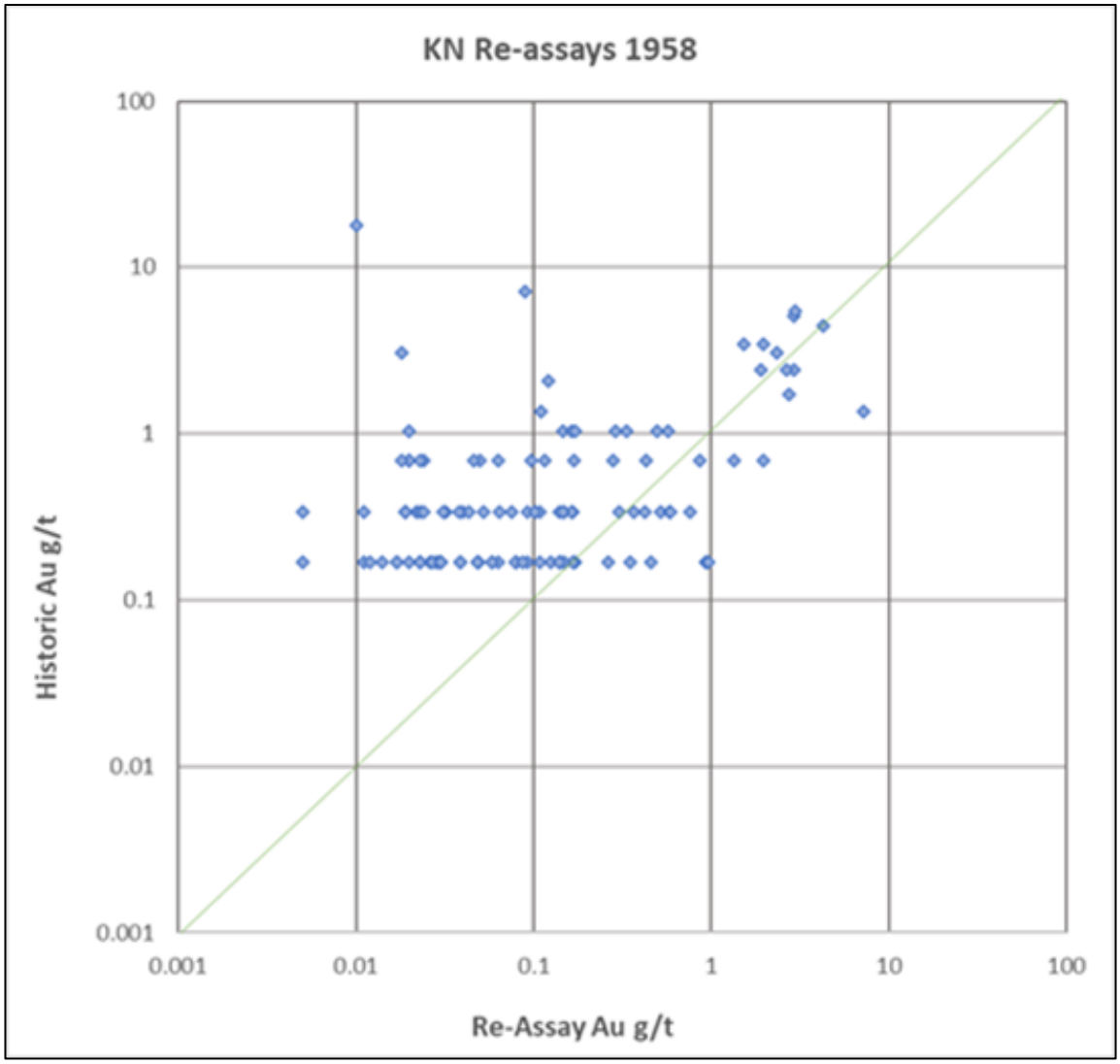
Year	Number of Re-assays	Mean of Re-assay g/t	Mean of historic assay g/t	% Difference (mean)	Median of Re-assay g/t	Median of Historic assay g/t	% Difference (median)
1958	116	0.46	0.90	49%	0.09	0.34	73%
1964	420	1.19	1.34	11%	0.31	0.34	9%
1980	112	1.02	1.22	17%	0.16	0.34	52%
1981	185	0.72	1.51	52%	0.01	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**	

12.4.2.1 Kierens-Norlartic Pre-1986 Data

Of the samples in the “potential pit” area, the Historic assays before 1986 have an original gold assay detection limit (DL) of 0.01 opt (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 detection limit, or 0.17 g/t. As has been discussed previously, the problem of the detection limit being greater than the range of interest of gold assay values is material and cannot be minimized. The pre-1986 historical data is also shown to be biased and therefore it should not be included in the data for resource modeling.

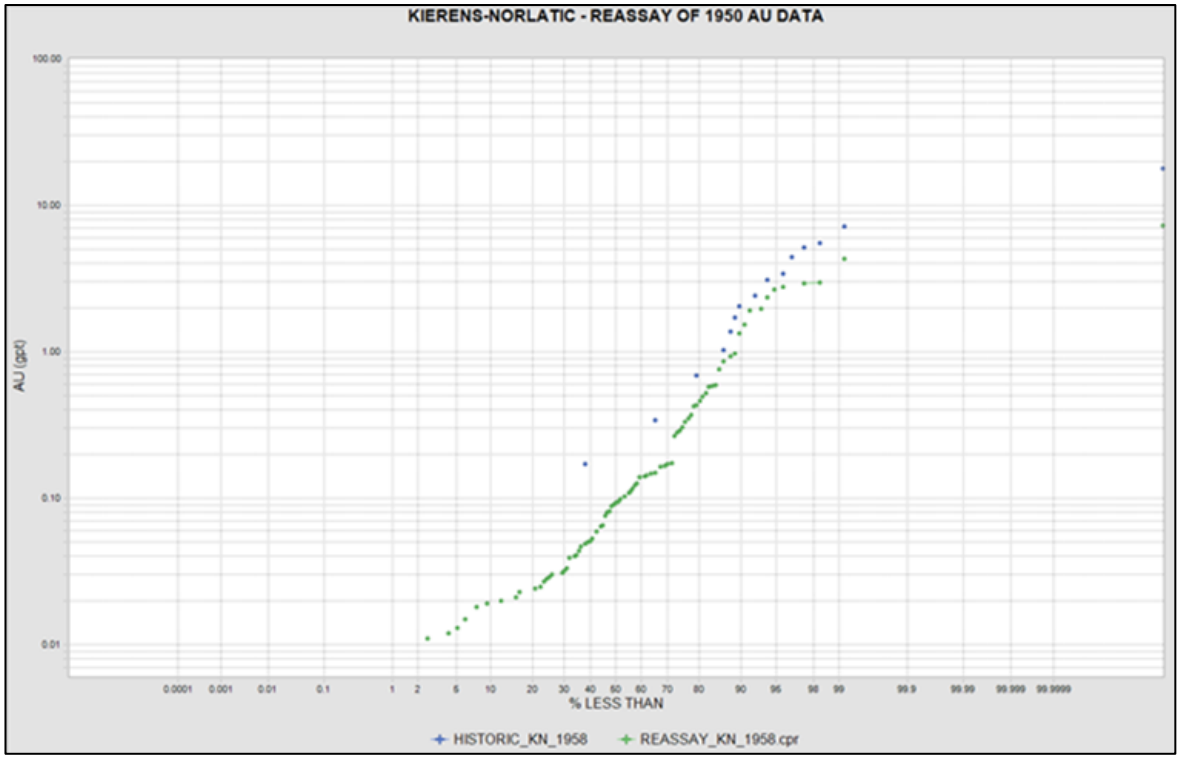
Figure 12 4 shows a scatter plot of the 1958 re-assays with the original assay on the y-axis and the re-assay on the x-axis. It is clearly seen that the data is clustered above the 1:1 line shown in green, indicating the historic assay data is biased high as confirmed by mean and median values presented earlier in Table 12 3. The cumulative probability plot presented in Figure 12 5 also illustrates the bias throughout the distribution.

Figure 12-4: Kierens-Norlartic Scatter Plot of Historic 1958 vs Re-Assay



Source: MMTS, 2020

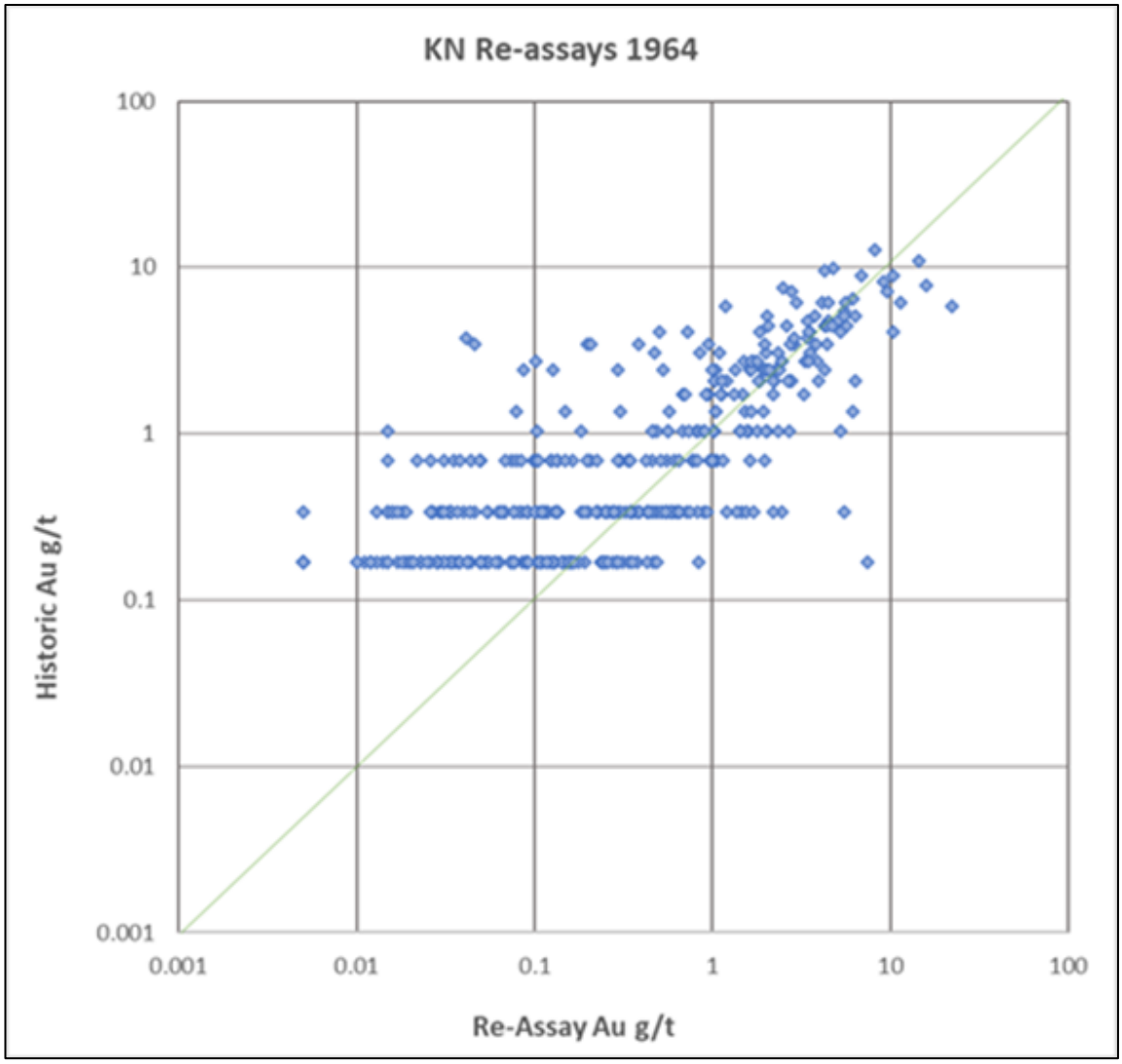
Figure 12-5: Kierens-Norlartic Comparison of 1958 and Re-Assay CPP



Source: MMTS, 2020

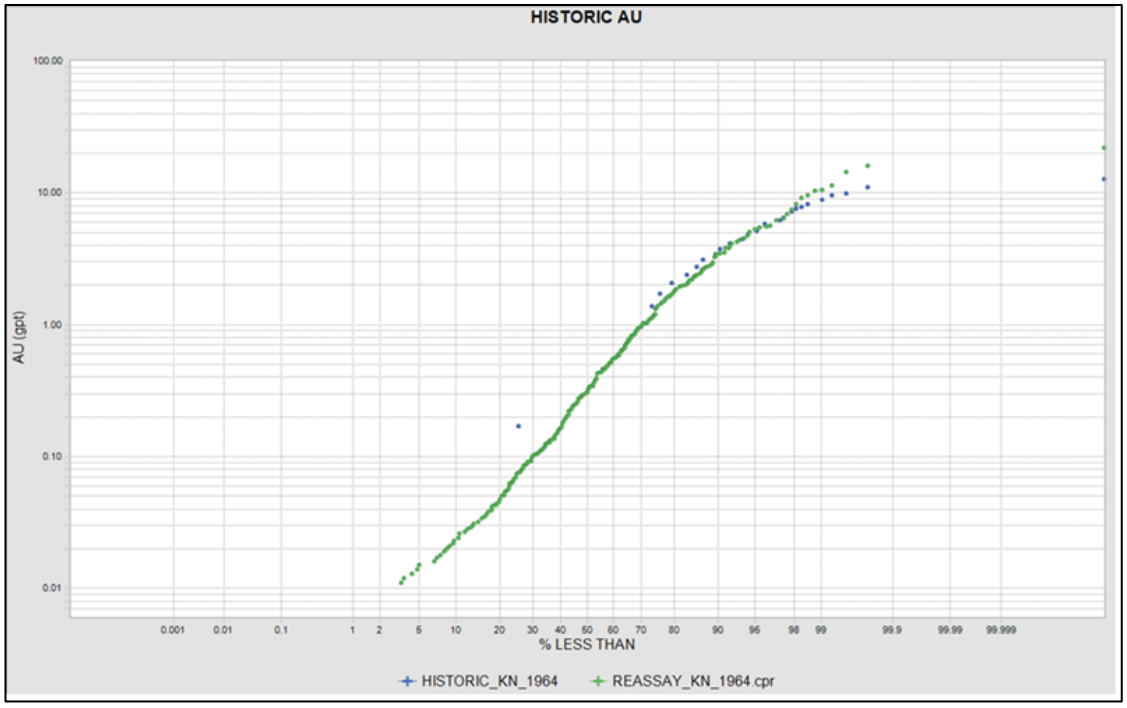
A scatter plot of the 1964 re-assays is presented in Figure 12 6. There is a clear bias as demonstrated by the concentration of data above the 1:1 line. The cumulative probability plot (CPP) presented in Figure 12 7 also illustrates the high bias.

Figure 12-6: Kierens-Norlartic Scatter Plot of 1964 vs Re-Assay



Source: MMTS, 2020

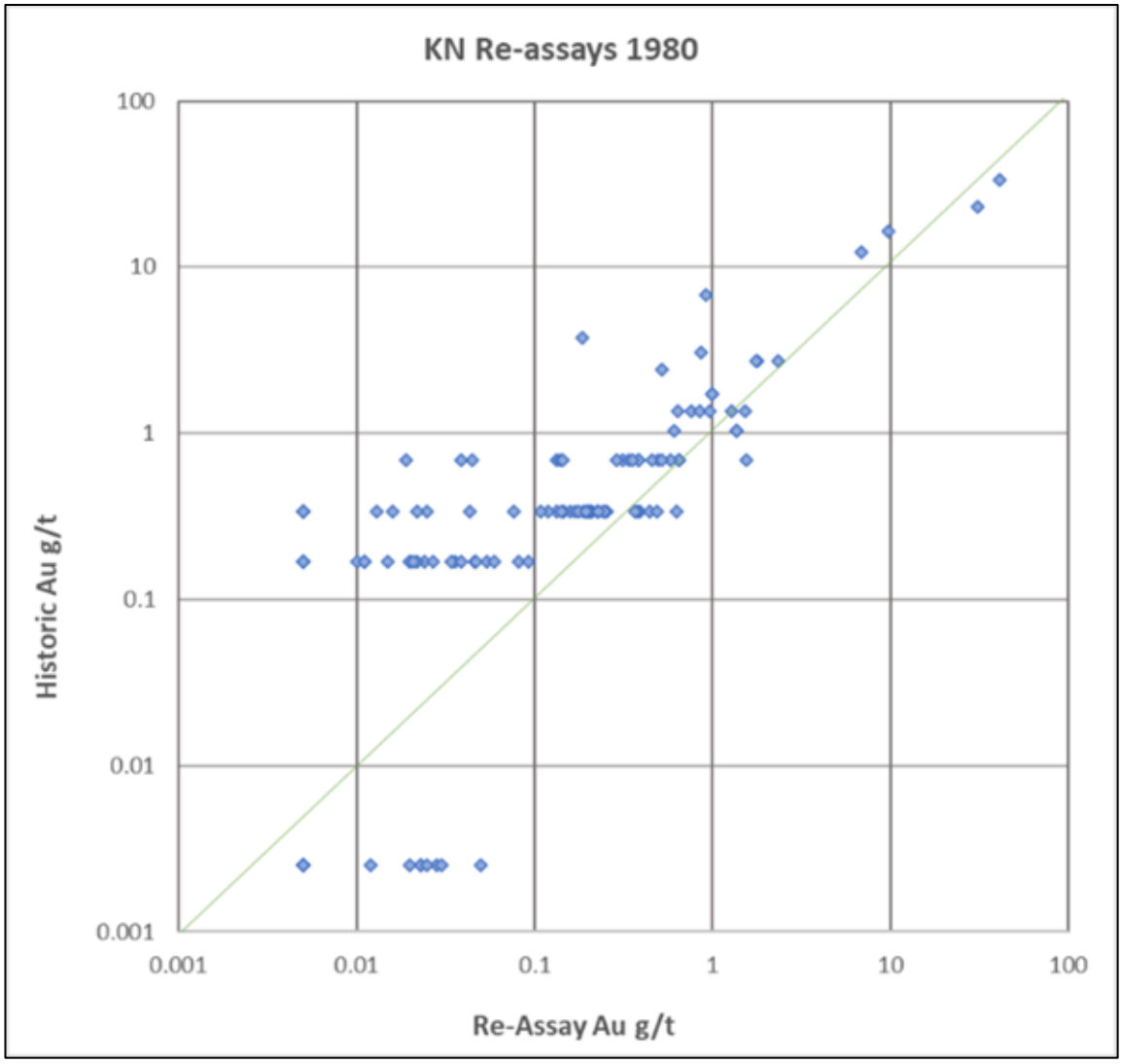
Figure 12-7: Kierens-Norlartic Comparison of 1964 and Re-Assay CPP



Source: MMTS, 2020

The 1980 scatterplot of the re-assays is presented in Figure 12 8. Here, a clear bias is seen with significantly more data points above the 1:1 line than below, the bias is supported by the higher mean and median of the historic assays compared the re-assays. This data also shows a group of historic data on the 0.0025, this is due to the historic data being reported as “nil” instead of “tr” and being entered as 0.0025 instead of 0.17. Because of the high bias and the detection limit issue, the 1980 data should not be included.

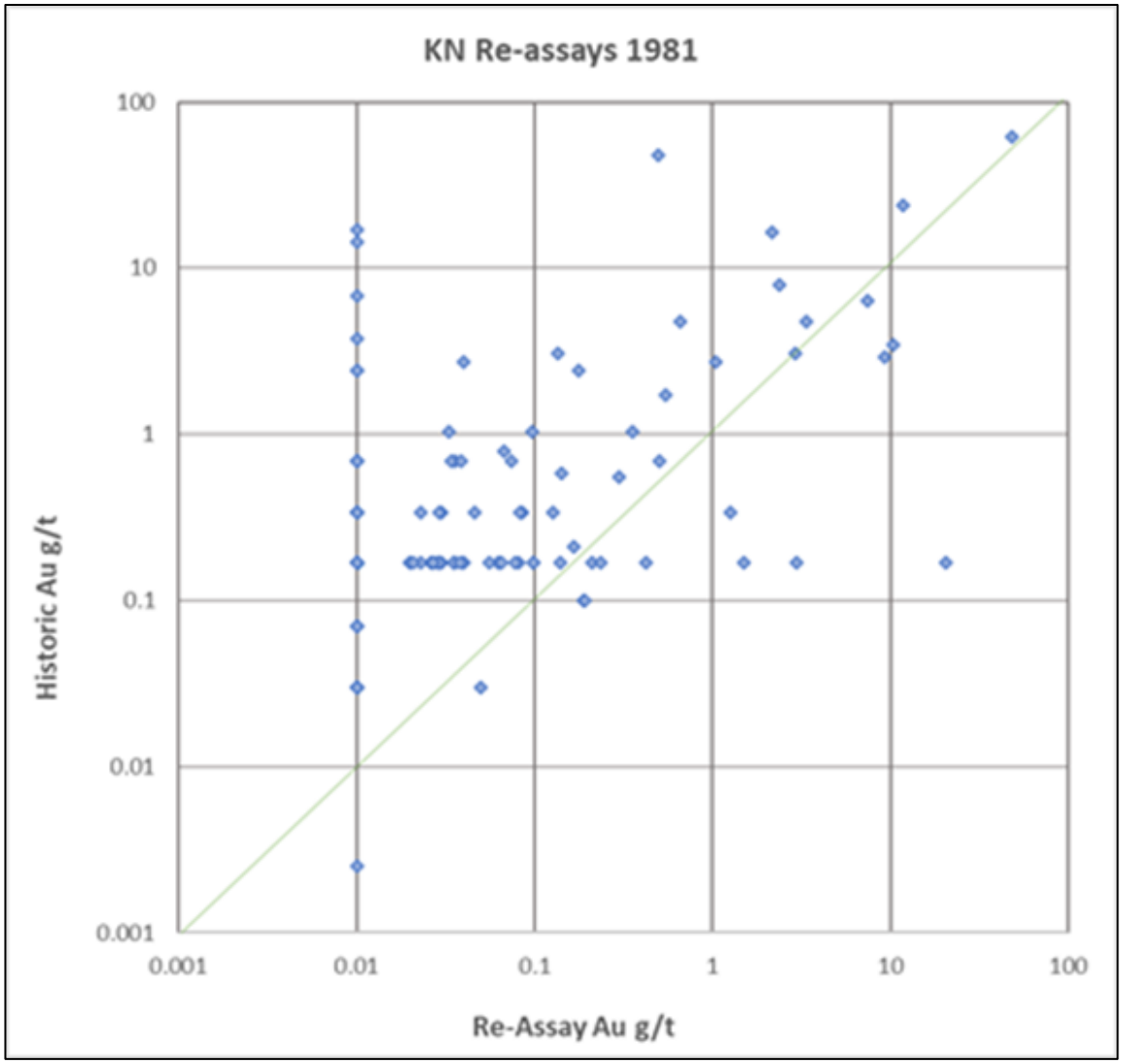
Figure 12-8: Kierens-Norlartic Scatter Plot of 1980 vs Re-Assay



Source: MMTS, 2020

The 1981 re-assays duplicate pairs are presented in Figure 12 9. Here a concentration of values along a re-assay value of 0.01 g/t results from a 0.02g/t detection limit of these re-assays, entered at half the detection value. Again, is it clearly seen that most of the historic data lies above the 1:1 line indicating clear bias.

Figure 12-9: Kierens-Norlartic Scatter Plot of 1981 vs Re-Assay



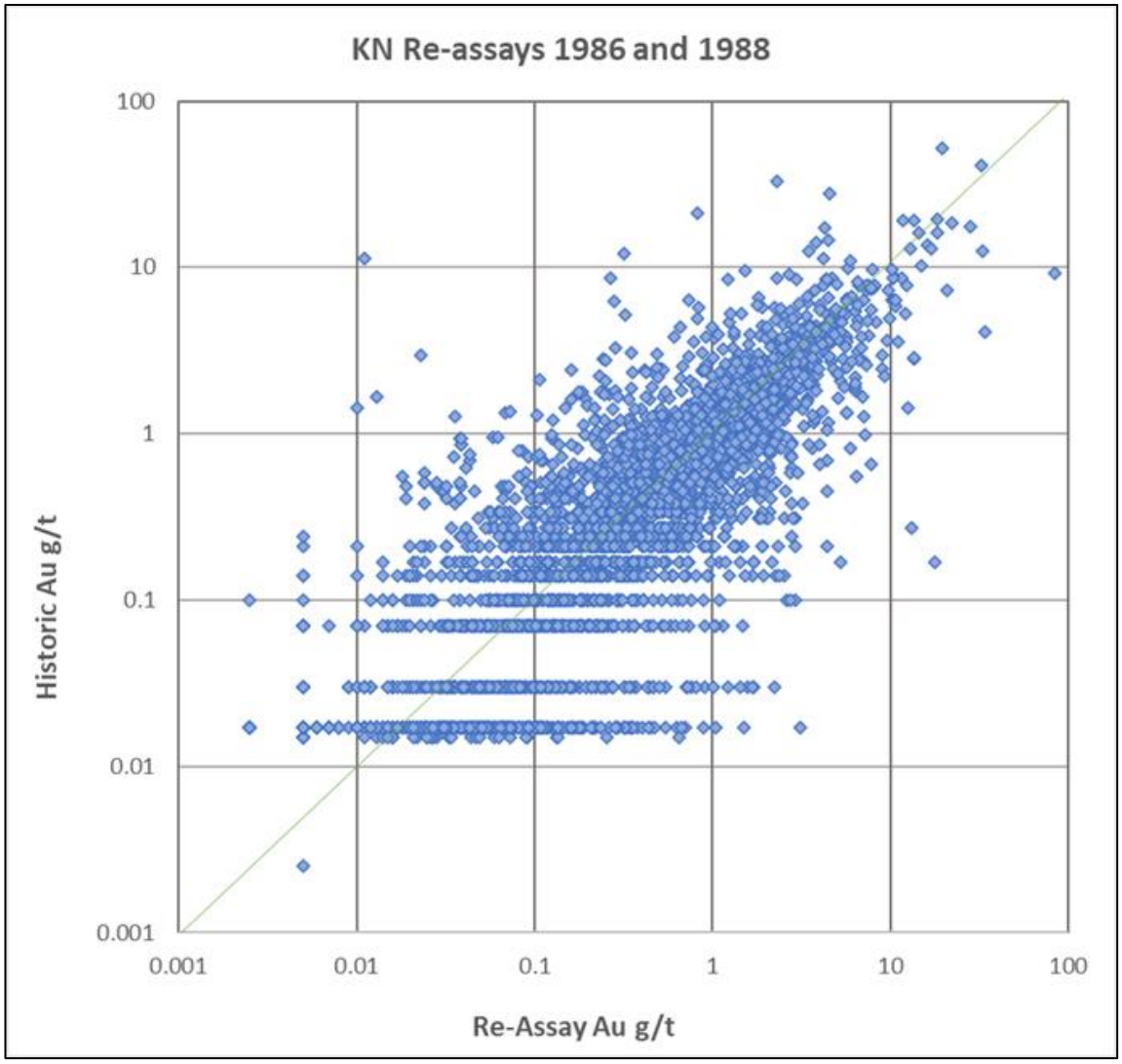
Source: MMTS, 2020

12.4.2.2 Kierens-Norlartic - Late 1980s Data

The statistics of the late 1980s data within the “potential pit” zone show less % difference in the mean and medians (see Table 12 3). The scatter plot of 1986 and 1988 data is given in Figure 12 10. The detection limit of the late 1980s data is lower than the early 1980s data, at 0.001opt or 0.034g/t. The data does show significant scatter likely attributed to the “nugget effect” but it also does not indicate significant bias, as it appears balanced around the 1:1 line.

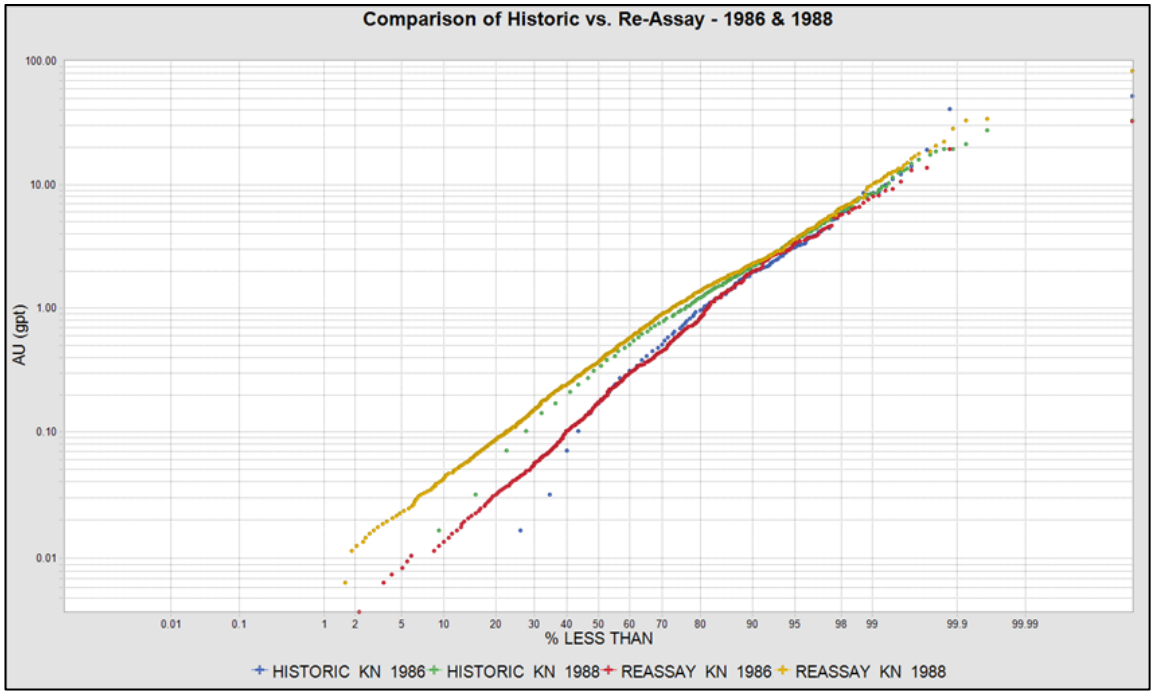
The CPP plot of all the late 1980s data is presented in Figure 12 27. Based on the agreement of the distributions of these data sets and the detection limit of 0.034 g/t in this more recent data, this later set of data is considered acceptable to be used for resource modelling.

Figure 12-10: Kierens-Norlartic Scatter Plot of 1986 and 1988 vs Re-Assay



Source: MMTS, 2020

Figure 12-11: CPP of Kierens-Norlartic Comparison of Au - 1986 and 1988 with Re-Assay

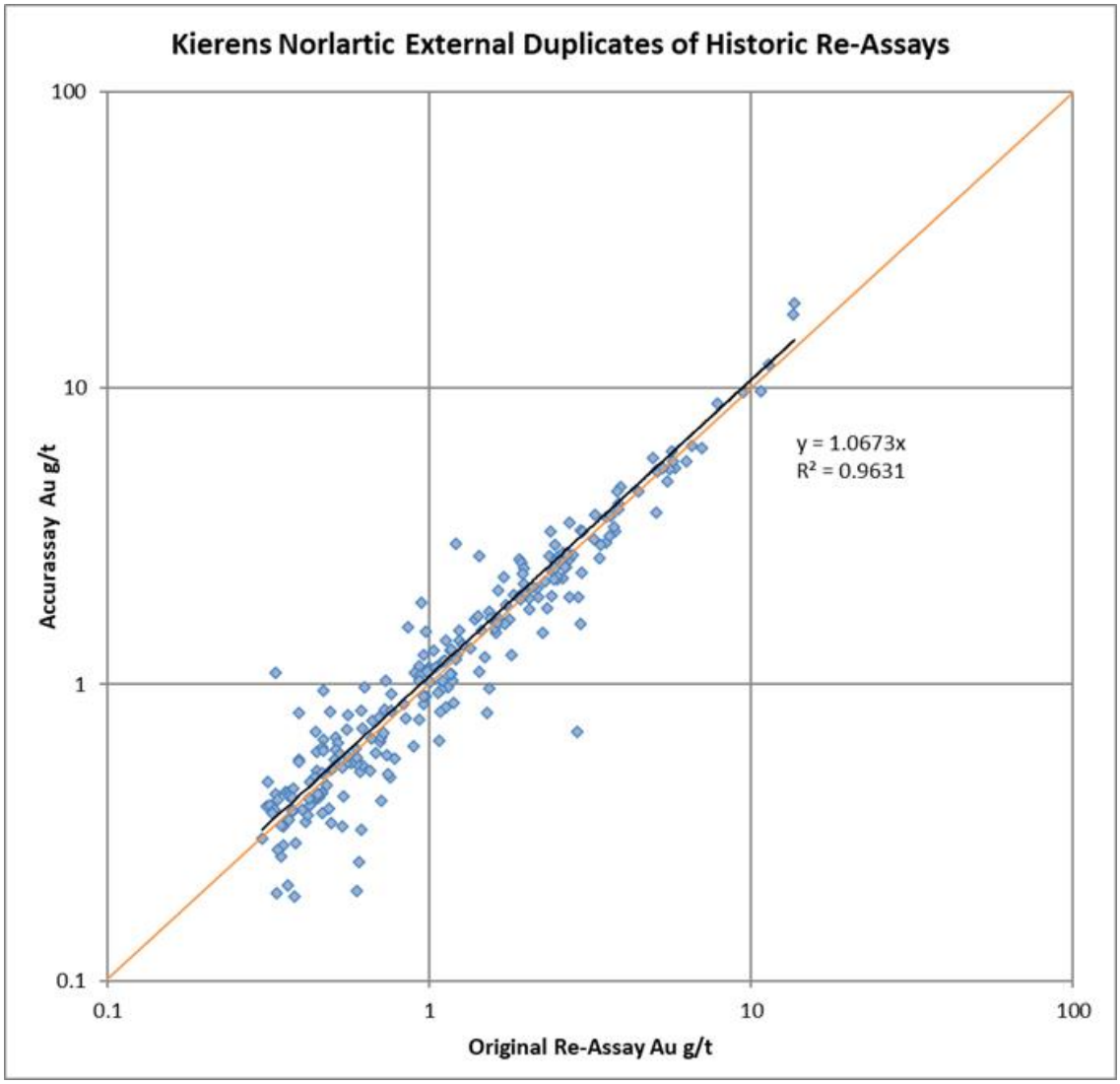


Source: MMTS, 2020

12.4.2.3 Kierens-Norlartic Historic Verification - External Duplicates

A set of 248 external duplicate re-assays by Accurassay was done in 2016 of pulps of historic re-assays with assays greater than 0.3g/t. Figure 12 12 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 with a slope greater than 1.0 indicating the Accurassay assays were slightly higher. The mean of the original historic re-assay data set is 1.78g/t and the Accurassay set is 1.82, an error of 2.6% higher. Because the error is in the conservative direction, meaning the values in the assay database are lower than the re-assay values, it is not of material concern for the purposes of resource estimation.

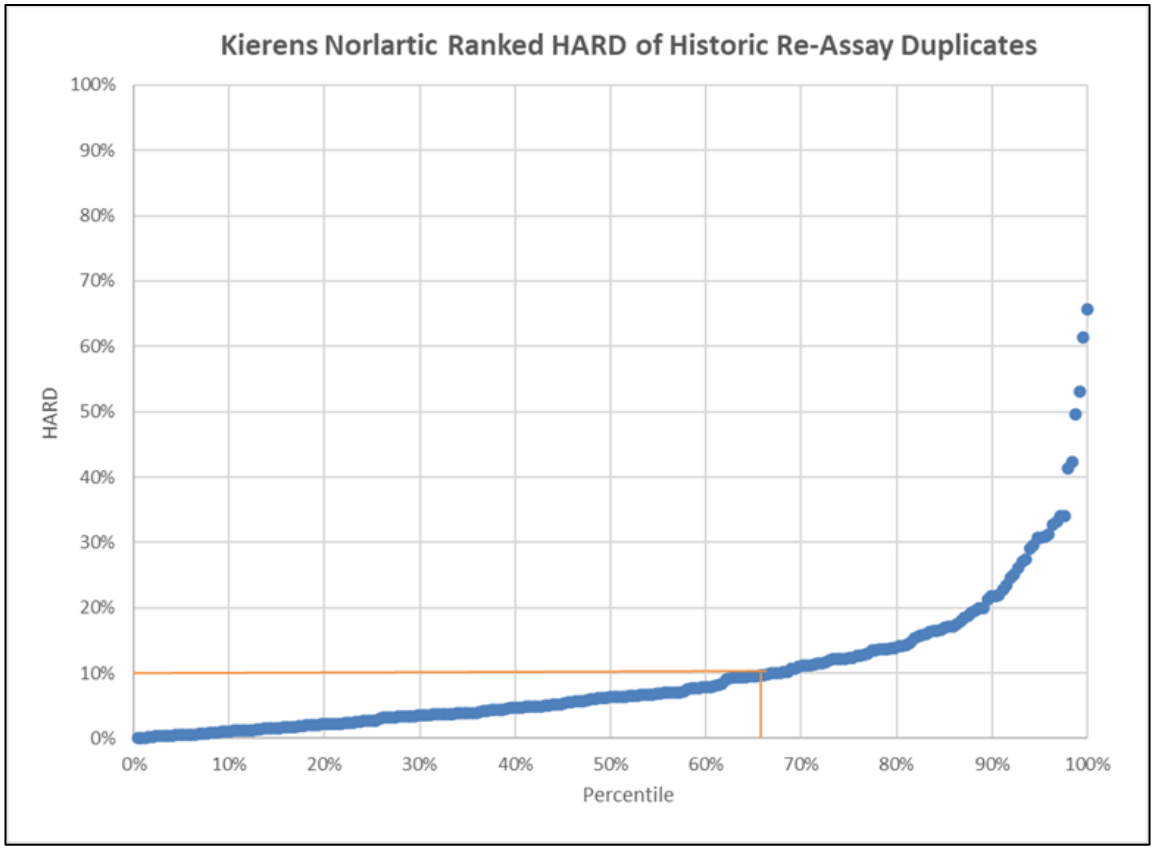
Figure 12-12: Kierens-Norlartic External Duplicates of Historic Re-Assays



Source: MMTS, 2020

Figure 12 13 gives the ranked “Half the Absolute Relative Difference” (HARD) plot for the external re-assays which shows approximately 34% greater than 10% HARD. This is more than the desired 10% for pulp duplicates, but not unexpected or unacceptable, given the nature of the gold mineralization in this deposit.

Figure 12-13: Kierens-Norlartic Ranked HARD of Historic Re-Assay External Duplicates



Source: MMTS, 2020

12.4.3 Specific Gravity

Specific gravity measurements have been completed at both areas on all the main rock types. A summary of the number of test and weighted mean sg values is provided in Table 12-4. These values have been coded to the resource model based on the lithologic model.

Table 12-4: Summary of sg Measurements by Area and Lithology

Area	Rock Type	Number of Samples	Min	Max	Weighted mean
Marban	Felsics	53	2.65	2.86	2.71
	Basalt	539	2.12	3.04	2.81
	Ultramafics	164	2.18	3	2.85
Kierens-Norlartic	Diorite	37	2.68	2.9	2.74
	Granodiorite	203	2.63	2.94	2.69
	Granodiorite-Komatiite	1	2.67	2.67	2.67
	Komatiite	37	2.59	2.96	2.70
	Komatiite-Diorite	86	2.66	2.98	2.77

12.4.4 Database Verification Conclusions

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 modeling because it is shown to be biased and unreliable in a range of significance to the modelling.
- The Kierens-Norlartic historic data earlier than 1986 is shown to be biased and unreliable and should not be included in the data set for resource modeling.
- The Marban data since the 1980s and the Kierens-Norlartic data from 1986 forward is considered of acceptable quality for resource modeling.
- All assay data subsequent to the dates above for both areas, including re-assays of the historic drilling has been included.

13 Mineral Processing & Metallurgical Testing

13.1 Introduction

Metallurgical testwork programs were conducted on mineralised samples from the O3 Mining Inc, between 2015 and 2017, as can be noted in Section 13.2. The testwork programs were performed on the Marban and Norlartic deposits.

The testwork programs investigated different processing options to develop a potential flowsheet, these include:

- comminution testing
- gravity separation (E-GRG)
- gravity and flotation of gravity tailing
- gravity and cyanidation of gravity tailing
- whole ore cyanidation

13.2 Source of Testwork Information

The following sources of technical and project information were referenced in developing the process plant design for the preliminary economic assessment:

- 2012 SGS Revised Preliminary Metallurgy Study Marban report 13333-001
- 2015 FLSmidth Gravity Circuit Modelling and scale-up report. NioGold Mining Corp. KRT20951
- 2015 Report – Marban Gold Mineralogy NG15-01
- 2016 SGS The Recovery of Gold from Marban Project Samples report 14947-002
- 2016 SGS The Grinding Circuit Design Based on Ten Bench-Scale Grindability Test results from the Marban Project report 149647-003
- 2016 SGS The Recovery of Gold from Norlartic Project Samples report 14947-004

13.3 Mineral Processing Testwork 2012

Preliminary metallurgical testing was carried out in 2012 at SGS located in Lakefield, Canada, on two composites named ECH-1 and ECH-2 using fifty drill core samples for each composite, collected from the Marban deposit.

The testwork program included head analysis, mineralogy, gravity separation and cyanidation and flotation testing on gravity tailings and whole ore cyanidation on the two composites.

Head analysis is summarised in Table 13-1. The direct gold head grades (by 1kg screened metallics +/- 150 mesh) were 1.24 g/t to 4.59 g/t for ECH-1 and ECH-2 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.

Table 13-1: Head Grades

Element		ECH #1	ECH #2
		Composite	Composite
Au	g/t	1.24	4.59
S	%	0.46	0.46
$S^{=}$	%	0.33	0.45
SO_4	%	0.1	< 0.1
S°	%	< 0.05	< 0.05
$C_{(t)}$	%	2.28	2.17
$C_{(g)}$	%	< 0.01	< 0.01
TOC	%	0.25	0.21
CO_3	%	11.0	10.6
ICP-Scan			
Ag	g/t	< 2	< 2
Al	g/t	57300	53900
As	g/t	< 30	< 30
Ba	g/t	56.5	57.5
Be	g/t	0.24	0.26
Bi	g/t	< 20	< 20
Ca	g/t	61400	61200
Cd	g/t	< 2	< 2
Co	g/t	54	46
Cr	g/t	323	328
Cu	g/t	157	97.8
Fe	g/t	74800	64800
K	g/t	1600	1700
Li	g/t	< 5	< 5
Mg	g/t	56100	46200
Mn	g/t	1330	1170
Mo	g/t	< 5	< 5
Na	g/t	20400	20800
Ni	g/t	328	272
P	g/t	256	202
Pb	g/t	< 20	< 20
Sb	g/t	< 10	< 10
Se	g/t	< 30	< 30
Sn	g/t	< 20	< 20
Sr	g/t	86.9	86.2
Ti	g/t	3280	3250
Tl	g/t	< 30	< 30
U	g/t	< 20	< 20
V	g/t	205	189
Y	g/t	15.9	14.1
Zn	g/t	69	63

Head samples from composites ECH 1 and ECH-2 were submitted for QEM-ARMS (automated rapid mineral scan). The QEM-ARMS results are shown in Table 13-2 and the mineral by mass percentage is shown in Table 13-3. The major minerals were silicates predominantly plagioclase

and the proportion of pyrite present in the ECH-1 and ECH-2 composites is 1.2% and 0.8% respectively.

Table 13-2: Summary of Qualitative X-ray Diffraction Results

Sample	Major	Moderate	Minor	Trace
(1) ECH #1	plagioclase	quartz, chlorite, dolomite, calcite	talc	*mica, *potassium feldspar,*amphibole
(2) ECH #2	plagioclase	quartz, chlorite, dolomite, calcite	talc	*mica, *potassium feldspar, *amphibole

** tentative identification due to low concentrations, diffraction line overlap or poor crystallinity*

Table 13-3: Mineral by Mass Percentage

	Survey	Aurizon Mines - ECH	
	Project	13333-001/ MI6000-SEP11	
	Sample	ECH# 1	ECH# 2
Mineral Mass (%)	Pyrite	1.2	0.8
	Other Sulphides	0.1	0.0
	Fe-Ti Oxides	1.1	1.0
	Titanite/Sphene	0.5	0.8
	Quartz	19.5	25.2
	Chlorites	24.7	22.2
	Plagioclase	22.9	25.6
	K-Feldspar	0.4	0.2
	low-Fe Dolomite	11.1	8.4
	Calcite	6.7	7.6
	Talc	8.5	4.4
	Micas	1.5	2.2
	Amphibole	1.4	1.5
	Clays	0.2	0.3
	Other	0.1	0.1

The laboratory gravity separation testwork equipment included a Knelson concentrator with a Mozley table upgrade at an 80% passing size (k_{80}) of 150 µm for each composite. The recovery of gold was 41.3% and 56.5% for the ECH-1 and ECH-2 composites respectively, with the higher-grade sample reporting the higher recovery.

The cyanidation testing examined whole ore and gravity tailings using the following test conditions:

- Cyanide (NaCN) concentration 0.5 g/L
- Slurry density 40%
- pH range 10.5 to 11.0
- 48-hour retention time (kinetic samples at 6 and 24 hours)
- Grind targets (k_{80} values) approximately 160, 80 and 60 µm

The whole ore leach tests shown in Table 13-4 on two composites and displayed gold extractions which ranged from 95.4 to 97.6% for the low grade head grade composite (ECH 1), the high grade composite (ECH-2) was 95.7 to 97.3%.

Table 13-4: Whole Ore Leach Test Results

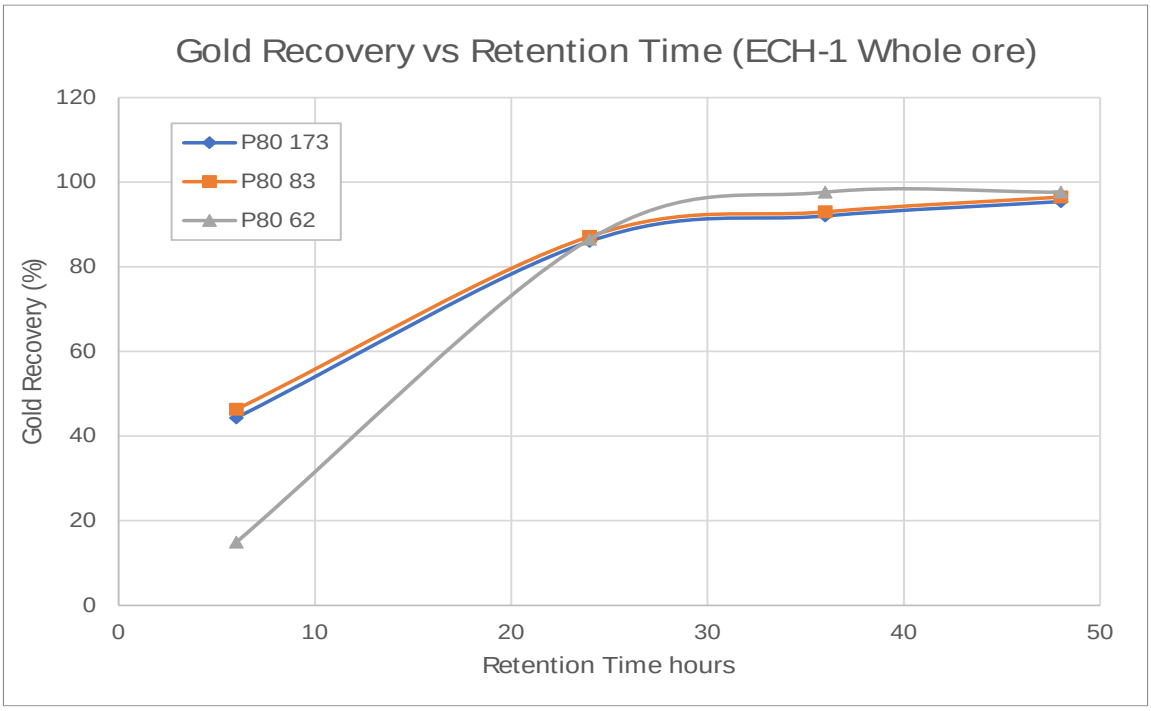
Sample	Test No.	Size P_{80} µm	Reagent Addition kg/t CN Feed		Regent Cons. kg/t CN Feed		% Extraction Au			Residue g/t Au	Head (calc) g/t Au	Head (direct) g/t Au
			NaCN	CaO	NaCN	CaO	6h	24h	48h			
ECH-1	CN 1	173	0.95	0.43	0.18	0.43	44.3	86.0	95.4	0.06	1.19	1.24
ECH-1	CN 2	83	0.96	0.47	0.27	0.47	46.3	87.2	96.5	0.04	1.15	1.24
ECH-1	CN 3	62	1.95	0.29	1.26	0.25	14.9	86.5	97.6	0.03	1.24	1.24
ECH-2	CN 4	153	0.89	0.40	0.16	0.40	37.2	76.6	95.7	0.22	4.95	4.59
ECH-2	CN 5	82	1.26	0.33	0.51	0.30	33.0	83.6	97.3	0.13	4.82	4.59
ECH-2	CN 6	62	1.95	0.27	1.20	0.24	18.2	76.2	97.0	0.15	4.90	4.59

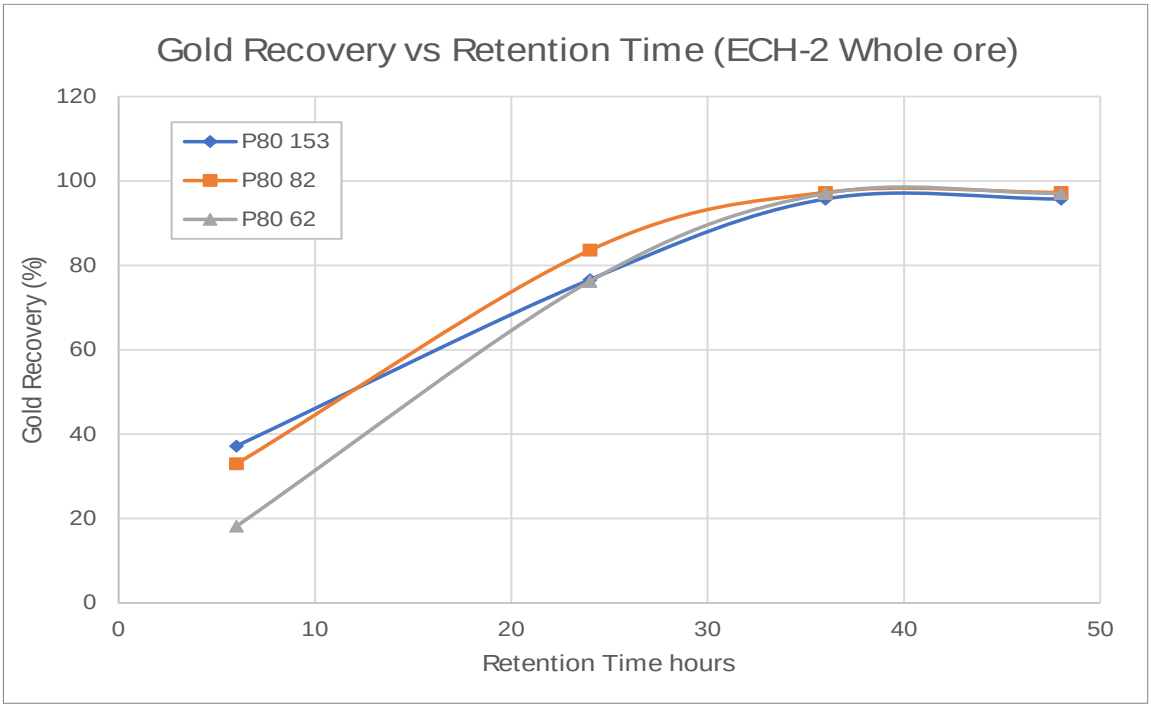
All cyanidations were conducted at 40% solids, 0.5g/L NaCN, pH 10.5-11.0 and for 48 hours

The leach kinetics for the whole ore leach tests are shown in Figure 13-1 indicate that for the low grade head grade composite (ECH 1) the gold extraction does not increase significantly between 30 to 48 hours. Both composites indicate that after 24 hours, grind size did not influence gold extraction.

The leach tests on the gravity tailings appear to have examined the same grind size and report very similar residue grades for the respective targeted grind size, as illustrated in Table 13-5. Gold extraction ranged from 88.0% to 95.2%. The overall gold extraction ranged from 92.8% to 97.5%. The leach kinetics of the gravity tailings shown in Figure 13-3 also show similar gold extractions over time compared to whole ore leaching.

Figure 13-1: Whole Ore Leach Kinetics





Source: SGS Revised Preliminary Metallurgy Study Marban report 13333-001 (2012)

Table 13-5: Gravity Tailings Leach Test Results

Sample	Test No.	Size P_{80} µm	Reagent Addition kg/t CN Feed NaCN	CaO	Regent Cons. kg/t CN Feed NaCN	CaO	% Extraction Au 6h	24h	48h	Residue g/t Au	Head (calc) g/t Au	Head (direct) g/t Au	Au Extraction Gravity + CN %
ECH-1	CN 7	134	0.80	0.42	0.09	0.42	57.8	85.4	87.8	0.07	0.57	0.60	92.8
ECH-1	CN 8	76	0.94	0.43	0.21	0.43	55.7	86.2	95.2	0.03	0.63	0.60	97.2
ECH-1	CN 9	54	1.59	0.30	0.85	0.28	39.4	86.8	94.0	0.04	0.59	0.60	96.5
ECH-2	CN 10	135	0.84	0.40	0.09	0.40	51.4	79.5	88.0	0.22	1.79	1.83	94.8
ECH-2	CN 11	79	1.03	0.42	0.26	0.42	51.3	83.5	94.2	0.11	1.82	1.83	97.5
ECH-2	CN 12	59	1.52	0.32	0.77	0.31	38.0	79.4	92.4	0.14	1.85	1.83	96.7

All cyanidations were conducted at 40% solids, 0.5g/L NaCN, pH 10.5-11.0 and for 48 hours..

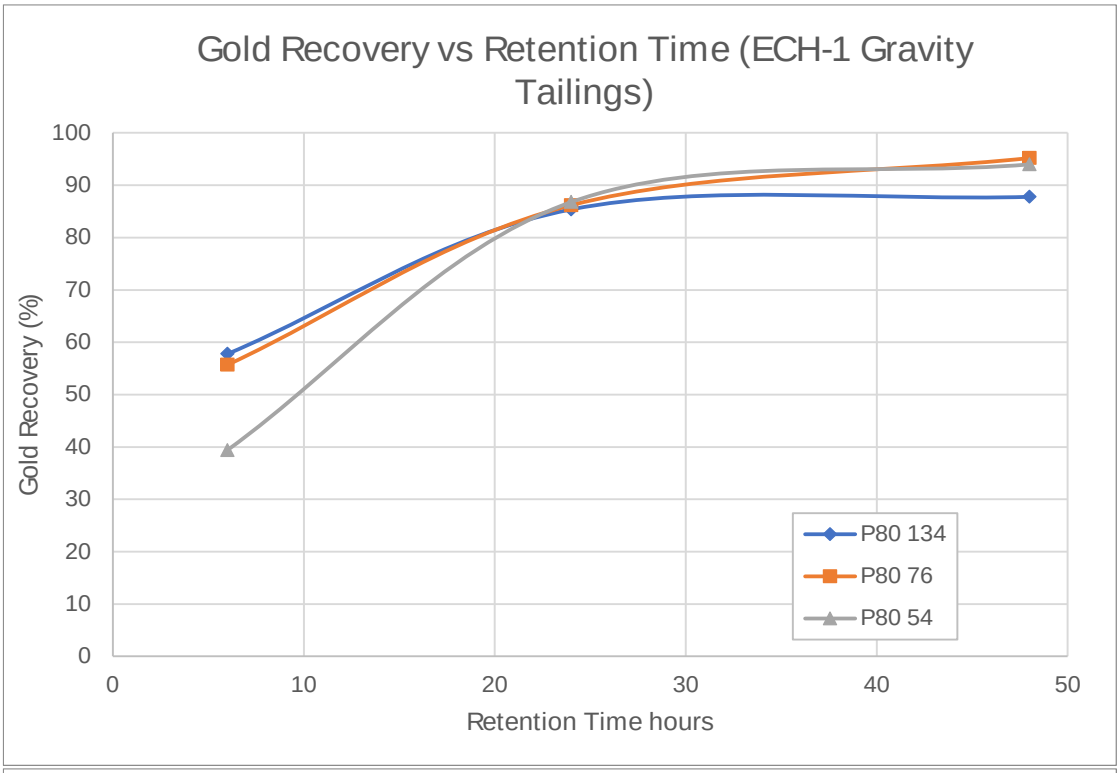
13.4 Mineral Processing Testwork 2015

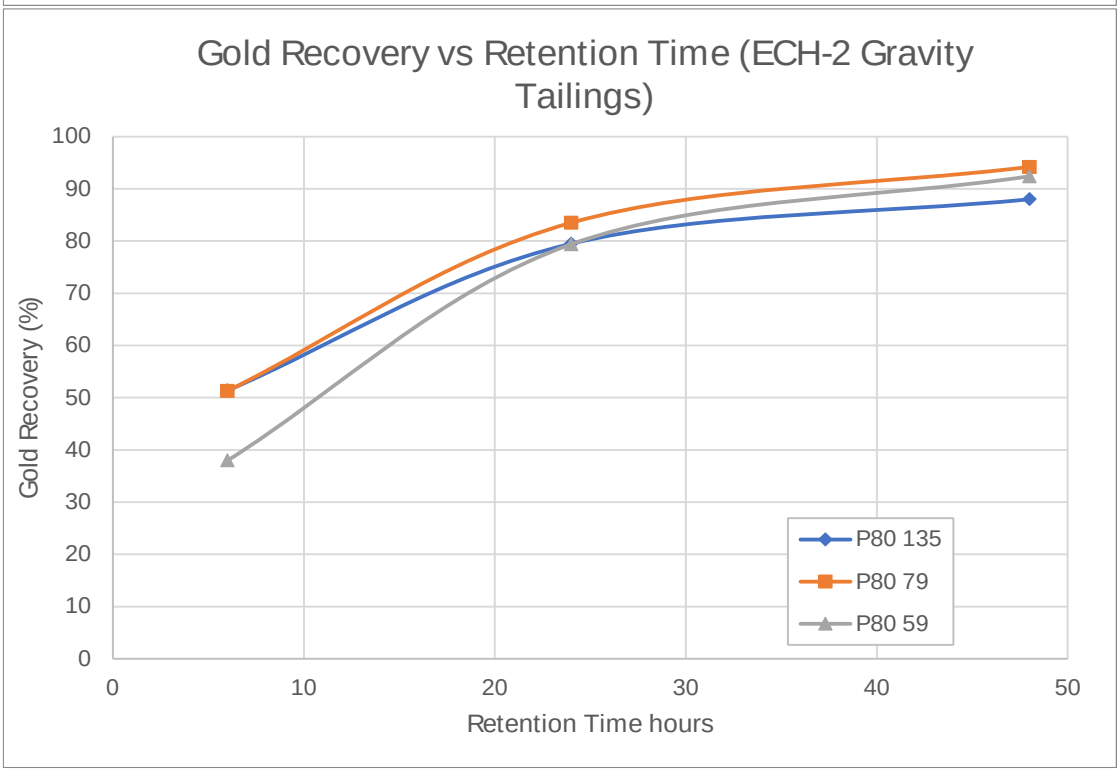
A Gravity testing campaign was carried out in 2015 at FLSmidth located in Dawson Laboratories in Salt Lake City, USA. Ten composites named GRAV-01 to GRAV-10 were prepared from a total of 281 drill cores. E-GRG recovery was estimated and was simulated for three different 80% passing targets. The results are presented in Table 13-5. Equipment recommendations are included in that analysis.

Figure 13-2: Gravity Testing Summary

Knelson units	% of Circulating load treated	Tons Treated t/h	Grind Size (p80 microns)	Upgrade step	Recovery Total Gold (%)	Knelson Con produced (Kg/day)	Acacia size
1 X QS48	24	350	100	Acacia	**31**	2210	CS2000
			100	Table	**25**		NA
			150	Acacia	**24**		CS2000
			150	Table	**18**		NA
			200	Acacia	**19**		CS2000
			200	Table	**15**		NA
2 X QS48	49	700	100	Acacia	**40**	4420	CS3000
			100	Table	**31**		NA
			150	Acacia	**32**		CS3000
			150	Table	**24**		NA
			200	Acacia	**27**		CS3000
			200	Table	**21**		NA
4 X QS48	95	1400	100	Acacia	**48**	8840	CS6000
			100	Table	**37**		NA
			150	Acacia	**42**		CS6000
			150	Table	**32**		NA
			200	Acacia	**36**		CS6000
			200	Table	**27**		NA

Figure 13-3: Gravity Tailings Leach Kinetics





Source: SGS Revised Preliminary Metallurgy Study Marban report 13333-001 (2012)

13.5 Mineral Processing Testwork 2016

13.5.1 SGS 104947-002 Cyanidation Testing

In Q1CY2016 SGS (Lakefield) performed cyanidation testing examining the effect of pre-aeration on gold extraction and reagent consumption. The MBR-1 to MBR-100 composites were made from combining 511 of separate drill core samples ranging from 78365 to 699847.

The program consisted of 211 individual whole ore leach tests and examining the application of pre-aeration (1 hour) using the following test conditions:

- NaCN concentration of 0.5 g/L
- Slurry density of 40%
- pH range 10.5 to 11.0
- 36-hour retention time
- Grind target k_{80} of 65 µm

The results of the cyanidation testing is summarised in Table 13-6. The results indicate the application of a pre-aeration stage prior to cyanide leaching in terms of reagent consumption and an improvement in the average gold extraction.

Table 13-6: Cyanidation Testing Summary (Pre-aeration Examination)

Results	P_{80}	Reag. Add'n., kg/t		Reag. Cons., kg/t		Extr'n	Residue Assay, g/t Au			Head, calc
	µm	NaCN	CaO	NaCN	CaO	Au, %	1	2	Avg.	Au, gt
Tests with Preaeration										
Average	64	0.79	0.69	0.07	0.66	91.1	0.14	0.22	0.17	1.42
Max	76	0.92	1.15	0.18	1.12	97.9	0.79	6.54	3.33	4.37
Min	51	0.75	0.50	0.01	0.48	67.6	0.01	0.01	0.02	0.24
Tests without Preaeration										
Average	65	0.98	0.62	0.26	0.56	87.3	0.20	0.20	0.20	1.44
Max	77	1.55	1.06	0.80	1.00	98.1	1.44	1.00	0.86	4.39
Min	57	0.68	0.26	0.01	0.21	61.5	0.02	0.02	0.02	0.22

13.5.2 SGS 104947-002 Comminution Testing

In 2016 SGS (Lakefield) composited 16 samples named MBR-1 to MBR-16 and these were submitted for comminution testing. The samples represent both Marban (samples MBR-1 to MBR-10) and Norlartic (samples MBR-11 to MBR-16). The Marban samples are from drill holes MBPQ-15-003, MBPQ-15-005, MBPQ15-007, MBPQ-15-008 and MBPQ-15-009. The Norlartic samples are from drill holes NLPQ-16-001, NLPQ-16-002 and NLPQ-16-003. The comminution testing included:

- Bond Low-energy impact test (CWI)

- SMC (abbreviated version of the JW Drop Weight test)
- Note: (one sample was submitted for full drop weight testing)
- Bond Rod Mill Work Index (RWI)
- Bond Ball Mill Work Index (BWI)

The results are presented in Table 13-7. The crushing work index (CWI) of the samples ranged from 11.3 to 23.3 kWh/t. The A x b value for the SMC test on the samples ranged from 89.5 to 23.8, which is classified from very low competency to extremely competent. The Bond Rod Mill work index (RWI) ranges from 10.3 to 16.5 kWh/t. The Bond Ball Mill work index (BWI) ranges from 9.4 to 13.5 kWh/t.

Table 13-7: Comminution Testing Summary

Statistics	Relative	JK Parameters			Work Indices (kWh/t)		
	Density	A x b	t_a	SCSE	CWI	RWI	BWI
Average	**2.78**	**38.5**	**0.53**	**10.59**	**15.1**	**14.1**	**12.0**
Std. Dev.	0.03	14.8	0.21	1.28	3.2	1.9	1.5
Rel. Std. Dev.	1	39	40	12	21	13	13
Minimum[1]	2.72	89.5	1.13	7.23	11.3	10.3	9.4
10th Percentile	2.74	45.0	0.69	9.50	11.6	12.2	10.4
25th Percentile	2.76	40.0	0.63	10.04	13.3	12.6	10.8
Median	**2.78**	**35.6**	**0.54**	**10.66**	**14.7**	**15.0**	**11.6**
75th Percentile	2.80	31.8	0.42	11.20	16.6	15.5	13.2
80th Percentile	2.80	30.5	0.34	10.91	16.9	15.6	13.5
90th Percentile	2.81	28.2	0.28	11.89	18.4	16.0	13.8
Maximum[1]	2.85	23.8	0.26	12.92	23.3	16.5	14.7

[1]Minimum and Maximum refer to the softest and hardest results, respectively

13.5.3 SGS 104947-002 Cyanidation Testing

In 2016 SGS (Lakefield) performed cyanidation testing on additional samples using with pre-aeration added to the leach test conditions described in section 13.5.1. The MBR-101 to MBR-135 composites were made from combining 433 of separate drill core samples ranging from 557803 to 742059.

The program consisted of 35 individual whole ore leach tests using the following test conditions:

- NaCN concentration of 0.5 g/L
- Slurry density of 40%

- pH range 10.5 to 11.0
- 36-hour retention time
- Grind target k_{80} of 65 µm

The average results from these 35 tests for gold extraction was 93.2% and reagent consumptions of 0.05 kg/t of cyanide and 0.53 kg/t of lime. These test results do not significantly differ from the previous testwork.

13.6 Other Testwork

- 2015 – COREM conducted HPGR testing and gravity testing coarse bottle rolls testing and flash flotation on the HPGR discharge.
- 2016 – FLSmidth conducted flash flotation testing
- 2017 – McClelland Laboratories conducted agglomeration testing, coarse bottle rolls and column leaching.

13.7 Comminution Design Parameters

The comminution tests presented in section 13.5.2 were used to develop parameters for the crushing and grinding circuit design. The details of the parameters are presented in Table 13-8. The designed values are derived from the 75th percentile of the testing completed.

Table 13-8: Comminution Design Parameters

Design Parameter	Units	Value
Bond Crusher Work Index (CWi)	kWh/t	16.5
SMC	A x b	31.8
Bond Rod Mill Work Index (RWi)	kWh/t	15.5
Bond Ball Mill Work Index (BWi)	kWh/t	13.2

13.8 Gold Recovery

The flowsheet selected from the testwork includes standard gravity concentration and cyanide gold processing plant to produce doré.

The gold recoveries for unit operations and overall is presented in Table 13-9.

Table 13-9: Gold Recovery

Unit Operation	Units	Value
Gravity	%	30.4
Leaching	%	92.5
Overall Gold Recovery	**%**	**93.9**

13.9 Reagent Consumption

The reagent additions for processing were derived from the testwork and are displayed in Table 13-10. The cyanide and, lime addition in the leaching is based on the outcome of the pre-aeration tests discussed in 13.5.1. The cyanide destruction reagent dosages were determined from the calculated CN_{WAD} at the end of the leach and comparison with stoichiometric dosages and conservative estimates.

Table 13-10: Reagent Additions

Reagent	Units	Value
Cyanide Addition	kg/t	0.77
Lime Addition	kg/t	1.12
Oxygen consumption	t O_2/d	8.5
Sulphur addition to cyanide destruct	g/g CNWAD	6.0
Lime addition to cyanide destruct	g/g SO_2	0.75
Copper addition to cyanide destruct	ppm Cu+2	20
Oxygen addition to cyanide destruct	g/g CNWAD	3.0
Flocculant to Tailings Thickener	g/t	25

13.10 Mineral Processing and Metallurgical Testing Conclusions

In the opinion of the QP:

- The samples selected and used for the metallurgical testing are appropriate for use in testing to support a preliminary economic analysis.
- The metallurgical testing included in this report is suitable for preliminary flowsheet selection and recovery estimation.
- Reagent consumptions derived from testing are reasonable for the nature of the samples tested.
- Assumed reagent consumptions used for process operating costs are typical for similar operations.

14 Mineral Resource Estimates

The Mineral Resource estimate for O3 Mining's Marban and Kierens-Norlartic deposits have been updated since the previous estimate in 2016 due to updated geologic modelling, updated QAQC and updated gold prices. The Mineral Resource with an effective date of 12 August, 2020 is listed in Table 14-1. Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (CIM, 2014) were followed for the Mineral Resource Estimate, as well as the updated CIM Guidelines (CIM, 2019).

The base case cut-off grade is 0.30 g/t Au based on a cut-off grade mill recovery of 87%, Processing + G&A costs of CDN$15/tonnes and a US$1400/oz Au price, with smelter terms as detailed in the notes below. The underground resource is within a 3.5 g/t gradeshell. At these cut-offs, the total Measured and Indicated Mineral Resource is estimated at 54.1 Mt at 1.10 g/t Au for a total of 1.9 Moz.

Table 14-1: Mineral Resource Estimate (effective date: 12 August, 2020)

Class	Source	Tonnage (kt)	Au (g/t)	Au Metal Ounces
Measured	Marban - Pit	65	1.32	2,762
	Kierens-Norlartic - Pit	450	1.03	14,900
Indicated	Marban - Pit	46,260	1.03	1,536,671
	KN-Pit	6,646	1.15	246,430
	Marban - UG	220	7.77	54,982
	Kierens-Norlartic - UG	510	3.57	58,504
Meas. + Ind.	All	54,151	1.10	1,914,249
Inferred	Marban - Pit	6,465	1.09	227,226
	Kierens-Norlartic - Pit	6,299	1.42	286,724
	Marban - UG	304	8.73	85,317
	Kierens-Norlartic - UG	119	3.02	11,560
	All	13,187	1.44	610,827

Notes:

1. *The Mineral Resource estimate has been prepared by Sue Bird, P.Eng., an independent Qualified Person.*
2. *Resources are reported using the 2014 CIM Definition Standards and were estimated in accordance with the CIM 2019 Best Practices Guidelines.*
3. *Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.*
4. *The open pit Mineral Resource has been confined by a "reasonable prospects of eventual economic extraction" pit shell generated using the following assumptions: US$1,800/oz. Au at a currency exchange rate of 0.75 US$ per CDN$; 99.95% payable Au; CDN$4.30/oz Au offsite costs (refining, transport and insurance); a 3% NSR royalty; $16/t process and G&A costs; $2.60/t mining costs; grade dependent mill process recoveries; and pit slopes varying from 25 to 50 degree overall depending on geotechnical characteristics.*
5. *The underground Mineral Resource reports all material within a continuous 3.5g/t Au gradeshell*
6. *Metallurgical recovery is based on the formula: ln(Au)*0.0372+0.9017, maximum 96.7%*
7. *The specific gravity of the deposit has been determined by lithology as being between 2.67 and 2.81.*
8. *Numbers may not add due to rounding.*

The QP 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.

14.1 Key Assumptions and Basis of Estimate

The total number of holes completed for the entire Marban and Kierens-Norlartic Project 5,809 as summarized in Chapter 10. The drilling by deposit area within each of the resource model bounds and used for the Resource Estimate is summarized in Table 14-2 and Table 14-3 for Marban and Kierens-Norlartic respectively.

The assay database has been limited during interpolation based on the QAQC presented in Section 11 and data verification presented in Section 12 to exclude data prior to 1980 based on detection limit issues and a bias in the Au grade for the 1960s and earlier drilling. This was noted in analysis done for this report and by previous authors (Belzile, 2016; Gustin and Ronning, 2013). The data has also been limited to exclude high grade assays within or immediately adjacent to previous underground workings and stopes. This is considered prudent due to uncertainties in the exact location of historic workings and also to avoid over-estimation by inclusion of already mined high grade zones. Where there has been drilling adjacent to stopes since underground mining for each area, the assay data has been used for the Resource Estimate.

Table 14-2: Summary of DHs and Assay used for the Marban Resource Estimate

Year	Number of DHs	Number of Assays	Total Metres with Au Assays
1960	11	756	1,028.22
1986	8	625	906.55
1987	6	1,661	2,060.40
1988	3	997	1,273.13
1993	4	264	302.24
2006	2	1,129	1,005.30
2007	27	11,248	10,492.95
2008	61	19,413	18,356.70
2010	44	12,832	11,743.65
2011	94	28,194	25,403.30
2012	79	26,509	24,375.36
2013	1	467	406.00
2014	36	12,859	11,312.45
2015	210	55,961	51,146.99
2016	1	335	288.60
2017	4	610	557.40
Total	591	173,860	160,659.24

Table 14-3: Summary of DHs and Assay used for the Kierens-Norlartic Resource Estimate

Year	Number of DH	Number of Assays	Total Metres with Au Assays
1958	5	102	96.90
1964	8	342	398.41
1980	3	102	115.94
1983	1	1	6.10
1986	318	23,356	24,083.16
1987	93	5,950	5,344.55
1988	321	14,608	13,899.37
1989	1	24	26.88
1990	369	18,058	16,430.06
1993	1	74	80.47
2006	63	5,253	4,688.85
2007	5	461	434.85
2011	31	6,623	5,729.26
2012	9	4,002	3,641.80
2013	32	11,641	10,622.41
2014	10	2,412	2,118.60
2016	2	332	299.40
Total	**1,272**	**93,341**	**88,017.01**

Separate block models were created for Marban and Kierens-Norlartic with a block size of 5m x 5m x 5m. Block model extents are presented in Tables 14-4 and 14-5 for each deposit area. In each case the model is a “mineralized percent model” with up to two mineralized domains per block. Both the Au grade for each domain and the percent of each block within each domain is stored in the model and used for the Resource Estimate.

Table 14-4: Marban Model Extents

Direction	Minimum	Maximum	Block size	# Blocks
Easting	277,500	279,150	5	330
Northing	5,335,300	5,336,500	5	240
Elevation	-425	325	5	150

Table 14-5: Kierens-Norlartic Model Extents and Rotation

Direction	Minimum	Maximum	Block size	# Blocks
Easting	0	1,500	5	300
Northing	0	2,300	5	460
Elevation	-425	325	5	150
Rotated:	(ccw)	-56.75		
Origin:	277,543.59E	5,336,206.50N		

Similarly, the percent of each block below the topography, below the overburden surface and within each major lithology is stored to limit interpolation to only below the overburden and to assign sg according to lithology. The underground voids provided by the client have been used to also code the percent of the block previously mined with the mineralized portion of the block calculated as the total percent within domains minus the total percent within voids.

14.2 Geologic Model

The geologic models for each of deposit areas consisted of creating solids for potentially mineralized zones. Mineralization within each of the deposits is within interpreted regional shears which acted as conduits to fluid flow. Mineralization and the relation of the geology to the potentially mineralized wireframes used in the block model interpolation are illustrated and discussed in detail in Section 7 of this report with the wireframes and corresponding search ellipses used during interpolation illustrated in later in this section.

Lithology solids as provided by Kenneth Williamson for Marban and by O3 Mining for Kierens-Norlartic have been used to code the block model with up to two lithologies per block and assign an sg value based on the lithology, as discussed later in this section.

14.3 Assays, Composites, Capping and Outlier Restriction

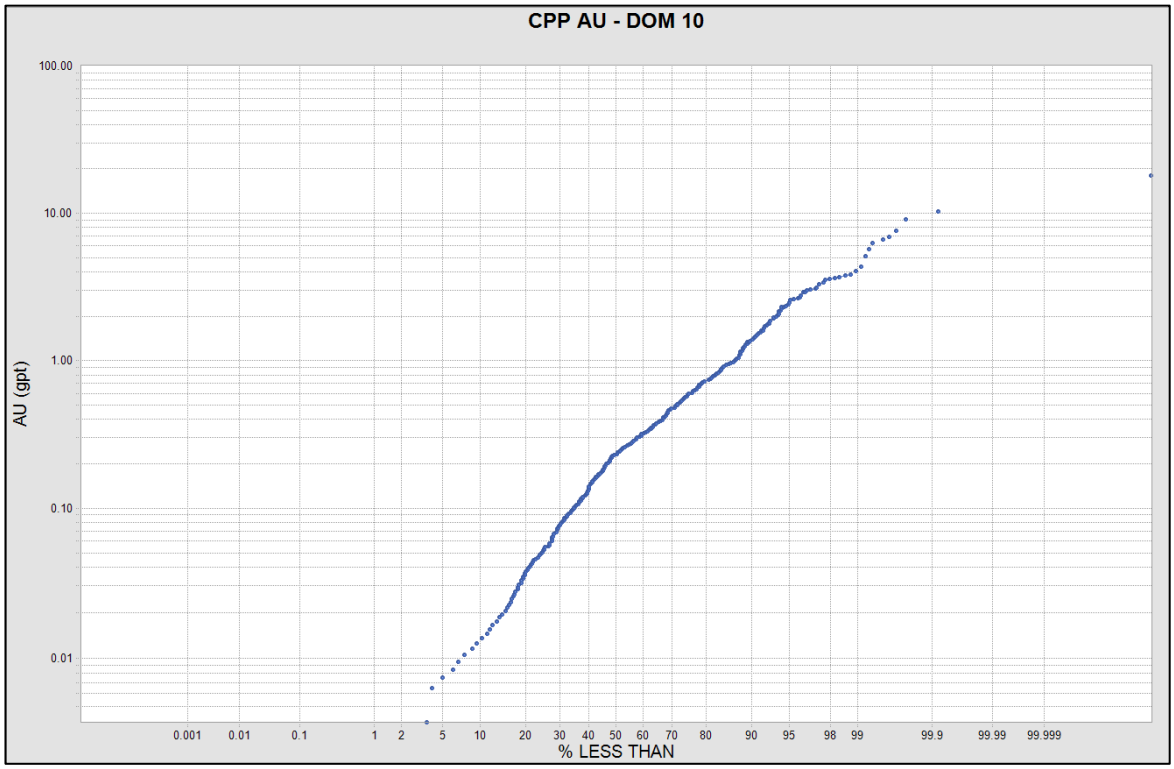
The assays and composites have been examined statistically using boxplots, histograms, and cumulative probability plots (CPP) to determine the capping values, the composite lengths, and to aid in determining the outlier restriction values. Comparison of the assay and composite statistics is done to ensure compositing has introduced no bias and that the data shows a reasonable distribution for use of linear interpolation methods.

The grade distribution for Au within the modelled grade shells is generally lognormal except at very low grades approaching the lower detection limits and at the upper end of the distribution where high-grade outliers are apparent. Capping of the Au assays has been implemented as summarized in Table 14-6 to limit high grade outliers, as indicated by the CPP plots shown as examples for each deposit. Figures 14-1 and 14-2 are examples from Marban from two of the main domains. Figure 14-3 shows all of the Kierens-Norlartic data, the majority of which is capped at 100g/t, with figure 14-4 illustrating the CPP as justification for the lower capping value required for domain 200 of Kierens-Norlartic. Table 14-6 also summarizes the Outlier Restriction used on the composites during interpolation, with the outlier value and distance that the value is used in the interpolations both summarized.

Table 14-6: Assay Capping and Composite Outlier Restriction

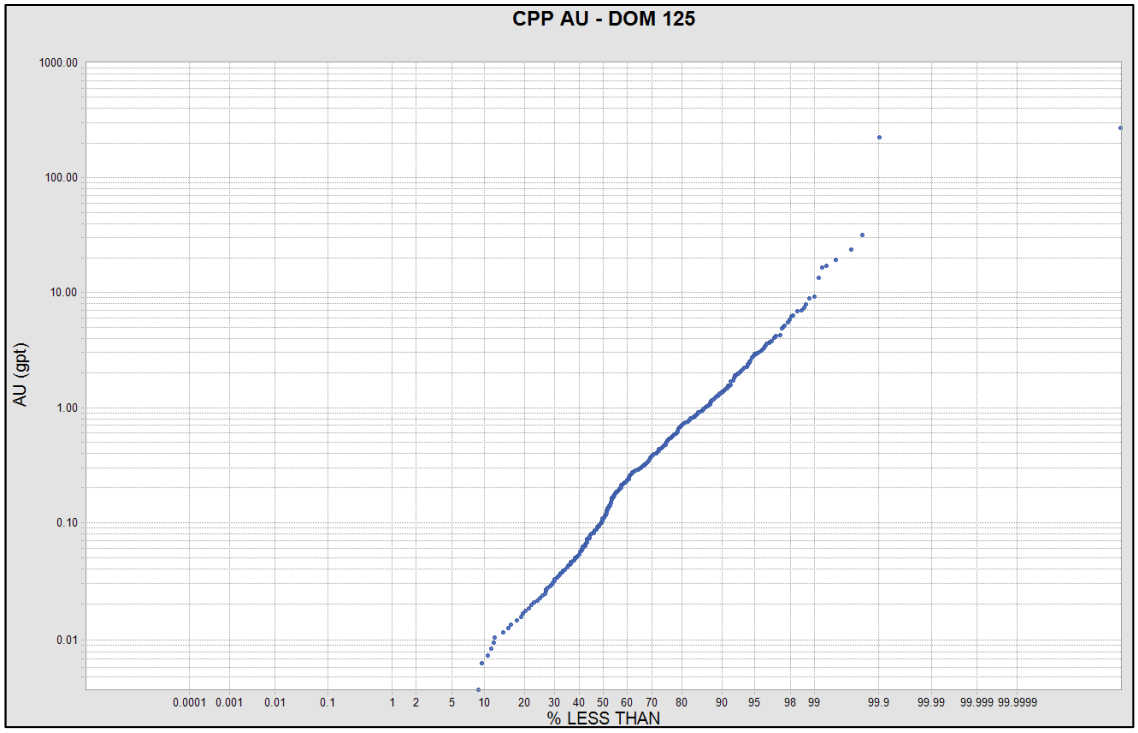
Marban				Kierens-Norlartic			
DOMAIN	Au Cap value (g/t)	Au Outlier Restriction (g/t)	Outlier Distance (m)	DOMAIN	Au Cap value (g/t)	Au Outlier Restriction (g/t)	Outlier Distance (m)
4	10	---	10	100	100	---	10
5	35	---	10	101	100	25	10
6	20	---	10	102	100	1.5	10
7	50	---	10	103	100	7	10
10	15	10	10	104	100	---	10
15	50	50	10	105	100	---	10
16	30	---	10	106	100	---	10
20	60	---	10	200	20	---	10
30	---	10	10	201	100	10	2
60	10	---	10	202	100	4	2
70	80	---	10	203	100	1.5	2
100	50	---	10	312	100	2.5	2
110	30	---	10	315	100	5	2
111	30	10	10	316	100	---	10
117	30	20	10	322	80	5	10
120	50	20	10	323	80	---	10
125	100	20	10				

Figure 14-1: CPP – Domain 10 – Marban (source: MMTS, 2020)



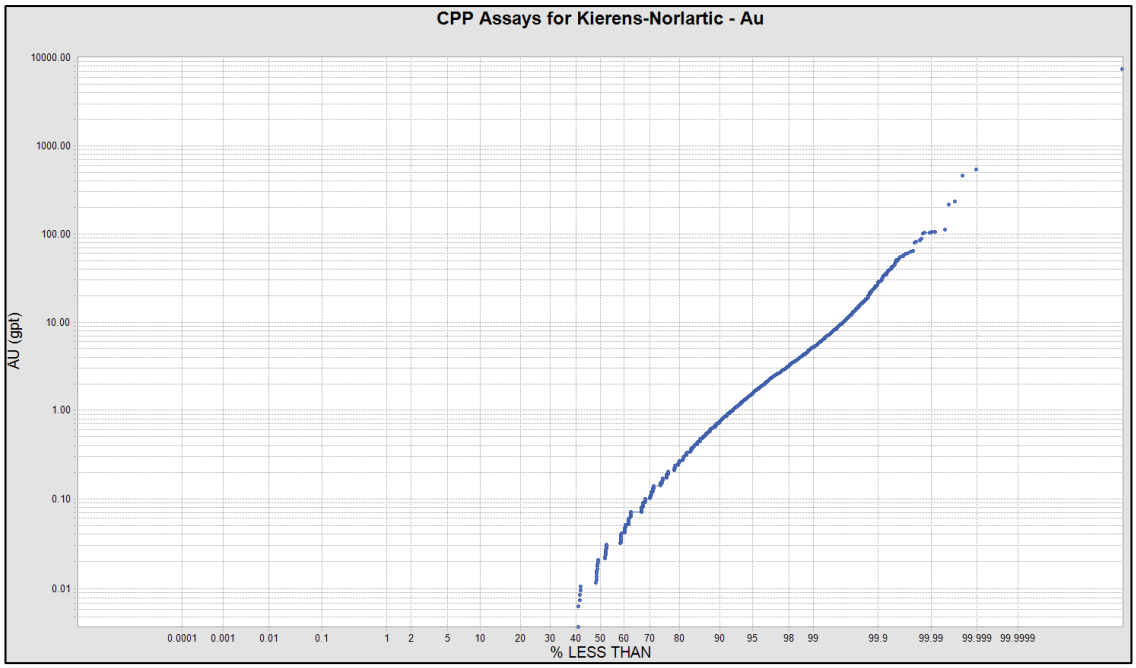
Source: MMTS, 2020

Figure 14-2: CPP – Domain 125 – Marban (source: MMTS, 2020)



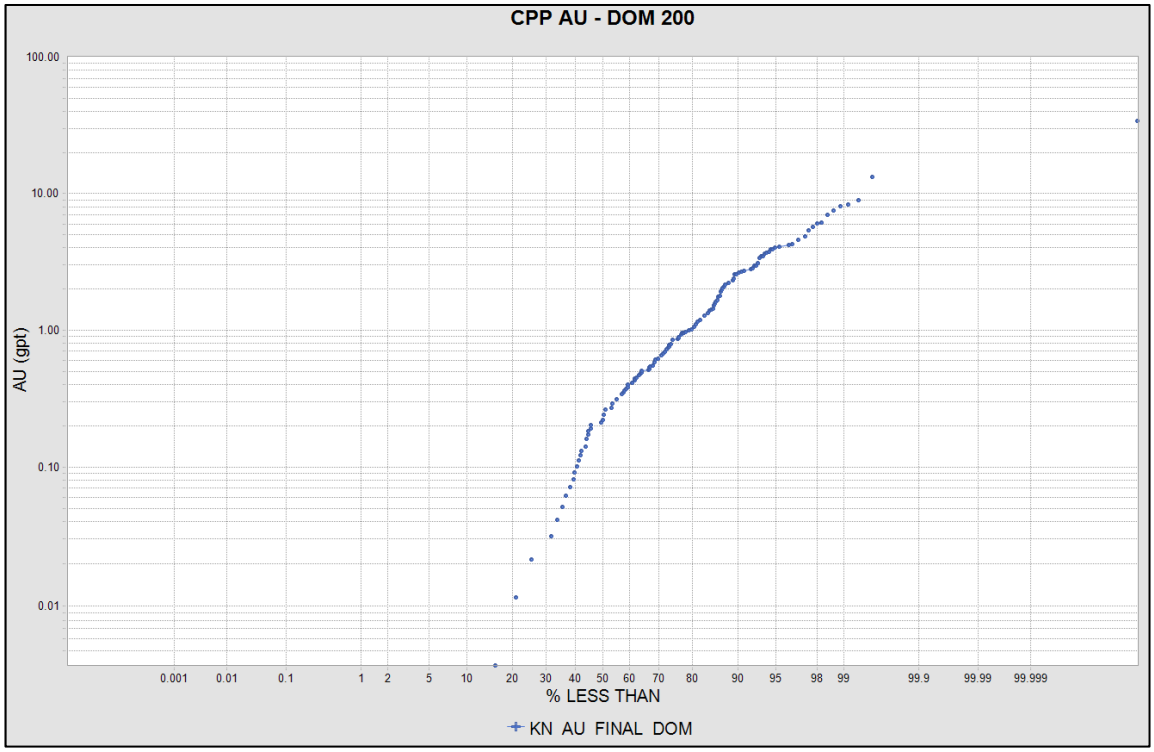
Source: MMTS, 2020

Figure 14-3: CPP – All Domains – Kierens-Norlartic (source: MMTS, 2020)



Source: MMTS, 2020

Figure 14-4: CPP – All Domains – Kierens-Norlartic (source: MMTS, 2020)

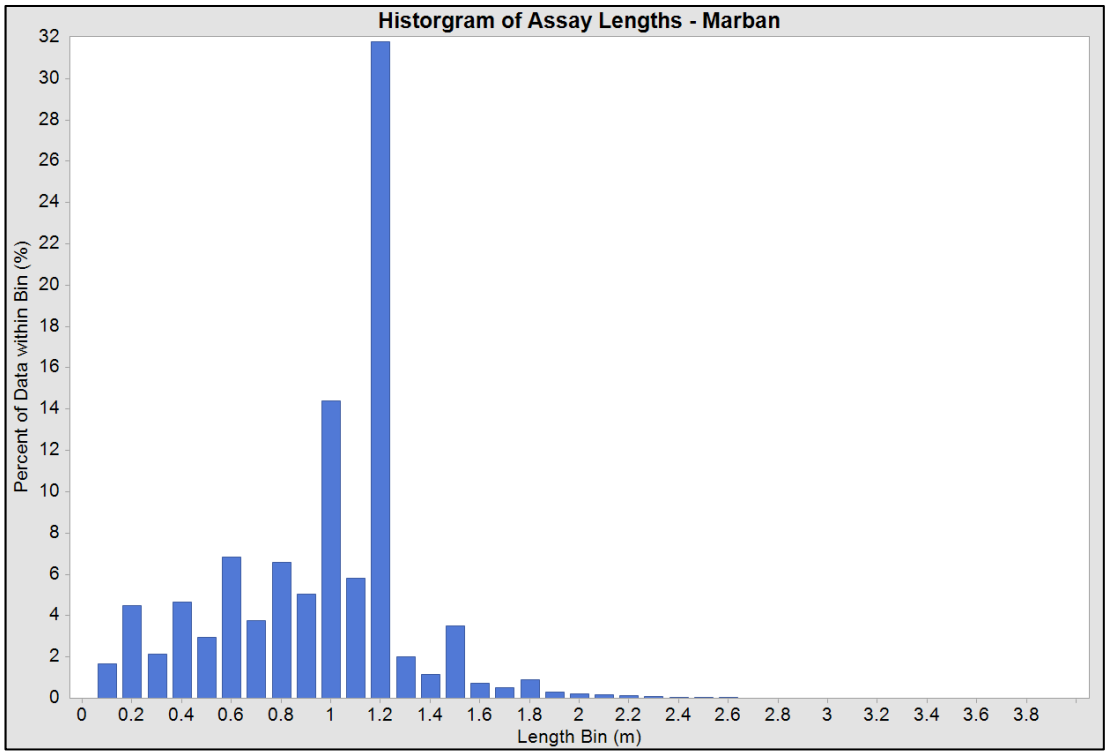


Source: MMTS, 2020

14.3.1 Compositing

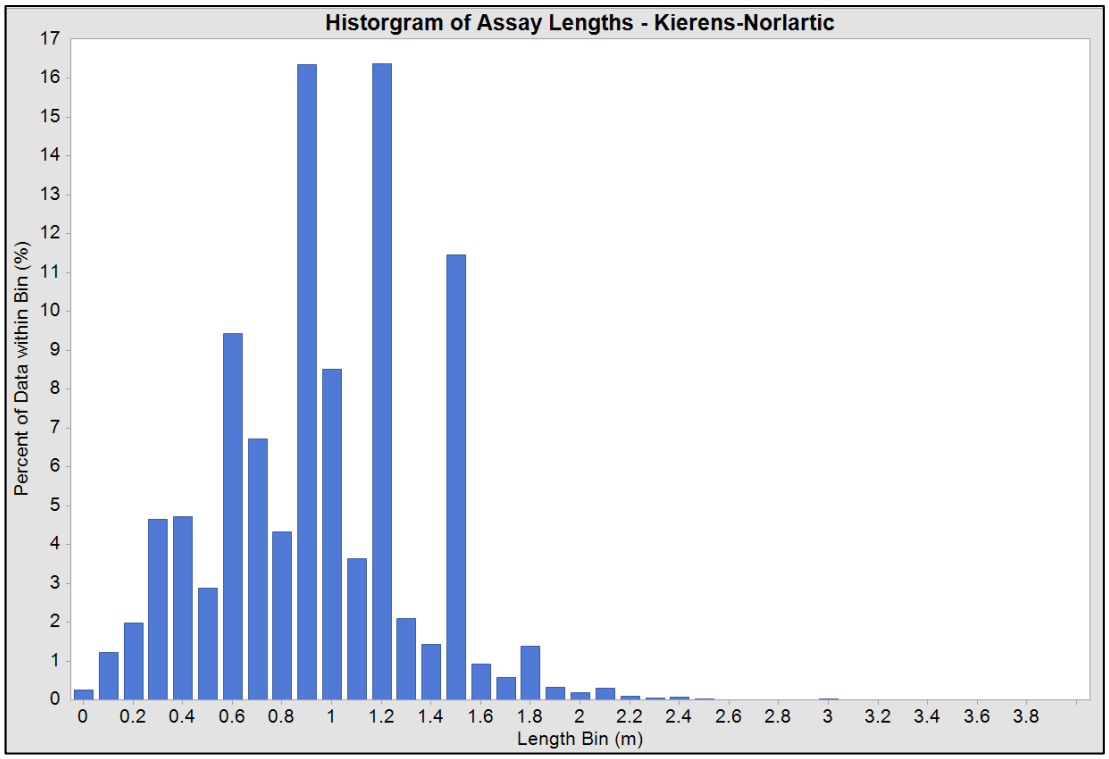
Compositing has been done for both deposit areas using a 1.5 m composite length to standardize the sample lengths used in interpolation. Based on the two histograms of assay length illustrated below, the assay length of The domain boundaries have been honoured during compositing, with any assay length less than 0.75 m added to the previous composite to avoid large numbers of small “remnant” composites at the domain boundaries.

Figure 14-5: Histogram of Assay Lengths – Marban (source: MMTS, 2020)



Source: MMTS, 2020

Figure 14-6: Histogram of Assay Lengths – Kierens-Norlartic (source: MMTS, 2020)



Source: MMTS, 2020

Assay statistics for each deposit area for the capped Au grades are summarized and compared to the capped Au composite statistics in Tables 14-7 and 14-8, illustrating that the weighted mean grades for each domain are essentially the same, and that the Coefficient of Variation (C.V.) is reasonable for linear interpolation methods for the deposits as a whole, and for the vast majority of domains. For the few domains where the C.V. is above about 3, sharing of composite within a drillhole is used to further reduce the variation in the final model.

Table 14-7: Assay and Composite Statistics of Capped Au – Marban

Dom	Assays				Composites				Difference (%)	
	Num Samples	Max Au g/t	Wtd Mean Au g/t	Wtd C.V.	Num Samples	Max Au g/t	Wtd Mean Au g/t	Wtd C.V.	Wtd Mean Au	Wtd C.V.
4	1,356	37.10	0.24	6.44	1,354	25.67	0.24	5.17	-0.7%	-24.6%
5	770	100.00	1.06	4.16	770	53.38	1.06	3.06	0.0%	-36%
6	311	33.40	0.81	3.16	293	20.55	0.81	2.45	0.0%	-29%
7	1,902	100.00	0.70	4.19	1,891	66.85	0.68	3.42	-2.4%	-22%
10	763	19.60	0.59	2.22	711	15.73	0.59	1.88	0.0%	-18%
15	3,661	85.53	0.64	3.15	3,631	62.80	0.64	2.50	-0.3%	-26%
16	742	100.00	0.33	9.38	742	33.51	0.33	6.30	-0.8%	-49%
20	3,563	70.55	0.68	3.34	3,393	43.70	0.68	2.73	0.5%	-22%
30	802	26.90	0.72	2.33	785	26.90	0.72	2.02	0.0%	-15%
40	958	20.28	0.48	2.80	957	19.82	0.48	2.21	-0.2%	-27%
45	1,598	66.00	0.93	3.39	1,572	43.78	0.93	2.67	-0.1%	-27%
50	655	36.01	1.05	2.47	655	21.29	1.05	1.91	-0.1%	-30%
60	317	83.70	0.37	9.45	317	39.07	0.37	6.26	0.0%	-51%
70	837	81.10	1.03	3.26	829	43.27	1.03	2.54	-0.1%	-28%
80	61	9.72	0.92	2.15	61	7.21	0.90	1.70	-1.8%	-27%
85	472	29.43	0.73	2.67	465	18.34	0.73	2.18	0.0%	-22%
90	675	28.98	0.70	2.38	647	18.73	0.70	1.97	0.0%	-21%
100	964	100.00	0.85	4.49	950	68.70	0.85	3.86	-0.1%	-16%
110	372	100.00	0.86	5.64	372	50.93	0.86	4.20	-0.3%	-34%
111	631	51.40	0.27	8.13	624	29.76	0.26	6.01	-1.3%	-35%
115	155	3.07	0.20	2.04	155	2.81	0.20	1.82	-0.2%	-12%
117	1,191	100.00	0.31	8.92	1,187	80.22	0.30	7.98	-1.1%	-12%
118	326	2.56	0.08	2.94	308	1.44	0.08	2.20	-0.1%	-34%
120	948	53.50	0.45	4.28	926	36.72	0.45	3.60	-0.1%	-19%
125	643	100.00	0.79	5.01	605	40.60	0.79	3.49	-0.2%	-44%
130	539	4.19	0.11	3.05	519	2.00	0.11	2.31	0.0%	-32%
135	138	3.70	0.15	2.47	131	3.35	0.15	2.25	0.0%	-10%
All	**5,350**	**100.00**	**0.63**	**4.10**	**5,350**	**80.22**	**0.63**	**3.24**	**-0.4%**	**-26%**

Table 14-8: Assay and Composite Statistics of Capped Au – Kierens-Norlartic

Domain	Assays				Composites				Difference (%)	
	Num Samples	Max Au g/t	Wtd Mean Au g/t	Wtd C.V.	Num Samples	Max Au g/t	Wtd Mean Au g/t	Wtd C.V.	Wtd Mean Au	Wtd C.V.
100	10,502	83.60	0.91	2.35	6,344	48.30	0.91	1.80	0.4%	-30.5%
101	8,348	100.00	1.02	3.06	4,382	60.43	1.04	2.17	1.7%	-40.8%
102	66	3.92	0.46	1.55	46	3.29	0.47	1.26	0.4%	-23.4%
103	653	26.40	1.07	2.03	408	14.75	1.07	1.57	0.0%	-29.5%
104	88	34.79	0.95	3.06	43	11.06	0.95	2.06	0.0%	-48.4%
105	162	100.00	1.08	5.01	73	20.65	1.08	2.31	0.0%	-116.8%
106	77	4.09	0.54	1.61	36	3.56	0.54	1.45	0.0%	-11.0%
200	470	20.00	0.77	2.18	321	14.81	0.77	1.81	0.0%	-20.1%
201	950	79.41	1.04	3.97	612	39.24	1.04	3.05	-0.1%	-30.1%
202	101	16.46	0.71	2.37	66	11.40	0.71	2.09	0.0%	-13.3%
203	55	2.11	0.46	1.02	39	1.60	0.46	0.84	0.0%	-21.3%
312	104	18.31	0.84	2.63	64	13.18	0.84	2.13	0.0%	-23.3%
315	224	13.15	0.46	2.35	145	7.43	0.46	1.80	0.0%	-30.3%
316	1,191	40.70	1.08	2.23	711	16.05	1.08	1.54	0.0%	-45.2%
322	145	57.09	1.07	4.00	94	17.12	1.07	2.65	0.0%	-50.7%
323	363	80.00	1.44	5.17	181	40.51	1.37	3.10	-5.1%	-67.0%
All	**23,499**	**100.00**	**0.96**	**2.8911**	**13,565**	**60.43**	**0.97**	**2.08**	**0.7%**	**-39.2%**

14.4 Density Assignment

Model blocks have been assigned a density based on the lithology as the average sg for the lithologies by block as summarized in Table 14-9. The SG of the overburden is given a value of 2.2.

Table 14-9: Summary of SG by Area and Lithology

Marban		Kierens-Norlartic	
Lithology	SG	Lithology	SG
Felsics	2.71	Diorite	2.74
Basalts	2.81	Granodiorite	2.69
Ultramafic	2.85	Granodiorite-Komatiite	2.67
		Basalt	2.81
		Komatiite	2.72
		Komatiite-Diorite	2.77

14.5 Historic Underground

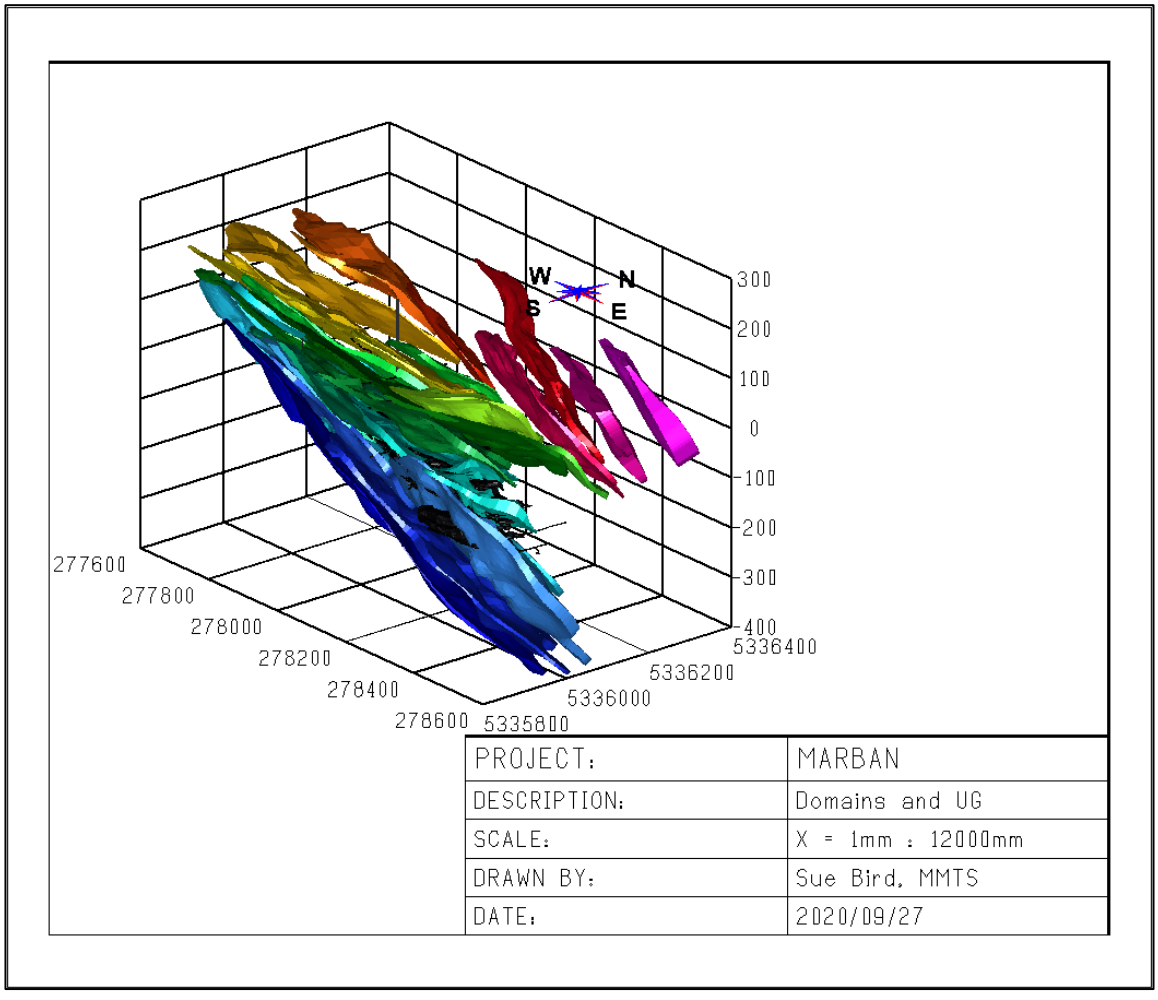
Historic Underground shapes were provided by the client as three-dimensional solids. The voids for Marban and Kierens-Norlartic are illustrated in the following two figures, also showing the interpolations wireframes (domains) for reference. Model coding includes the block percent within

the voids, which has been removed from the resource. A reconciliation of the shapes provided has been done to ensure that all historic underground is accounted for. The underground solid volume has been compared to the historic production with immaterial volume differences for all areas. In the Kierens pit area, an amount of 33ktonnes at a grade of 6.3g/t has been removed from the resource estimate to match the production records, as summarized in Table 14-10.

Table 14-10: Summary of Ounces Removed from Kierens to match Historic Production

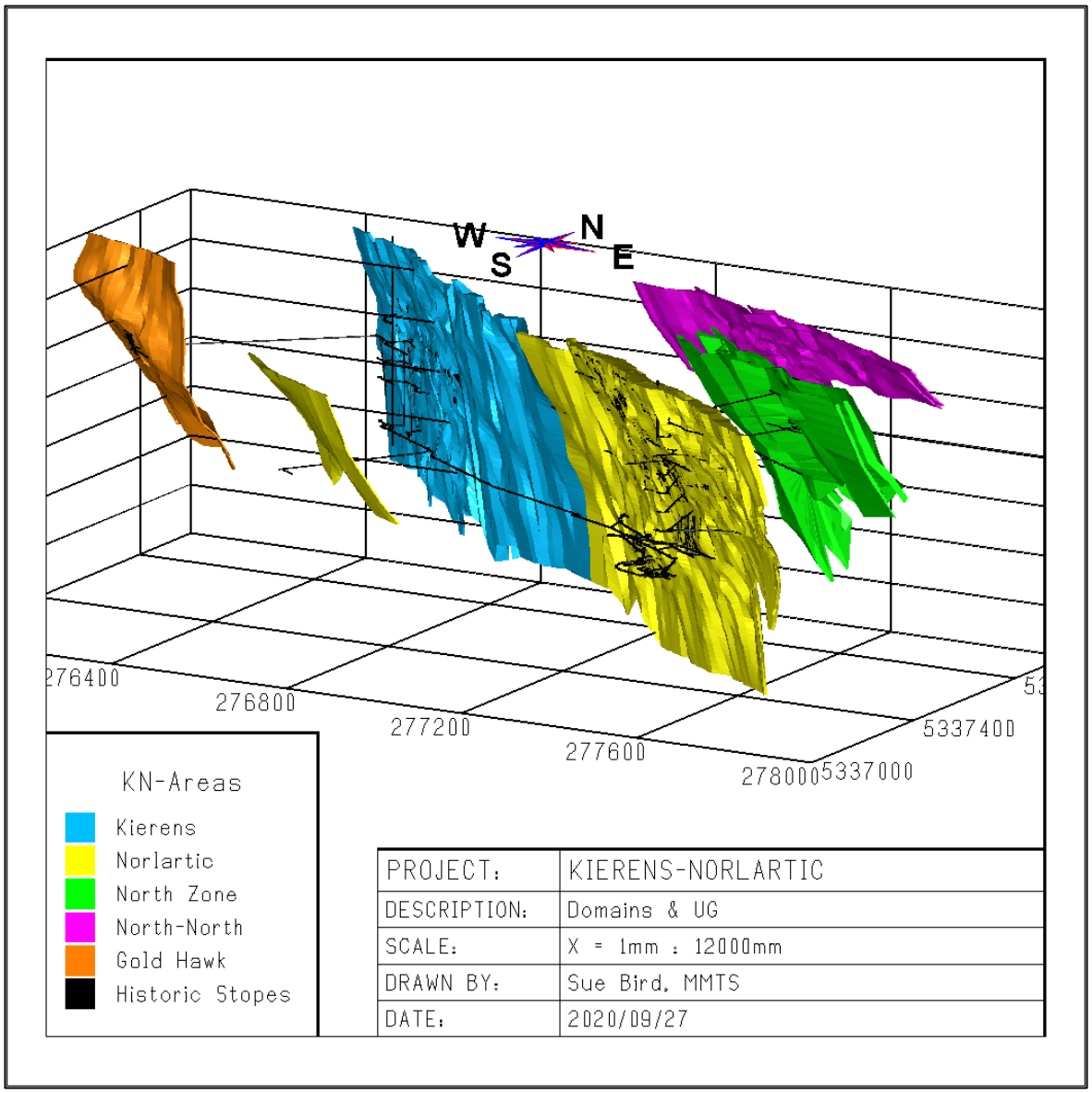
Unaccounted Au Ounces for Historic UG in Kierens			
Source	**tonnage (KT)**	**grade**	**ounces**
Production Record	251	6.3	52,000
Solid Provided	218	na	na
Difference	**33**	**6.3**	**6,684**

Figure 14-7: 3D View Looking Northwest of the Mineralized Domains and Historic Underground (black) – Marban



Source: MMTS, 2020

Figure 14-8: 3D View Looking Northwest of the Mineralized Domains and Historic Underground (black) – Kierens-Norlartic

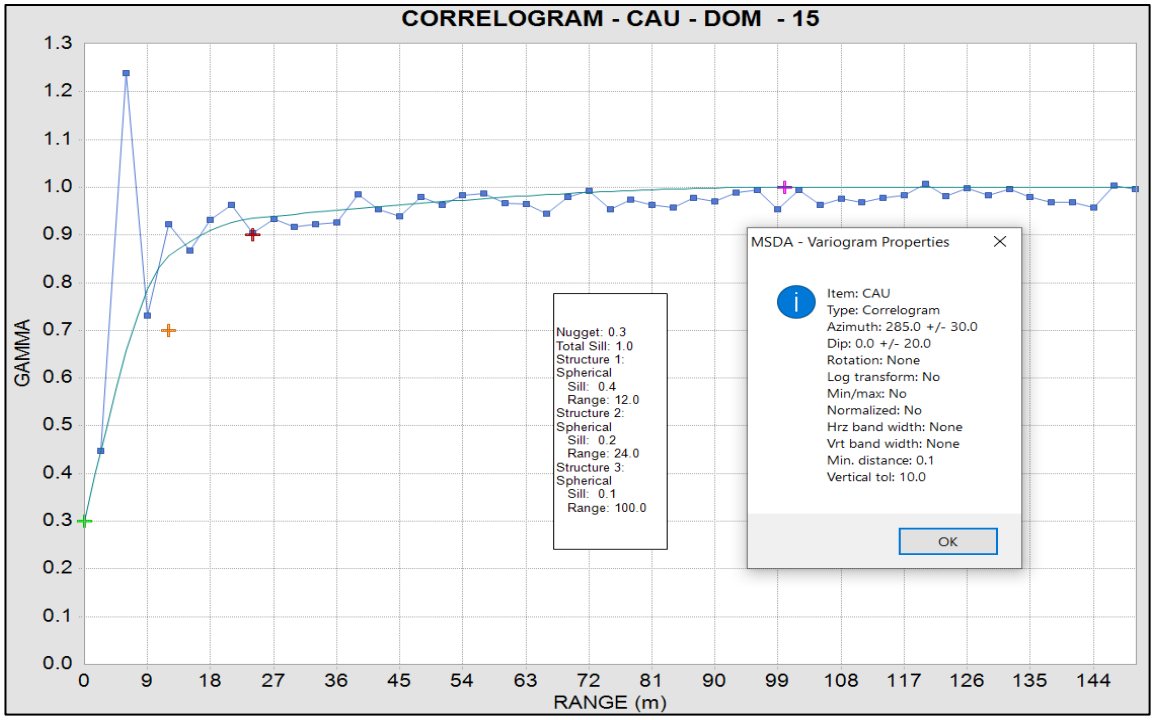


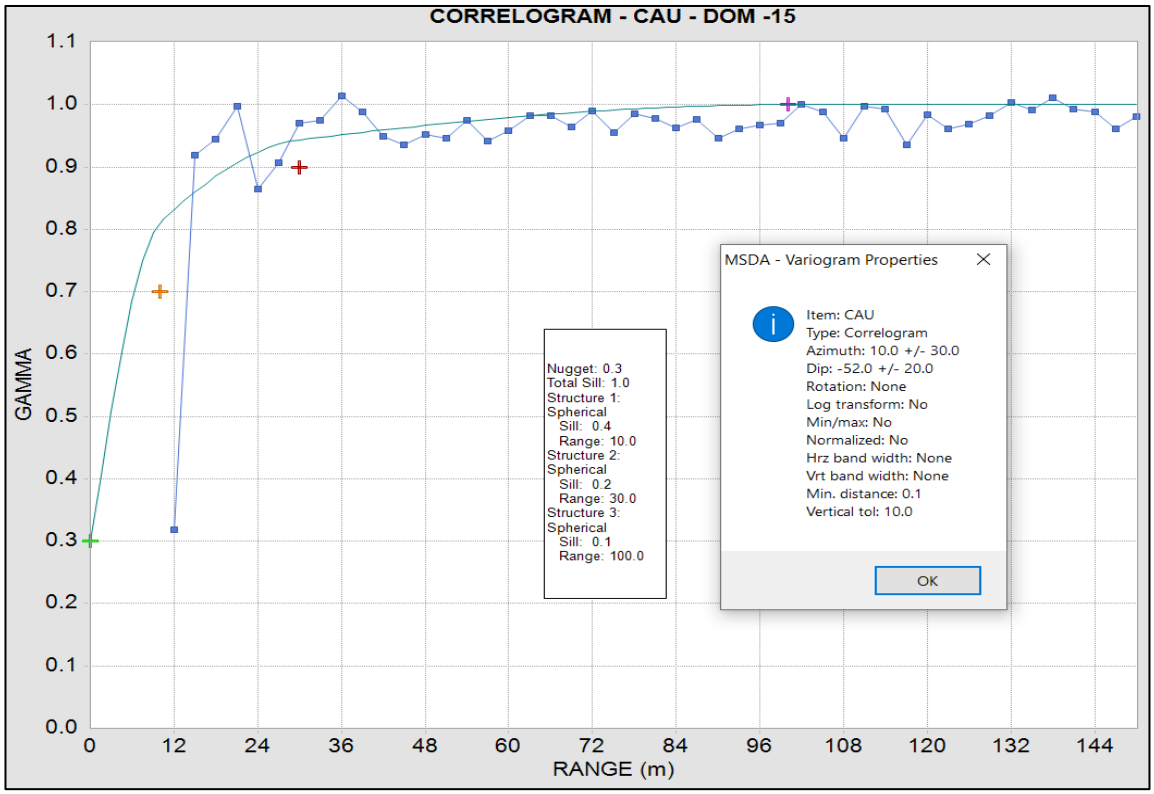
Source: MMTS, 2020

14.6 Block Model Interpolations

Interpolations have been done based on variography completed where possible. The interpolations have been done by Ordinary Kriging (OK) in four passes.

Figure 14-9: Marban Variography – Major and Minor Axes





Source: MMTS, 2020

Table 14-11: Variogram Summary – Domain 15

Rotation (GSLIB-MS)		Axis	Nugget	Sill1	Sill2	Sill3	Range 1 (m)	Range 2 (m)	Range 3 (m)
ROT	295	Major	0.3	0.4	0.2	0.1	3.7	7.3	30.5
DIPN	0	Minor					3.0	9.1	30.5
DIPE	-45	Vert					1.5	3.0	4.6

14.6.1 Search Parameters

The search parameters are based on the variography with the maximum search distances equal to the maximum variogram range for the domains which were analyzed. Anisotropic search distances are used, with the rotation parameters of the search ellipses varied by domain to align with the general structure and mineralization of each domain. The tables below summarize the ellipse's rotation and search parameters respectively. Composite selection shown in Table 14-13 is defined to ensure at least 2 drillholes and no more than 5 drillholes are used for the first 3 passes and that at least two "split quadrants" are used for the first 3 passes. Figure 14-10 and 14-11 illustrate the domains and search ellipses for each domain (for Pass=3) for Marban and Kierens-Norlartic respectively.

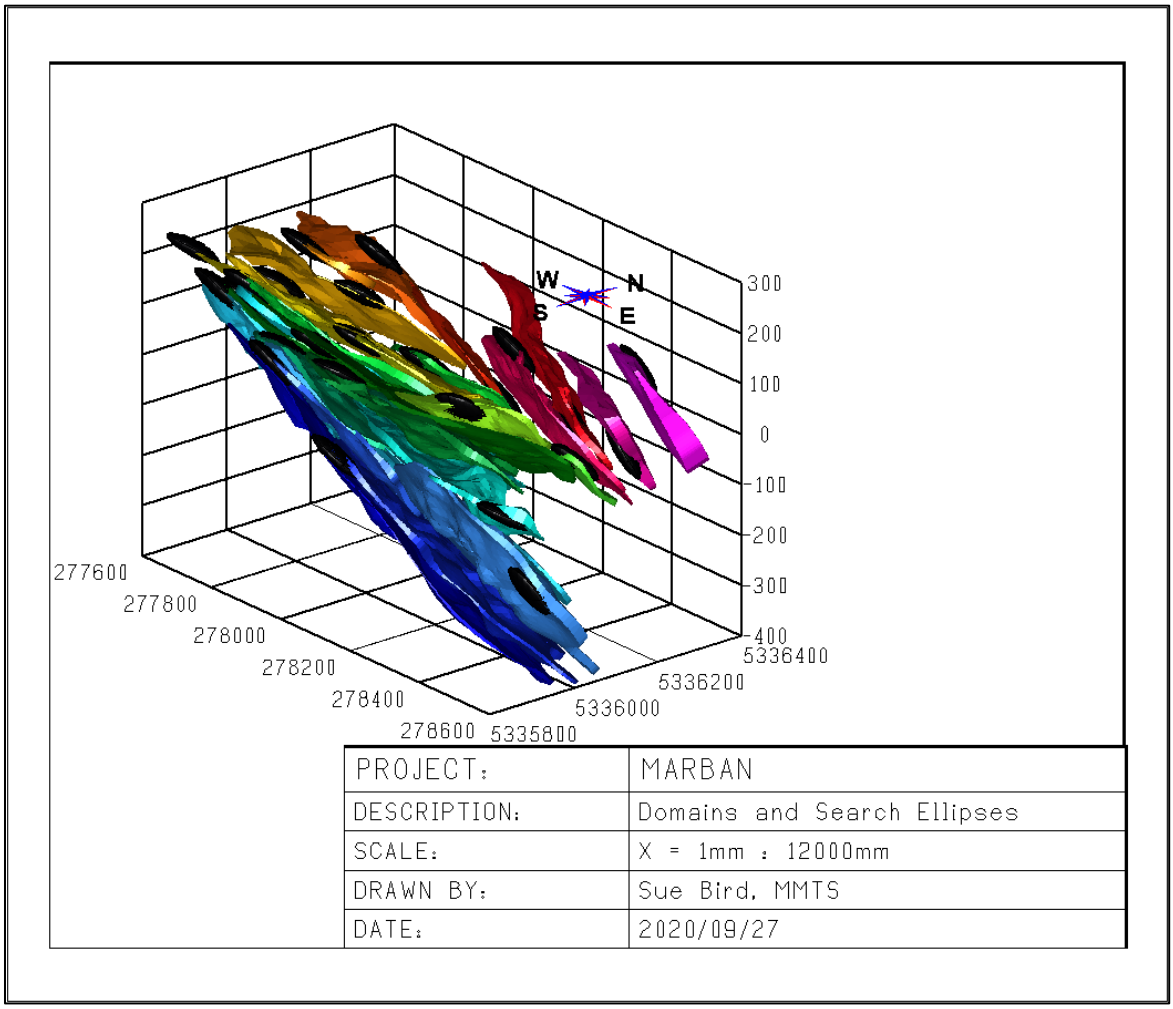
Table 14-12: Summary of Ellipse Rotations for Search Parameters

Marban Domains								Kierens-Norlartic Domains			
Domain	Rot-Y	Rot-X	Rot-Z	Domain	Rot-Y	Rot-X	Rot-Z	Domain	Rot-Y	Rot-X	Rot-Z
4	295	0	-52	90	278	0	-38	100	303	0	-70
5	295	0	-45	100	278	0	-35	101	300	0	-78
6	285	0	-53	110	278	0	-35	102	304	0	-77
7	280	0	-56	111	265	0	-37	103	317	0	-82
10	290	0	-50	115	282	0	-40	104	301	0	-85
15	285	0	-50	117	280	0	-46	105	295	0	-82
16	273	0	-43	118	291	0	-55	106	303	0	-82
20	275	0	-32	120	298	0	-60	200	303	0	-60
30	285	0	-35	125	286	0	-52	201	303	0	-60
40	270	0	-33	130	298	0	-61	202	305	0	-60
45	270	0	-38	135	292	0	-58	203	305	0	-60
50	280	0	-45					312	313	0	-55
60	272	0	-36					315	297	0	-50
70	272	0	-43					316	291	0	-55
80	272	0	-45					322	105	0	-85
85	247	0	-43					323	306	0	-57

Table 14-13: Summary of Interpolation Parameters

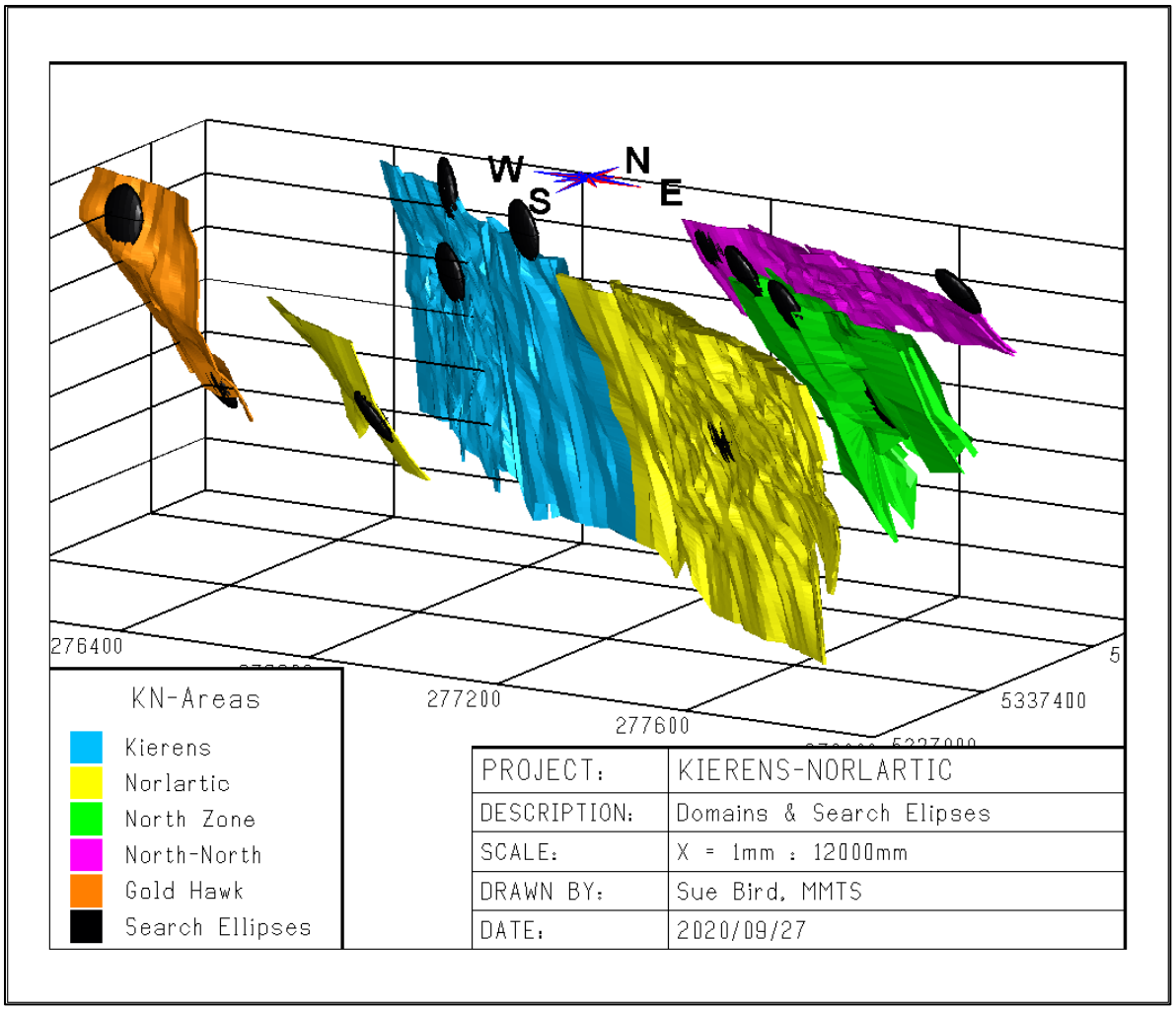
Parameter	Interpolation Pass			
	1	2	3	4
Dist-Y	15	30	48	100
Dist-X	15	35	60	100
Dist-Z	5	10	15	15
Min Comps	3	3	3	1
Max Comps	9	9	9	4
Max/DH	2	2	2	2
Max/Quad	2	2	2	na

Figure 14-10: 3D View Looking North of the Mineralized Domains and Search ellipses – Marban



Source: MMTS, 2020

Figure 14-11: 3D View Looking North of the Mineralized Domains and Search ellipses – Kierens-Norlartic



Source: MMTS, 2020

14.7 Block Model Validation

A nearest neighbour model (NN model) has been created in each deposit area in order to compare the OK modelled grades for Au with the de-clustered composite grades. The NN model has been created using composites of 3m intervals, which is approximately the minimum mining width. For the NN models, the uncapped values are used in the comparison.

14.7.1 Global Bias Check

A comparison of global mean values (at zero cut-off) with the de-clustered composite data is provided in Tables Table 14-14 and Table 14-15 for Marban and Kierens-Norlartic respectively. The table illustrate no global bias as the modelled grades are only 0.4% lower for Marban and 0.3% lower for Kierens-Norlartic compared to the de-clustered composites. Domains with higher differences in grade are generally domains in which either the grade is low, or the C.V. of the NN model is high. In these cases, the OK modelled grades are conservative compared to the NN model.

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

Domain	NN Model - Au				Modelled OK - Au				Difference (%)	
	Num Samples	Max Au g/t	Wtd Mean Au g/t	Wtd C.V.	Num Samples	Max Au g/t	Wtd Mean Au g/t	Wtd C.V.	Wtd Mean Au	Wtd C.V.
4	62,446	25.67	0.25	62,446	12.68	0.24	2.83	-5.9%	4	62,446
5	34,968	104.84	1.31	34,968	27.76	1.21	1.79	-8.4%	5	34,968
6	7,220	20.55	0.91	7,220	9.04	0.84	1.25	-8.3%	6	7,220
7	51,847	106.27	0.59	51,847	28.28	0.56	1.72	-5.0%	7	51,847
10	12,834	15.73	0.61	12,834	7.16	0.64	0.97	4.9%	10	12,834
15	69,459	62.80	0.68	69,459	21.66	0.67	1.12	-0.7%	15	69,459
16	11,023	104.55	0.59	11,023	22.12	0.34	3.68	-71.1%	16	11,023
20	70,758	43.70	0.68	70,758	21.26	0.69	1.57	2.4%	20	70,758
30	16,715	26.90	0.72	16,715	10.78	0.75	0.98	3.6%	30	16,715
40	18,927	19.82	0.49	18,927	8.46	0.50	1.09	1.0%	40	18,927
45	35,816	43.78	0.94	35,816	18.93	0.98	1.28	3.6%	45	35,816
50	10,051	21.29	1.11	10,051	9.65	1.12	0.91	1.0%	50	10,051
60	8,045	39.07	0.35	8,045	17.91	0.34	2.82	-4.0%	60	8,045
70	17,491	43.27	0.90	17,491	14.17	0.97	1.26	7.2%	70	17,491
80	2,216	7.21	0.89	2,216	3.76	0.71	0.92	-24.7%	80	2,216
85	7,222	18.34	0.74	7,222	10.45	0.70	1.17	-5.1%	85	7,222
90	13,095	18.73	0.72	13,095	10.68	0.74	1.17	2.3%	90	13,095
100	28,210	110.24	0.91	28,210	32.83	0.89	1.95	-2.3%	100	28,210
111	16,034	29.76	0.20	16,034	13.14	0.19	1.93	-1.3%	111	16,034
115	2,604	2.81	0.26	2,604	1.59	0.21	1.03	-22.9%	115	2,604
118	10,567	1.44	0.10	10,567	1.14	0.09	1.45	-6.6%	118	10,567
120	20,531	36.72	0.49	20,531	12.43	0.47	1.41	-3.4%	120	20,531
125	21,397	109.41	1.04	21,397	25.39	1.10	1.63	5.4%	125	21,397
130	14,997	2.00	0.12	14,997	1.41	0.11	1.35	-7.5%	130	14,997
135	17,013	3.35	0.07	17,013	1.58	0.07	2.06	2.8%	135	17,013
All	**581,486**	**110.24**	**0.62**	**609,764**	**32.83**	**0.61**	**1.77**	**-1.4%**	**All**	**581,486**

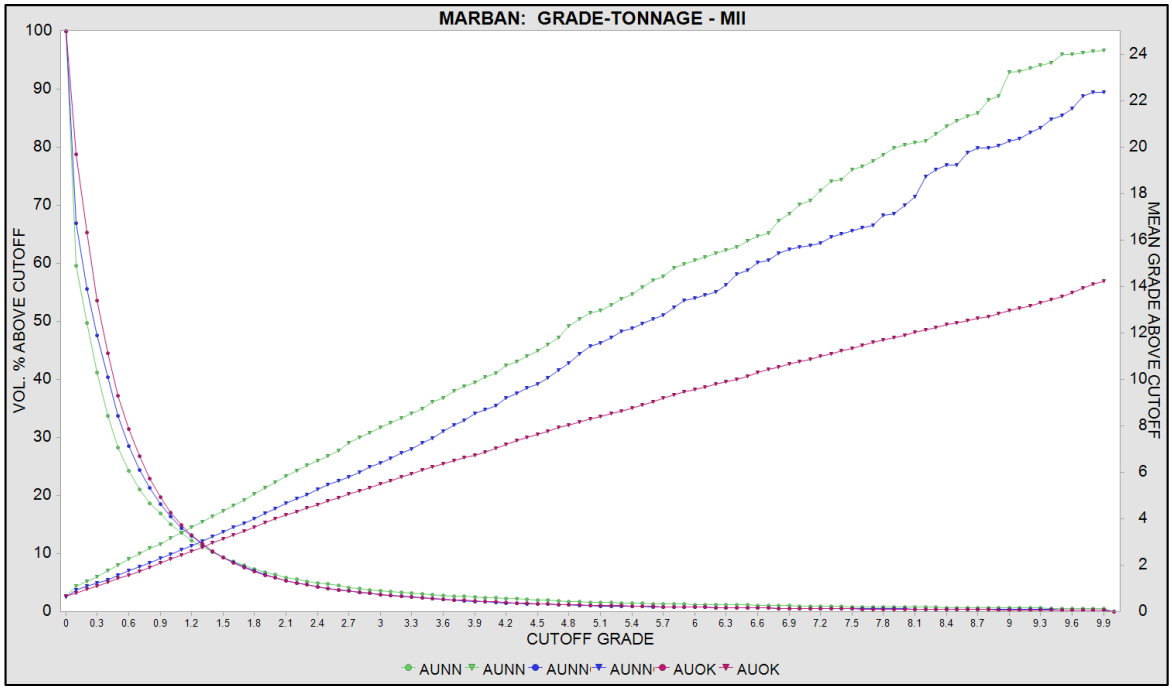
Table 14-15: Comparison of De-clustered Composites to Model Grades – Kierens-Norlartic

Domain	NN Model - Au				Modelled OK - Au				Difference (%)	
	Num Samples	Max Au g/t	Wtd Mean Au g/t	Wtd C.V.	Num Samples	Max Au g/t	Wtd Mean Au g/t	Wtd C.V.	Wtd Mean Au	Wtd C.V.
100	101,954	48.30	0.89	1.90	101,954	17.45	0.89	1.18	0.1%	-61.0%
101	62,581	60.43	0.86	1.90	62,581	27.80	0.88	1.00	2.2%	-89.1%
102	253	2.18	0.41	1.13	253	1.43	0.41	0.65	0.4%	-72.8%
103	2,231	14.75	1.04	1.66	2,231	7.19	1.04	0.74	0.2%	-124.8%
104	1,552	11.06	1.04	2.04	1,552	4.42	0.95	1.03	-10.4%	-98.4%
105	715	21.37	0.79	2.06	715	7.00	0.77	0.92	-2.8%	-123.9%
106	481	3.56	0.38	1.57	481	2.01	0.38	0.86	-0.5%	-82.6%
200	26,787	25.15	0.62	1.74	26,787	9.13	0.63	1.08	0.3%	-61.3%
201	26,613	39.24	0.81	2.52	26,613	7.08	0.78	0.98	-4.4%	-158.3%
202	3,852	11.40	0.58	2.58	3,852	2.63	0.50	0.86	-17.4%	-200.7%
203	1,882	1.60	0.32	0.87	1,882	1.05	0.37	0.73	14.5%	-19.6%
312	8,915	13.18	0.43	2.24	8,915	1.81	0.48	0.81	9.9%	-177.5%
315	6,950	7.43	0.36	1.70	6,950	2.38	0.38	0.84	5.3%	-103.3%
316	19,172	16.05	0.90	1.65	19,172	5.69	0.87	0.89	-3.0%	-85.6%
322	3,923	17.12	0.90	2.57	3,923	7.84	0.77	1.36	-16.4%	-88.6%
323	6,606	72.34	0.77	5.14	6,606	15.35	0.62	1.80	-24.3%	-185.8%
All	**274,467**	**72.34**	**0.82**	**2.0494**	**274,467**	**27.80**	**0.82**	**1.11**	**-0.3%**	**-83.9%**

14.7.2 Grade-Tonnage Curves

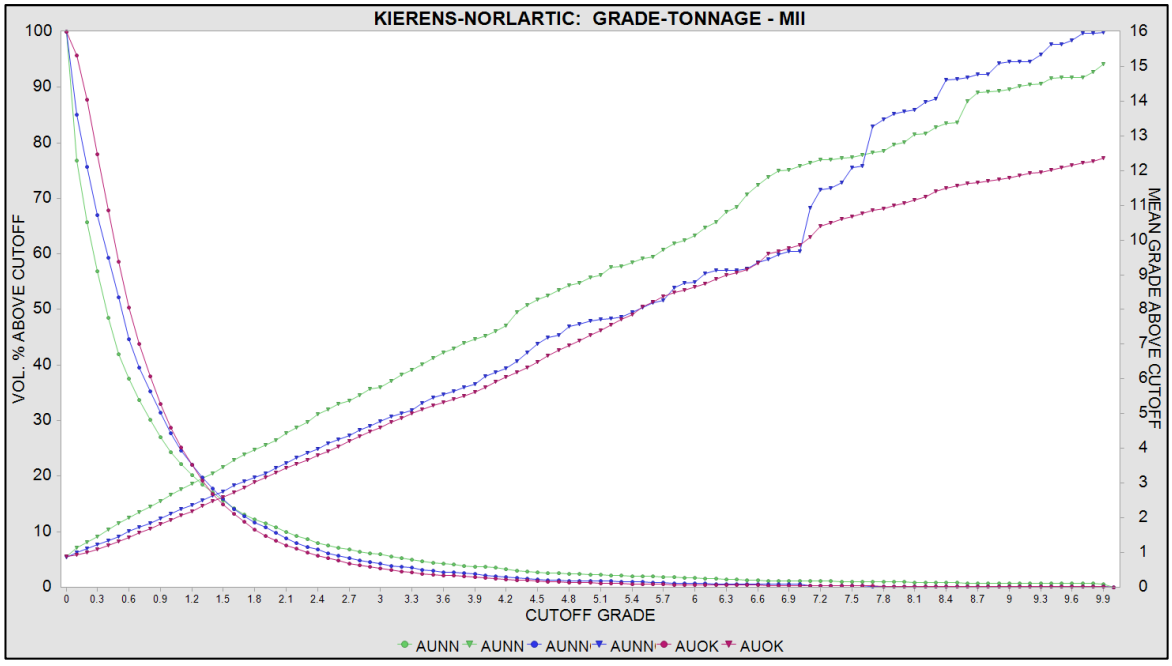
Grade-tonnage curves have been created to be sure that the amount of internal model "smoothing", or dilution of grades is appropriate. To account for a change of support from the composite size to the block size, a theoretical Indirect Lognormal correctio (ILC) calculation is performed based on the bock variance, the mean grade and C.V. of the NN model in each domain. A comparison of the tonnage grade curves for the OK modelled grades, the de-clustered composites (NN) and the ILC corrected grades (NNC) is provided Figure 14-12 and 14-13. For each deposit, the final modelled grades indicate slightly higher tonnages and lower grades than the NN and NNC, indicating that interpolations are not optimistic, but rather have some conservatism built in. Additional dilution has been added at the mine planning stage.

Figure 14-12: Grade-Tonnage Curves Comparison – Marban



Source: MMTS, 2020

Figure 14-13: Grade-Tonnage Curves Comparison – Kierens-Norlartic

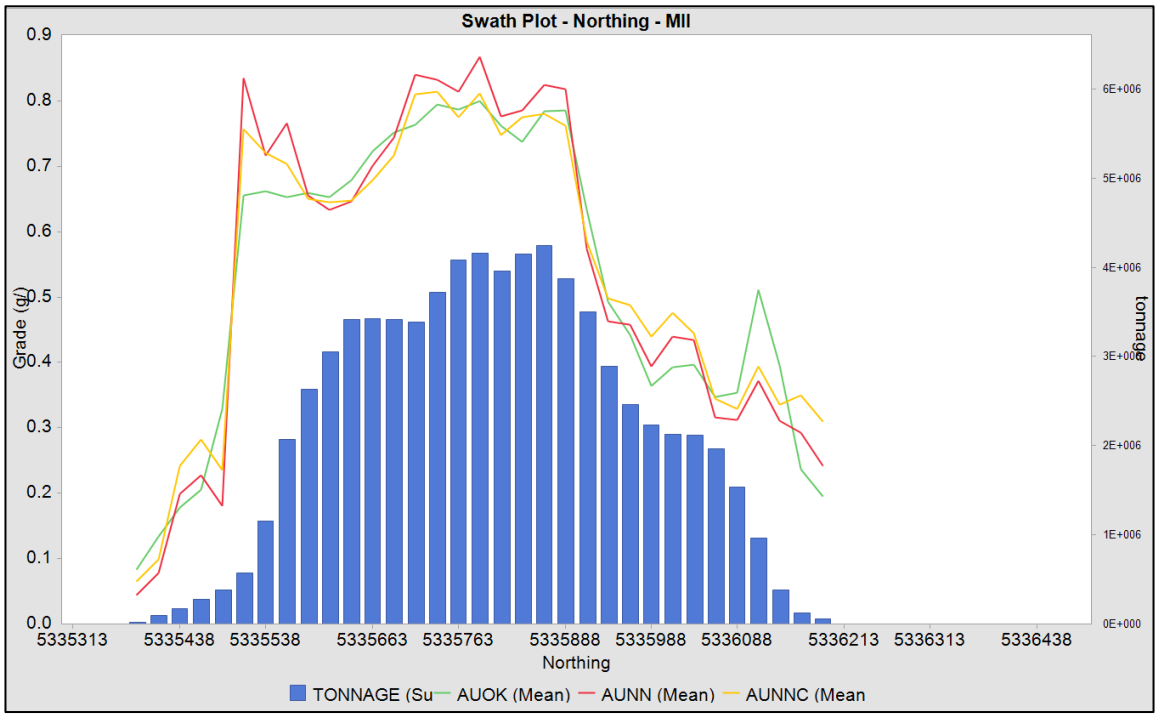


Source: MMTS, 2020

14.7.3 Swath Plots

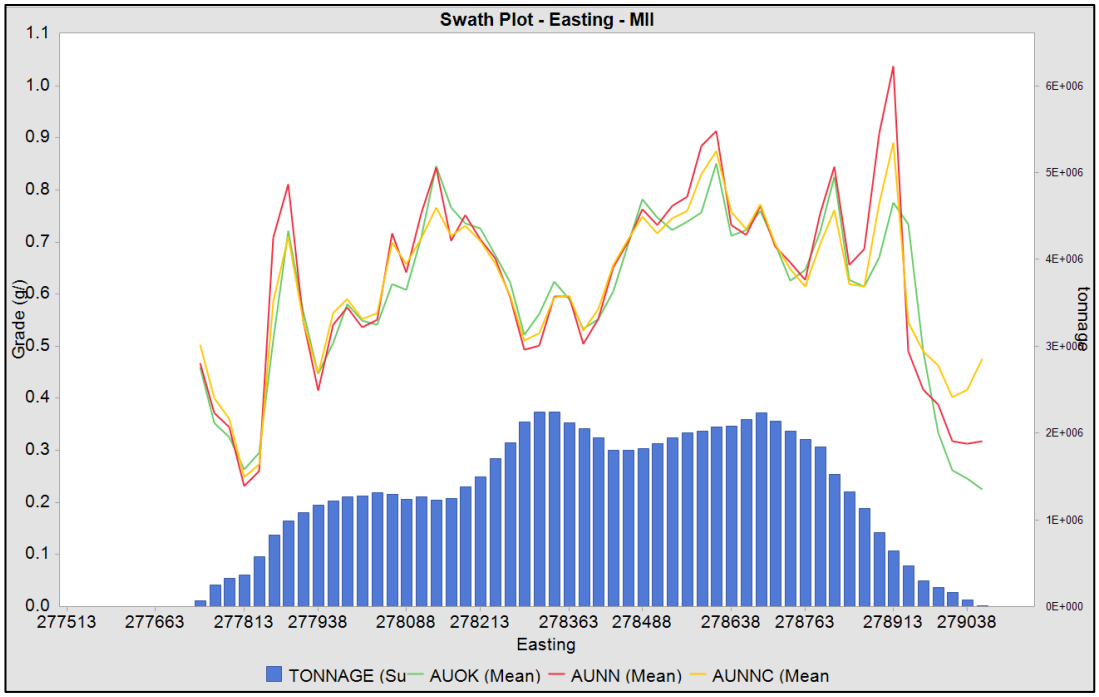
Swath plots have ben crated for each deposit area in the Northing, easting and elevation directions in order to ensure that there are no unexpected bias in the modelled grades compared to the data. It is evident from the following plots of each area and direction, that the modelled grades are generally less than the NN grades, particularly in the areas of the model with material tonnages (as shown in the blue bars with the scale for the tonnage on the right axis of each plot).

Figure 14-14: Swath Plot - Northing – Marban



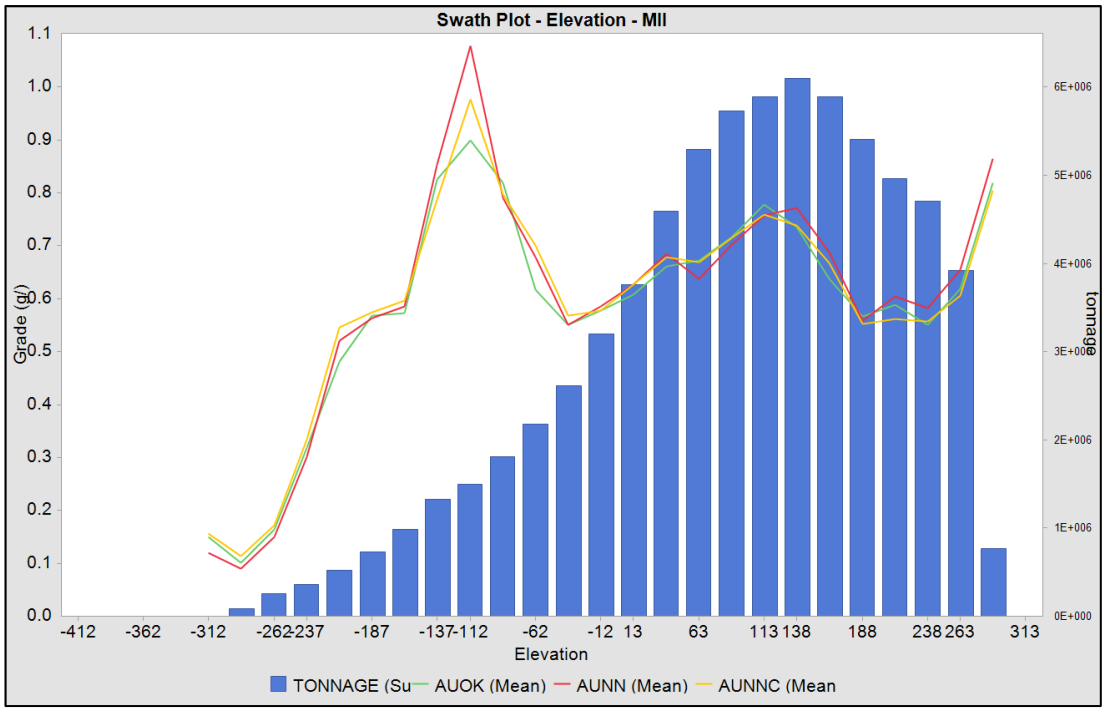
Source: MMTS, 2020

Figure 14-15: Swath Plot - Easting – Marban



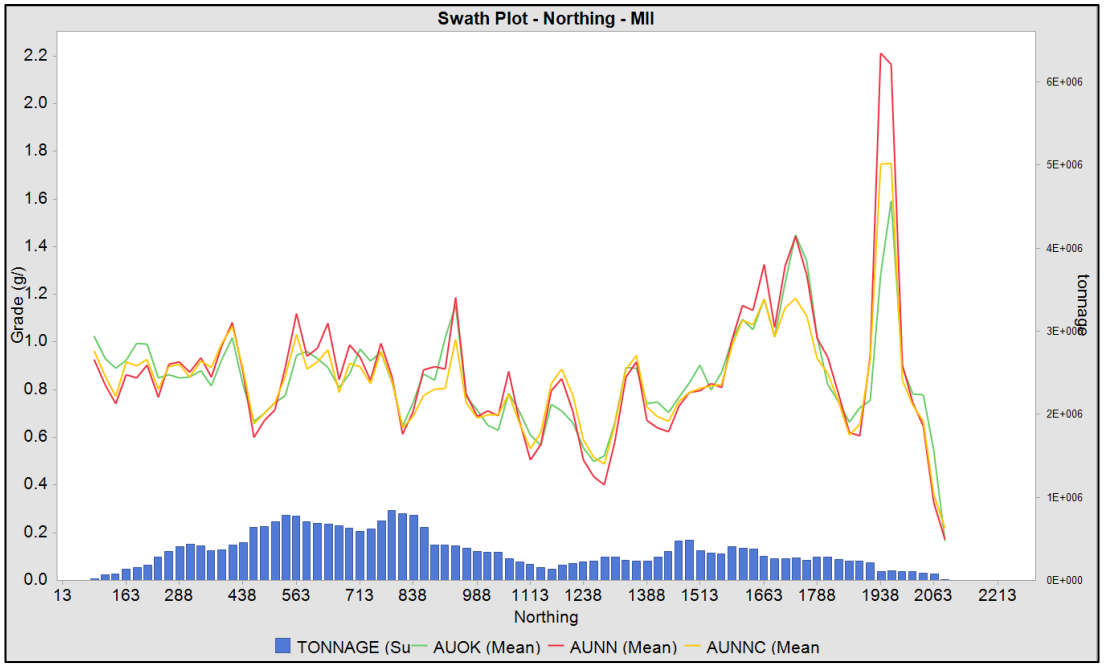
Source: MMTS, 2020

Figure 14-16: Swath Plot - Elevation – Marban



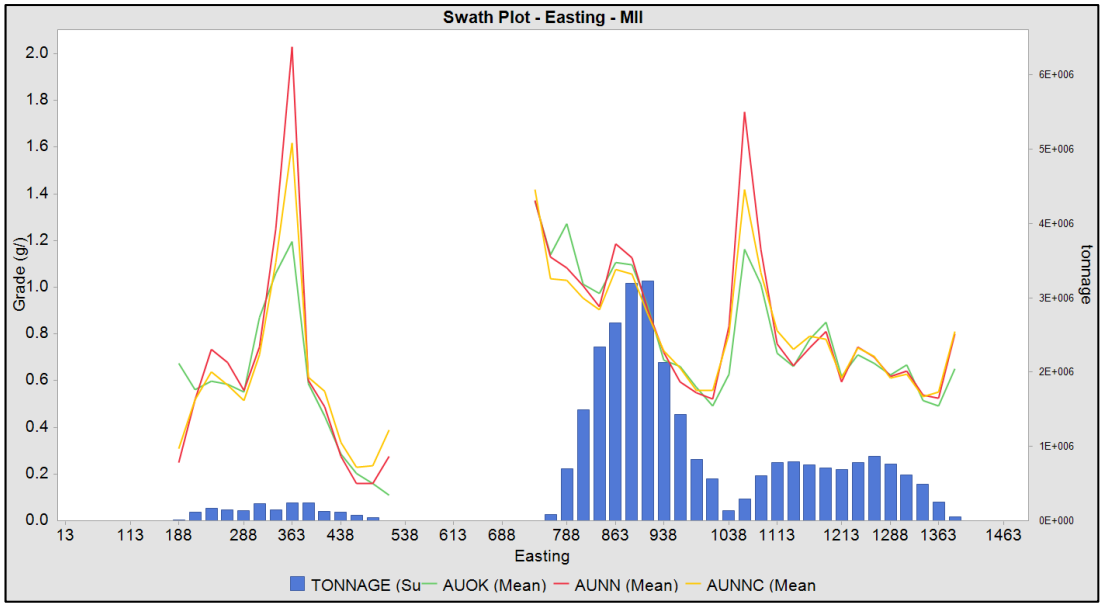
Source: MMTS, 2020

Figure 14-17: Swath Plot - Northing – Kierens-Norlartic



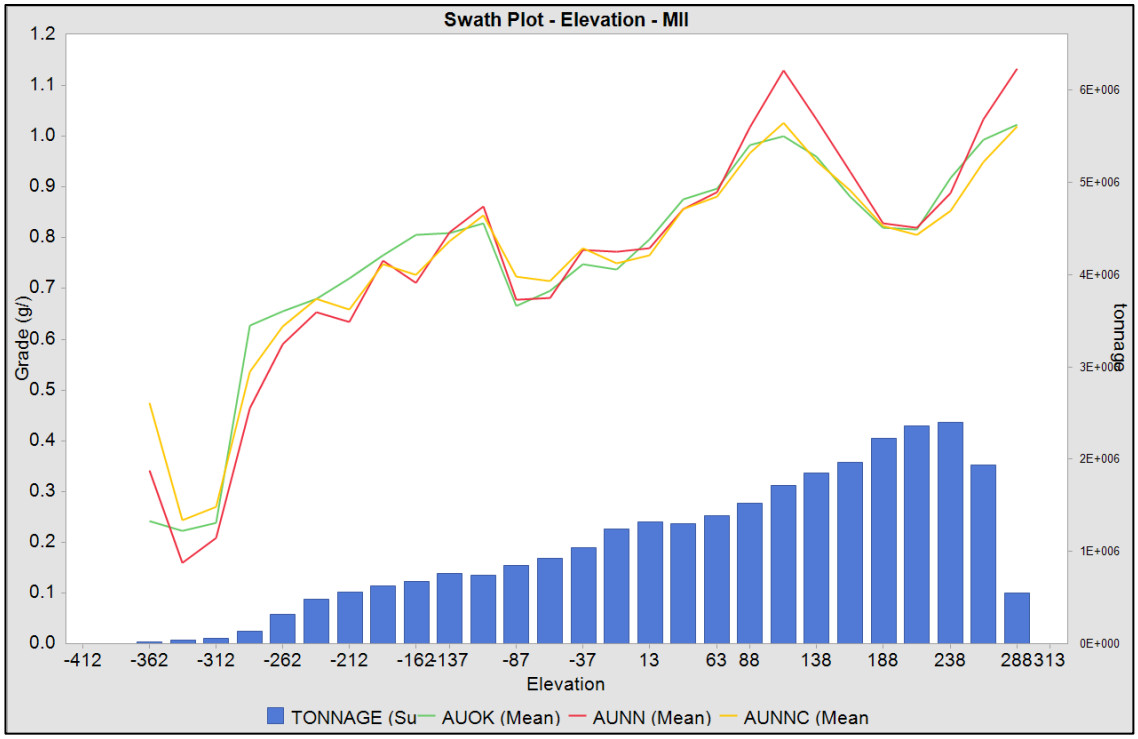
Source: MMTS, 2020

Figure 14-18: Swath Plot - Easting – Kierens-Norlartic



Source: MMTS, 2020

Figure 14-19: Swath Plot - Elevation – Kierens-Norlartic

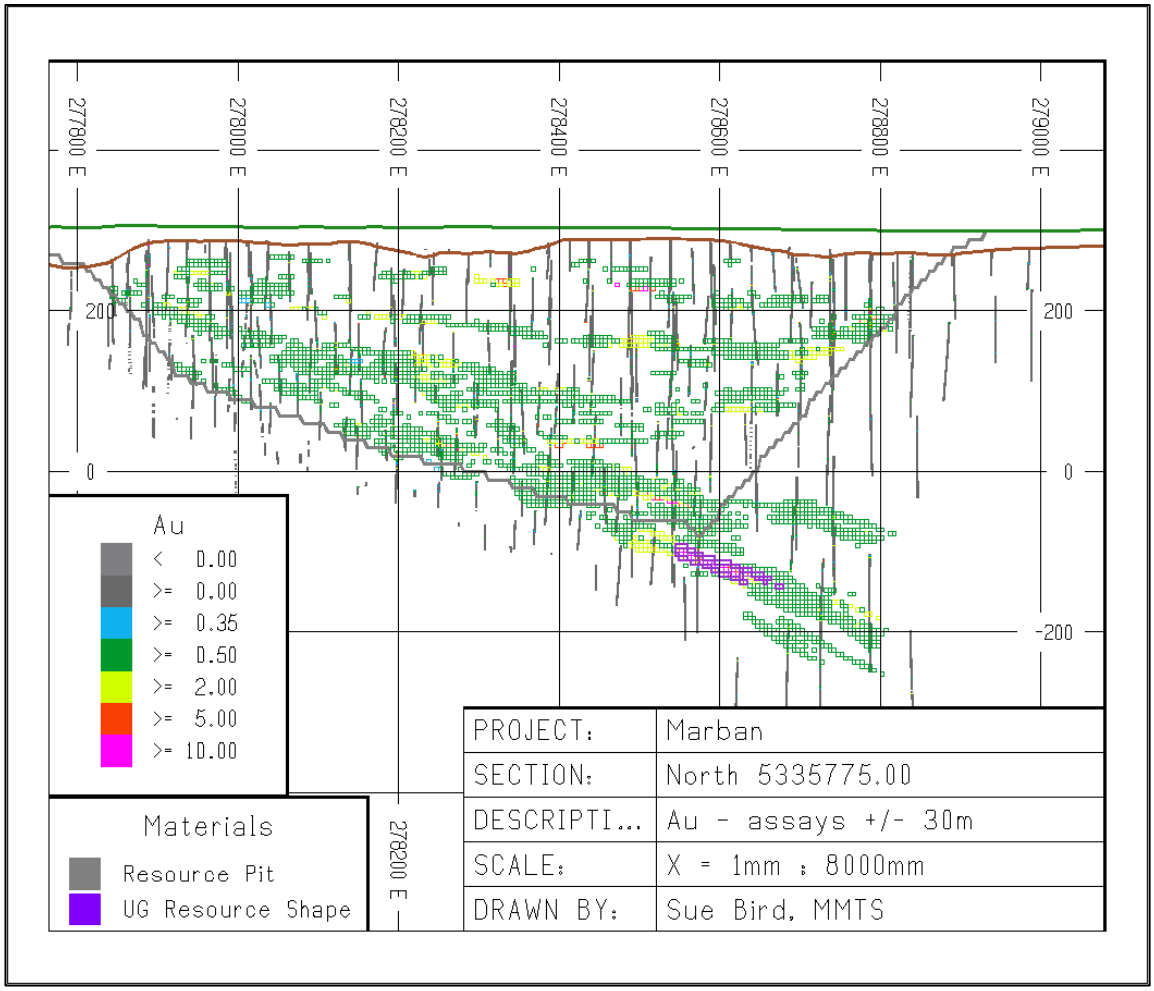


Source: MMTS, 2020

14.7.4 Visual Inspection

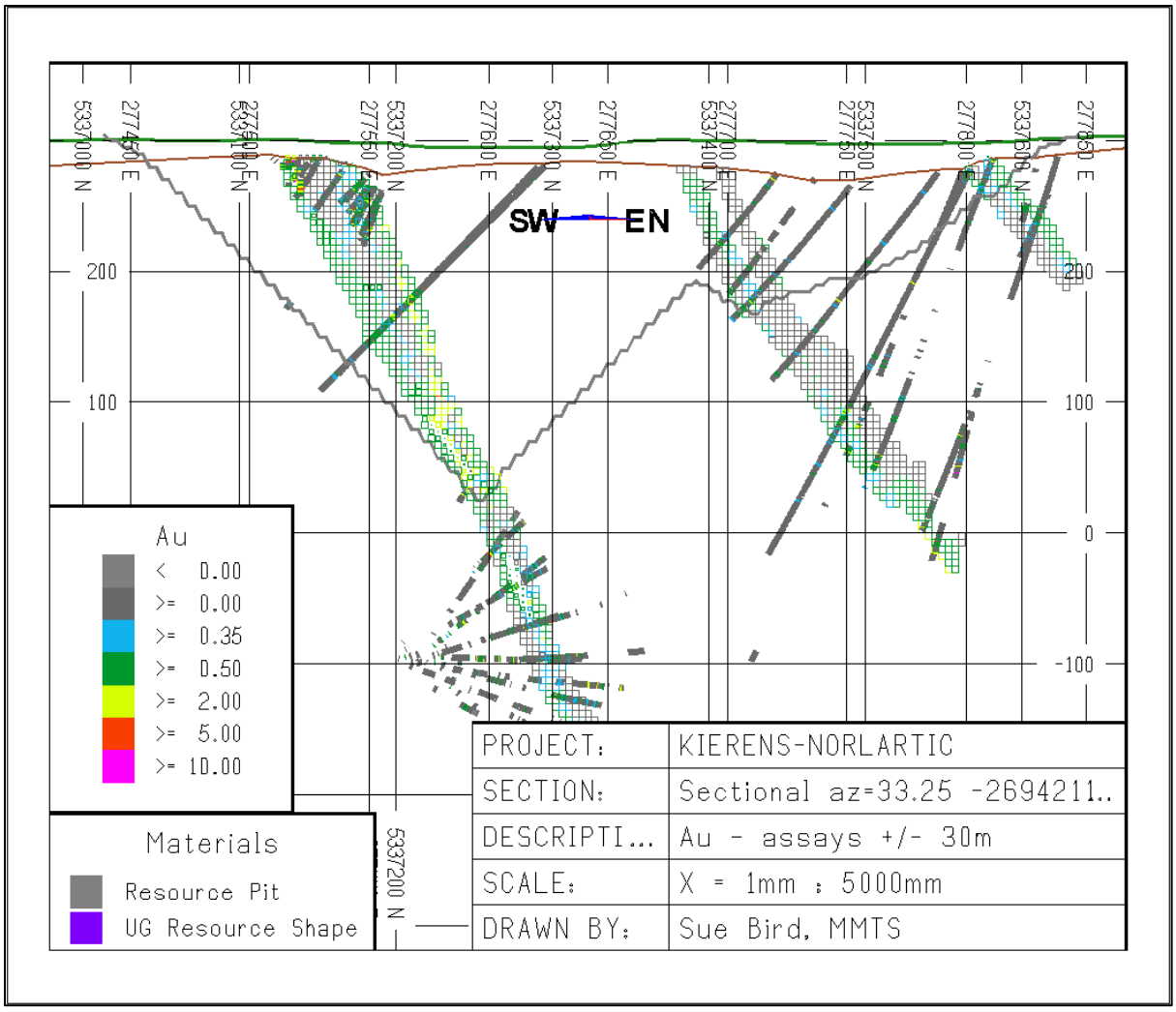
Visual inspection of the modelled grades to the assays has been done for both model areas. Illustrated below are example sections for each area comparing the modelled Au grade to the assays. Only the drillholes and assays used are shown, including re-assays only for years prior to 1984, not including assay within previous underground. The model blocks are scaled based on the percent of the block within the wireframe used for interpolation with the previous underground mining removed.

Figure 14-20: Visual Comparison of Au Grade – Marban Section 5335775N



Source: MMTS, 2020

Figure 14-21: Visual Comparison of Au Grade – Kierens-Norlartic – Model EW Section



Source: MMTS, 2020

14.8 Classification of Mineral Resources

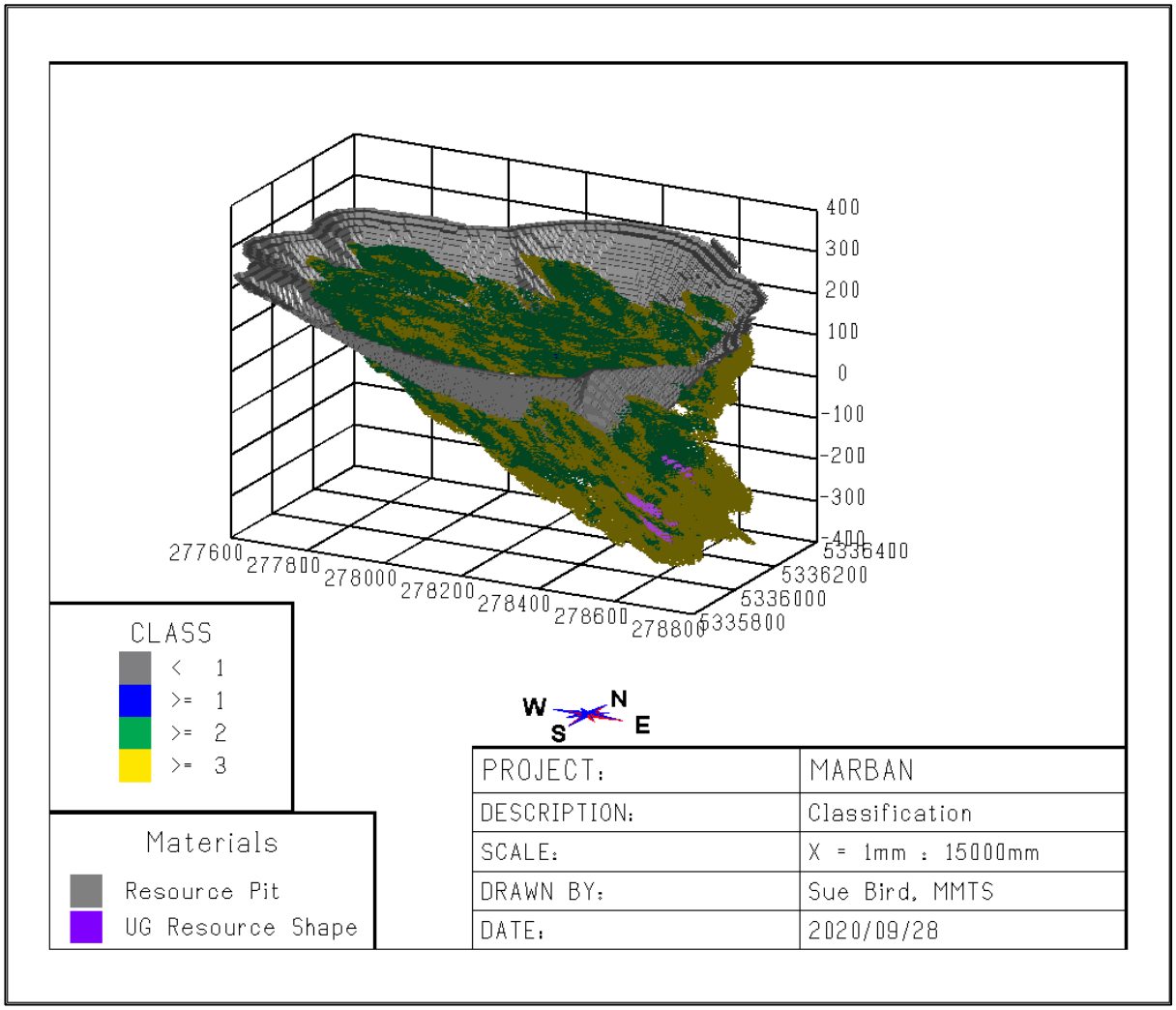
The blocks were classified according to CIM (2014 and 2019) definitions. All Classified material must be within a potentially mineralized wireframe and within the "reasonable prospects of eventual economic extraction" shapes. The following criteria have been used:

- Blocks within a wireframe and within an anisotropic search ellipse with dimensions of 100mx100mx15m are assigned a preliminary classification of Inferred.
- Indicted blocks are required to two drillholes within 22m
- Measured blocks are required to have three drillholes within 10m

The distances used for Classification are based on the variography, with the measured distance the range at 70% of the sill (10m) and the Indicated distances the range at 90% of the sill. These

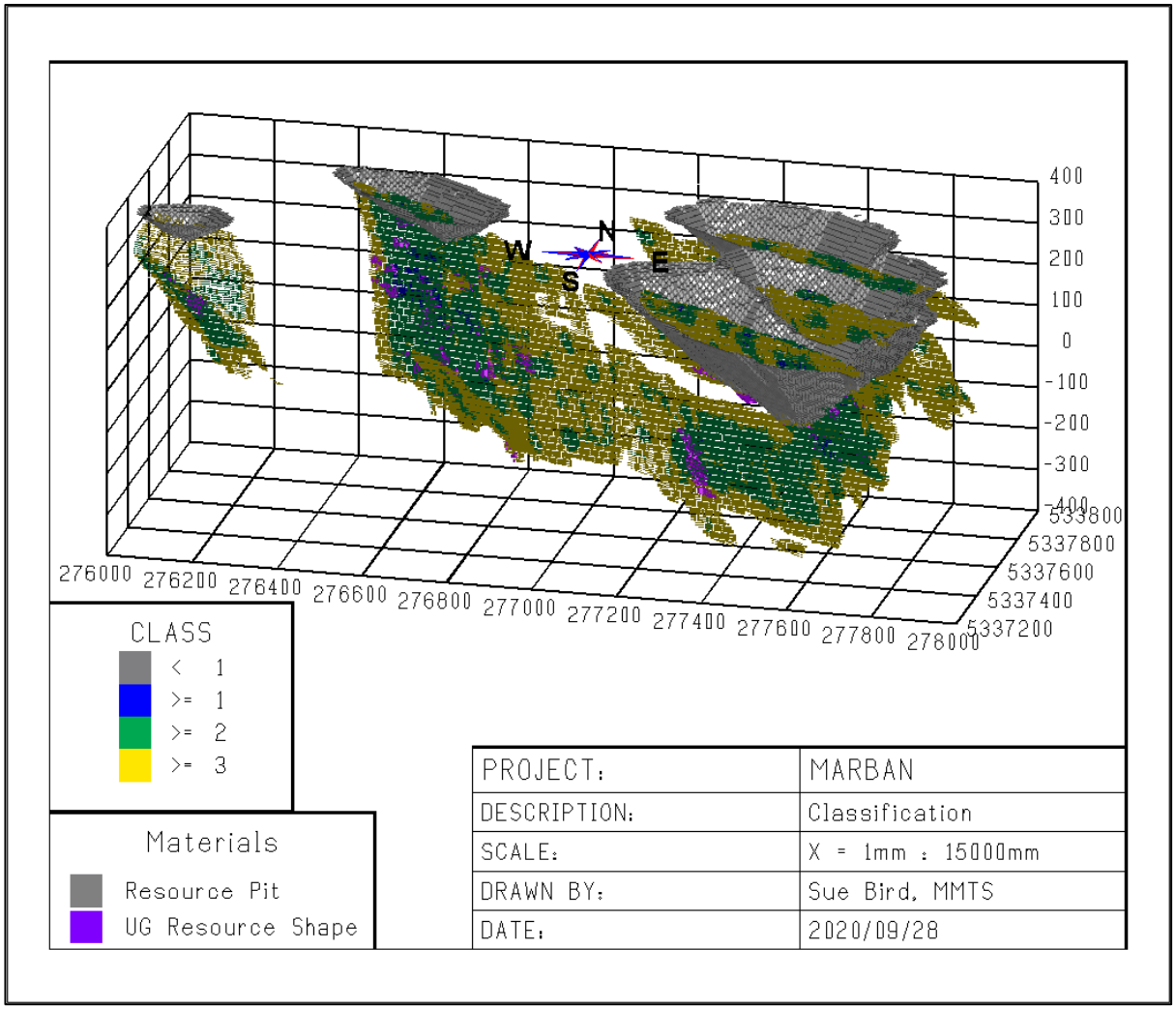
distances are considered reasonable for this type of deposit that are also constrained within wireframes. Data prior to 1984 has not been used in the resource estimate of in the Classification of the resource. The figures below illustrated the Measured, Indicated and Inferred blocks for Marban and Kierens-Norlartic respectively. Also illustrated is the resource pit and underground shapes, pertinent to the following section.

Figure 14-22: Block Classification – Marban



Source: MMTS, 2020

Figure 14-23: Block Classification – Kierens-Norlartic



Source: MMTS, 2020

14.9 Reasonable Prospects of Eventual Economic Extraction

As illustrated above, the Marban and Kierens-Norlartic resources have been constrained by either a pit shell or an underground resource shape. The majority of the resource in each area is within the resource open pit shape. There is a small portion of underground material also included in the resource which has been constrained by a 3.5g/t Au gradeshell.

Areas of the gradeshell that are considered too remote or too small to be considered as having a “reasonable prospects of eventual economic extraction” have been removed. All remaining material within the gradeshell are reported for the underground portion of the resource estimate.

To define the resource pits, Net Smelter Prices and Processing and G&A costs have been used as summarized in Table 14-20. Pit slopes have been defined based on Golder’s recommendations (Golder, 2020) as summarized in Table 14-21.

Table 14-16: Resource Pit Inputs

Lerchs-Grossman Pit Shell Input	Value	Unit
Gold Price	$1,800.00	US$/Oz
Forex	0.75	US$:CDN$
Gold Payable	99.95%	%
Gold Refining	2.00	US$/Oz
Gold Transport	2.00	US$/Oz
Insurance	0.30	US$/Oz
Royalty	3.0%	CDN$/tonne processed
Process Costs	$14.00	CDN$/tonne processed
G&A and Site Costs	$2.00	CDN$/tonne processed
Pit Rim Mining Cost	$2.40	CDN$/tonne mined
Incremental Mining Cost / 5m below 300m elevation	$0.015	CDN$/tonne mined

Table 14-17: Resource Pit Slope Inputs for LG

Pit Area	Overall Slope (degrees)	Azimuth Range
Overburden	25	All
Marban Footwall	35	165 to 205
Marban Main	50	205 to 165
Norlartic Footwall	40	195 to 235
Norlartic Main	45	235 to 195
Kierens Footwall	40	195 to 235
Kierens Main	40	235 to 195

The recovery formula used for the Lerchs-Grossman (LG) pit is:

$$Recovery = \ln(Au\ Grade) * 0.0336 + 0.9111, maximum\ 93.7\%$$

The cut-off grade is estimated using a gold price of US$1,400/oz, Processing and G&A costs of CDN$15.00 and the same smelter terms as those used for the LG pit to give a net smelter price of CSN$57.86. Using an estimated overall metallurgical recovery of 88%, this gives a cut-off grade of 0.30 g/t Au, calculated as:

$$Cutoff\ Grade = \frac{(15.00)}{57.86 * 88\%} = 0.30\ g/t$$

Figure 14-20 and 14-21 illustrate the Resource pits and underground shapes used in the Resource Estimate for Marban and Kierens-Norlartic respectively.

14.10 Mineral Resource Sensitivity to Cut-off Grade

The sensitivity of the pit resource to cu-toff grade is summarized in Table 14-16. Sensitivity of the underground to cut-off grade was not done because, by definition, all material within an underground shape is reported as a resource.

Table 14-18: Sensitivity of the Pit Resource to Cut-off Grade

CLASS	Cut-off	MARBAN		KIERENS-NORLARTIC		Total in Pit	
		ROM tonnage	In situ	ROM tonnage	In situ	ROM tonnage	In situ
	(g/t)	(kt)	AU (g/t)	(kt)	AU (g/t)	(kt)	AU (g/t)
Measured	0.30	65	1.32	450	1.03	515	1.07
	0.35	56	1.47	433	1.06	489	1.10
	0.40	50	1.60	414	1.09	464	1.14
	0.45	47	1.69	385	1.14	432	1.20
	0.50	42	1.82	355	1.20	397	1.26
	0.60	38	1.98	306	1.30	344	1.37
	0.80	28	2.45	224	1.52	252	1.62
	1.00	23	2.73	163	1.76	186	1.88
Indicated	0.30	46,260	1.03	6,646	1.15	52,906	1.05
	0.35	42,205	1.10	6,321	1.20	48,526	1.11
	0.40	38,513	1.17	5,955	1.25	44,468	1.18
	0.45	35,056	1.24	5,624	1.29	40,680	1.25
	0.50	31,970	1.32	5,324	1.34	37,294	1.32
	0.60	26,839	1.47	4,711	1.44	31,550	1.46
	0.80	19,325	1.77	3,618	1.67	22,943	1.75
	1.00	14,263	2.08	2,796	1.90	17,059	2.05
Measured + Indicated	0.30	46,325	1.03	7,096	1.15	53,421	1.05
	0.35	42,261	1.10	6,754	1.19	49,015	1.11
	0.40	38,563	1.17	6,369	1.24	44,932	1.18
	0.45	35,103	1.25	6,009	1.28	41,112	1.25
	0.50	32,012	1.32	5,679	1.33	37,691	1.32
	0.60	26,877	1.47	5,017	1.44	31,894	1.46
	0.80	19,353	1.77	3,842	1.66	23,195	1.75
	1.00	14,286	2.08	2,959	1.89	17,245	2.05
Inferred	0.30	6,465	1.09	6,299	1.42	12,764	1.25
	0.35	5,920	1.16	6,050	1.46	11,970	1.31
	0.40	5,443	1.23	5,809	1.51	11,252	1.37
	0.45	5,030	1.30	5,595	1.55	10,625	1.43
	0.50	4,664	1.36	5,393	1.59	10,057	1.48
	0.60	3,991	1.50	4,865	1.70	8,856	1.61
	0.80	2,900	1.81	4,040	1.91	6,940	1.86
	1.00	2,038	2.19	3,358	2.11	5,396	2.14

Notes:

1. *The Mineral Resource estimate has been prepared by Sue Bird, P.Eng., an independent Qualified Person.*
2. *Resources are reported using the 2014 CIM Definition Standards and were estimated in accordance with the CIM 2019 Best Practices Guidelines.*
3. *Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.*
4. *The open pit Mineral Resource has been confined by a "reasonable prospects of eventual economic extraction" pit shell generated using the following assumptions: US$1,800/oz. Au at a currency exchange rate of 0.75 US$ per CDN$; 99.95% payable Au; CDN$4.30/oz Au offsite costs (refining, transport and insurance); a 3% NSR royalty; $16/t process and G&A costs; $2.60/t mining costs; grade dependent mill process recoveries; and pit slopes varying from 25 to 50 degree overall depending on geotechnical characteristics.*
5. *The underground Mineral Resource reports all material within a continuous 3.5g/t Au gradeshell*
6. *Metallurgical recovery is based on the formula: ln(Au)*0.0372+0.9017, maximum 96.7%*
7. *The specific gravity of the deposit has been determined by lithology as being between 2.67 and 2.81.*

8. *Numbers may not add due to rounding.*

There are no other known 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.

The mine plan is partly based on Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the PEA based on these Mineral Resources will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

14.11 Comparison to the Previous Resource Estimate

The previous resource estimate was completed in 2016 (Belzile, 2016). The resource is compared below using a cut-off grade for the current in pit resource of 0.40 g/t Au for comparison purposes only (Note that the current resource cut-off for open pit is 0.30 g/t Au). The differences in the resource are determined to be:

1. An overall increase in tonnage of 26% and the same overall grade (at cut-off of 0.40 g/t for pit and 3.5 g/t for underground) for both deposits.

2. Increase in tonnage due primarily to:

    a) Re-interpretation of the Marban area mineralization to better conform to geologic controls and to include more lower grade mineralization suitable to open pit mining

    b) Re-interpretation of part of the North zone in Kierens-Norlartic to include additional mineralization

    c) Larger “reasonable prospects of eventual economic extraction” pits due to a higher Au price.

3. Less Measured + Indicated material for Kierens-Norlartic due to removal of historic assays which had no QAQC and were shown to contain a bias in the data.

Table 14-19: Comparison of 2020 Model with 2016 Resource Model

Source	CLASS	Cut-off	2020 - MMTS			2016			Difference (1-2016/2020)		
			Tonnage	Au	Ounces	Tonnage	Au	Ounces	Tonnage	Au	Ounces
		(gpt)	(kt)	(gpt)	(kOz)	(kt)	(gpt)	(kOz)	(%)	(%)	(%)
Pit	Meas	0.4	50	1.60	3	7,653	1.47	362			
	Ind		38,513	1.17	1,450	20,801	1.13	756			
	M+I		38,563	1.17	1,453	28,454	1.22	1,117	26%	-4%	23%
	Inf		5,443	1.23	216	3,599	1.15	133	34%	7%	38%
	All in Pit		44,006	1.18	1,668	32,053	1.21	1,250	27%	-3%	25%
Underground	Meas	3.5			0	18	3.49	2			
	Ind		220	7.77	55	279	4.24	38			
	M+I		220	7.77	55	297	4.19	40	-35%	46%	27%
	Inf		304	8.73	85	502	3.67	59	-65%	58%	31%
	All Underground		524	8.33	140	799	3.86	99	-52%	54%	29%
Marban - Pit and UG			44,530	1.26	1,809	32,852	1.28	1,350	26%	-1%	25%
Kierens-Norlartic											
Pit	M+I	0.4	6,369	1.24	253	8,975	1.30	375	-41%	-5%	-48%
	Inferred		5,809	1.51	281	25	1.32	1	100%	12%	100%
	ALL - Pit		12,178	1.36	534	9,000	1.30	377	26%	4%	29%
Underground	M+I	3.5	529	3.58	61	432	3.68	51	18%	-3%	16%
	Inferred		119	3.02	12	12	3.61	51	90%	-19%	-341%
	All Underground		648	3.47	72	444	3.68	53	31%	-6%	27%
Kierens-Norlartic - Pit and UG			12,826	1.47	607	9,444	1.42	430	26%	4%	29%
All - Marban + Kierens-Norlartic											
All - Pit and UG		varies	57,356	1.31	2,415	42,296	1.31	1,780	26%	0%	26%

15 Mineral Reserve Estimates

This section is not relevant to the Technical report

16 Mining Methods

16.1 Introduction

Open pit mine designs, mine production schedules, mine capital and operating costs have been developed for the Marban and Kierens-Norlartic (KN) deposits at a scoping level of engineering.

The open pit operations are designed for approximately fifteen years of operation. Mine planning is based on conventional open pit methods suited for the project location and local site requirements. The subset of Mineral Resources contained within the designed open pits are summarized in Table 16-1, with a 0.35 g/t gold cut-off, and form the basis of the mine plan and production schedule.

Table 16-1: PEA Mine Plan Production Summary

Description	Value
PEA Mill Feed	60,356 kt
Mill Feed Gold Grade	0.97 g/t
Waste Overburden and Rock	355,627 kt
Strip Ratio	5.9

Notes:

1. *The PEA Mine Plan and Mill Feed estimates are a subset of the August 12, 2020 Mineral Resource estimates and are based on open pit mine engineering and technical information developed at a Scoping level for the Marban and Kierens-Norlartic deposit.*
2. *PEA Mine Plan and Mill Feed estimates are mined tonnes and grade, the reference point is the primary crusher.*
3. *Waste/mineralization contact edge dilution of 27% at 0.14 g/t for Marban 20% at 0.15 g/t for KN is applied to the insitu Mineral Resources. Mining Recovery of 100% of diluted tonnages is assumed.*
4. *Cut-off grade of 0.35 g/t assumes US$1,400/oz. Au at a currency exchange rate of 0.75 US$ per C$; 99.95% payable gold; $4.30/oz offsite costs (refining, transport and insurance); a 3.0% NSR royalty; and a variable metallurgical recovery for gold that is anticipated to be 88% at cut-off.*
5. *The cut-off grade covers processing costs of $14.00/t, administrative (G&A) costs of $2.00/t, and low grade stockpile Rehandle costs of $1.50/t.*
6. *Estimates have been rounded and may result in summation differences.*

Ultimate pit limits are split up into phases or pushbacks to target higher economic margin material earlier in the mine life. The Marban deposit is split into four phases, and the KN deposit is split into six phases, two for Kierens, two for Norlartic, one for North Zone and one for Gold Hawk. Pit designs are configured on 5 m bench heights, with 8 to 9 m wide berms placed every four benches, or quadruple benching. Bench face angles, and subsequent inter-ramp angles, are varied based on prescribed geotechnical zones.

The mill will be fed with material from the pit at an average rate of 4.0 Mtpa (11 ktpd). Cut-off grade optimization is employed, which feeds a low-grade stockpile south of the Marban deposit, which is planned to be rehandled as feed to the mill in the later years of the mine life. Overburden, for use upon mine closure, will be placed in a stockpile directly north of the Marban pit. Waste rock will be placed in stockpiles adjacent to all pits. Waste rock will also be used for construction of the tailings and water management pond northwest of the deposits.

Mining operations will be based on 365 operating days per year with two 12-hour shifts per day. An allowance of 10 days of no mine production has been built into the mine schedule to allow for adverse weather conditions.

The mining fleet will include diesel-powered down the hole (DTH) drills with 165 mm bit size for production drilling, diesel-powered reverse circulation (RC) drills for bench-scale grade control

drilling, 12-m^3 bucket size diesel-hydraulic excavators and 13-m^3 bucket sized wheel loaders for production loading, 91-t payload rigid-frame haul trucks and 36-t articulated trucks for production hauling, plus ancillary and service equipment to support the mining operations. In-pit dewatering systems will be established for each pit. All surface water and precipitation in the pits will be handled by submersible pumps.

The mine equipment fleet is planned to be purchased via a lease financing arrangement. Maintenance on mine equipment will be performed in the field with major repairs to mobile equipment in the shops located near the plant facilities.

16.2 Key Design Criteria

The following mine planning design inputs were used:

- topography is based on a LiDAR survey of the region
- resource block model on 5 m spacing in all three dimensions. Resource model contains mineralized block percentages based on 3D modelled domains, mineralized gold grades, specific gravities, lithologies, resource classifications, and measurements of underground voids based on historic mining
- Measured, Indicated and Inferred class Mineral Resource Estimates included in pit optimizations and mill feed estimates
- stockpile and haul roads are planning to minimize wetland, waterbody and watercourse disturbance
- pit crests and stockpile toes planned to fall within existing O3 Mining claim boundaries.
- a grade dependant gold process recovery is used for the pit optimisation and cut-off grade estimations:
    - process recovery = 0.0336 * LN (gold head grade) + 0.9111, capped at 93.7%

16.2.1 Net Smelter Price and Cut-off Grade

Net Smelter Price (NSP) is used for mine planning, in place of the market price for gold, to consider all offsite costs and determine revenue potential at the mine gate. The NSP calculation uses the inputs shown in Table 16-2.

Table 16-2: Net Smelter Price

Item	Value
Gold Price	US$1,400/oz
US Exchange Rate	0.75 US$: 1 C$
Payable Gold	99.95%
Gold Offsite Costs (Refining, Transport, Insurance)	US$4.30/oz
Royalty	3.0%
Net Smelter Price	**C$57.86/g**
	C$1,800/oz

The economic cut-off grade is chosen as the gold grade required to pay for processing costs, general and administration costs, and low-grade stockpile reclaim costs. The cut-off grade calculation uses the inputs shown in Table 16-3:

Table 16-3: Economic Cut-off Grade

Item	Value
Net Smelter Price	57.86/g
Process Recovery at Cut-off	88%
Process Costs	$14.00/t
G&A and Site Costs	$2.00/t
Stockpile Rehandle Costs	$1.50/t
Economic Cut-off Grade	**0.35 g/t**

16.2.2 Mining Loss & Dilution

The Mineral Resources are modelled as mineralized percentages within 5 m block dimensions. For mine planning, a contact dilution skin is added to waste contacts in the resource model, which accounts for planned open pit mine operating conditions.

These waste dilution contacts are determined wherever the block mineralized percent is less than 100% in the block, or wherever a neighbouring block is less that the 0.35 g/t cut-off grade.

The applied contact dilution is 27% at 0.14 g/t for the Marban deposit and 20% at 0.15 g/t for the KN deposit.

Additional mining operational losses have not been accounted for and are assumed to be considered within the scoping level accuracy of the dilution measurements.

16.2.3 Pit Slopes

Pit designs are configured on 5 m bench heights, with 8 to 9 m wide berms placed every four benches, or quadruple benching. Bench face angles, and subsequent inter-ramp angles, are varied based on prescribed geotechnical zones.

The pit slope criteria are based on a 2020 geotechnical report by Golder Associates Ltd. (Golder, 2020). The majority of the rock mass at the Marban Project is strong and can be described as typical Abitibi belt hard rock. Overall rock mass failure is not a concern for the proposed 85 m to 420 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. It is anticipated that the dip of schistosity/foliation will be the dominant control on bench face angle. It is assumed that foliation is aligned with the orientation of the lithological units.

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 location and behavior of schists is not well defined at this stage for any of the Marban Project deposits. More field data and analysis will better define these zones and their potential modes of failure and the bench configurations will be updated accordingly in future studies. Local experience indicates that these zones will require single-benching for safe excavation and less problematic operations. The design slopes and bench configuration are estimated to account for the presence of these weaker rock units, specifically in the Marban footwall, Norlartic footwall and Kierens hangingwall and footwall, blending potential shallower single benching arrangements with steeper multiple benching arrangements into an average pit wall configuration. Adjustments to the slope designs are likely and should be anticipated in subsequent studies.

Bench face and inter-ramp slopes in defined zones are listed in Table 16-4 for Marban pit and Table 16-5 for Norlartic and North zones pits and Table 16-6 for Kierens and Gold Hawk pits.

Table 16-4: Marban Pit Bench Face & Inter-Ramp Angle Inputs

Domain	Azimuth	Bench Face Angle (º)	Inter-Ramp Angle (º)	Overall Slope* (º)
Overburden	All	37	30	25
Hangingwall and End Walls	205º – 0º – 165º	69	51	50
Footwall	165º – 205º	52	40	35

*Overall slope angles are inputs for pit optimisations only

Table 16-5: Norlartic and North Zone Pits Bench Face Inter-Ramp Angle Inputs

Domain	Azimuth	Bench Face Angle (º)	Inter-Ramp Angle (º)	Overall Slope * (º)
Overburden	All	37	30	25
Hangingwall and End Walls	235º – 0º – 195º	75	55	45
Footwall	195º – 235º	65	48	40

*Overall slope angles are inputs for pit optimisations only

Table 16-6: Kierens and Gold Hawk Bench Face Inter-Ramp Angle Inputs

Domain	Azimuth	Bench Face Angle (º)	Inter-Ramp Angle (º)	Overall Slope * (º)
Overburden	All	37	30	25
Hangingwall and End Walls	235º – 0º – 195º	69	51	40
Footwall	195º – 235º	65	48	40

*Overall slope angles are inputs for pit optimisations only

In-pit haul roads and geotechnical berms (20 m wide) are added to the pit designs and flatten the inter-ramp slopes out to shallower overall slopes. Geotechnical berms are placed on 140 m vertical spacing, wherever in-pit ramps are not present.

16.3 Pit Optimisation

The economic pit limits are determined using the Lerchs-Grossman (LG) algorithm. This algorithm uses the resource gold grades, mineralized block percentages, and specific gravity (SG) for each block of the 3D block model and evaluates the costs and revenues of the blocks within potential pit shells. The routine uses input economic and engineering parameters and expands downwards and outwards until the last increment is at break-even economics.

Additional cases are included in the analysis to evaluate the sensitivities of open pit mined resources to strip ratio and high-grade/low-grade areas of the deposits. In this study, the various cases or pit shells are generated by varying the input gold price and comparing the resultant waste and mill feed tonnages and gold grades for each pit shell.

By varying the economic parameters while keeping inputs for metallurgical recoveries and pit slopes constant, various generated pit cases are evaluated to determine where incremental pit shells produce marginal or negative economic returns. This drop-off is due to increasing strip ratios, decreasing gold grades, increased mining costs associated with the larger or deeper pit shells, and the value of discounting costs before revenues.

The economic margins from the expanded cases are evaluated on a relative basis to provide payback on capital and produce a return for the project. At some point, further expansion does not provide significant added value. A pit limit can then be chosen that has suitable economic return for the deposit.

For each pit shell, an undiscounted cashflow (UCF) is generated based on the shell contents and the economic parameters listed in Table 16-7. The UCFs for each case are compared to reinforce the selected point at which increased pit expansions do not increase the project value. Note that the economics are only applied for comparative purposes to assist in the selection of an optimum pit shell for further mine planning; they do not reflect the actual financial results of the mine plan.

The chosen pit shell is then used as the basis for more detailed design and economic modelling.

Price inputs for the LG runs are listed Table 16.2 above and operating cost assumptions are provided in Table 16-7Table 16-7. The input gold price is varied from US$200/oz to US$2,500/oz.

Table 16-7: Operating Cost Inputs into Pseudoflow Shell Runs

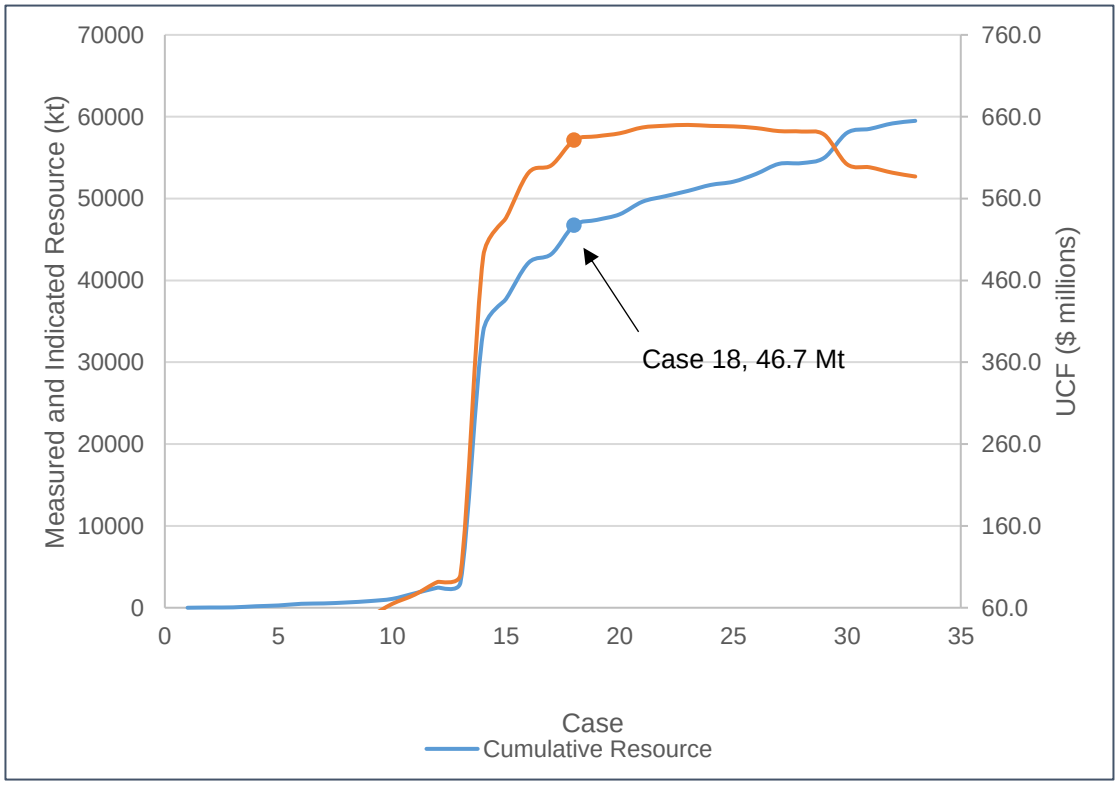
Item	Unit
Pit rim mining cost	$2.40/t, pit rim of 300 m in all deposits
Incremental haulage cost	$0.015 per every 5 m bench below pit rim
Processing cost	$14.00/t
General/Administration cost	$2.00/t

16.3.1 Marban Pit Limit

Figure 16-1 shows the contents of the generated LG pit shells for the Marban deposit. An inflection point can be seen in the curve of cumulative resources and UCF by pit case. This point indicates Case 18 as a point at which larger pit shells will not produce significant increases to project value.

The pit shell generated from Case 18 is selected as the ultimate pit limits for Marban and is used for further mine planning as a target for detailed open pit designs with berms and ramps.

Figure 16-1: Marban Pseudoflow Pit Shell Resource Contents by Case



Source: Moose Mountain, 2020.

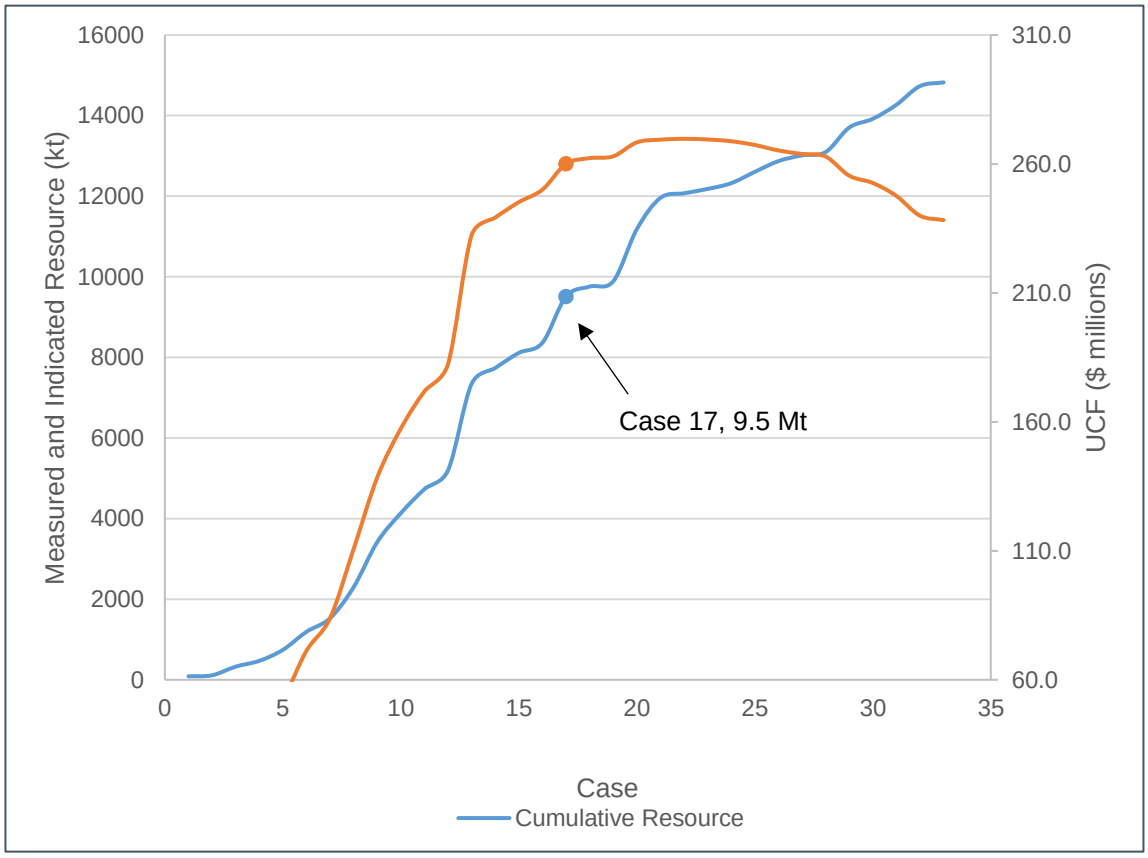
The Marban pit is restricted by the claim boundaries in the east.

16.3.2 KN Pit Limit

Figure 16-2 shows the contents of the generated LG pit shells for the KN deposits. An inflection point can be seen in the curve of cumulative resources and UCF by pit case. This point indicates Case 17 as a point at which larger pit shells will not produce significant increases to the project value.

The pit shells generated from Case 17 are selected as the ultimate pit limits for the KN deposits and are used for further mine planning as a target for detailed open pit designs with berms and ramps.

Figure 16-2: KN Pseudoflow Pit Shell Resource Contents by Case



Source: Moose Mountain, 2020.

Future studies should consider running pit optimization analysis on each KN deposit separately: Kierens, Norlartic, North Zone, and Gold Hawk.

The Norlartic pit is restricted by the claim boundaries in the east.

16.4 Pit Designs

Contents of the designed open pits are presented in Table 16-8. The contents for each designed pit phase are presented graphically in Figure 16-3.

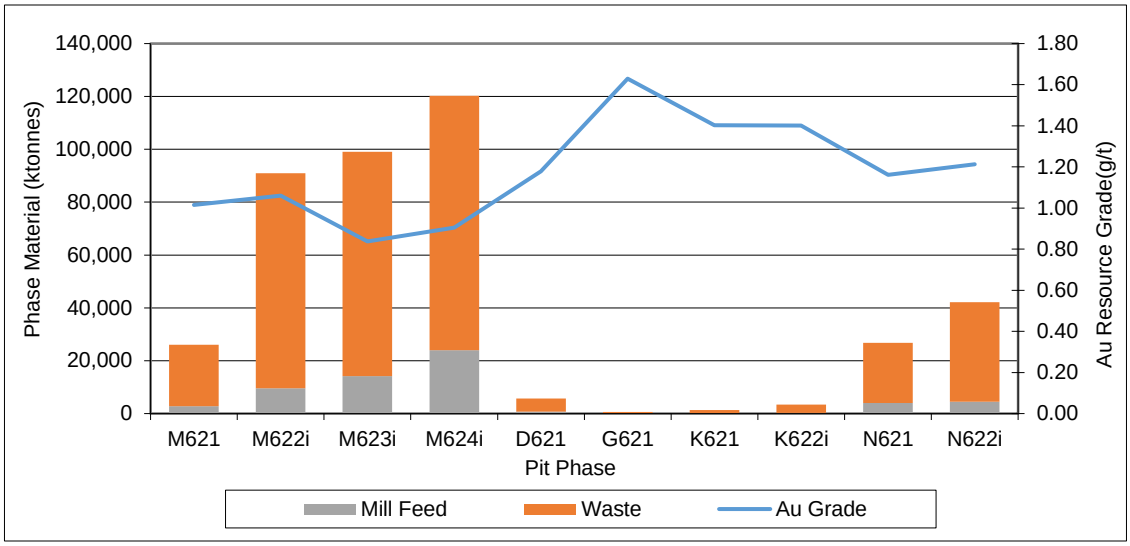
Table 16-8: Contents of Designed Pit Phases

Pit Phase	Pit Name	Mill Feed (Mt)	Diluted Gold Grade (g/t Au)	Waste (Mt)	Strip Ratio (t/t)
Marban Phase 1	M621	2.8	1.02	23.2	8.2
Marban Phase 2	M622i	9.6	1.06	81.4	8.5
Marban Phase 3	M623i	14.2	0.84	84.9	6.0
Marban Phase 4	M624i	24.0	0.90	96.2	4.0
Total Marban	**M624**	**50.6**	**0.92**	**285.6**	**5.6**
North Zone	D621	0.8	1.18	4.9	6.3
Gold Hawk	G621	0.0	1.63	0.5	18.0
Kierens Phase 1	K621	0.1	1.40	1.2	8.9
Kierens Phase 2	K622i	0.3	1.40	3.1	10.3
Norlartic Phase 1	N621	4.1	1.16	22.7	5.6
Norlartic Phase 2	N622i	4.5	1.21	37.7	8.4
Total KN	**N622**	**9.8**	**1.20**	**70.0**	**7.2**
Grand Total		**60.4**	**0.97**	**355.6**	**5.9**

Notes:

1. *The PEA Mine Plan and Mill Feed estimates are a subset of the August 12, 2020 Mineral Resource estimates and are based on open pit mine engineering and technical information developed at a Scoping level for the Marban and KN deposits.*
2. *PEA Mine Plan and Mill Feed estimates are mined tonnes and grade, the reference point is the primary crusher.*
3. *Waste/mineralization contact edge dilution of 27% at 0.14 g/t for Marban 20% at 0.15 g/t for KN is applied to the insitu Mineral Resources. Mining Recovery of 100% of diluted tonnages is assumed.*
4. *Cut-off grade of 0.35 g/t assumes US$1,400/oz. Au at a currency exchange rate of 0.75 US$ per C$; 99.95% payable gold; $4.30/oz offsite costs (refining, transport and insurance); a 3.0% NSR royalty; and a variable metallurgical recovery for gold that is anticipated to be 88% at cut-off.*
5. *The cut-off grade covers processing costs of $14.00/t, administrative (G&A) costs of $2.00/t, and low-grade stockpile Rehandle costs of $1.50/t.*
6. *Estimates have been rounded and may result in summation differences.*

Figure 16-3: Designed Phase Pit Contents



Source: Moose Mountain, 2020.

16.4.1 In-Pit Haul Roads

Two-way haul roads of 29 m width are designed. Haul road grades are limited to a maximum of 10%. Access ramps are not designed for the last two benches of the pit bottom, on the assumption that the bottom ramp segment will be removed using some form of retreat mining. The bottom two ramped benches of the pit use one-way haul roads of 21 m width and 12% grade since bench volumes and traffic flow are reduced.

16.4.2 Pit Phases

Ultimate pit limits are generally split up into phases or pushbacks to target higher economic margin material earlier in the mine life. Minimum pushback distances of 75 m are honoured. The Marban pit is split into four phases with the higher-grade first two phases mined ahead of the two pushbacks to the southeast and northeast. Targets for the first two phases use Case 14 of the optimisation runs described in Section 16.3.1.

The Kierens and Norlartic pits are split into two phases with the lower strip ratio first phases mined ahead of north pushback second phases. Targets for the first phases use Case 9 of the optimisation runs described in Section 16.3.2. The North Zone and Gold Hawk pits are not phased.

16.4.3 Marban Pit Designs

The Marban pit designs are shown in Figure 16-4 (final pit phase) and Figure 16-5. Sections through the deposit showing the resource model grades and mineralized percentages are illustrated in Figure 16-6 and Figure 16-7.

Future design iterations will need to be made to accommodate anticipated updates to geotechnical design inputs, as described in Section 16.2.3. These updates are not estimated to materially impact the economics presented in this study.

16.4.3.1 Marban Phase 1, M621

This phase targets the high-grade, near surface portions of the deposit. This phase mines from the pit exit in the south at the 305 m elevation, down to the various pit bottoms between the 200 m and 230 m elevations. The main ramp runs counter-clockwise down from the pit exit. The southwest portion of the pit is accessed from the 270 m elevation down to the 220 m elevation from a secondary ramp. This pit bottom will be mined before the remainder of the pit progresses below the 260 m elevation.

16.4.3.2 Marban Phase 2, M622

This phase targets deeper, high-grade mineralisation west of the phase 1 pit. It does not push any of the phase 1 pit walls out, so can be mined independently of the phase 1 pit. Phase 2 mines to the final highwall in the west part of the pit, while leaving enough room for a final pushback to the phase 3 and 4 pits to the north, east and south. This phase mines from the pit exit at the 305 m elevation, down to the pit bottom at the 60 m elevation. The main ramp runs counter-clockwise from the pit exit in the west of the pit, switchbacking at the 180 m and 140 m elevations.

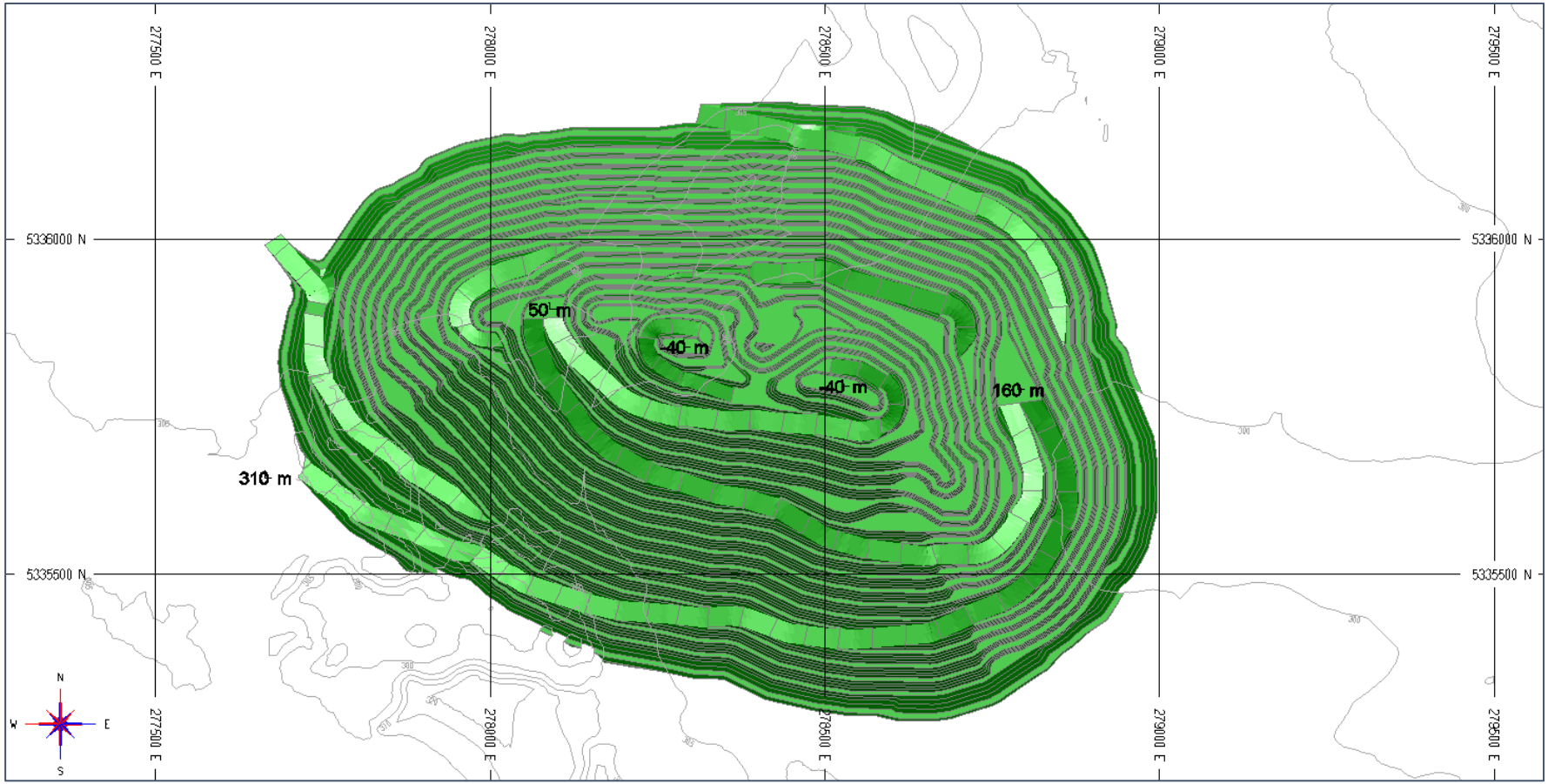
16.4.3.3 Marban Phase 3, M623

This phase pushes the pit out in the north and east directions, targeting the deep mineralisation in the north. This phase mines from the pit exit at the 305 m elevation, down to the pit bottom at the 40 m elevation. The main ramp runs clockwise down from the pit exit in the north of the pit, switchbacking at the 160 m elevation.

16.4.3.4 Marban Phase 4, M624

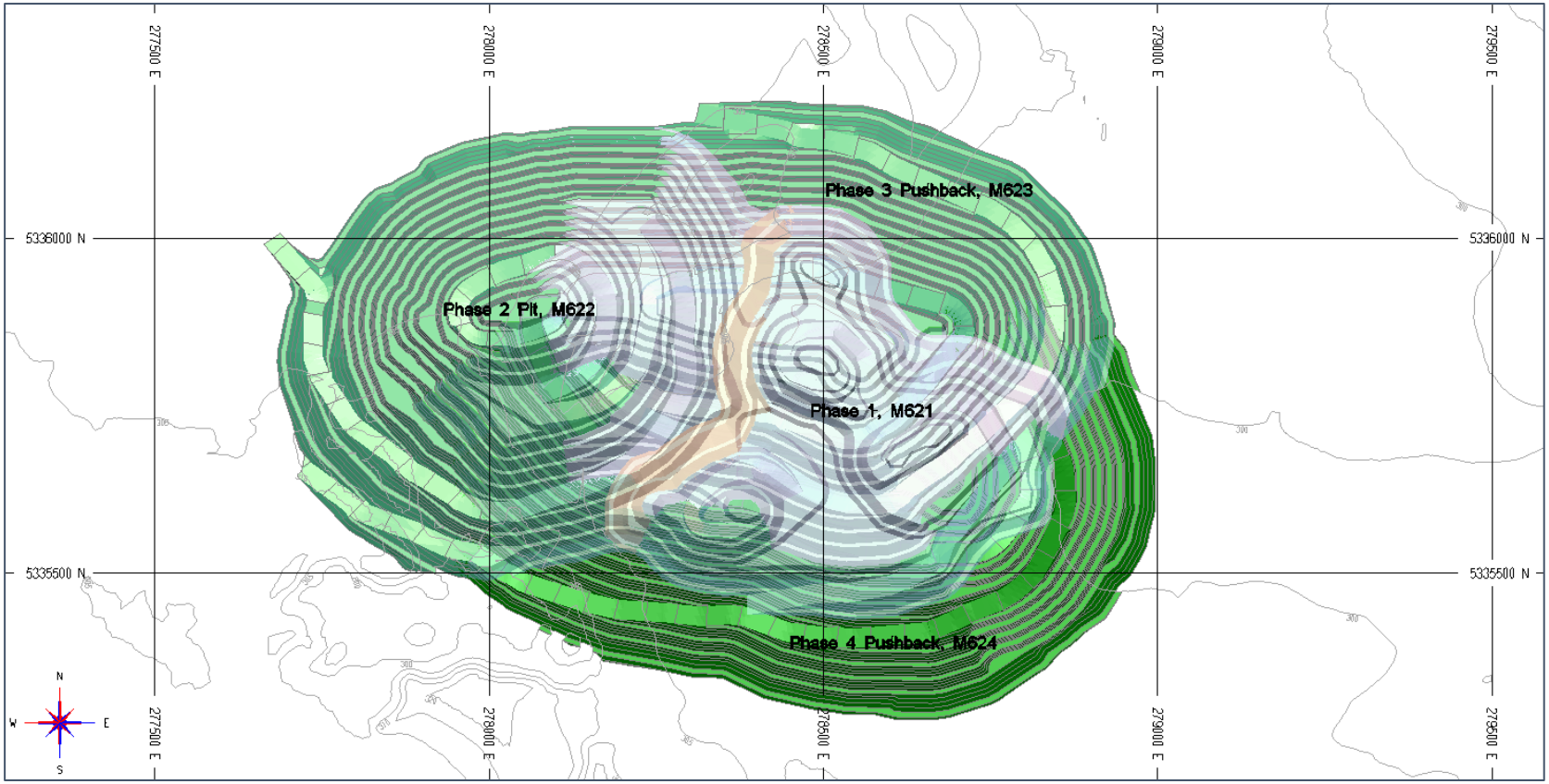
This phase pushes the pit out in the south and east directions, targeted the deepest economic mineralisation. This phase mines from the pit exit at the 310 m elevation, down to the pit bottom at the -40 m elevation. The main ramp runs clockwise down from the pit exit in the west of the pit, switchbacking at the 160 m and 50 m elevations.

Figure 16-4: Marban Pit Design Phase 4 Pit, M624



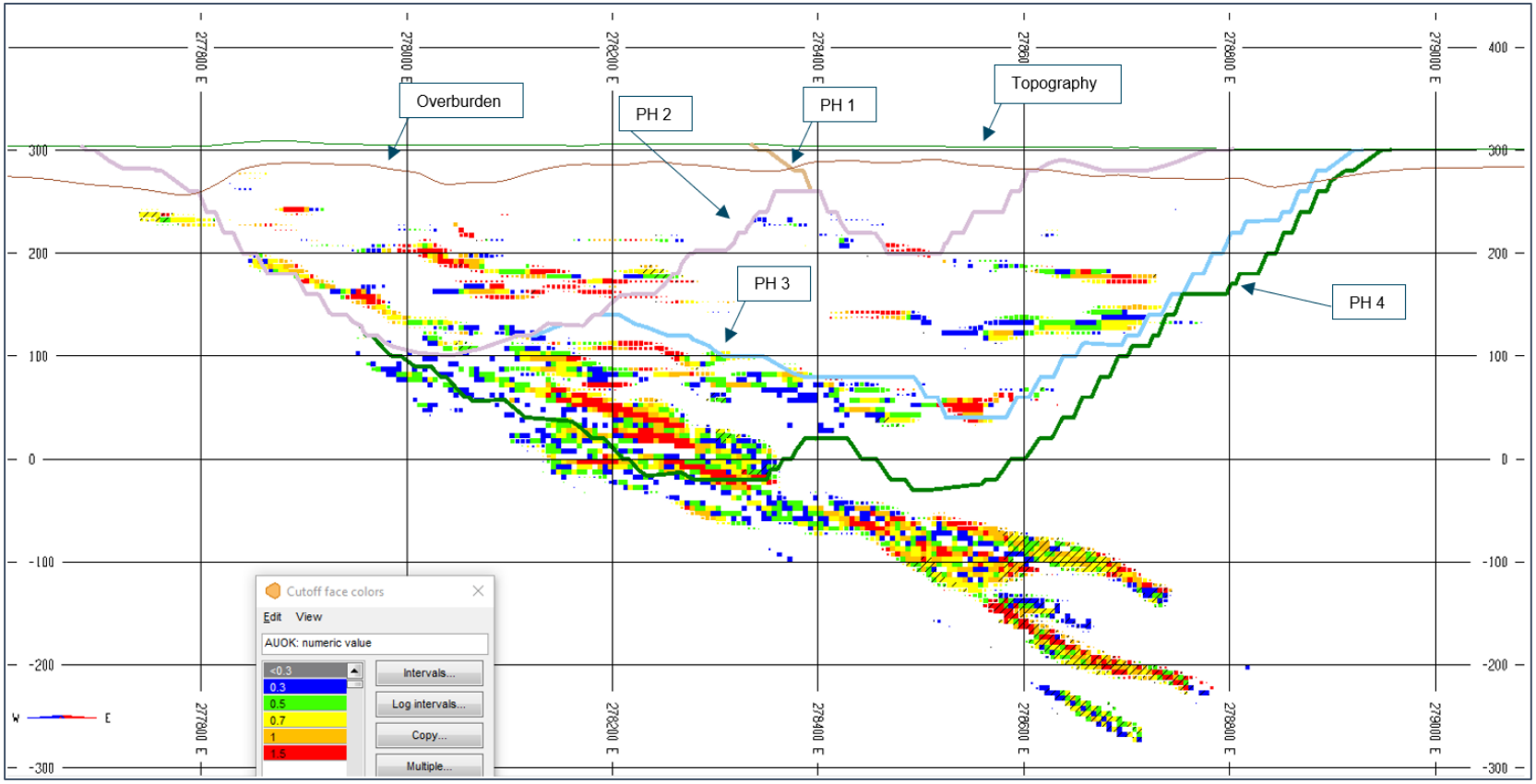
Source: Moose Mountain, 2020

Figure 16-5: Marban Phased Pit Designs



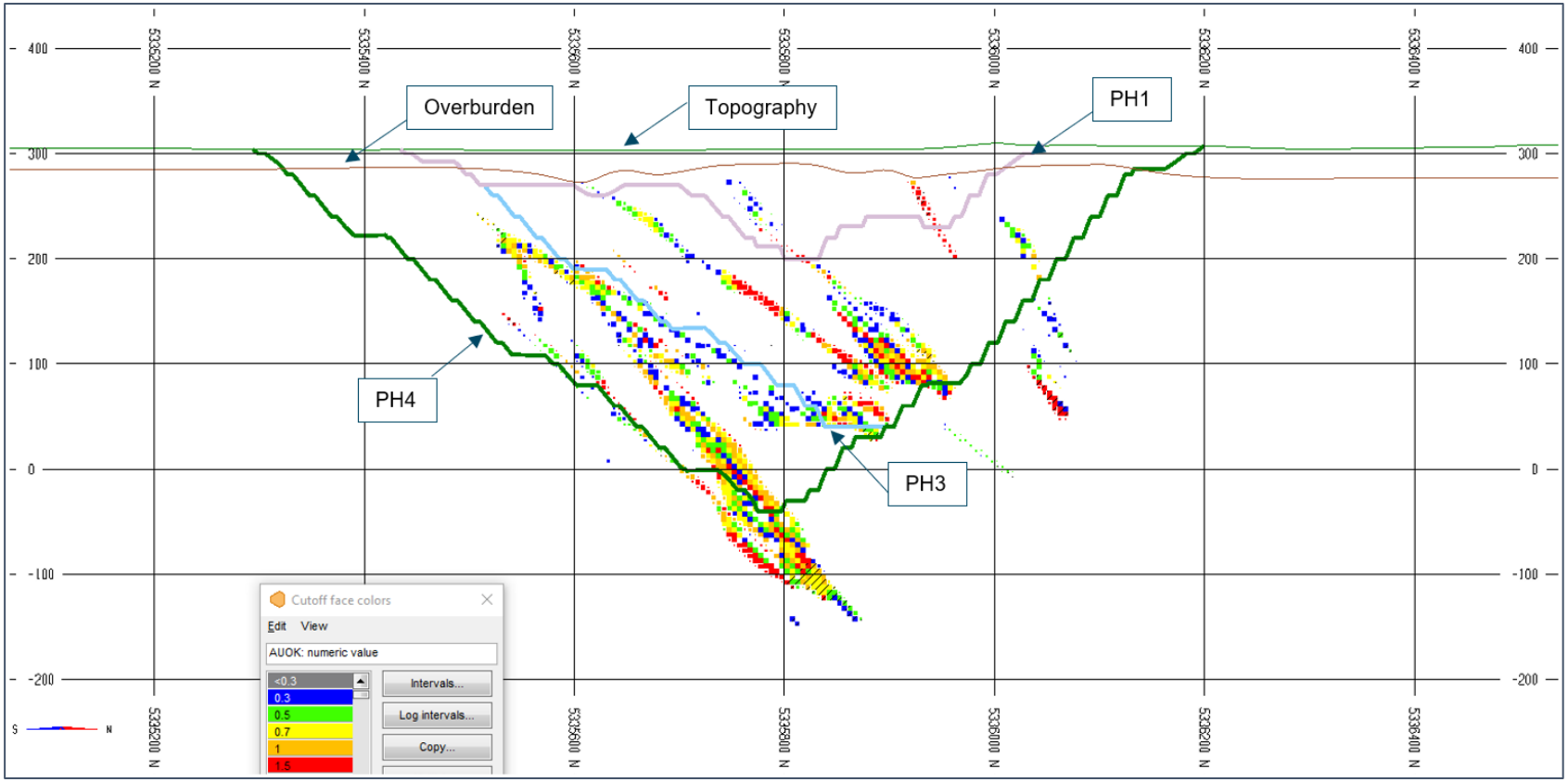
Source: Moose Mountain, 2020

Figure 16-6: Marban Pit Designs, West-East Section, 5,335,815N



Source: Moose Mountain, 2020

Figure 16-7: Marban Pit Designs, North-South Section, 278,500E



Source: Moose Mountain, 2020

16.4.4 KN Pit Designs

The phased KN pit designs are shown in Figure 16-8 to Figure 16-9. Sections through the deposit showing the resource model grades and mineralized percentages are illustrated in Figure 16-10 to Figure 16-13.

Future design iterations will need to be made to accommodate anticipated updates to geotechnical design inputs, as described in Section 16.2.3. These updates are not estimated to materially impact the economics presented in this study.

16.4.4.1 North Zone Pit, D621

This phase targets the near surface portion of the North Zone deposit. This phase mines from the pit exit at the 300 m elevation, down to the pit bottom at the 220 m elevation. The main ramp runs counter-clockwise down from the pit exit in the east.

16.4.4.2 Gold Hawk Pit, G621

This phase targets the near surface portion of the Gold Hawk deposit. This phase mines from the pit exit at the 300 m elevation, down to the pit bottom at the 270 m elevation. The main ramp runs counter-clockwise down from the pit exit in the east.

16.4.4.3 Kierens Phase 1, K621

This phase targets the lower strip ratio portion of the Kierens deposit. This phase mines from the pit exit at the 300 m elevation, down to the pit bottom at the 260 m elevation. The main ramp runs counter-clockwise down from the pit exit in the east.

16.4.4.4 Kierens Phase 2, K622

This phase pushes out in the north, west, and east directions, targeted deeper Kierens mineralisation. This phase mines from the pit exit at the 300 m elevation, down to the pit bottom at the 230 m elevation. The main ramp runs clockwise down from the pit exit in the south of the pit.

16.4.4.5 Norlartic Phase 1, N621

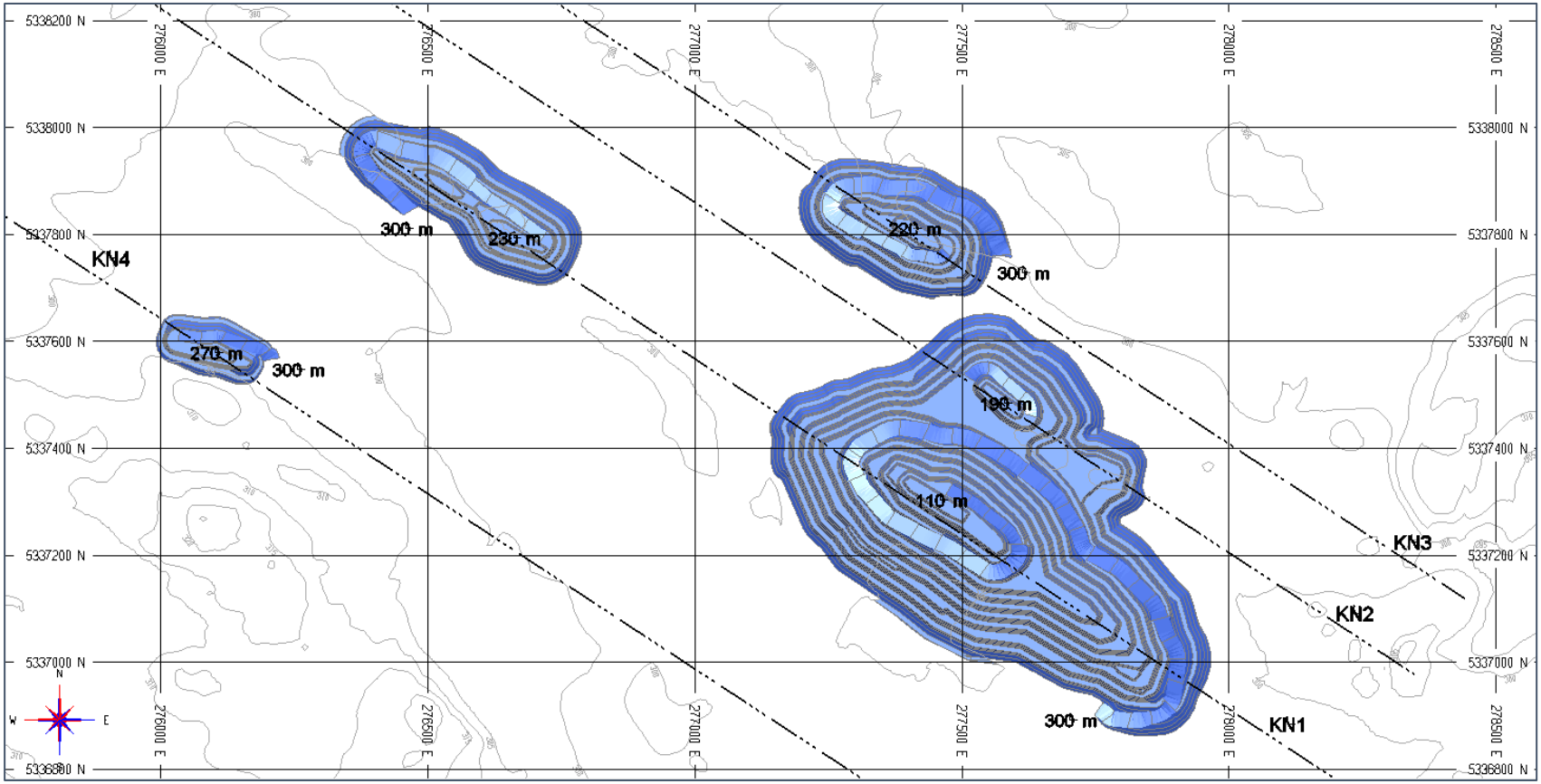
This phase targets the lower strip ratio portion of the Norlartic deposit. This phase mines from the pit exit at the 295 m elevation, down to two separate pit bottoms at the 220 m and 180 m elevations. The main ramp runs clockwise down from the pit exit in the west. The southeast portion of the pit is accessed from the 260 m elevation down to the 220 m elevation from a secondary ramp. This pit bottom will be mined out before the remainder of the pit progresses below the 250 m elevation.

16.4.4.6 Norlartic Phase 2, N622

This phase pushes out in the north and east directions, targeting deeper mineralisation, as well as the higher strip ratio northern portion of the Norlartic deposit. This phase mines from the pit exit at the 300 m elevation, down to two separate pit bottoms at the 190 m and 110 m

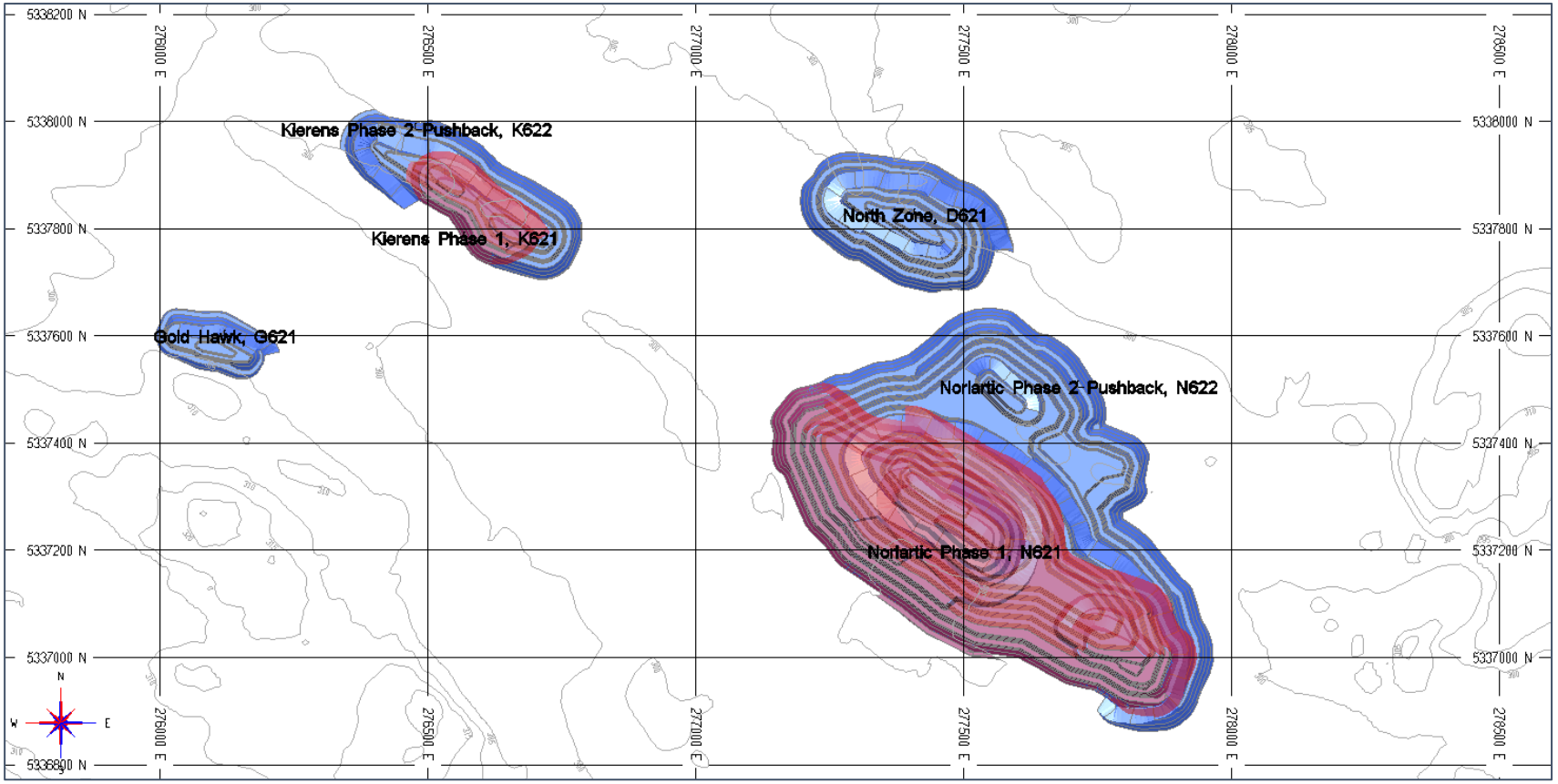
elevations. The main ramp runs counter-clockwise down from the pit exit in the south of the pit.

Figure 16-8: KN Pit Designs



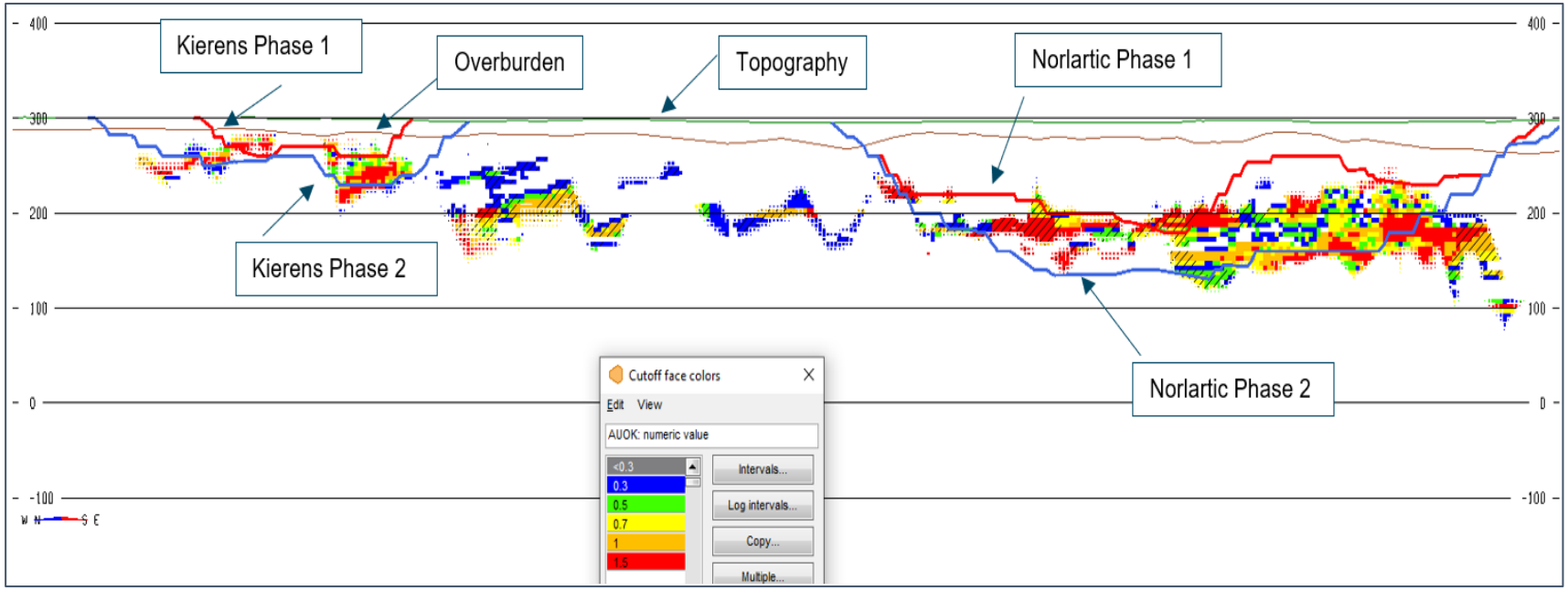
Source: Moose Mountain, 2020

Figure 16-9: KN Phased Pit Designs



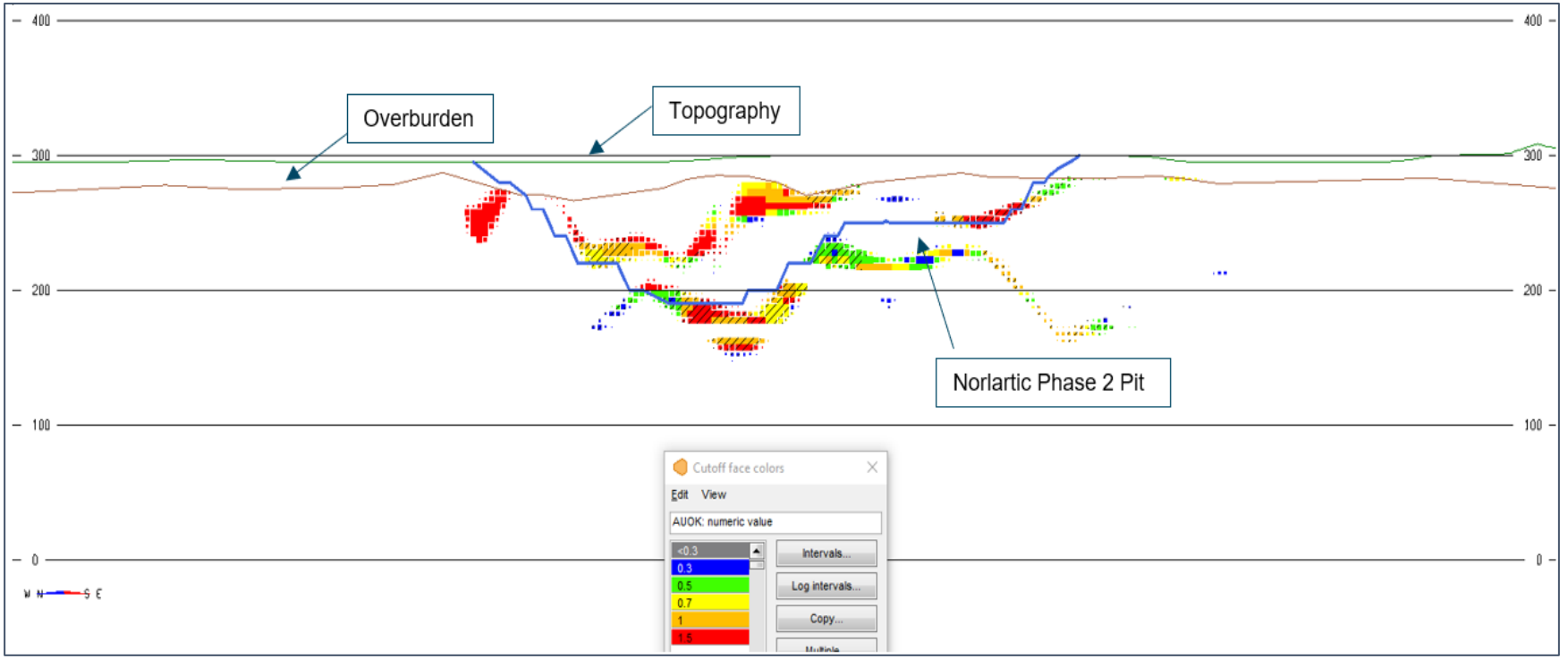
Source: Moose Mountain, 2020

Figure 16-10: KN Pit Designs, West-East Section, KN1 (section location in Figure 16-8)



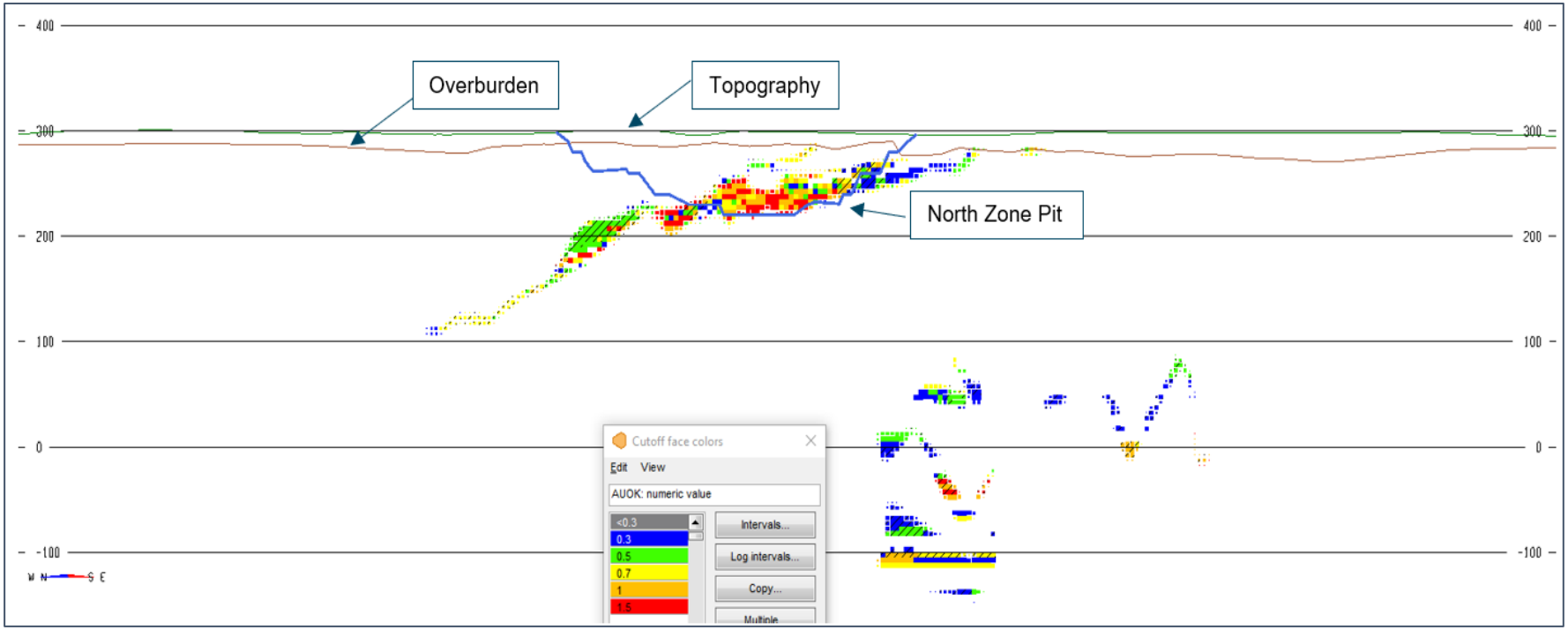
Source: Moose Mountain, 2020

Figure 16-11: KN Pit Designs, West-East Section, KN2 (section location in Figure 16-8)



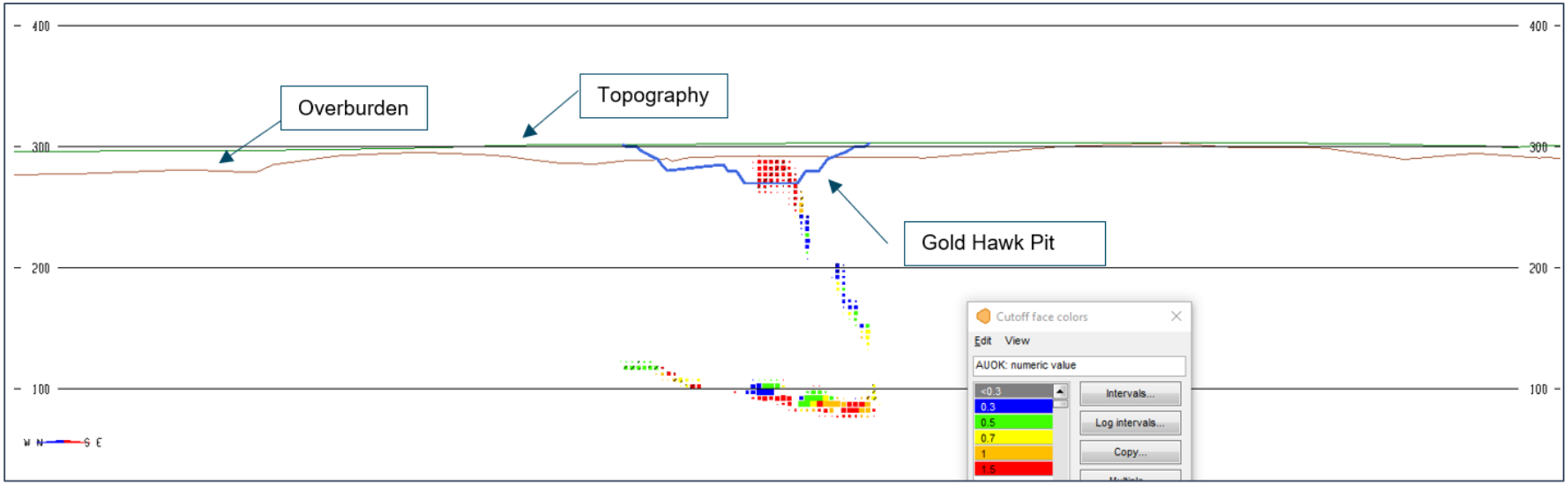
Source: Moose Mountain, 2020

Figure 16-12: KN Pit Designs, West-East Section, KN3 (section location in Figure 16-8)



Source: Moose Mountain, 2020

Figure 16-13: KN Pit Designs, West-East Section, KN4 (section location in Figure 16-8)



Source: Moose Mountain, 2020

16.5 Ex-Pit Haul Roads

Mine haul roads external to the open pits are planned to haul resource and waste materials from the open pits to the scheduled destinations. Centerline routes for the ex-pit mine haul roads have been laid out with the following conceptual features:

- 29 m wide ex-pit haul roads that incorporate a dual-lane running width and berms on both edges of the haul road
- sized to handle 180 tonne payload rigid-frame haul trucks
- 8% maximum grade

The ex-pit haul road centerlines are shown in the project layout drawing Figure 16-20.

16.6 Low-Grade Storage Facilities

When resources are mined from the pit, it will either be delivered to the crusher, the ROM stockpile located next to the crusher, or the low-grade stockpiles.

The crusher and ROM stockpiles are located 0.5 km south of the Marban pit limits and 2.0 km southeast of the Norlartic pit limits.

Cut-off grade optimisation on the mine production schedule sends resource between 0.35 g/t and 0.70 g/t Au to a stockpile in certain planned periods. This stockpile is located just south of the Marban pit limits and just north of the crusher and ROM stockpile.

The stockpiled resources are planned to be re-handled back to the crusher during the mine life. The higher grade, between 0.50 g/t and 0.70 g/t Au, will be re-handled during pit operations and the lower grade, between 0.35 g/t to 0.50 g/t Au, once the pits are exhausted.

The low-grade stockpiles are shown in the project layout drawings in Figure 16-20.

16.7 Waste Rock Storage Facilities

Waste rock and overburden/topsoil storage facilities are planned for waste materials from the open pit. In general, design considerations assumed:

- bottom-up construction
- 5:1 overall slopes
- Maximum 80 m height on the waste rock stockpiles
- Maximum 60 m height on the overburden stockpiles

It is assumed that the waste rock from both deposits is net acid neutralising and there has been no consideration for segregation of different rock types in the planned stockpiles.

All stockpiles are planned to avoid existing waterbodies and water courses and minimize disturbance of wetlands.

Waste rock from all pits will be stored in a pile directly west of the Marban pit limits and 0.5 km south of the Norlartic pit limits. The waste rock stockpile will be built up to a crest elevation of 385 m. The waste rock stockpile will backfill the Gold Hawk pit after it is mined out.

Overburden from the Marban and KN deposits will be stored in a stockpile directly north of the Marban pit limits and 0.5 km southeast of the Norlartic pit limits. Overburden from the KN deposits will also be stored in a secondary pile direct west of the Norlartic pit limits. The overburden stockpiles will be built to a crest elevation of 355 m and 310 m respectively.

Topsoil stockpiles have not yet been sited.

Waste rock and overburden from the KN deposit will also be used to support construction of the tailings storage facility 3.0 km northwest of the Norlartic pit limits. Details of this facility are described in Section 18.

The waste rock and overburden storage facilities are shown in the project layout drawings in Figure 16-20.

16.8 Production Schedule

Production requirements by scheduled period, mine operating considerations, product prices, recoveries, destination capacities, equipment performance, haul cycle times and operating costs are used to determine the optimal production schedule from the phased pit contents.

The overall production schedule is included as Table 16-9.

The open pit mine production schedule for all of the deposits is included as Figure 16-14 and shows the production tonnage and grade forecast; Figure 16-15 provides an illustration of the projected material mined and strip ratio. This is illustrated for each individual deposit in Figure 16-16 to Figure 16-19.

The production schedule is based on the following parameters:

- The Mineral Resource and associated waste material quantities are split by phase and bench.
- The operations are scheduled on annual periods.
- An annual mill feed rate of 4,000 kt/a (11 kt/d)is targeted.
- Target throughput rates start at 85% (3,400 kt) in Year 1 considering mill ramp-up.
- Within a given phase, each bench is fully mined before progressing to the next bench.
- Pit phases are mined in sequence, where the second pit phases do not mine below the first pit phases.
- Pit phase vertical progression is limited to no more than 45 m in each year; average annual phase progression is 30 m.
- Pre-production (Y-1) mining targets construction materials for haul roads and starter tailings dam.
- Resource tonnes released in excess of the mill capacity will be stockpiled.

- Low-grade resource is stockpiled and re-handled to the primary crushers at the end of mine life.

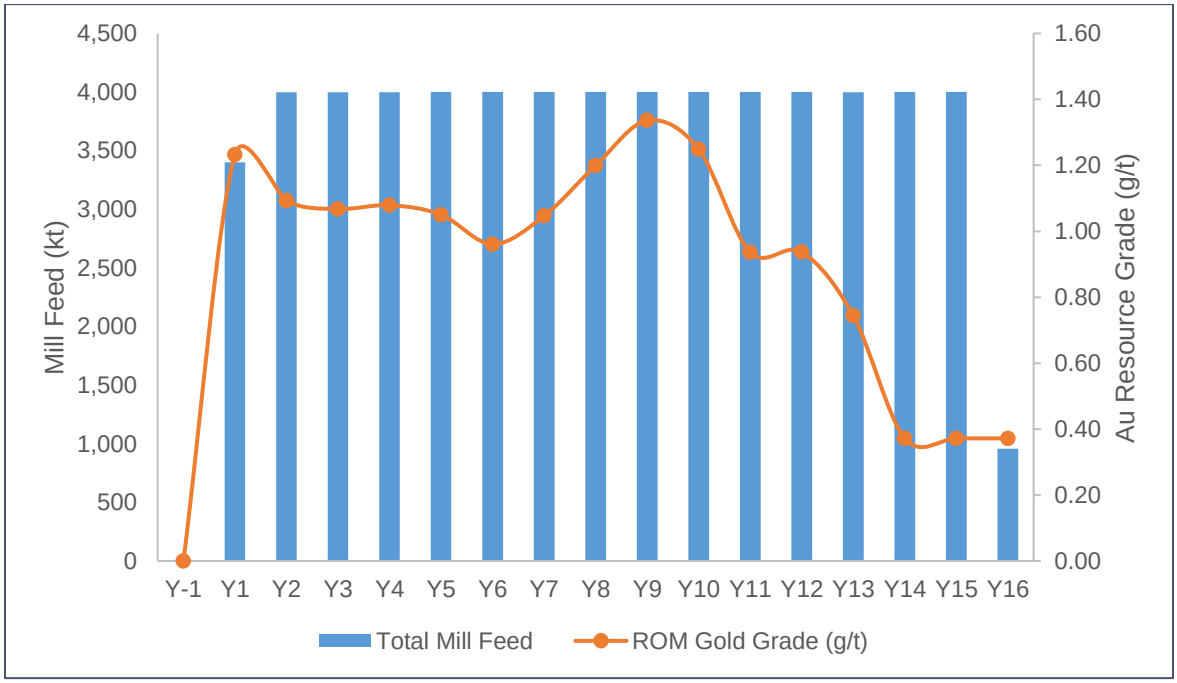
16.8.1 Mining Sequence

The pit operations will run for fourteen years, including one year of pre-production. The KN deposit pits will be exhausted after six years of mill operations. Following pit operations, stockpile re-handling operations will continue for three years. The general mine sequence through the various pit phases is illustrated in Table 16-10. The final layout plan is illustrated in Figure 16-20.

Table 16-9: Mine Production Schedule

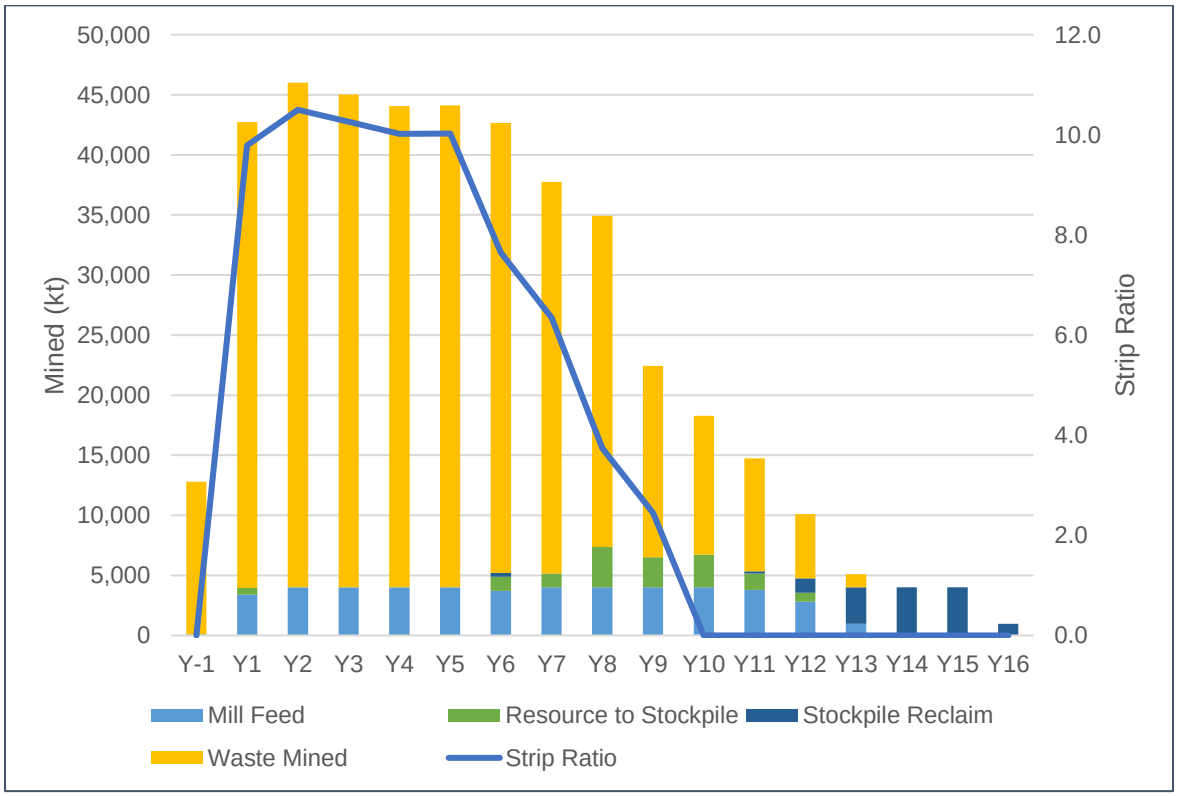
Total Mine Production	Year	LOM	Y -1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16
Mill Feed	kt	60,356	—	3,400	4,000	4,000	4,000	4,000	4,000	4,000	4,000	4,000	4,000	4,000	4,000	4,000	4,000	4,000	957
Au	g/t	0.97	—	1.23	1.09	1.07	1.08	1.05	0.96	1.05	1.20	1.34	1.25	0.94	0.94	0.74	0.37	0.37	0.37
Mill feed gold	koz.	1,875	—	135	140	137	139	135	124	135	154	172	161	120	121	96	48	48	11
Resource mined	kt	60,356	84	3,959	4,000	4,000	4,000	4,000	4,896	5,140	7,368	6,508	6,723	5,148	3,558	971	—	—	—
Au	g/t	0.97	0.98	1.11	1.09	1.07	1.08	1.05	0.84	0.89	0.85	0.99	0.92	0.81	0.97	1.51	—	—	—
Stockpile retrieval to mill	kt	13,686	—	—	—	—	—	—	300	—	—	—	—	200	1,200	3,029	4,000	4,000	957
Au	g/t	0.42	—	—	—	—	—	—	0.55	—	—	—	—	0.50	0.50	0.50	0.37	0.37	0.37
Waste mined	kt	355,627	12,711	38,768	42,000	41,037	40,062	40,108	37,451	32,622	27,559	15,917	11,567	9,387	5,336	1,102	—	—	—
Total Mined from Pits	kt	415,983	12,795	42,728	46,000	45,036	44,062	44,108	42,348	37,762	34,927	22,425	18,290	14,534	8,894	2,073	—	—	—
Total Moved	kt	429,669	12,795	42,728	46,000	45,036	44,062	44,108	42,648	37,762	34,927	22,425	18,290	14,734	10,094	5,102	4,000	4,000	957
Marban																			
Mill Feed	kt	50,575	—	414	1,580	2,878	2,225	3,139	3,873	4,000	4,000	4,000	4,000	4,000	4,000	4,000	4,000	4,000	467
Au	g/t	0.92	—	0.86	0.94	1.09	1.09	0.85	0.93	1.05	1.20	1.34	1.25	0.94	1.20	1.34	1.25	0.94	0.37
Resource mined	kt	50,575	0	498	1,580	2,878	2,225	3,139	4,839	5,140	7,368	6,508	6,723	5,148	3,558	971	—	—	—
Au	g/t	0.92	0.00	0.77	0.94	1.09	1.09	0.85	0.81	0.89	0.85	0.99	0.92	0.81	0.97	1.51	—	—	—
Stockpile retrieval to mill	kt	13,125	—	—	—	—	—	—	229	—	—	—	—	200	—	—	—	200	467
Au	g/t	0.41	—	—	—	—	—	—	0.35	—	—	—	—	0.50	—	—	—	0.50	0.37
Waste mined	kt	285,593	4,589	14,712	23,783	30,505	33,188	37,908	37,418	32,622	27,559	15,917	11,567	9,387	5,336	1,102	—	—	—
KN																			
Mill Feed	Kt	9,782	—	2,986	2,420	1,122	1,775	861	128	—	—	—	—	—	—	—	—	—	490
Au	g/t	1.20	—	1.28	1.19	1.01	1.07	1.78	2.04	—	—	—	—	—	—	—	—	—	0.37
Resource mined	kt	9,782	84	3,462	2,420	1,122	1,775	861	57	—	—	—	—	—	—	—	—	—	—
Au	g/t	1.20	0.98	1.16	1.19	1.01	1.07	1.78	3.02	—	—	—	—	—	—	—	—	—	—
Stockpile retrieval to mill	kt	561	—	—	—	—	—	—	71	—	—	—	—	—	—	—	—	—	490
Au	g/t	0.48	—	—	—	—	—	—	1.11	—	—	—	—	—	—	—	—	—	0.37
Waste mined	kt	70,034	8,122	24,056	18,216	10,531	6,874	2,200	33	—	—	—	—	—	—	—	—	—	—

Figure 16-14: Mine Production Schedule, Mill Feed Tonnes & Grade (All Deposits)



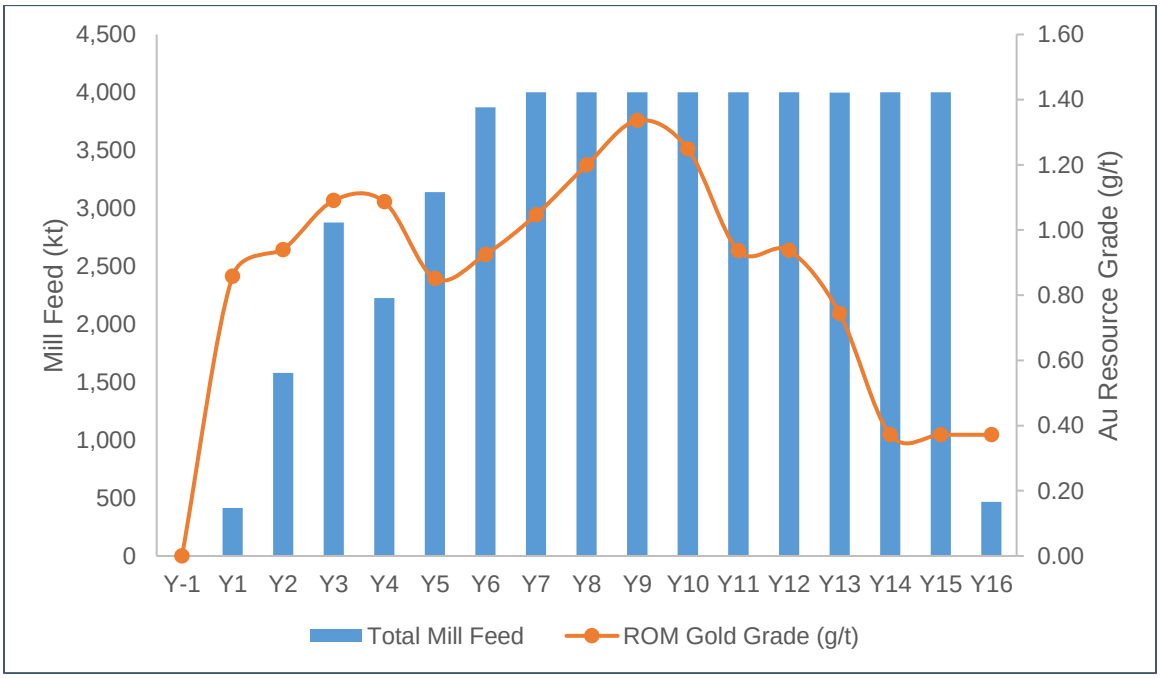
Source: Moose Mountain, 2020

Figure 16-15: Mine Production Schedule, Material Mined & Strip Ratio (All Deposits)



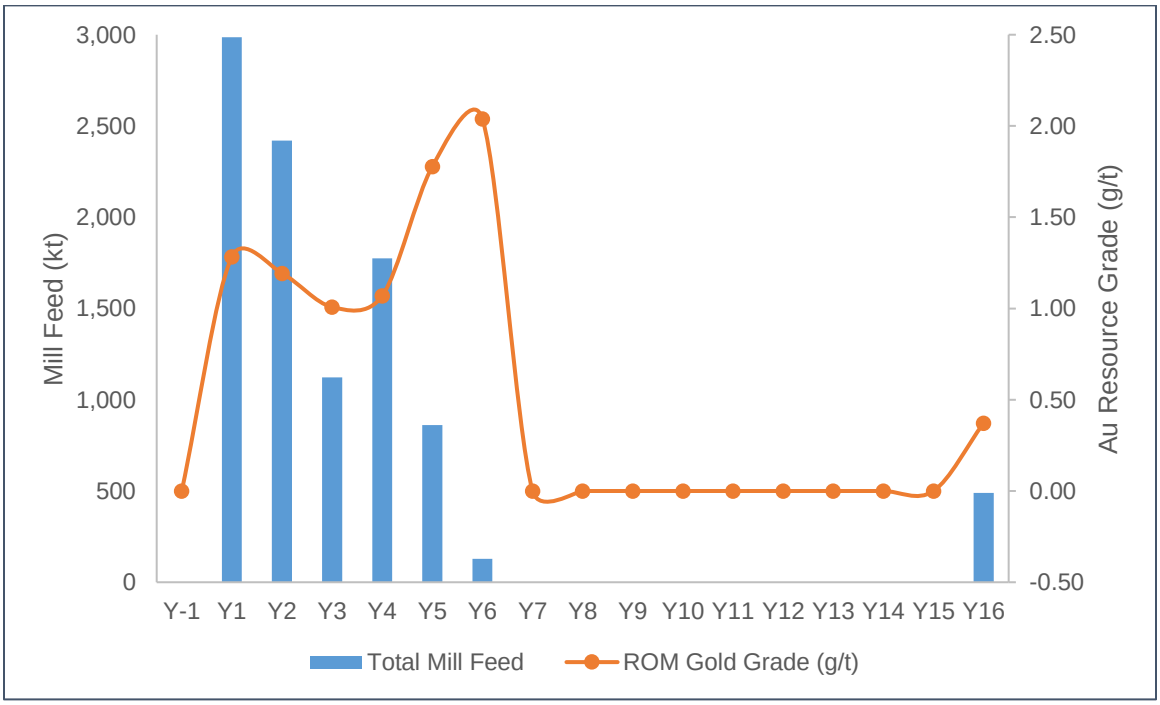
Source: Moose Mountain, 2020

Figure 16-16: Marban Mine Production Schedule, Mill Feed Tonnes & Grade



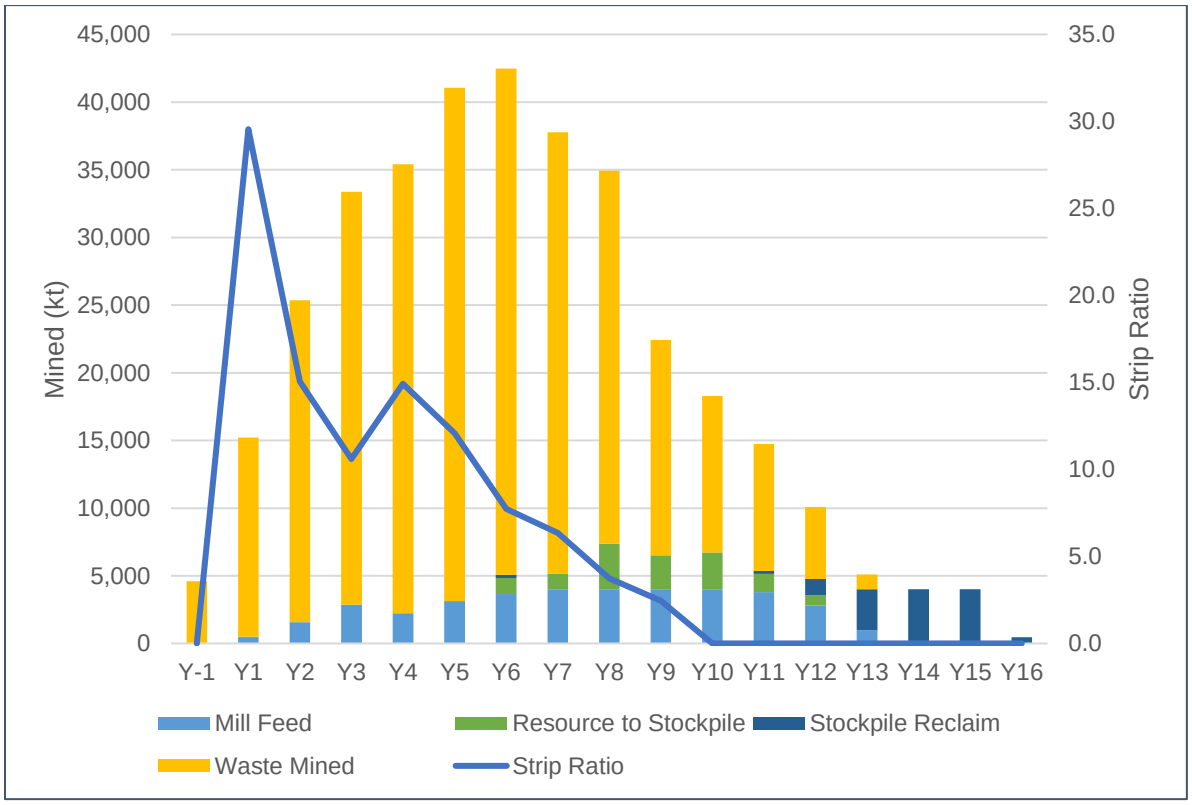
Source: Moose Mountain, 2020

Figure 16-17: KN Mine Production Schedule, Mill Feed Tonnes & Grade



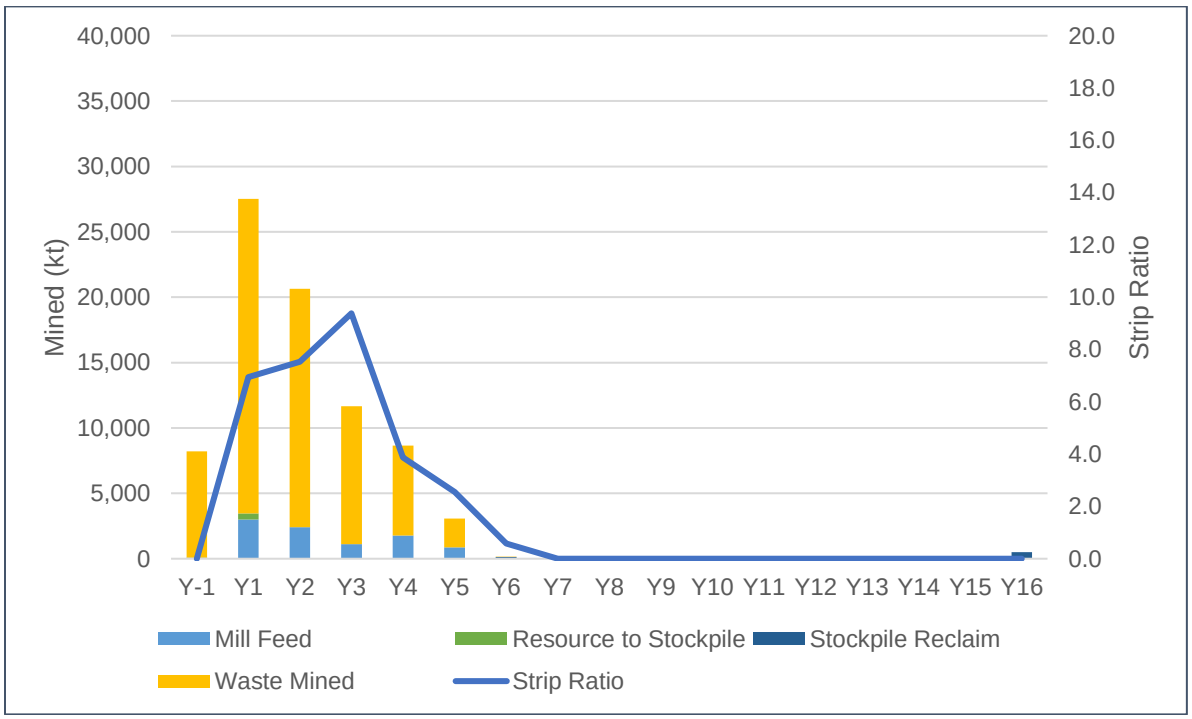
Source: Moose Mountain, 2020

Figure 16-18: Marban Mine Production Schedule, Material Mined & Strip Ratio



Source: Moose Mountain, 2020

Figure 16-19: KN Mine Production Schedule, Material Mined & Strip Ratio

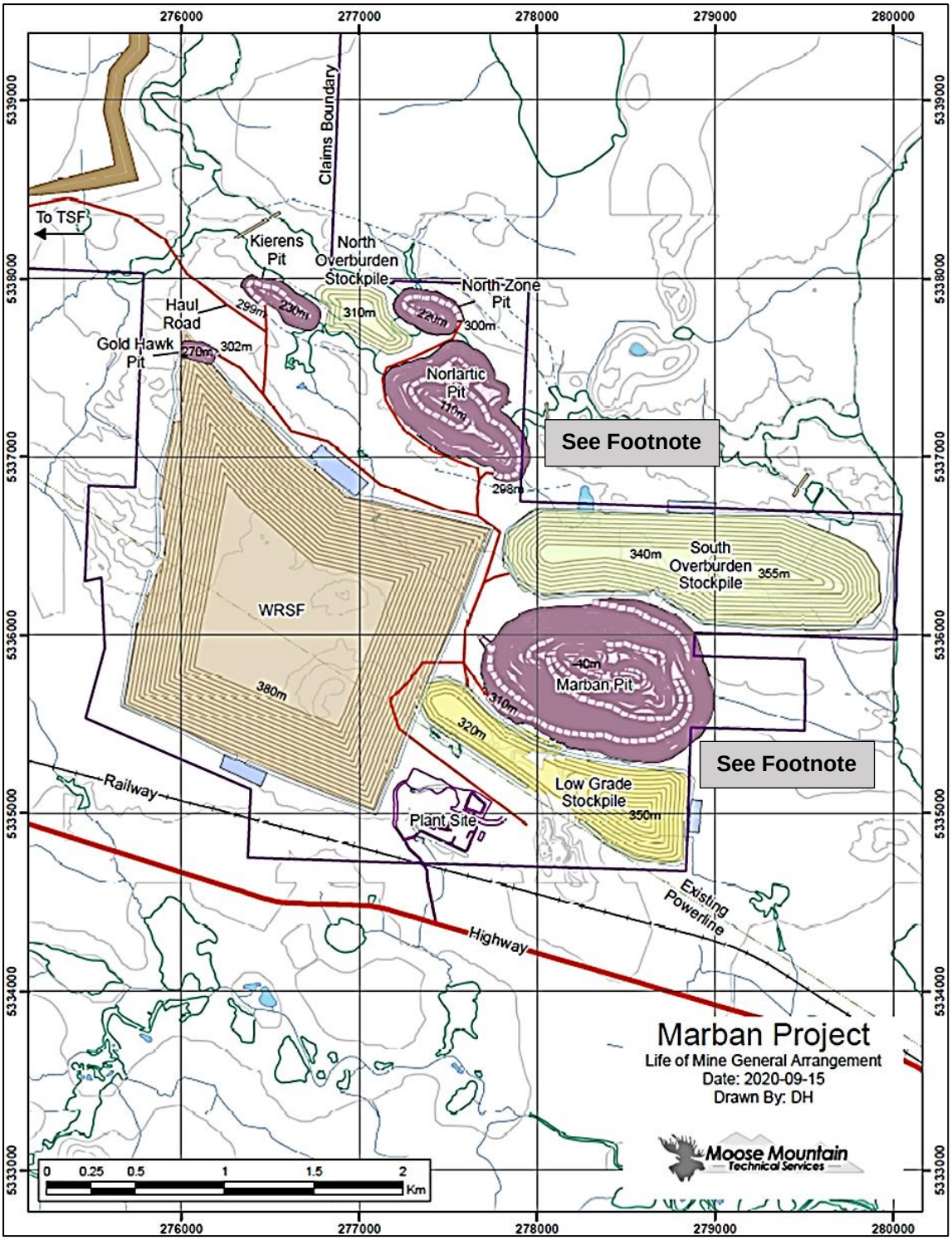


Source: Moose Mountain, 2020

Table 16-10: Pit Phase Sequence

Deposit	Phases Mined	Y-1	Y01	Y02	Y03	Y04	Y05	Y06	Y07	Y08	Y09	Y10	Y11	Y12	Y13
Marban	Marban Phase 1														
	Marban Phase 2														
	Marban Phase 3														
	Marban Phase 4														
Kierens Norlartic	Norlartic Phase 1														
	Norlartic Phase 2														
	Kierens Phase 1														
	Kierens Phase 2														
	North Zone														
	Gold Hawk														

Figure 16-20: Marban Project Mine Operations PEA Layout



Note: To be redesigned in future studies to not cross the claim boundary – modifications to mill feed and waste tonnages will be insignificant.

16.9 Operations

Owner operating and managed open pit mine operations are planned to be typical of similar operations in the region.

Grade control drilling is carried out to better delineate the resource in “near term” benches to be mined. A grade control system is planned to provide field control for the loading equipment to selectively mine resource-grade material separately from the waste.

In-situ rock is drilled and blasted on 5 m benches to create suitable fragmentation for efficient loading and hauling of both resource and waste rock. It is assumed that overburden material does not require blasting. Powder factors of 0.30 kg/t in resource and waste rock are proposed for all deposits. The blasting activities are planned to fall under a contract service agreement with the explosive supplier. Bulk blasting materials will be stored off site and delivered as need. An onsite magazine is planned for initiation systems.

Loading in resource zones will be completed with hydraulic excavators on 5 m benches, and in waste zones with hydraulic excavators and wheel loaders on 5 m or 10 m benches, depending on grade control requirements.

Each deposit contains historic underground voids that should be mined out as much as possible, and backfilled when necessary for pit floor stability. The mine plan should avoid leaving any underground void openings behind interim or final pit walls.

Resource and waste rock will be hauled out of the pit and to scheduled destinations with off-highway rigid-frame haul trucks.

Mine pit services include:

- haul road maintenance
- pit floor and ramp maintenance
- waste dump maintenance
- ditching
- dewatering
- secondary blasting and rock breaking
- snow removal
- reclamation and environmental control
- lighting
- transporting personnel and operating supplies

Direct mining operations and mine fleet maintenance are planned as an Owner’s fleet; equipment ownership and labour are undercharged to mine operations.

Mining operations are based on 365 operating days per year with two 12-hour shifts per day. An allowance of 10 days of no production has been built into the mine schedule to allow for adverse weather conditions.

The number of hourly mine operations personnel, including maintenance crews, peaks at 285 persons. Due to the shift rotation, only one-quarter of full personnel complement will be on shift at a given time. Salaried personnel of approximately 35 persons will be required for mine operations, including the mine and maintenance supervision, mine engineering and geology.

16.10 Mining Equipment

The following mine equipment descriptions are based on typical fleet contingents utilized in the North American open pit mine operations. It should be expected that equipment specifications and fleet sizes will be altered with further project engineering and optimization.

Grade control drilling will be carried out with diesel hydraulic RC drills. Production drilling will be carried out with 165 mm (6.5″) diesel driven down-the-hole (DTH) drills.

Reliable mining equipment commonly found in the construction and open pit mining industry has been selected for the loading and hauling fleet. Hydraulic excavators (12.0-m^3 bucket) are proposed based on their ability to minimise losses and dilution for the grade control operations. Front-end wheel loaders (13.0-m^3 bucket) are proposed based on their ability to load the haulers in three to four passes, and their ability to load the crusher when required. Rigid-frame haulers (91 tonne payload) are proposed to be flexible enough to use on the smaller pit benches and in selective mining scenarios but are not so small that the fleet size is excessive. Two articulated haulers (36 tonne payload) are proposed to supplement the fleet and provide additional flexibility for construction of the pits, haul roads, and tailings dam.

Graders will be used to maintain the haul routes for the haul trucks and other equipment within the pits and on all routes to the various waste storage locations and the crusher. On-highway trucks that are outfitted with a water tank and gravel spreader are included for haul road maintenance. Track dozers (447 kW and 325 kW) are included to handle waste rock, overburden, and topsoil to the various waste storage locations. Front-end wheel loaders (4.5-m^3 bucket) and hydraulic excavators (3.8-m^3 and 3.0-m^3 bucket) are included as pit support, loading tools for the articulated haulers, and back-up loaders for the main fleet. Custom fuel/lube trucks are included for mobile fuel/lube support. Various small mobile equipment pieces are proposed to handle all other pit service and mobile equipment maintenance functions.

Pits will be dewatered with conventional dewatering equipment (submersible pumps place in pit bottoms or in underground workings). Preliminary pit inflows of 4,000 m^3 per day are estimated for when the pits will be at their deepest. This is based on the assumptions that there are no major pit inflows from the esker running below the Kierens stream and that the rock mass is not hydraulically conductive. Pit water will be pumped to collection ponds adjacent to the pits, where it will be managed as per the overall site water management plan in section 18.

Mine fleet maintenance activities are generally performed in the maintenance facilities located near the plant site.

Primary mining equipment requirements are summarised in Table 16-11. The equipment classes, as well as number of units, are preliminary scoping level estimates, and modifications in future studies should be anticipated.

Table 16-11: Primary Mining Fleet Schedule

	Y -1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14-16
Drilling															
Diesel DTH tracked drill 165 mm (6.5”) holes	2	4	5	6	6	6	6	6	6	4	3	3	2	2	0
Loading															
Wheel loader 13.0 m^3 bucket	1	2	2	2	2	2	2	2	2	2	1	1	1	1	1
Hydraulic excavator 12.0 m^3 bucket	1	4	4	4	4	4	4	3	3	3	3	3	2	1	0
Hauling															
Rigid frame haul truck 91 t payload	6	26	26	26	26	26	26	26	26	18	18	15	12	5	5

17 Recovery Methods

17.1 Overall Process Design

The unit operations were selected for the respective processing routes based on the outcome of the testwork and preliminary financial evaluations. The unit options for the options are also are very common to the industry.

The project focused on capital cost-effective mill design, with a target grind of 80% passing (P_{80}) 100 µm based on the testwork reviewed in Section 13.

Key process design criteria listed in Table 17.1 were derived from the comminution testwork.

Table 17-1: Key Process Design Criteria

Design Parameter	Units	Value
Throughput	Mt/a	4.0
	t/d	10,960
Material Specific Gravity	t/m³	2.78
Primary Grind size (P80)	µm	100
Moisture Content	%	3.0
Crushing Area Availability	%	67
Crushing Operating Throughput	t/h	685
Process Plant Availability	%	92
Process Plant Operating Throughput	t/h	496
Bond Crusher Work Index (CWi)	kWh/t	16.5
Bond Ball Mill Work Index (BWi)	kWh/t	11.0
Gravity Circuit Feed Source		Ball mill discharge
Gravity Circuit Feed Rate	%cyclone recirc	25
L-CIL Residence Time	h	28
L-CIL Extraction	Au (%)	92.5
L-CIL Operating Density	% w/w	42.5
L-CIL DO Target	mg/L	20
L-CIL pH Target		10.5 – 11.0
L-CIL Carbon Concentration	g/L	10
L-CIL Sodium Cyanide Addition	kg/t	0.3
L-CIL Hydrated Lime Addition	Ca(OH)2 kg /t	1.0
Pre-aeration, Leach & CIL Tanks	#	1 + 1 + 6
Overall Recovery	Au (%)	93.7
Design Parameter	Units	Value

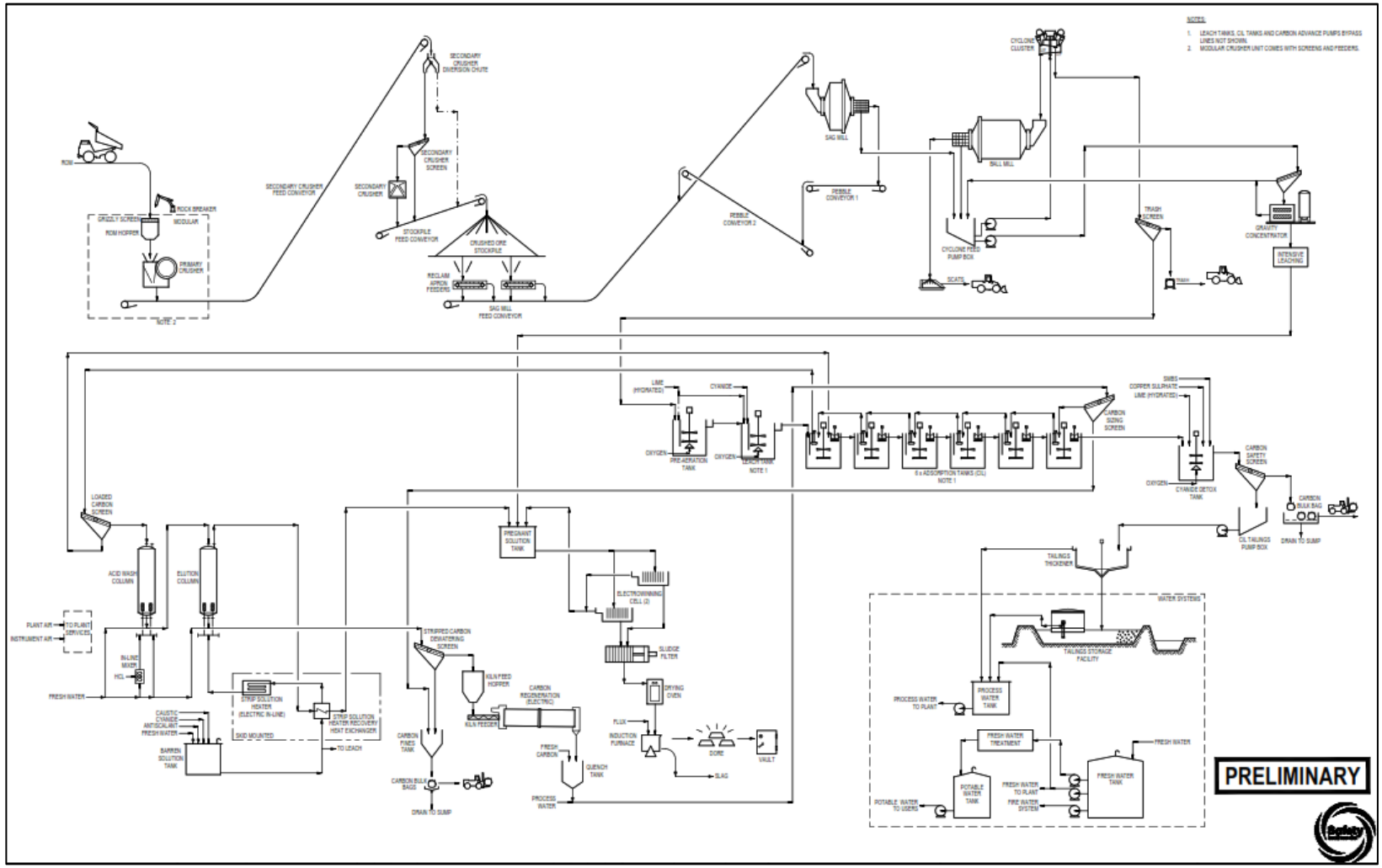
17.2 Process Plant Description

The process design is comprised of the following:

- primary and secondary crushing of run-of-mine (ROM) material

- a covered crushed material stockpile to provide buffer capacity for the process plant
- SAG mill with trommel screen followed by a ball mill with cyclone classification
- gravity recovery of ball mill discharge by one semi-batch centrifugal gravity concentrator, followed by intensive cyanidation of the gravity concentrate and electrowinning of the pregnant leach solution.
- trash screening
- Pre-aeration, leach and carbon in leach (CIL) adsorption
- acid washing of loaded carbon and Anglo-American Research Laboratory (AARL) type elution followed by electrowinning and smelting to produce doré
- carbon regeneration
- cyanide destruction of tailings using SO2 air process
- carbon screening, tailings thickening and disposal

Figure 17-1: Process Flowsheet



Source: Ausenco, 2020

17.2.1 Primary Crushing & Stockpiling

Material is hauled from the mine and direct tipped into to the crusher feed hopper. Material from the feed hopper is crushed by a primary jaw crusher mounted on a modular crushing mobile unit, which also includes a feeder and vibrating grizzly screen.

A fixed rock breaker is utilised to break oversize rocks at the feed to the jaw crusher. The crushed material is conveyed to a covered stockpile that provides approximately 16 hours of live storage. The residence time of the stockpile permits opportunity to perform maintenance on the crushing circuit without affecting the process plant.

Crushed material is withdrawn from the stockpile by two apron feeders that discharge onto the SAG mill feed conveyor. Pebbles from the SAG mill are fed to a recycle circuit via conveyors and discharged on the SAG mill feed conveyor to recycle to the SAG mill.

17.2.2 Grinding Circuit

The grinding circuit consists of a SAG mill followed by a ball mill in closed circuit with hydrocyclones. The SAG mill slurry discharges through a trommel where the pebbles are screened and recycled to the SAG mill via conveyors. Trommel undersize discharges into the cyclone feed pumpbox.

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. Trommel undersize discharges into the cyclone feed pumpbox. The gravity tailings discharge to the cyclone pumpbox.

Water is added to the cyclone feed pumpbox to obtain the required cyclone feed density. This hopper also has a dedicated pump to feed the gravity circuit scalping screen. Cyclone overflow gravitates over the trash screen. Trash screen oversize is collected in a bunker and periodically removed. Trash screen undersize reports to the leach/CIL circuit.

17.2.3 Gravity Circuit

The gravity circuit includes centrifugal concentration with a feed scalping screen. Feed to the circuit is pumped from the cyclone feed pumpbox via a dedicated pump to the scalping screen. Gravity scalping screen oversize at +2 mm reports to the gravity tails pumpbox, from where the gravity tails pump directs the material back to feed 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. The tails from the gravity concentrator also reports to the gravity tails pumpbox.

17.2.4 Intensive Leach Reactor

Concentrate from the gravity circuit reports to the intensive cyanidation reactor (ICR). The concentrate from the centrifugal gravity concentrator is directed to the ICR gravity concentrate storage cone and de-slimed before transfer to the ICR.

ICR leach solution (mixture of NaCN, NaOH and LeachAid® - an oxidant) is made up within the heated ICR 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 recovered to the reaction vessel feed tank, and then the gravity leach tailings are returned to the grinding circuit.

The ICR pregnant leach solution is pumped from the reaction vessel feed tank to the ICR dual purpose pregnant solution tank. The pregnant solution tank also receives elution pregnant solution.

Combined ICR and elution pregnant solution is treated in the gold room for gold recovery as gold sludge using two electrowinning cells. The sludge is then collected and then smelted into doré.

17.2.5 Leach/CIL Circuit

The leach/CIL circuit consists of one pre-aeration tank, a single leach tank and six CIL tanks, each with four hours retention time, providing total time of 36 hours at 42.5% w/w density.

Oxygen is sparged to the pre-aeration and leach tank to maintain adequate dissolved oxygen levels for leaching of 20 mg/L. Hydrated lime is added to further refine the operating pH to the desired set point of 10.5 to 11.0. Cyanide solution is added to the leach tank, and the first two CIL tanks. Fresh/regenerated carbon from the carbon regeneration circuit is returned to the last tank of the CIL circuit and is advanced counter-currently to the slurry flow by pumping slurry and carbon. Slurry from the last CIL tank gravitates to the cyanide detoxification tanks.

Each CIL tank has a mechanically swept carbon retention screen to retain the carbon while allowing the slurry to flow by gravity to the downstream tank. Loaded carbon is transferred from the first CIL tank to carbon elution. Recessed impeller pumps are used to transfer slurry between the CIL tanks and to the loaded carbon screen.

17.2.6 Cyanide Detox

The CIL tailings gravitate to cyanide detox. Cyanide detox includes two tanks operating in parallel with total residence time of approximately 60 minutes. Cyanide detox reduces weak acid dissociable cyanide (CN_{WAD}) concentration from approximately 150 mg/L to less than 1.0 mg/L to comply with environmental requirements prior to deposition in the TSF.

Cyanide destruction is based on the SO_2 air method. Oxygen, lime, copper sulphate, and sodium metabisulphite (SMBS) are dosed to both tanks. The cyanide detox tanks agitators provide a high degree of mixing to ensure the reaction proceeds to completion.

Detox tailings report to the carbon safety screen. Screen undersize feeds the tailings thickener, while screen oversize (recovered carbon) is collected in a fine carbon.

17.2.7 Tailings Thickening

Carbon safety screen undersize flows into the thickener feed well where it is mixed with flocculant to promote settling. Thickener underflow is pumped to the tailings management facility (TSF). Decant water from the TSF is recovered and pumped to the process water tank. Excess water in the TSF is transferred to an effluent treatment plant prior to discharging to the environment.

Thickener overflow flows to the process water tank along with reclaim water and fresh water (as required) for use throughout the process.

17.2.8 Carbon Acid Wash, Elution & Regeneration Circuit

17.2.8.1 Carbon Acid Wash

Slurry from the first CIL tank is pumped to the loaded carbon screen. Screen undersize gravitates back to the first CIL tank while screen oversize discharges to the acid wash vessel.

Loaded carbon is washed with a weak hydrochloric acid solution to remove calcium, magnesium, and other salt deposits. The carbon is soaked in the hydrochloric acid solution for a predetermined time. Post soaking stage the spent acid is transferred to the cyanide destruction tank along with waster washing of the carbon to remove any retained acid.

The acid-washed carbon is then hydraulically transferred to the elution column for gold stripping.

17.2.8.2 Carbon Elution

Carbon elution circuit is based on the AARL process.

The acid washed carbon is first soaked with a strong sodium cyanide solution and sodium hydroxide solution in the elution column. The caustic/cyanide solution is then re-circulated through the elution column via a heater, until a temperature of 95°C is achieved. At the end of the pre-soak cycle, the eluate is transferred to electrowinning.

The balance of the elution cycle is passing water heated to 120°C, through the elution column single pass to the electrowinning.

17.2.8.3 Carbon Reactivation

The eluted or stripped carbon now is dewatered by a screen over a five-tonne feed hopper, which feeds an electric rotary kiln via a screw feeder. The kiln is operated at 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 CIL tanks to remove undersized carbon fragments.

Make up 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.

17.2.9 Electrowinning & Gold Room

Gold is recovered from the combined ICR and elution pregnant solution by electrowinning and smelted to produce doré bars. The pregnant solution is pumped through one of two electrowinning cells fitted with stainless steel mesh cathodes. The gold is deposited onto the cathodes, with the barren solution transferred to the leach circuit.

The gold-rich sludge is washed off the steel cathodes in the electrowinning cells using high-pressure spray water and gravitates to the sludge hopper. The sludge is filtered, dried, mixed with fluxes, and smelted in an electric induction furnace to produce gold doré. The electrowinning and smelting process takes place within a secure and supervised gold room equipped with access control, intruder detection, and closed-circuit television equipment.

17.3 Reagent Handling & Storage

The following reagent systems are required for the typical cyanidation gold processing plants:

- grinding media
- hydrated lime
- sodium cyanide
- sodium hydroxide
- hydrochloric acid
- copper sulphate pentahydrate
- sodium metabisulphite
- activated carbon
- smelting reagents (doré production)
- oxygen
- flocculant

18 Project Infrastructure

18.1 Introduction

The Marban project is located in Abititi-Témiscamingue, between Val-d'Or and Malartic in the province of Québec in Eastern Canada. The site is accessible by a short well-maintained gravel road from Highway 117.

Infrastructure to support the Marban project will consist of site civil work, site facilities/building, a water system, and site electrical. These are indicated in Figure 18-1.

Site civil work includes designs for the following infrastructure:

Light vehicle and heavy equipment roads;

- Access road;
- Overburden stripping and stockpiling;
- Mine facility platforms and process facility platforms;
- Water management ponds;
- Tailings storage facility (TSF);
- Waste storage facilities;
- High voltage substation platform.

Site facilities will include both mine facilities and process facilities:

- The mine facilities will include the administration offices, truckshop and warehouse, tire repair shop, mine workshop, mine dry, and miscellaneous facilities;
- The process facilities will include the process plant, crusher facility, process plant workshop and assay laboratory;
- Both the mine facilities and process facilities will be serviced with potable water, fire water, compressed air, power, diesel, communication, and sanitary systems.

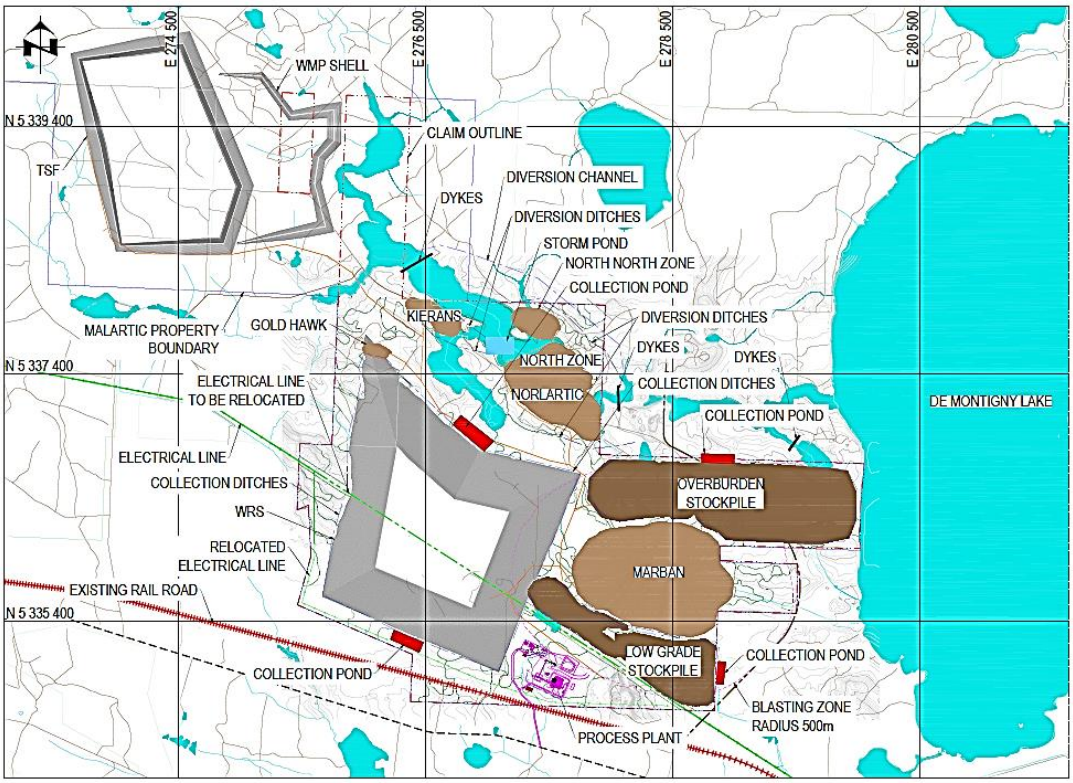
The site location is located east of De Montigny Lake. The Kierens and Norlartic (KN) open pits require a diversion of Kierens Creek water with two dams either side of the planned excavations. The tailings storage facility and water treatment pond are located to the north-west, outside of the Marban Project Property, but within the Malartic Block. Waste rock is stockpiled in a central location south of the KN pits and west of the Marban pit. The processing plant is located south of the waste rock stockpile. The arrangement of the project site is illustrated in Figure 18-1.

The selection of the site was based on the following observations and factors:

- Within the claim boundary
- Limit building on wetlands
- Locate the waste rock stockpile close to the mine pits to reduce haul distance

- Locate the process plant in an area safe from flooding
- Existing roads south and east of the site (Highway 117 and Chemin Gervais)
- Arrange administration, processing plant and offices in close proximity to reduce to limit walking distances (important during extreme cold weather)
- Proximity of the processing plant to the existing power line (Hydro-Québec)
- Maintain distance from CN railway

Figure 18-1: Overall Site Plan



Source: Ausenco, 2020

18.1.1 Site Preparation

The existing roads will be connected to the project to enable site access. Typical method of clearing and topsoil removal, excavation, drains, safety bunds and aggregates will be employed to construct additional roads.

Similarly, it is expected the forest clearing and topsoil removal will be required for the processing plant and other buildings and facilities.

Where the opportunity presents to utilise and exploit existing earthworks and concrete pads it will be examined and determined if it viable and or worth preserving.

18.2 Roads & Logistics

The accesses to process plant site and the three pits areas and tailings management facility will consist of a 0.5km gravel-based road connected to Highway 117.

The roads within the project site, connecting the process plant, crushing area, mining operation structures and all mill buildings will be around 6 m in width. These will also be connected to the process pad and be designed to include draining and safety berms/bunds where appropriate. Haul roads connecting the TSF, mine pits, and waste rock stockpile are 29m wide.

18.3 Built Infrastructure

18.3.1 Administrative

The administration offices will comprise of a single storey building and is anticipated to be prefabricated modular construction, placed on precast concrete block footings and will have HVAC. The administrative building will contain the include offices, meeting rooms, lunchroom, washrooms, men's and women's dry, lockers, first-aid and showers.

18.3.2 Crushing Area Buildings

Crushing area buildings will be of modular design.

The primary crushing building will house the ROM hopper, vibrating grizzly feeder primary jaw crusher, chutes and platework. The rock breaker will also be within the building. In addition, access platforms and reinforced concrete will be utilised for the pad to support the primary jaw crusher.

The secondary crushing building will house the secondary crusher screen and secondary crusher and all associated chutes and platework.

A dome cover and a concrete reclaim tunnel will be used for the crushed ore stockpile.

18.3.3 Process Plant Building

The process plant complex is comprised of the following separate buildings:

- grinding and gravity
- reagents
- goldroom

Most buildings will be constructed from pre-engineered metal, supported on reinforced concrete footings and are complete with concrete slabs and pedestals. Buildings will also be with insulated metal panel (IMP) roof and wall cladding. Area crane will also be present for equipment servicing.

The grinding and gravity building includes a ground floor, one elevated concrete floor. The various equipment will be accessed by purpose-built mezzanine platforms for maintenance, service and sampling. The grinding and gravity building will contain the SAG and Ball Mill, cyclone feed hopper/pumps, cyclone cluster, trash screen, and liner handler, as well as dedicated areas for the gravity circuit equipment, acid wash column, the elution column and regeneration equipment.

The reagent building will contain the reagent mixing tanks, and dosing tanks (where applicable). The reagent profile comprises of cyanide, lime, sodium hydroxide, hydrochloric acid, carbon, copper sulphate, sodium metabisulphite, flocculant and antiscalant. Where possible tote of reagents will be used directly, to conserve space and tankage.

The gold room will house pregnant solution tank, electrowinning cells, sludge filters, furnace, drying oven and vault. The building will be a two-story concrete wall structure.

External parts of the processing plant include a pre-aeration, pre-leach and six carbon in leach tanks, all of which are 1.43 m in diametre and detoxification tanks (7.4 m diametre). The tanks will be access by a purpose-built mezzanine platform and walkway to allow servicing, sampling and maintenance. Area crane will also be present will access to screens, tanks, pumps and agitators. The tailings thickener (22 m diametre) will also be located outside and the underflow from the thickener will be pumped to the tailings management facility.

18.3.4 Truck Shop / Truck Wash

The truck shop and wash building will be located next to each other. The pre-engineered buildings will be supported on a reinforced concrete raft foundation and will function to provide maintenance to the truck fleet (highway and haul).

18.3.5 Maintenance Shop & Warehouse Building

The plant maintenance shop and storage building will be located close to the process plant. Buildings will be comprised of a reinforced concrete raft foundation and fabric.

18.3.6 Laboratory

The laboratory a prefabricated, single-storey, modular building on precast concrete blocks.

18.3.7 Security Gate

The security gatehouse will comprise of a small prefabricated building, with a single boom gate. Site inductions for visitors and new employee can be conducted at this point.

18.4 Water Management

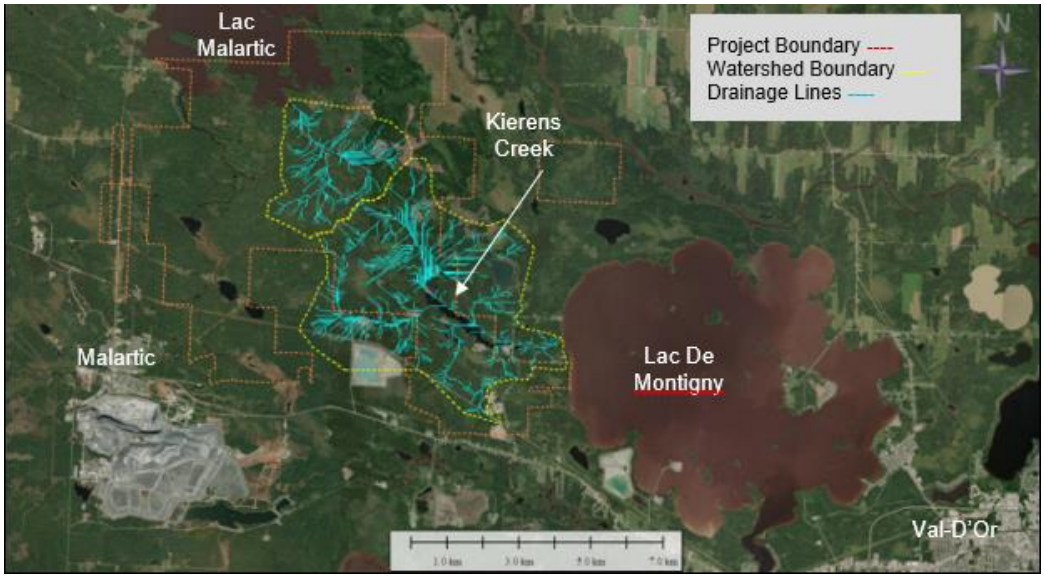
18.4.1 Overview

The site wide management is designed to collect and manage all contact water (essentially any water that comes into contact with the mine and processing plant facility). Water collection strategy will incorporate ditched, sumps and ponds and these will be administered based on the water quality and water destination (i.e. like the TSF or the water management pond).

The Marban Gold Project is located between the two lakes of Malartic and De Montigny as illustrated in (Figure 1). The project boundary and the major drainage paths within the study area were delineated through GIS analysis of publicly available National Topographic Survey of Canada (NTS) 1:50,000 scale.

The watershed boundary between Lac Malartic and Lac De Montigny is shown in Figure 18-2, with distinct drainage paths to the northwest and southeast respectively. The facilities including pits, plant site, Waste Rock Stockpile (WRS), Low Grade Stockpiles (LGS) and Overburden Stockpiles (OBS) are located within the southern watershed with Kierens Creek flowing south east through the facilities. The Kierens Creek has a watershed size of approximately 29 km^2.

Figure 18-2: Major Drainage Paths



Source: Ausenco, 2020

18.4.2 Water Management Structures

For the design of water management structures, the same design criteria were applied as those for the TSF.

In order to allow mining of the Kierens and Norlartic (KN) pits, the Kierens Creek must be diverted. A diversion channel is planned to divert Kierens Creek from its current course along an alignment to the north of the planned KN pits. The channel is designed for a probable maximum flood (PMF) event and must divert runoff from an approximately 22 km^2 of drainage in the upstream catchment area.

Diversion ditches are required to divert clean runoff away from the facilities and to minimize the amount of contact runoff to be collected and managed. The design criteria for the diversion ditches is conveyance of 1:100-year peak flow without overflow.

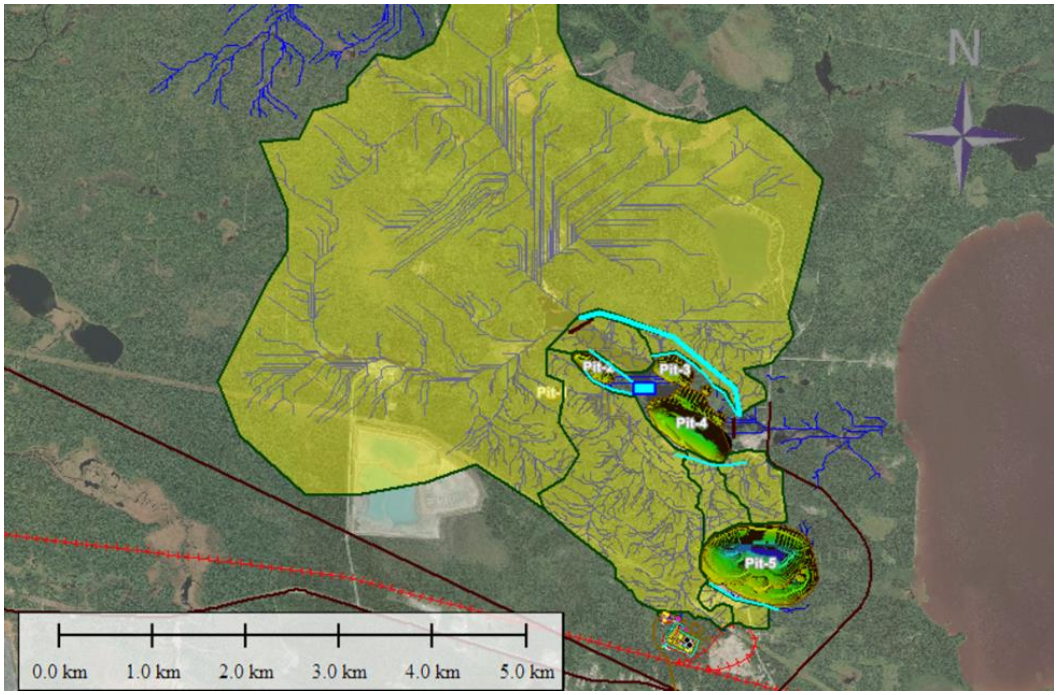
Collection ditches collect contact runoff from the plan site, WRS, OBS, LGS and from the portions of the facility catchment runoff that are not diverted by the diversion ditches. The design criteria for the collection ditches is conveyance of 1:100-year peak flow without overflow.

Storm water ponds are included to store clean runoff from diversion ditches which do not divert runoff outside of the project site, or where the clean water storage is required as reclaim water for plant use. The design criteria for the storm ponds is to store 1:100-year flood with a minimum 0.5 m freeboard.

Collection ponds are included to store contact runoff from the collection ditches. The stored contact water should be either treated and released to the environment or reused

for process purposes. The design criteria for the collection ponds is to store 1:100-year flood with a minimum 0.5 m freeboard. Water collected in catchment ponds is directed to the process plant for use in the mill or sent to the effluent treatment plant.

Figure 18-3: Water Management Structures for the KN Pits



Source: Ausenco, 2020

18.4.2.1 Catchment Delineation

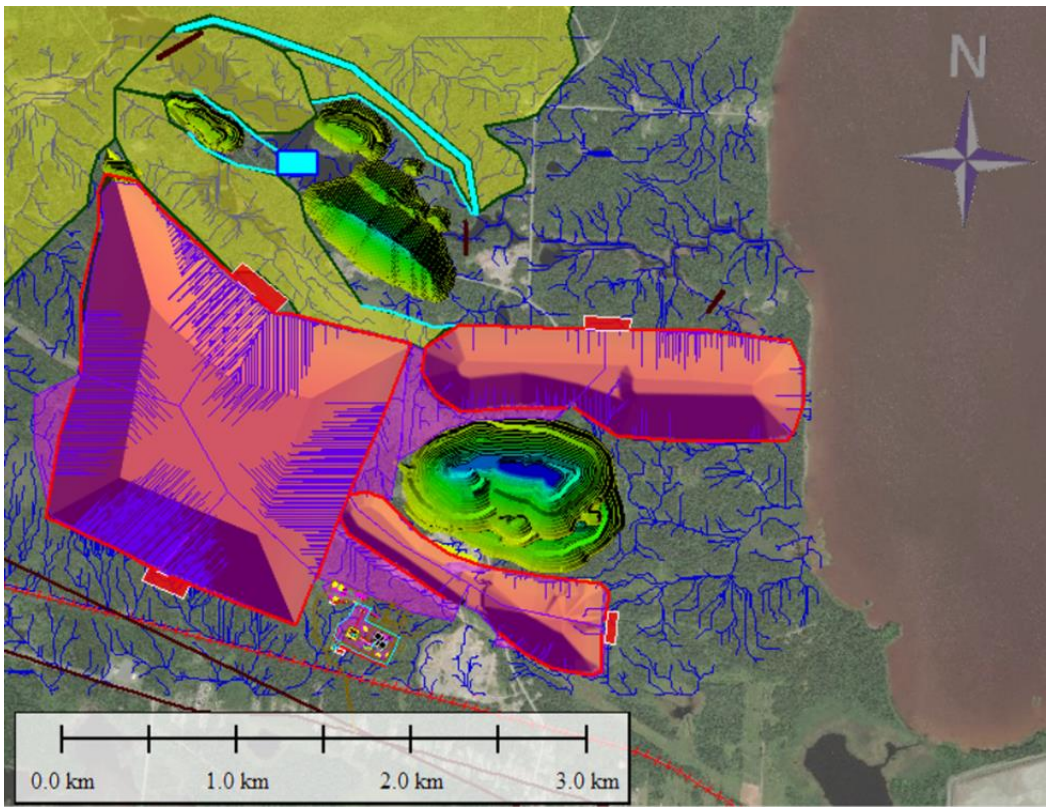
The catchments were delineated separately for each project layout. Figure 18-3 shows the water management structures for the pits layout, and Figure 18-4 shows the water management structures for the WRS, OBS and LGS layout.

The proposed water management structures together with catchment physical characteristics are summarized in Table 18-1. Slopes, elevations, and channel lengths were derived from the Digital Elevation Model (DEM) to estimate the time of concentration and lag time for each sub-catchment. The curve numbers were modified for an Antecedent Moisture Condition III (rain and low temperatures over the previous five days causing saturated conditions).

18.4.2.2 Rainfall Runoff Modelling

The hydrologic model used in the runoff analysis was HEC-HMS version 4.5.0, developed by U.S. Army Corps of Engineers. The US Soil Conservation Service (SCS) unit hydrograph method was applied to determine the runoff hydrograph from the design rainstorm. The SCS Type II distribution was selected to define the distribution of design rainfall total over 24 hours (Rivard, 1996).

Figure 18-4: Water Management Structures for the Stockpiles



Source: Ausenco, 2020

In general, the catchment area was mainly forested in a condition between fair and poor. Soil Type B, representing soil composed of shallow loess and sandy loam, was chosen for the study area. A curve number of 75 considering an antecedent moisture condition III was used to estimate the pick runoff from the WRSF, OBS and LGS. However, to estimate the flow volume, it was assumed that these facilities are lined and the entire runoff shall be collected. The results of rainfall runoff analysis for various water management structures at Marban Gold project is summarized in Table 18-2.

18.4.2.3 Water Diversion Design

Using the results of rainfall runoff model, hydraulic analyses were carried out to preliminarily size the water management structures at Marban Gold project. A high-level estimate of excavation and fill volumes was also completed using the proposed geometries of the structures. The results of rainfall runoff analysis and quantity estimates for the diversion channel, diversion and collection ditches, storm and collection ponds are summarized in Tables 18-3, 18-4 and 18-5, respectively.

18.5 Waste Rock Storage Facilities

Stockpiles are described in Section 16 of this Report.

18.6 Stockpiles

Stockpiles are described in Section 16 of this Report.

Table 18-1: Catchment Physical Characteristics

Catchment ID	Catchment Area Km2	Hydraulic Length m	Elevation max m	Elevation min m	Slope m/m	SCS Curve Number	Time of Concentration minute	Lag Time minute
Diversion Channel	22.00	4,800	330	294	0.008	78	158	95
Pit-2 North Diversion Ditch	0.38	500	302	295	0.014	80	28	17
Pit-2 South Diversion Ditch	2.90	3,100	314	296	0.006	78	129	77
Pit-3 North Diversion Ditch	0.15	1,000	305	295	0.010	80	51	30
Pit-4 South Diversion Ditch	0.62	1,200	310	295	0.013	79	53	32
Pit-5 South Diversion Ditch	0.16	300	317	305	0.040	81	15	9
Plant Site Collection Ditch	0.10	350	306	304	0.006	98	29	17
WRSF NW Collection Ditch	0.77	2,100	384	300	0.040	125	56	34
WRSF NE Collection Ditch	1.55	2,400	384	300	0.035	60	63	38
WRSF SW Collection Ditch	0.81	1,500	384	305	0.052	60	40	24
WRSF SE Collection Ditch	0.34	1,000	384	305	0.079	60	28	17
OBS W Collection Ditch	0.57	2,000	340	301	0.020	60	67	40
OBS E Collection Ditch	0.71	2,800	355	301	0.019	60	86	52
LGS N Collection Ditch	0.25	1,800	351	302	0.027	60	58	35
LGS SW Collection Ditch	0.10	800	320	307	0.016	60	38	23
LGS SE Collection Ditch	0.20	900	351	302	0.054	60	29	17

Table 18-2: 24-hour Peak Discharge & Flow Volume

	Design Event	Catchment km2	Peak Discharge m3/s	Volume 1,000 m3
Diversion Channel	PMF	22	445	6,200
Pit2 South Diversion	100year	2.9	9.4	114.9
Pit2 North Diversion	100year	0.38	3.2	16.4
Pit3 North Diversion	100year	0.15	1	6.5
Pit4 South Diversion	100year	0.69	2.8	20.6
Pit5 South Diversion	100year	0.16	2	7.2
Plant Site Collection Ditch	100year	0.1	1.5	8.5
WRSF NW Collection Ditch	100year	0.77	3.6	70.07
WRSF NE Collection Ditch	100year	1.55	6.8	141.05
WRSF SW Collection Ditch	100year	0.81	4.7	73.71
WRSF SE Collection Ditch	100year	0.34	2.3	30.94
OBS W Collection Ditch	100year	0.57	2.4	51.87
OBS E Collection Ditch	100year	0.71	2.5	64.61
LGS N Collection Ditch	100year	0.25	1.2	22.75
LGS SW Collection Ditch	100year	0.1	0.6	9.1
LGS SE Collection Ditch	100year	0.2	1.3	18.2

Table 18-3: Conceptual Design of the Diversion Channel

Design Flood	Channel Shape	Channel Depth m	Bottom Width m	Channel Grade m/m	Side Slope	Excavation Volume m3	Dyke Fill Volume m3
PMF	Trapezoidal	3	70	0.0012	2H:1V	1,000,000	113,000

Table 18-4: Conceptual Design of the Diversion & Collection Ditches

Structure	100-year Peak Discharge m3/s	Side Slope	Bottom Width m	Bottom Grade m/m	Depth m	Length m	Excavation Volume m3
Pit2 South Diversion	9.4	2H:1V	2	0.005	1.1	600	3,900
Pit2 North Diversion	3.2	2H:1V	1.5	0.003	0.8	600	2,100
Pit3 North Diversion	1	2H:1V	1	0.003	0.5	1,360	2,000
Pit4 South Diversion	2.8	2H:1V	2	0.01	0.6	785	6,000
Pit5 South Diversion	2	2H:1V	2	0.003	0.7	810	2,700
Plant Site Collection Ditch	1.5	2H:1V	1	0.003	0.6	350	647
WRSF NW Collection Ditch	3.6	2H:1V	1	0.001	1.2	2,100	11,995
WRSF NE Collection Ditch	6.8	2H:1V	1.5	0.003	1.1	2,400	13,675
WRSF SW Collection Ditch	4.7	2H:1V	1	0.003	1	1,500	6,300
WRSF SE Collection Ditch	2.3	2H:1V	1	0.003	0.7	1,000	2,352
OBS W Collection Ditch	2.4	2H:1V	1	0.003	0.8	2,000	5,824
OBS E Collection Ditch	2.5	2H:1V	1	0.003	0.85	2,800	8,996
LGS N Collection Ditch	1.2	2H:1V	1	0.003	0.5	1,800	2,520
LGS SW Collection Ditch	0.6	2H:1V	1	0.003	0.6	800	1,478
LGS SE Collection Ditch	1.3	2H:1V	1	0.003	0.4	900	907

Table 18-5: Conceptual Design of the Storm & Collection Ponds

Structure	Freeboard m	100-year Flow Volume 1000m3	Side Slope	Depth m	Top Width m	Top Length m	Excavation Volume m3
Storm Pond - Pits	0.5	151.9	2H:1V	3	210	310	234,500
Collection Pond - Plant Site	0.5	23.8	2H:1V	3	65	75	15,800
Collection Pond - WRSF N	0.5	70.4	2H:1V	3	210	460	349,800
Collection Pond - WRSF S	0.5	31.7	2H:1V	3	160	310	177,300
Collection Pond - OBS	0.5	14.1	2H:1V	3	160	330	188,900
Collection Pond - LGS	0.5	8.8	2H:1V	3	110	210	80,700

18.7 Tailings Storage Facility (TSF)

18.7.1 TSF Design Criteria

The TSF has a preliminary design to accommodate 60.4 Mt (~41.3 Mm^3) of tailings solids that will be produced over a 15 year mine life, based on an annual throughput of 4 Mtpa (11,000 tpd) for the mill. The first seven years of tailings production (approximately 19.3 Mm^3) will be stored in a surface TSF with the remaining 8 years of tailings production (approximately 22.0 Mm^3) stored in the mined out Kierens Pit #4.

The overall design objective of the TSF is to safely contain tailings solids while protecting groundwater and surface water resources during both operations and long term (post-closure). The design of the TSF has taken into account the following:

- Canadian Dam Association (CDA, 2013). Dam Safety Guidelines and Ministère du Développement Durable, de l'Environnement et des Parcs (MDDEP, 2012). Directive 019 sur l'industrie minière.
- Minimizing the impact and risks to the surrounding environment.
- Permanent and secure containment of all solid waste materials.
- Collection and management of water from the tailings during operations for recycling of process water to the maximum practical extent.
- Avoid development of mine waste infrastructure on fish habitat.
- Staged development of the TSF over the life of the project to defer capital cost and allow for efficient use of waste materials from pit stripping as construction materials.
- The TSF was designed based on the following information:
- Material Specific Gravity = 2.78
- Thickened Tailing Solids Content = 60% (by weight)
- Assumed Void Ration of Deposited Tailings = 0.9
- Calculated Density of Deposited Tailings = 1.46 t/m^3
- Calculated Maximum Volume of Tailings = 41.3 Mm^3

A downstream embankment TSF concept was used based on the availability of waste rock with the proposed mine plan. The TSF is sited northwest of the Marban Project Property, within the Malartic Property. The TSF perimeter containment embankments will be constructed primarily with mine waste rock, which will be hauled directly by mine haul trucks from the open pit mines as they are developed.

The TSF perimeter embankment will be free draining and have an upstream crushed rockfill transition layer and a locally sourced sand and gravel filter to prevent tailings solids from migrating through the embankment. Seepage and runoff from the TSF will be collected in perimeter ditches that drain to a Water Management Pond (WMP). Water will be reclaimed from the TSF Water Management Pond and/or the Kierens Pit #4 (Years 8 through 15) for reuse as process water in the mill.

Both static and pseudo-static stability analyses were performed on the TSF and WMP and both met or exceeded CDA (2013) guidelines based on a preliminary dam hazard classifications of high.

18.7.2 TSF Construction

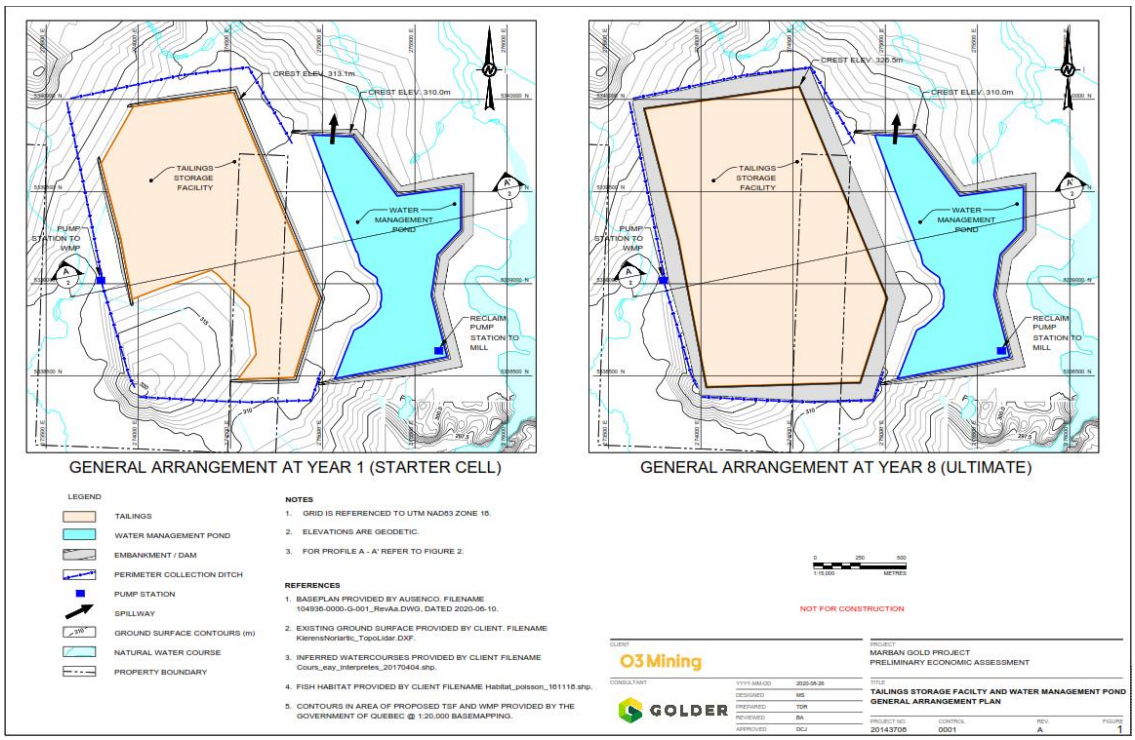
The TSF embankment will be constructed continuously over the life of mine using the downstream method. The TSF will have an initial starter cell embankment with a crest elevation of 313.1 m that will be constructed prior to commencement of mining to store tailings from the first year of operation. The TSF embankment will be raised continuously over the life of mine with the final crest elevation of the TSF at elevation 326.5 m. The layout of the TSF is shown in Figure 18.1, illustrating the Starter Cell (Year 1) and Ultimate Stage (Year 7).

Perimeter water collection ditches will be constructed along the toe of the ultimate TSF embankment that will drain to the Water Management Pond (WMP). A pump station will be required on the west side of the TSF to pump water collected by the west collection ditch to the WMP. The WMP will have an emergency spillway design to pass the PMF. The TSF perimeter ditches and WMP will be constructed prior to commencement of mining operations. Figure 18.2 illustrates a profile through the TSF and WMP.

The TSF will be constructed using mine waste rock materials and local borrow materials. The embankment design concept consists primarily of pit run rockfill with a crushed rockfill transition zone and sand and gravel filter on the upstream slope. The WMP dam will be constructed primarily of silty clay till overburden from open pit stripping with erosion protection on the slopes. A crest width of 10 m has been assumed for both the TSF embankment and WMP dam to allow for vehicle and equipment movement. The slopes for the TSF embankment will be 2H:1V upstream and 5H:1V downstream for stability. Both the upstream and downstream slopes of the WMP dam will be 5H:1V for stability. Typical cross-sections of the TSF embankment and WMP dam are shown in Figure 18.3.

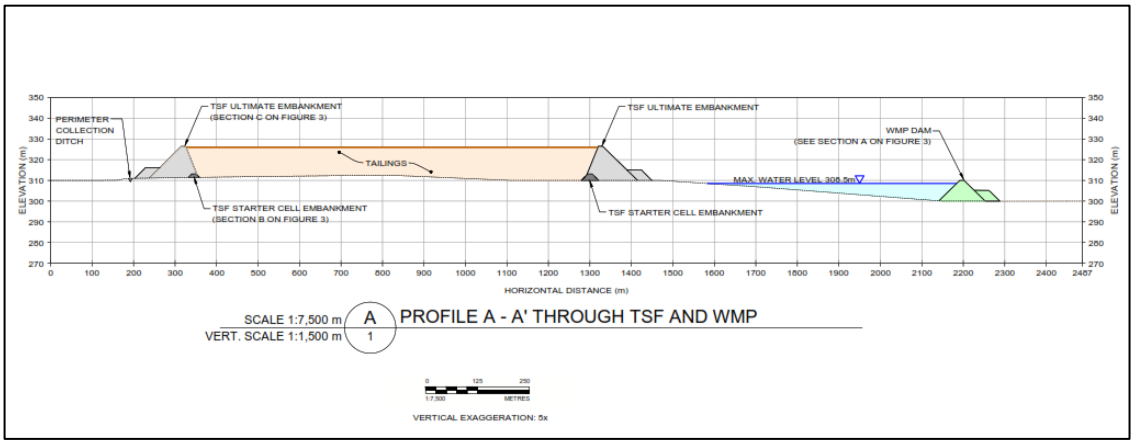
No geotechnical soil investigations have been carried out in the proposed TSF or WMP areas. Based on available information, slope stability analyses for the PEA have assumed 15 m thick layers of silty clay over glacial till (i.e., 30 m total overburden) over bedrock. Both the TSF embankment and WMP dam will require toe berms for stability purposes (as illustrated in Figure 18.3). The PEA design assumes that wick drains will be required over a 75 m width beneath the TSF embankment where the silty clay foundation is deeper than 6 metres. Deep silty clay foundation (e.g., greater than 6 m thick) requiring the installation of wick drains has been assumed for 50% of the TSF perimeter. A waste rock shear key excavation/installation may be sufficient where there is thinner foundation clay (e.g., 3 to 6 m) and ground improvement may not be required where the clay is very thin (e.g., less than 3 m).

Figure 18-5: Tailings Storage Facility Starter Cell & Ultimate Plan



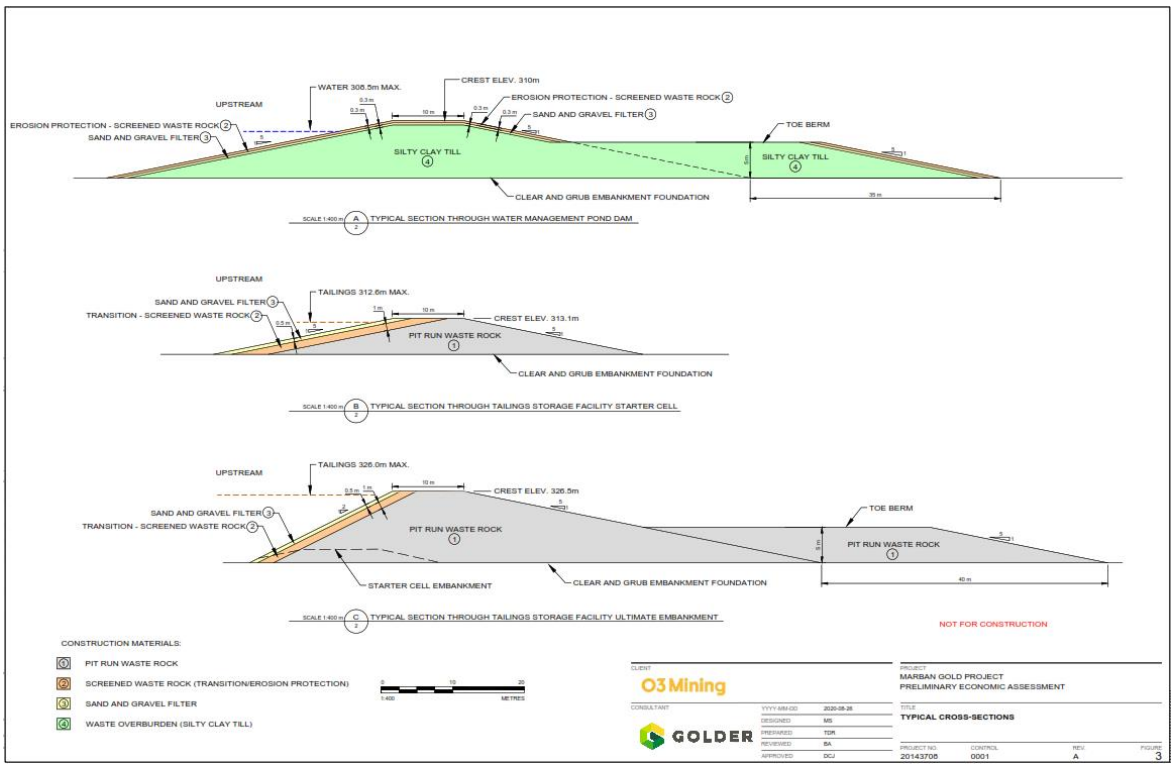
Source: Golder, 2020

Figure 18-6: Tailings Storage Facility & Water Management Pond Profile



Source: Golder, 2020

Figure 18-7: Typical TSF Embankment & WMP Dam Cross-sections



Source: Golder, 2020

18.7.3 Operation

The operational plan for the TSF is to deposit tailings from spigots along the west perimetre embankment with additional spigots that extend to the centre of the TSF to optimize deposition efficiency. Excess process water and runoff from the tailings surface will drain through the rockfill perimetre embankment to the perimetre collection ditch and report to the Water Management Pond. Water will be reclaimed from the Water Management Pond for re-use in the mill. Excess water in the water management pond will be treated in an effluent treatment plant and discharged.

The TSF construction and operation will be planned to align with mine development and operations. Raising of the TSF embankments will be timed to ensure sufficient tailings storage capacity with consideration of equipment accessibility, tailings distribution pipeline relocation and availability of construction materials.

18.8 Power Supply & Electrical

The processing plant will require an electrical power supply. The supply is provided by Hydro-Québec and will be delivered to the site by an existing 120-kV power line which transverses the Marban property and runs adjacent to the processing plant. The peak demand is estimated at 12 MW. Power will be distributed at 4.16 kV.

The 120 kV power line that currently crosses through the Marban Property will be re-aligned along the south property boundary. Re-aligning the power line will free up room for a single continuous waste stockpile which can completely contain all of the mine waste generated from open pit mining the Marban and KN resources. The length of the new power line alignment is approximately 4.3 km.

19 Market Studies & 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 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 Commodity Price Projections

The economics analysis for the Marban Project was calculated at a gold price of US$1,450/oz Au. As of late August 2020, the median consensus price forecast from 30 investment dealers estimated a gold price of US$1,600/oz in 2023 and US$1,500/oz long-term. As of September 3, 2020, the trailing two-year gold price was US$1,470/oz and the trailing three-year gold price was US$1,410/oz. For the purpose of the PEA, a gold price of US$1,450/oz was assumed. The exchange rate used in the study is C$1.00:US$0.74.

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 PEA, a cost of $2.5/oz Au was assumed for transportation, and refining.

20 Environmental Studies, Permitting & 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 the mine closure requirements and costs.

20.1 Environmental Studies

The environmental information provided in this chapter were collected from public databases and from the field inventories and environmental surveys (baseline studies) undertaken in the Marban Project site, mainly in 2016 by WSP.

The studied environmental components include the following:

- Hydrological conditions;
- Surface water and sediment quality;
- Soil natural background content assessment;
- Soil environmental assessment of three old mines sites;
- Biological components (vegetation and wetlands, small mammals, bats, avian fauna, herpetofauna and species at risk, fish and fish habitat);
- Ambient air quality;
- Ambient noise and vibration levels;
- Waste rock geochemical characterization (static).

Additional field inventories and environmental assessments will have to be completed within the project area such as:

- Hydrogeology study and groundwater quality;
- Spring period components (hydrological surveys, surface water quality, species at risk, herpetofauna);
- Biological components (water fowl);
- Soil environmental assessment on potentially contaminated lands;

Soil environmental assessment on potentially contaminated lands (refer to Section 20.1.2.1). Since the TSF and WMP are located outside of the study area considered in the previous environmental studies, all the above baseline studies will have to be done for this new sector.

The following environmental studies and assessments will be required to comply with the EIA requirements:

- Air modelling;

- Greenhouse gas emissions assessment;
- Climate change resilience
- Noise and vibration modelling;
- Ambient night lighting;
- Exhaustive ore, waste rock and tailings geochemical characterization.

20.1.1 Baseline Conditions

The following sub-sections summarize the project's current biophysical environmental conditions. Unless mention otherwise, the information comes from WSP's studies (WSP, 2018).

20.1.1.1 Hydrological Conditions

The project area is mainly located within Kierens Creek watershed, which flows into the north-east 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 Kierens Creek and the Piché River are both part of the De Montigny Lake watershed, which is connected to the Harricana River.

The Kierens Creeks 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, is observed since 1926 (Corriveau, 2010) and over a distance of 3.2 km upstream the Gervais Road crossing passage.

At the Gervais Road crossing passage, the Kierens 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.

20.1.1.2 Surface Water & Sediment Quality

The sampling was carried out during the period from July to August 2016 and in May 2017 in order to obtain representativeness of the annual variability on four selected stations, i.e. ES-01 (Kierens Creek, inside the Marban property), ES-02 (Kierens Creek, outside the Marban property), ES-03 (De Montigny Lake) and ES-04 (tributary of De Montigny Lake). The sediments were sampled once at same four stations during the summer of 2016.

The water is rather acidic ranging from 4.94 to 7.53, 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 CPC (EO)[1], CVAC[2] and the criteria and the recommendations of the Canadian Council of Ministers of the Environment (CCME) for the protection of freshwater aquatic life long-term exposure, as well.

Also, in the case of station ES-04, a high ammoniacal nitrogen value was recorded (5.03 mg L-1) in July 2017 and did not meet the CVAC criterion. Total phosphorus also did not meet the

[1] CPC(EO): Prevention of contamination of aquatic organisms, MELCC (2017)
[2] CVAC: Protection of aquatic life, chronic effect, MELCC (2017)

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, 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 2013, Golder also sampled surface water from the Kierens Creek and found exceedance of the CPC(EO) criteria in one of the three stations for mercury (Golder, 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 occupied 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.

By considering the CER[3] and COE[4] criteria, seven metals were in exceedance: arsenic, cadmium, chromium, copper, mercury, lead and zinc. Soil leaching could contribute to the presence of metals in the sediments of rivers, especially considering the presence of old mining sites along the shores. In 2013, Golder also sampled sediment from the Kierens Creek and found exceedance of the COE criteria at two of the three stations for chromium, copper and nickel (Golder, 2013).

20.1.1.3 Soil Quality Assessment

Soil Natural Background

The assessment of the soil natural metal background was established with 30 soil samples taken from trenches distributed over the study area and from 3 samples of the surface horizon to detect possible airborne contamination. Statistics were conducted for the parameters with at least 10 results detected above the laboratory detection limits. Those metals are aluminum, barium, calcium, chromium, cobalt, iron, manganese, nickel, potassium, sodium, titanium and vanadium. The natural background results can be used as reference and comparison purposes with results from an accidental spill or following the restoration and rehabilitation closure works.

Additionally, when generic criteria of the MELCC's Intervention Guide – Soil Protection and Rehabilitation of Contaminated Sites (Beaulieu, 2016) were available, the results were compared to the generic criteria "A", "B" and "C". The natural background levels evaluated for barium, chromium, cobalt and manganese are lower than criterion "A", while for nickel, it has been evaluated between criteria "A" and "B". The Intervention Guide does not include criteria for the other parameters.

Samples from the surface horizon showed exceedances of the generic criteria "A" and "B" for silver, cadmium, manganese and lead. For these metals, the maximum concentrations measured at the surface were significantly higher than the maximum concentrations measured in the deeper samples. Therefore, the surface layer may have been affected by airborne contamination.

Land Contamination

[3] CER: Rare effect concentration (Environnement Canada et MDDEP, 2007)
[4] COE: Occasional effect concentration

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 on all sites can be considered contaminated with chromium and nickel and locally with copper. They are not considered as acidogenic but 2 out of 6 samples are leachable for copper with the TCLP test. However, no metal mobility was detected with SPLP (acid rain) and CTEU-9 (neutral pH) tests.

The groundwater taken from the 3 wells at Kierens and 3 wells at Norlartic show exceedances of applicable criteria for manganese, zinc and BOD_5. 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 (2 out of 3 samples) for both sites and the criteria for a resurgence of groundwater in surface water (1 out of 3 samples) for the Kierens site. The exceedances are small, and the source is unknown. The exceedance for BOD_5 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 and several wells. These high values can be related to the geological lithologies as the wells were dug in the rock.

20.1.1.4 Vegetation & Wetlands

A total of 170 characterization plots or validation points (97 in wetlands and 73 in terrestrial environments) were carried out to describe the natural environments within the study area.

Across the study area (1,005 ha), 54% (544 ha) is occupied by terrestrial environments while 39% (389 ha) are wetlands. Bodies of water (32 ha) and anthropic lands (40 ha) represent 3% and 4% of the study area.

The terrestrial environments are mainly represented by balsam fir stands, followed by conifer-dominant mixed stands, deciduous stands and 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.

20.1.1.5 Terrestrial Fauna

Trapping statistics for the fur-bearing animal management unit (UGAF) 03 for the seasons 2017-2018 and 2018-2019 (MFFP, 2020) confirm the presence of 14 species, i.e. weasel (*Mustela*), beaver (*Castor canadensis*), coyote (*Canis latrans*), red squirrel (*Tamiasciurus hudsonicus*), grey wolf (*Canis lupus*), otter (*Lontra canadensis*), Canadian Lynx (*Lynx canadensis),* bobcat *(Lynx rufus)*, American marten (*Martes americana*), American black bear (*Ursus americanus*), fisher (*Martes pennanti*), muskrat (*Ondatra zibethicus*), raccoon (*Procyon lotor*), red fox (*Vulpes vulpes*) and mink (*Neovison vison*).

Inventories carried out in August and September 2016 confirmed the presence of 8 species of micromammals in the study area, the most abundant being the Gapper's red-backed vole (Myodes gapperi) and the Meadow vole (*Microtus pennsylvanicus*).

Inventories of avian fauna were carried out in June 2016. A total of 84 species (28 families) were observed. Of this number the nesting was confirmed for 7 species, was judged probable for 9 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, 6 species were observed during the 2016 inventories.

20.1.1.6 Fish & Fish Habitat

Fish habitat description was carried out De Montigny Lake, Kierens Creek and 9 water courses. Water bodies with potential for fish presence and that were potentially impacted by the project's layout at that time were then selected for fish inventories: De Montigny Lake, Kierens Creek and 3 water courses. Ten species were captured, among them, walleye (*Sander vitreus*), sauger (*Sander canadensis*), yellow perch (*Perca flavescens*), northern pike (Esox lucius) and brown bullhead (*Ameiurus nebulosus*) which are of interest for sport fishing. De Montigny Lake has a well-diversified fish population compared to the other water bodies of the study area. WSP considers that only the sections having a direct link with the Kierens Creek or De Montigny Lake should be considered as fish habitat. These are the first segments of the CE-1, CE-2, CE-3, CE-4, CE-7 and CE-9 water courses.

20.1.1.7 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 a vulnerable to harvest.

Four at-risk avian species were observed in the study area: bald eagle (*Haliaeetus leucocephalus*), rusty blackbird (*Euphagus carolinus*), barn swallow (*Hirundo rustica*) 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 common nighthawk (*Chordeiles minor*), the whip-poor-will (*Caprimulgus vociferous*), the bobolink (*Dolichonyx oryzivoru*), the evening grosbeak (*Coccothraustes vespertinus*), the short-eared owl (*Asio flammeus*), the olive-sided flycatcher (*Contopus cooperi),* the eastern wood-pewee (*Contopus virens*) and the eastern meadowlark (*Sturnella magna*).

In 2015, the southern bog lemming (*Synaptomys cooperi*) was captured in the Malartic area (WSP, 2015). So, this species could potentially be present in the study area as preferential habitats are found in the study area (wetlands).

20.1.1.8 Ambient Air Quality

Air sampling was carried out over a period of five 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 to 21 µg/m³ with an average of 10 µg/m³. For the South station, total particle concentration ranged from 4 to 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% to 45.4% of the standard limits for the North station and ranging from 0.01% to 0.7% of the standard limits for the South station.

20.1.1.9 Ambient Noise & Vibration Levels

Ambient Noise

Sound measurements were carried out in July 2016 near residences located on Gervais and Lac-Vassan roads (3 stations), as well as within the study area (1 station) 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 all stations, the measured residual noise sound levels are lower than those provided for in the Instruction Note 98-01 (45 dB_A at day and 40 dB_A at night). The highest records were 41 dB_A at day and 25 dB_A at night.

Vibration Levels

Vibration measurements were also carried out in July 2016 and located approximately in the same places as the sound measurement stations. At all stations, the measured residual vibration levels are low with average values recorded between 0.032 mm/sec. at 0.552 mm/sec.

20.1.2 Environmental Issues

20.1.2.1 Contaminated Land

A landfilling 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, where activities designated under the *Land Protection and Rehabilitation Regulation* were carried out, are located within the proposed layout: 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 a private property located within the proposed layout: garbage, reservoirs, cars and other types of waste (WSP, 2018c).

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.

20.1.2.2 Wetlands

A preliminary evaluation estimates at 450 ha of wetlands that will be directly impacted by the project.

Under the *Regulation respecting compensation for adverse effects on wetlands and bodies of wate*r, O3 Mining will have to compensate for the loss of wetlands. In accordance with the second paragraph of Section 46.0.5 of the EQA, the replacement of 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 in the case of the work for mining mineral substances within the meaning of Section 1 of the *Mining Act*. A compensation program will have to be submitted to the MELCC for approval.

20.1.2.3 Fish & Fish Habitat

As the diversion of the Kierens 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 provided Fisheries and Oceans Canada (DFO). If required, O3 Mining will work with DFO on the details of the offsetting requirements.

According to available data and preliminary assumptions, the mine waste disposal facilities (tailings and waste rock) are not located on fish habitat. However, O3 Mining will have to confirm with DFO if waste rock disposal inside the open pits, after the Kierens Creek diversion is completed, will be considered as a mine waste disposal on fish habitat.

Using a natural water body frequented by fish for mine waste disposal requires an amendment to the *Metal and Diamond Mining Effluent Regulations* (MDMER), which is a federal legislative action, to add 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 alternatives for mine waste disposal for consideration (according to the federal guidelines) and prepare a fish habitat compensation plan for consideration as part of the EIA.

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.

An air modelling will be required to evaluate the potential effects, to identify optimization measures and, if required, to elaborate 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. A noise and vibration modelling will be required to evaluate the potential effects, to identify optimization measures and, if required, to elaborate mitigation, management and monitoring measures.

20.1.2.6 Hydrogeological Conditions

Since the hydrogeological conditions have not yet been established and considering that the mining activities could lower the water table, effects on wetlands, water courses and drinking wells can be expected.

20.2 Ore, Waste Rock, Tailings & Water Management Requirements

The following sections describe the environmental requirements for mining materials storage management facilities based on available information. The Directive 019 is the main guideline

for ore, waste rock, tailings and water environmental management requirements. 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 classified them. This classification serves to determine the design parameters of the storage areas to ensure groundwater protection according to Directive 019, as well as the closure requirements to prevent and/or manage acid rock drainage (ARD) or contaminated neutral drainage (CND).

20.2.1.1 Ore

In 2020, 10 ore samples selected within the two main geological units (five samples for each unit) were selected based on recent mineral resources information and available drilling and geochemical data. The material was tested for acid rock drainage (ARD) potential, total metals content and leachability potential (static tests) (WSP, 2020).

The MABA (Modified acid base accounting) test results indicate that the ore is not potentially acidic rock drainage generating (non-PGA). Their sulfur content is >0,04% but the NNP are >20 kg$CaCO_3$/t and NP/AP are <2. As recommended by the new guideline (MELCC, 2020), kinetic tests must be performed to confirm the ARD potential.

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, the TCLP test was also performed. About 20% of the ore samples show copper exceedance for the groundwater 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 TLCP test. Therefore, the ore must be considered as leachable. None of the ore samples are considered as high-risk. However, as recommended by the *Guide de caractérisation*, kinetic testing must be performed to confirm ore leaching potential.

20.2.1.2 Waste Rock

In 2016, waste rock from Marban pit was characterized based on the geological and operation information provided in 2016-17 (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 a classification of mine materials. The waste rock classification was reviewed as per the new guideline (WSP, 2020).

MABA testing was performed on 22 samples. The results indicate that the waste rock is non-PGA, except for one sample of the V3B geological unit. Kinetic tests could confirm the ARD potential, as recommended by the new guideline.

Since the results of metals content testing exceed the generic criteria "A" of the MELCC for one or more extractible metals, 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. The SPLP test was then performed on the 14 samples that showed the higher metals mobility under the TLCP test and no exceedance of the groundwater criteria was noted. None of the waste rock sample are considered as high-risk.

Based on those results, and without further kinetic testing to confirm the ARD and leachability potential, the waste rock from the Marban pit is considered as leachable. However, if it is confirmed that the waste rocks are non-PAG by further investigations, only the SPLP and

CTEU-9 testing results must be considered. As there was no CTEU-9 testing performed, this might also be performed during further investigation. Leaching potential might also be assessed by kinetic testing.

The testing done by Golder of the Norlartic, Kierens and Marban on waste rock used as backfill also shows metal mobility under the TLCP test and were non-PAG (Golder, 2013). Therefore, all waste rock that will have to be manage must be considered as leachable until further testing is performed.

20.2.1.3 Tailings

The chemical composition, ARD and leachability potential of the tailings will be defined when samples become available. Thus, the results from the ore samples were used as preliminary assumptions of the tailings' geochemical characteristics to define the tailings management facilities requirements.

20.2.2 Ore & Waste Rock Management

20.2.2.1 Ore Storage

When the enriched ore or concentrate are 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 according to the characteristics of these materials must be taken 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 a majority of samples are non-PAG, since most of them are metal leaching, Level A groundwater protection measure 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 waste rock stockpile design proposed in Section 18 considers that the clay layer will allow a proper groundwater protection measure by limiting water infiltration into the ground.

The hydrogeological study will have to include a conceptual model of the hydrological context and groundwater flow to establish possible hydraulic links with the receiving environments. A model establishing that the measures proposed are sufficient to prevent groundwater quality degradation will also have to be done.

20.2.3 Tailings Management

A surface disposal 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 are 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 TSF design proposed in Section 18 considers that the clay layer will allow a proper groundwater protection measure by limiting water infiltration into the ground.

The hydrogeological study will have to include a conceptual model of the hydrological context and groundwater flow to establish possible hydraulic link with the receiving environments. A model establishing that the measures proposed are sufficient to prevent groundwater quality degradation will also have to be done.

20.2.4 Water Management

The waters to be managed are mine water from the pits, 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 and be treated before being released to the environment. Those water management infrastructures might have to be lined to limit water infiltration into the ground.

Water treatment will be required to ensure that the mining effluent discharge meets the Directive 019 and MDMER quality criteria. Additional environmental discharge objectives (OER) criteria could be added to the previous ones. Those OER criteria will 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 the mitigation or compensation measures proposed in the impact assessment. As part of the project, an environmental monitoring program will be implemented. The following are the main components of the environmental site monitoring program:

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

Since habitations are located less than 1 km from the project area, O3 Mining will have to install a monitoring network (between one and three stations) for ground vibrations and air pressures near residences or drinking wells (Directive 019).

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 PEA. The Provincial government is reviewing new regulation projects under the *Environmental Quality Act* (EQA). Changes in guidelines are also a possibility throughout the project development.

20.3.1 Environmental Impact Assessment Process

20.3.1.1 Provincial Authorities

With its average extraction rate of 12,702 t/d and a process plant with a capacity of 10,959 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 EQA, to the extent provided therein, and must obtain an authorization from the Government.

The MELCC oversees the EIA process and relies on the Bureau d'audience publique sur l'environnement (BAPE) du Québec for public hearings. The phases of the provincial EIA process are:

4. Project Notification: The Proponent must file a written notice with the Minister, in accordance with Section 31.2 of the EQA.

5. Evaluation: Within 15 days after receiving the project notice the Minister must send to the proponent a directive provided 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.

6. EIA Statement Preparation: The proponent conducts the studies and assessment according to the requirements of the guidelines and submits the EIA statement in 12 paper copies to the Minister as well as an electronic version.

7. 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.

8. 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.

9. 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 the project notice to the Government decision goes 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 CEAA 2012 to designate projects by type and thresholds prescribed by regulation.

The provisions in the schedule to 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 10,335 t/d and a process plant with a capacity of 10,959 t/d, the preliminary analysis of the project indicates that it is subject to the IAA. However, the Impact Assessment Agency of Canada (Agency) will have to render a decision on whether the designated project will be subject to an impact assessment.

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 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.

Tables 20.1 and 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 the 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 project and typical activities related to mining projects.

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 application of Section 32 of the Environment Quality Act (Q-2, r. 2)
Regulation respecting the application of the Environment Quality Act (Q-2, r. 3)
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 waste water 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)

Provincial Jurisdiction
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)
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)

Provincial Jurisdiction
Directives and Guidelines
Directive 019 sur l'industrie minière (2012)
Lignes directrices relatives à la valorisation des résidus miniers (2015)
Guidelines for preparing mine closure plans in Québec (2017)
Guide d'intervention – Protection des sols et réhabilitation des terrains contaminés (2019)
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)

Federal Jurisdiction
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)
Guidelines for the Assessment of Alternatives for Mine Waste Disposal (2016)
Strategic climate change assessment (draft version)

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, waste water 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

Activities	Type of Request	Authority
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	-	ECC

20.4 Social or Community Considerations

20.4.1 Consultation Activities

No information and consultation activities were initiated with First Nations, local communities, or stakeholders.

20.4.2 Social Components & Related Requirements

20.4.2.1 Land Planning & Development & Land Use

The Project is located in the administrative region of Abitibi-Témiscamingue (08) in the RMC 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 the purpose of carrying out the project.

According to the land use planning and development plan of the RMC 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 a compatible activity with both Rural and Forest land uses, but outside the urbanized area. The nearest urban perimetre 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.

Forest shelters and resorts 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 the RCM La Vallée de l'Or was estimated at 43,505 people in 2019 while in 2015, it numbered 43,435, a slight increase of 0.2%5. Val-d'Or also experienced an increase in its population during the same period, from 31,588 people to 32,896 (+4.1%). Malartic experienced a decline in its population which was 3,398 people in 2015 and is now estimated at 3,271 people in 2019 (-3.7%).

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 the commercial and institutional function. In addition, the

5 All population data are taken from the Institut de la statistique du Québec.

municipality has both local and regional recreational tourism facilities. The mining industry is the main economic driver.

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 13.8% of employment in 2016. Jobs related to the health care and retail sectors were respectively 13.3% and at 13.2% of total jobs (Statistics Canada, 2017). For Malartic, mining represents a high proportion of total jobs (26.3%), followed by health care (14.8%) and retail (11.5%) sectors.

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 databank. 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 Marban, Norlartic and Kierens former 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 the 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 (MCC), the land use planning and development plan of the RMC La Vallée-de-l'Or and in the Inventory of Archaeological Sites of Québec (ISAQ) are located within the proposed layout.

An evaluation of the archaeological potential was done in 2016 by an archeologist (Archeo-08, 2016) for the same study area as the environmental baseline studies. When an area presented a certain potential, it was rated low, medium or high. According to the report, the northern shore of the Kierens Creek has a high archaeological potential, the southern shore a low or high potential, and the De Montigny Lake a low potential.

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:

- Conseil de la Première Nation Abitibiwinni (52 km)
- Conseil de la nation Anishnabe du Lac-Simon (50 km)
- Conseil des Anicinapek de Kitcisakik (84 km)
- Long Point First Nation (76 km)
- Timiskaming First Nation
- Algonquins of Barriere Lake

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.

20.4.2.7 Social Related Requirements

Engagement Activities

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 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.

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.

Additional Studies

The following studies should be carried out as part of the EIA process:

- Economic Benefits Assessment
- Visual Integration (landscape, night light)
- Circulation and Roads Security Assessment
- Domestic Wells Inventory and Water Table Lowering Assessment

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 the 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);
- 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 Project is an open-pit mining operation, a cost-benefit analysis for backfilling the pit must be included in the closure plan.

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 the plan’s approval
- Each subsequent payment must be made on the anniversary of the plan’s approval

- The first payment represents 50% of the total amount of the guarantee, and the second and third payments represent 25% each

Total guarantee for the Marban Project evaluated by Golder is at approximately $12.2 M, including the direct and indirect costs, and 20% contingency.

The guarantee must remain in effect until the certificate of release provided for in Section 232.10 of the *Mining Act* has been issued.

A proponent whose closure plan has been approved must submit a revised plan every five years to the MERN, unless the latter has set a shorter period on approving the closure plan or the revised plan.

Closure work must begin within three 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;
- 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;
- 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.

21 Capital & Operating Costs

The capital and operating cost estimates presented in this PEA are for the Marban Project and are based on open pit mining of the Marban and Kierens, Norlartic deposits and the construction of a process plant, associated tailings storage facility, and infrastructure. The processing plant nameplate capacity is 10,959 tpd (4.0 Mtpy), with a life of mine of 15.2 years.

All capital and operating cost estimates are reported in Canadian dollars for this PEA.

The purpose of the capital estimate is to provide substantiated costs which can be used to assess the preliminary economics of the project.

21.1 Capital Costs

The capital cost estimate was developed in Q2CY2020 from Ausenco's in-house database of projects and studies and experience from similar operations to a level of accuracy of ±50% (Class 5) in accordance with the Association for the Advancement of Cost Engineering International (AACE International). The estimate includes mining, processing, utilities, TSF and Project site infrastructure. The capital cost summary is presented in Table 21-1. The total initial capital cost for the Marban Project is $256M and LOM sustaining costs are $189M. Closure costs are estimated at $10.2M.

Table 21-1: Capital Cost Summary

WBS	Cost Centre	Initial Capital	Sustaining	Total Capital
100	Mining	48.8	153.9	202.7
200	Crushing	19.2		19.2
300	Processing	75.5		75.5
400	Utilities	10.5		10.5
500	TSF	17.9	35.0	52.8
600	Infrastructure	14.1		14.1
700	Offsite Infrastructure	4.3		4.3
	Directs	190.3	188.9	379.2
800	Indirects	24.5	10.2	34.7
900	Owner's Cost	10.6		10.6
	Contingency	30.6		30.6
	Total Project Cost	**256.0**	**199.1**	**455.1**

The cost estimate is based on an engineering, procurement and construction management (EPCM) implementation approach. The following basic information pertains to the estimate:

- Base date is Q2, 2020;
- Expressed in Canadian dollars (C$);
- Currency exchange rate US$0.74:C$1.00;
- Accuracy is ±50%.

- No allowance has been made for exchange rate fluctuations.

Major cost categories including permanent equipment, equipment packages, modular equipment, material purchase, installation, subcontracts, indirect costs, as well as Owner's costs were identified and examined to calculate the capital cost estimate.

21.2 Basis of Capital Cost

The capital cost estimate has been developed to a conceptual level of accuracy based on Ausenco's in-house database of projects and studies and experience from similar operations. Contingency was estimated, per area, based on the confidence level of source information.

Data for the estimates have been obtained from numerous sources, including:

- mining schedule;
- Conceptual engineering design by Ausenco, MMTS, and Golder.
- mechanical equipment costs determined internally from benchmarking against similar projects for the eastern Canada region;
- civil, concrete, steel, electrical, earthworks and mechanical works have been benchmarked against similar projects with the equivalent technologies and unit operations;
- topographical information considered
- pre-economic-assessment level engineering design

21.2.1 Direct Costs – Mining

Mine capital costs have been derived from historic data collected by MMTS at other Canadian open pit mining operations, combined with the Marban PEA mine production schedule.

Pre-production mine operating costs (i.e., all mine operating costs incurred before mill start-up) are capitalised and included in the capital cost estimate. Pre-production pit operating costs include drill and blast, load and haul, support, and general mining expenses (GME) costs. All mine operations site development costs—such as clear and grub, topsoil stripping, wetland removal, haul road construction and explosive pad preparation—are capitalised.

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 capitalised through the initial and sustaining periods of the project.

Estimated fleet spare and estimated initial fuel, lube, and tire inventories are capitalised.

The following items are also capitalised:

- explosives magazine
- site GPS (global positioning system) and machine guidance systems
- mine survey gear and supplies

- geology, grade control, and mine planning software licenses
- maintenance tooling and supplies
- mine rescue equipment

Capital costs for the mine fleet maintenance facility, as well as the fueling station, are accounted for under general project infrastructure.

Table 21-2 summarizes the Mine Area Capital Cost estimates for the Marban PEA Project. It is the QP's opinion that these estimates are reasonable for the location and planned mine development and can be used for a PEA.

Table 21-2: Marban Mine Area Capital Cost Summary

WBS	Description	Initial C$M	Sustaining C$M	Total C$M
110	Mine Development	26.5		26.5
120	Mine Operations Services	5.7	2.0	7.7
130	Mine Mobile Equipment	13.3	151.9	165.2
140	Mining Truck Shop / Maintenance / Wash Bay	3.3		3.3
	Total Mining Capital Cost	**48.8**	**153.9**	**202.7**

21.2.2 Direct Costs – Process & Infrastructure

The definition of process equipment requirements was based on conceptual process flowsheets and process design criteria as defined in Section 17.

Process equipment costs were derived using recent similar projects, recent and historical budget quotes on file from vendors. Delivery and installation of process equipment is a factored cost relative to the total purchase price of equipment. A detailed estimate of man-hours was not completed for this study. The factors developed were based on installation by local contractors.

Bulk earthworks for the plant site, mine ancillary buildings, tailings storage facility and water management infrastructure were developed based on semi-detailed cut and fill volumes based on site layout and site topographical information. Unit rates were benchmarked based on recent projects within the region. An allowance for plant site perimeter fencing was included.

On-site infrastructure costs were developed based on in-house database of costs and include:

- Mine buildings including truck shop, mine workshop, tire repair shop and mine dry building;
- Process plant buildings including workshop and laboratory;
- Ancillary buildings including warehousing, administration and gatehouse;
- An allowance for HV powerline tie-in and substation

Off-site infrastructure includes:

- High voltage (HV) overhead power line 4.3 km x 120 kV re-alignment; benchmarked cost per distance;

Process and infrastructure costs are summarized in Table 21-3.

Table 21-3: Marban Process & Infrastructure Capital Cost Summary

WBS	WBS Description	Initial C$M	Sustaining C$M	Total C$M
200	Crushing	19.2		19.2
210	Primary Crushing	6.5		6.5
220	Secondary Crushing	5.7		5.7
230	Ore Storage & Reclaim	7.0		7.0
300	Processing	75.5		75.5
320	Milling	33.1		33.1
330	Leaching	16.8		16.8
340	Detox / Tailings	7.9		7.9
350	Desorption	6.6		6.6
360	Refining	4.1		4.1
390	Reagents	7.1		7.1
400	Utilities	10.5		10.5
430	Water Services	3.1		3.1
440	Water Treatment	3.7		3.7
450	Air Services	0.5		0.5
460	Electrical Services	2.5		2.5
480	Plant Area Lighting	0.2		0.2
490	Fuels	0.5		0.5
500	Tailings Management Facility	17.9	35	52.8
600	Infrastructure	14.1		14.1
601	Main Access Road and site prep	1.8		1.8
603	Security Fencing	0.1		0.1
604	Overland Process Piping	0.4		0.4
606	Water Management Piping + Ditching	8.3		8.3
650	Plant Buildings	3.5		3.5
700	Offsite Infrastructure	4.3		4.3
710	Powerline relocation	4.3		4.3
Total Process and Infrastructure Cost		**141.5**	**35.0**	**176.5**

21.2.3 Indirect costs

Indirect costs (Table 21-4) are those that are required during the project delivery period to enable and support the construction activities. Indirect costs include:

- Field indirect costs: construction distributable costs (tools, supplies, consumables) and temporary facilities;
- Commissioning support: vendor representatives, first fill and spares;
- EPCM: home office engineering; site and home office expenses.

The indirect cost estimate was factored based on previous Ausenco experience with similar-sized projects. EPCM was estimated at 12%, field indirect costs at 4%, and commissioning support at 1.5% of total direct costs (not including mining direct costs).

Table 21-4: Indirect Cost Summary

WBS	Indirect Costs	C$M
810	Field Indirects	5.7
850	EPCM Costs	16.7
860	Commissioning Support	0.7
870	First Fills/Spares	1.4
Total Indirect Capital Costs		**24.5**

Note: Closure costs of $10.2M have also been included as indirect sustaining costs.

21.3 Contingency

The contingency accounts for the difference in costs from the estimated and actual costs of materials and equipment. Typically, these costs become more identifiable at the further stage in the engineering design of the project (i.e., going from PEA to pre-feasibility study level). The contingency cost is from total installed costs based on the level of uncertainty for each area, using a deterministic approach. The total estimated for contingency is $30.6M, or 14% of the initial capital costs as illustrated in Table 21-5.

Table 21-5: Project Contingency

WBS	Description	Contingency %	Contingency C$M
100	Mining	10	2.2
200	Crushing	12	2.3
300	Processing	12	9.1
400	Utilities	20	2.1
500	Tailings Management Facility	20	3.6
600	Infrastructure	25	3.5
700	Offsite Infrastructure	20	0.9
800	Indirect Costs	20	4.9
900	Owners Project Costs	20	2.1
CONTINGENCY		**14**	**30.6**

21.4 Sustaining Capital Costs

These are the costs incurred post operation of the processing plant. As production proceeds, the mining fleet expansion and tailings management requirements, frequency of replacement equipment and materials, pit extensions and therefore surface water management. Sustaining costs are listed in Table 21-2 and 21-3.

21.5 Capital Cost Estimate Responsibilities

The capital cost estimate was developed in accordance with the responsibility breakdown presented in Table 21-6.

Table 21-6: Estimate Responsibility Summary

WBS	Cost Centre	Responsible Company / Consultant
100	Mining[A]	Moose Mountain Technical Services
300	Processing	Ausenco
400	Utilities	Ausenco
500	Tailings Management Facility	Ausenco / Golder
600	Plant Site Infrastructure	Ausenco
	Directs	Ausenco
800	Indirects	Ausenco
900	Owner's Costs	O3 Mining / Ausenco
	Contingency	Ausenco
	Total Project Cost	Ausenco

Note: [A] - WBS 140 - Mining Truck Shop / Maintenance / Wash Bay are Ausenco's responsibility

21.5.1 Exclusions

The following costs and scope will be excluded from the capital cost estimate:

- land acquisitions
- taxes not listed in the financial analysis
- sales taxes
- operating costs
- scope changes and project schedule change and the associated costs
- any facilities/structures not mentioned in the project summary description
- costs to advance the project from preliminary economic assessment to pre-feasibility study
- geotechnical unknowns / risks
- financing charges and interest during the construction period
- any costs for demolition or decontamination for the current site
- Third party costs

21.5.2 Owner's Costs (WBS: 900)

The Owner's costs were factored from the total directs and are 7.5% of total direct costs or $10.6M shown in Table 21-1. The following items are part of the Owner's costs:

- maintenance vehicles
- operations vehicles

- G&A vehicles
- Owner's project team and expenses
- Pre-production labour
- Mobile equipment leases
- insurance, finance, legal, and offices
- environmental consultation, management and treatment
- site security
- community relations
- recruiting, training, human resources and site visits
- administration, financial and costs
- mobile equipment

21.6 Operating Costs

The operating cost estimate was developed in Q2CY2020 from Ausenco's in-house database of projects and studies and experience from similar operations to a level of accuracy of ±50%. The estimate includes mining, processing, general and administration (G&A), mobile equipment and TSF.

The overall life-of-mine operating cost is $1,702M over 15.2 years, or $28.2/t of ore milled.

21.6.1 Operating Costs – Mining

Mine operating costs are built up from first principles and applied to the Marban PEA mine production schedule. Cost inputs are derived from historical data collected by MMTS. 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 PEA 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 21-7. 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 PEA.

Table 21-7: Marban Mine Operating Cost Summary

Item	$/t Mined	$/t Milled	$M
Drilling	$0.29	$1.95	$117.8
Blasting	$0.59	$3.97	$239.6
Loading	$0.29	$1.92	$116.1
Hauling	$0.98	$6.56	$395.9
Support	$0.32	$2.11	$127.6
Site	$0.03	$0.18	$10.6
Unallocated Labour	$0.05	$0.34	$20.2
DIRECT COSTS - Subtotals	**$2.55**	**$17.03**	**$1,027.8**
General Mining Expenses (GME)	**$0.14**	**$0.91**	**$55.0**
TOTAL MINE OPERATING COST	**$2.69**	**$17.94**	**$1,082.9**

21.6.2 Operating Costs – Process & Infrastructure & G&A

The operating cost estimates are presented in CAD and summarised in

Table 21-8. These are derived from benchmarking, against existing gold processing plants located in eastern Canada as well as in-house data and quotations. The various sections that are included in the operating cost estimate are discussed in 21.6.3.1 to 21.6.3.9.

The tailings management facility was jointly developed by Golder and Ausenco correspond to the 4.0 Mt/a plant throughput. The processing and infrastructure operating costs total $576.6M and the G&A and mobile equipment costs total $42.8M over the LOM.

Table 21-8: Operating Cost Summary

Cost Centre	C$M per Year	C$/t
Processing	36.57	9.14
Labour	7.16	1.79
Power	4.87	1.22
Reagents	14.61	3.65
Consumable	8.08	2.02
Maintenance	1.53	0.38
Lab Services	0.31	0.08
Effluent Treatment	1.09	0.27
General & Administration	2.35	0.59
Mobile Equipment	0.36	0.09
Total OPEX Costs	**40.38**	**10.09**

Common to all operating cost estimates are the following assumptions:

- For material sourced or benchmarked in US dollars, an exchange rate of 0.74 US dollar per Canadian dollar was assumed.
- Diesel costs used are C$1.00/L and gasoline costs are C$1.045/L.
- The annual power costs were calculated using a unit price of C$0.053/kWh applying Rate L.
- Labour is assumed to come mostly from Abitibi region, and locally from places such as Rouyn-Noranda, Val-D'or, Kirkland Lake and Amos and salaries are based on the region and benchmarked against similar gold projects
- The processing unit operations were benchmarked against similar or comparable processing plants. For example, the 2C_SAB crushing and grinding circuit was compared to circuits of the same configuration.
- Equipment and materials will be purchased as new
- Process plant operating costs are calculated based on labour, power consumption, and process and maintenance consumables.
- General and administration (G&A) costs were baselined against previous project experience.
- Grinding media consumption rates have been estimated based on the ore characteristics. Note: Abrasion Index is currently unavailable so was assigned
- Reagent consumption rates have been estimated based on the metallurgical testwork
- Mobile equipment cost provides for fuel and maintenance.

21.6.3 Basis of Processing Operating Cost

21.6.3.1 Power

The power will be provided by Hydro-Québec and the costs is calculated from the standing charge and power draw determined from the mechanical equipment list. Power cost was calculated assuming Hydro-Québec's "tariff L" (Rate L) which is a published rate for large-power industrial customers whose contract power is in excess of 5,000 kW.

21.6.3.2 Reagents

The reagent profile was developed from the testwork review in section 13. The testwork enabled estimation of addition and consumption rates of particular reagents, where testwork was not available, benchmarking against currently operating unit technologies was examined. Costs for each reagent were identified from other projects in eastern Canada. The details are presented in Table 21-9.

Section 17 discusses the reagents in detail for both processing. Reagents for the intensive leach reactor (ILR) and the elution process have been grouped together.

Table 21-9: Reagent Cost Summary

Cost Centre	C$M/a	$/t
Gravity - ILR	0.41	0.10
Cyanide - Leach	3.96	0.99
Lime - Leach	1.06	0.27
Carbon	0.15	0.04
Oxygen[A]	3.62	0.90
Elution	0.41	0.10
SMBS	4.31	1.08
Copper Sulphate	0.24	0.06
Lime - Detox	0.79	0.20
Flocculant	0.40	0.10
Misc.	0.10	0.02
Total Reagent Cost	**15.46**	**3.86**

Notes: A – delivered as liquid oxygen (used in leaching and detoxification)

21.6.3.3 Consumables

These were identified as non reagent requirements/replacements which were not listed in section 21.6.3.2. and related to the crushing and grinding circuit. The following items have been included under consumables:

- SAG and Ball Mill Grinding Media
- SAG and Ball Mill Liners
- Trommel and screen panels
- Jaw and Cone crusher sets

The grinding media were $5.88 M CAD per annum and linear costs were $2.95 M CAD per annum. The costs have been developed from Ausenco's in-house database and experience, industry practice, and peer-reviewed literature. The consumption rates were calculated internally.

21.6.3.4 Maintenance

The process plant annual maintenance costs were derived from the total installed mechanical cost and using a factor of 5%. Refer to Table 21-8.

21.6.3.5 Lab Services

The operating cost estimate for the laboratory and assay activities were based on the number of assays required per day and per year as well as the assays type (fire assay, atomic adsorption etc.). These assay costs arise from the monitoring of grade/recovery for unit operations to permit optimisation of the process plant, environmental analysis, cases of repeat analysis and are also important for metallurgical accounting. Refer to Table 21-8.

21.6.3.6 Labour

The labour estimate was determined from benchmarking against similar projects with comparable unit processes in terms for crushing, grinding, leaching, goldroom and laboratory as well burden costs/employee bonuses. A burden rate of 55% was applied to the various rates. Labour was estimated for both staff and hourly on a 12-hour shift basis.

The estimate includes the following areas:

- Management
- Metallurgical
- Operation
- Maintenance/Site services
- Laboratory
- Allowance for contract labour

An organisational roster outlining the labour requirement for the process plant is shown in Table 21-10

Table 21-10: Processing Plant Annual Labour Cost

Labour / Contractor Summary	Number of Personnel	C$M per Year
Mill Superintendent	1	0.23
Maintenance Superintendent	1	0.14
Chief Metallurgist	1	0.14
Process Upper Management	3	0.50
General Foreman	2	0.23
Crusher Operator	8	0.73
Grinding/Gravity Operator	4	0.42
Leach/Elution/Detox/Reagents Operator	12	1.05
Gold Refining Foreman/Operator	4	0.42
Mill Operations	30	2.85
Lab Managers	1	0.14
Lead Hands	2	0.24
Laboratory Technicians	4	0.18
Laboratory	7	0.56
Maintenance Planner	1	0.12
Millwrights	4	0.48
Electricians	4	0.48
Process Control/Instrument Tech.	2	0.24
Welders	2	0.24
Mill Maintenance	13	1.57
Total	**53**	**7.16**

21.6.3.7 Effluent Treatment

The water treatment plant size and cost are influenced by the water balance and the tailings management facility. The cost estimate included maintenance, labour, power and consumables. The cost breakdown is presented in Table 21-11.

Table 21-11: Effluent Treatment Plant Annual Operating Cost Summary

Cost Centre	C$M per Year	C$/t
Labour	0.87	0.22
Others (including consumables)	0.03	0.01
Power	0.10	0.03
Plant Maintenance	0.09	0.02
Total	1.09	0.27

21.6.3.8 General & Administration

General and administrative (G&A) costs were developed with Ausenco's in-house data on existing Canadian operations. The costs were estimated based on the following items:

- human resources (including recruiting, training, and community relations)
- infrastructure power (HVAC and Administrative Buildings)
- site administration, maintenance and security (including subscriptions, professional memberships and dues, first aid, office equipment, garbage disposal, bank and payroll fees)
- assets operation (including non-operation-related vehicles)
- health and safety (including personal protective equipment)
- environmental (including sampling and TSF operation)
- IT and telecommunications (including hardware)
- contract services (including insurance, consulting, sanitation, auditing, licences, freight, and legal fees)

21.6.3.9 Mobile Equipment

The costs were developed from the number of light vehicles and mobile equipment, maintenance, spares and tires. Also, fuel (diesel, gasoline) was included under mobile equipment.

Table 21-12: Annual Average G&A Operating Cost Summary

Cost Centre	C$M per Year	C$/t
G&A	0.07	0.014
Processing	0.14	0.034
Maintenance	0.17	0.04
Total	0.36	0.09

22 Economic Analysis

22.1 Cautionary Statement

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;
- Assumptions as to environmental, permitting and social risks.

Additional risks to the forward-looking information include:

- Changes to costs of production from what is assumed;
- Unrecognized environmental risks;
- Unanticipated reclamation expenses;
- Unexpected variations in quantity of mineralized material, grade or recovery rates;
- 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;
- Changes to tax rates.

The mine plan is partly based on Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the PEA based on these Mineral Resources will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

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 impact of variations in metal prices, head grades, 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 2020 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:

- Commercial production start-up in 2023;
- Construction period of one year;
- Mine life of 15.2 years;
- Base case gold price of US$1,450/oz was based on consensus analyst estimates and recently published economic studies. The forecasts used are meant to reflect the average metal price expectation over the life of the Project. No price inflation or escalation factors were taken into account. Commodity prices can be volatile, and there is the potential for deviation from the forecast;
- United States to Canadian dollar exchange rate assumption of 0.74 (US$/C$)
- Cost estimates in constant Q2 2020 C$ with no inflation or escalation factors considered;
- Results are based on 100% ownership with 1.5% NSR;
- Capital costs funded with 100% equity (i.e. no financing costs assumed);
- All cash flows discounted to beginning of construction January 01, 2022;
- All metal products are assumed sold in the same year they are produced;
- Project revenue is derived from the sale of gold doré;
- 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 PEA study and include estimates for O3 Mining's expenditures, and related impacts to various tax pool balances, between the PEA study and the assumed construction start date. At 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 Québec Mining Tax Act
- Total undiscounted tax payments are estimated to be C$484 M over the life of mine.

The tax evaluation was completed by applying the following assumptions:

- It has been assumed all royalty payments are deductible for income tax purposes and non-deductible for Québec Mining Duties purposes.
- It has been assumed that the 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 Québec income tax return filed for fiscal year 2019 ($13.2M). 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 balance of capital losses in the amount of $0.05M indicated on the 2019 T2 has not been considered in the model (as it can only be utilized to offset capital gains).
- The opening balance of CEE corresponds to the closing balance of regular and successor pools as per the FY 2019 T2 and Québec income tax return.
- The opening balance of CDE corresponds to the closing balance of regular and successor pools as per the FY 2019 T2 and Québec income tax return.
- The opening balance of ITCs corresponds to the closing balance as per the FY 2019 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 UCC corresponds to the closing balance as per the FY 2019 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 PP&E is expected to be disposed of by the end of the life of the mine and salvage proceeds of $6.9M are expected to be received. As a result, a terminal loss is being considered for all CCA classes in year 2039. The salvage proceeds were added back in the taxable income calculation.

- The model currently indicated that non-capital losses will be generated on FY 2038 and FY2039 (last years of operation). Such losses can be carried back to the three previous taxation years.
- For simplicity purposes, it has not been considered the Québec minimum tax in the model. As the minimum tax paid becomes a credit that can be applied against the Québec 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 2020 opening balance in the model corresponds to the closing balance as per the FY 2019 Québec mining tax return.
- The FY 2020 opening balance in the model corresponds to the closing balance as per the FY 2019 Québec 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 Working Capital

Working capital and initial fills are included as part of initial capital and assumed to be recovered in the final years of operation. The effective sum of working capital and initial fills over the life of mine is zero.

22.3.3 Closure Costs & Salvage Value

Closure costs and salvage value are applied at the end of the life of mine. Closure costs were estimated C$10.2M and salvage value of C$6.9M.

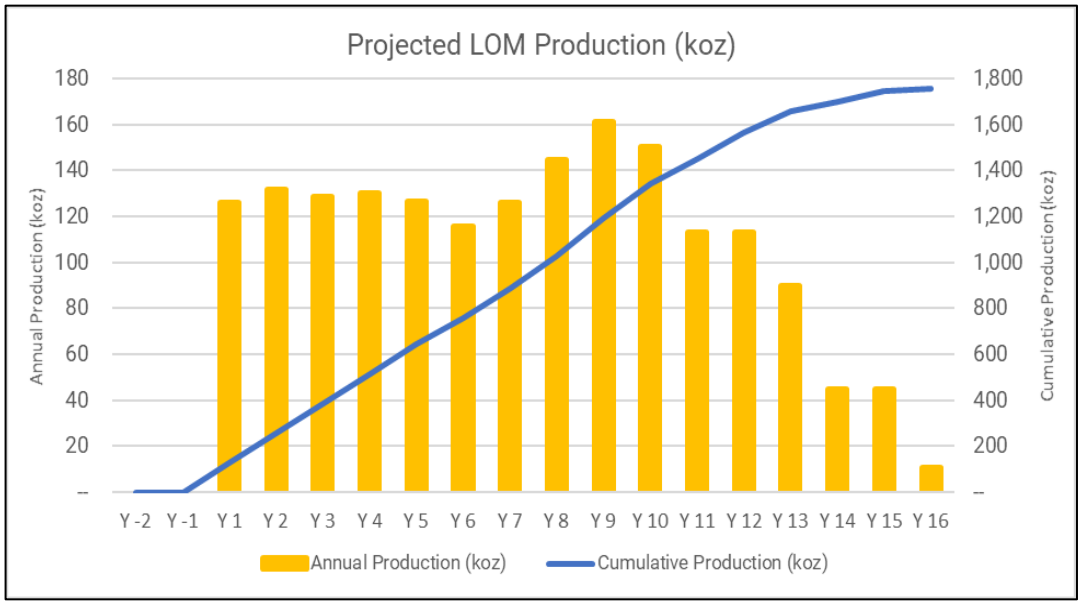
22.4 Royalties

Based on the agreements in place, a 1.5% NSR royalty has been assumed for the Project, resulting in approximately C$51.5 M in undiscounted royalty payments over life of mine.

22.5 Gold Production

Over the life of mine, a total of 1,757 koz of gold (average annual: 115,000 oz) will be produced. Figure 22-1 provides a summary of the payable gold production by year.

Figure 22-1: Annual Gold Production (koz)

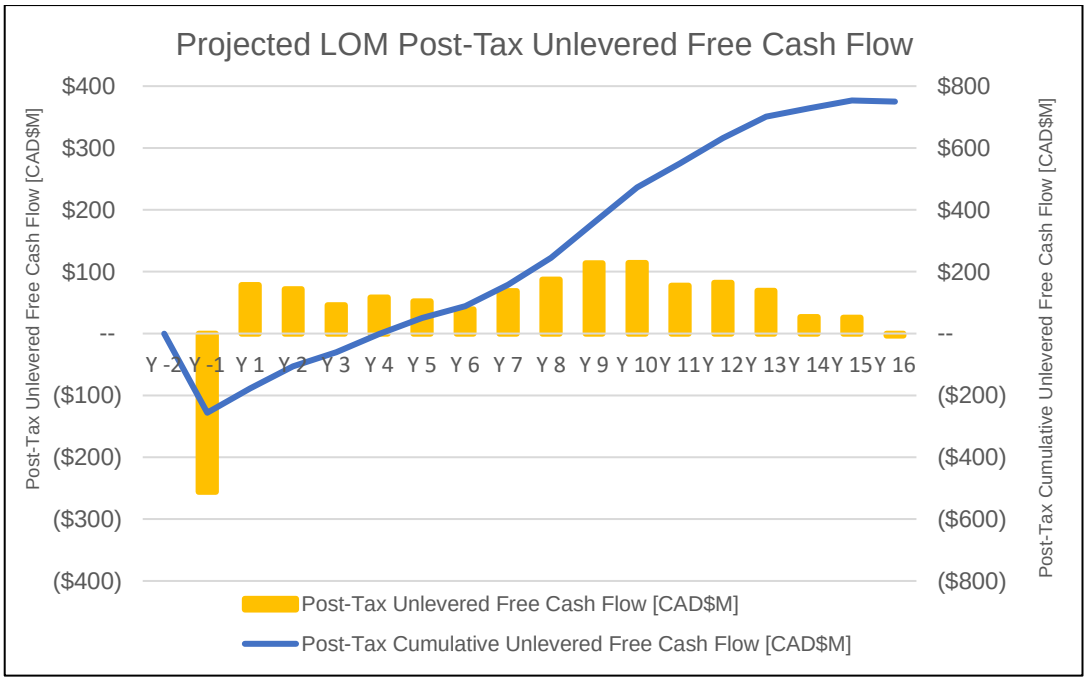


Source: Ausenco, 2020

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$715M, the internal rate of return IRR is 31.5%, and payback is 3.7 years. On an after-tax basis, the NPV5% is C$423M, the IRR is 25.2%, and the payback period is 4.0 years. A summary of the Project economics is included in Table 22.1 and shown graphically in Figure 22-2. The cashflow on an annualized basis is provided in Table 22.2.

Figure 22-2: Projected Post-Tax Cash Flow



Source: Ausenco, 2020

Table 22-1: Summary, Project LOM Cashflow Assumptions & Results

General	LOM Total / Avg.
Gold Price (US$/oz)	$1,450
Exchange Rate ($US:$CAD)	0.74
Mine Life (years)	15.2
Total Waste Tonnes Mined (kt)	355,627
Total Mill Feed Tonnes (kt)	60,356
Strip Ratio (waste: mineralization)	5.89
Production	**LOM Total / Avg.**
Mill Head Grade (g/t)	0.97
Mill Recovery Rate (%)	93.7%
Total Mill Ounces Recovered (koz)	1,757
Total Average Annual Production (koz)	115
Operating Costs	**LOM Total / Avg.**
Mining Cost (CAD$/t Mined)	$2.7
Mining Cost (CAD$/t Milled)	$17.9
Processing Cost (CAD$/t Milled)	$9.6
G&A Cost (CAD$/t Milled)	$0.7
Total Operating Costs (CAD$/t Milled)	$28.2
Refining Cost (CAD$/oz)	$2.5
Royalty NSR	1.5%
Cash Costs (US$/oz Au)	$741
AISC (US$/oz Au)	$822
Capital Costs	**LOM Total / Avg.**
Initial Capital (CAD$M)	$256
Sustaining Capital (CAD$M)	$189
Closure Costs (CAD$M)	$10
Salvage Costs (CAD$M)	$7
Financials - Pre Tax	**LOM Total / Avg.**
NPV (5%) (CAD$M)	$715
IRR (%)	31.5%
Payback (years)	3.7
Financials - Post Tax	**LOM Total / Avg.**
NPV (5%) (CAD$M)	$423
IRR (%)	25.2%
Payback (years)	4.0

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.	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
Production Summary																			
Resource Sent to Mill	kt	60,356	--	3,400	4,000	4,000	4,000	4,000	4,000	4,000	4,000	4,000	4,000	4,000	4,000	4,000	4,000	4,000	957
Head Grade (Au Diluted)	g/t	0.97	--	1.23	1.09	1.07	1.08	1.05	0.96	1.05	1.20	1.34	1.25	0.94	0.94	0.74	0.37	0.37	0.37
Gold Recovered	koz	1,757	--	126.2	131.6	128.6	130.0	126.6	115.8	126.1	144.5	161.0	150.5	112.8	112.9	89.7	44.8	44.8	10.7
Gold Payable	koz	1,756	--	126.2	131.6	128.6	130.0	126.5	115.7	126.1	144.5	160.9	150.4	112.8	112.9	89.6	44.8	44.8	10.7
Revenue																			
Gold Price	US$/oz	$1,450	$1,450	$1,450	$1,450	$1,450	$1,450	$1,450	$1,450	$1,450	$1,450	$1,450	$1,450	$1,450	$1,450	$1,450	$1,450	$1,450	$1,450
Exchange Rate	US$:CAD$	0.74	0.74	0.74	0.74	0.74	0.74	0.74	0.74	0.74	0.74	0.74	0.74	0.74	0.74	0.74	0.74	0.74	0.74
Gross Revenue	CAD$M	$3,441	--	$247	$258	$252	$255	$248	$227	$247	$283	$315	$295	$221	$221	$176	$88	$88	$21
Operating Costs																			
Mine Operating Costs Inc Rehandling Costs	CAD$M	$1,083	--	$87	$103	$122	$101	$109	$124	$93	$95	$79	$55	$47	$39	$16	$5	$5	$2
Mill Processing Inc. Water Treatment Costs	CAD$M	$577	--	$33	$38	$38	$38	$38	$38	$38	$38	$38	$38	$38	$38	$38	$38	$38	$16
G&A Costs	CAD$M	$43	--	$3	$3	$3	$3	$3	$3	$3	$3	$3	$3	$3	$3	$3	$3	$3	$2
Refining and Royalties																			
Refining	CAD$M	$4	--	$0.3	$0.3	$0.3	$0.3	$0.3	$0.3	$0.3	$0.4	$0.4	$0.4	$0.3	$0.3	$0.2	$0.1	$0.1	$0.0
Royalties	CAD$M	$52	--	$4	$4	$4	$4	$4	$3	$4	$4	$5	$4	$3	$3	$3	$1	$1	$0
Capital Expenditures																			
Initial Capital	CAD$M	$256	$256	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Sustaining Capital	CAD$M	$189	--	$41	$36	$36	$31	$20	$3	$3	$3	$3	$3	$3	$2	$3	$2	$0	$0
Closure Cost	CAD$M	$10	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	$10
Salvage Value	CAD$M	$7	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	$7
Pre-Tax Unlevered Free Cash Flow																			
Pre-Tax Unlevered Free Cash Flow	CAD$M	$1,234	($256)	$78	$74	$50	$79	$75	$56	$107	$140	$187	$191	$127	$136	$114	$39	$40	($3)
Pre-Tax Cumulative Unlevered Free Cash Flow	CAD$M		($256)	($178)	($103)	($54)	$25	$100	$156	$263	$403	$590	$781	$908	$1,044	$1,159	$1,197	$1,238	$1,234
Taxes																			
Unlevered Cash Taxes	CAD$M	$484	--	--	$3	$4	$20	$23	$17	$38	$53	$74	$77	$50	$54	$45	$12	$14	--
Post-Tax Unlevered Free Cash Flow																			
Post-Tax Unlevered Free Cash Flow	CAD$M	$750	($256)	$78	$72	$46	$58	$52	$39	$69	$88	$113	$114	$78	$82	$69	$27	$26	($3)
Post-Tax Cumulative Unlevered Free Cash Flow	CAD$M		($256)	($178)	($106)	($60)	($2)	$50	$89	$158	$245	$359	$473	$550	$632	$701	$728	$754	$750

Source: Ausenco, 2020

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, January 01, 2022 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 NPV, IRR and Payback of the Project, using the following variables: metal price, foreign exchange rate, discount rate, grade, recovery, total capital costs, and operating costs. Table 22-3 summarizes the sensitivity analysis results. Table 22-4 shows the pre-tax sensitivity analysis findings, and Table 22-5 shows the results post-tax.

Analysis revealed that the Project is most sensitive to changes in metal prices, foreign exchange rate, head grade and recovery, then, to a lesser extent, to operating costs and capital costs.

Table 22-3: Gold Price Sensitivity Summary

	Gold Price US$/oz	$1,200	$1,350	$1,450	$1,750	$1,950
Pre-Tax	**NPV (5%)**	C$318 M	C$556 M	C$715 M	C$1,191 M	C$1,508 M
	IRR	16.9%	25.6%	31.5%	49.9%	62.6%
	Payback	7.3 yrs	4.8 yrs	3.7 yrs	2.0 yrs	1.6 yrs
Post-Tax	**NPV (5%)**	C$174 M	C$327 M	C$423 M	C$707 M	C$892 M
	IRR	13.1%	20.5%	25.2%	38.5%	47.2%
	Payback	7.5 yrs	5.1 yrs	4.0 yrs	2.5 yrs	1.9 yrs

Table 22-4: Pre-Tax Sensitivity Analysis

Pre-Tax NPV Sensitivity To Discount Rate

Discount Rate \ Gold Price (US$/oz)	$715	$1,200	$1,350	$1,450	$1,750	$1,950
1.0%		$567	$890	$1,105	$1,751	$2,182
3.0%		$428	$704	$888	$1,440	$1,808
5.0%		$318	$556	$715	$1,191	$1,508
8.0%		$195	$389	$518	$905	$1,163
10.0%		$134	$304	$417	$757	$984

Pre-Tax IRR Sensitivity To Discount Rate

Discount Rate \ Gold Price (US$/oz)	$1,200	$1,350	$1,450	$1,750	$1,950
1.0%	16.9%	25.6%	31.5%	49.9%	62.6%
3.0%	16.9%	25.6%	31.5%	49.9%	62.6%
5.0%	16.9%	25.6%	31.5%	49.9%	62.6%
8.0%	16.9%	25.6%	31.5%	49.9%	62.6%
10.0%	16.9%	25.6%	31.5%	49.9%	62.6%

Pre-Tax IRR Sensitivity To FX

FX \ Gold Price (US$/oz)	$715	$1,200	$1,350	$1,450	$1,750	$1,950
(20.0%)		($62)	$128	$255	$636	$890
(10.0%)		$128	$342	$485	$913	$1,199
--		$318	$556	$715	$1,191	$1,508
10.0%		$509	$771	$945	$1,469	$1,818
20.0%		$699	$985	$1,175	$1,746	$2,127

FX \ Gold Price (US$/oz)	$1,200	$1,350	$1,450	$1,750	$1,950
(20.0%)	2.5%	9.9%	14.6%	28.5%	38.1%
(10.0%)	9.9%	17.7%	23.0%	39.0%	50.2%
--	16.9%	25.6%	31.5%	49.9%	62.6%
10.0%	23.8%	33.6%	40.3%	61.0%	75.3%
20.0%	30.9%	41.8%	49.3%	72.4%	88.1%

Pre-Tax IRR Sensitivity To Opex

Opex \ Gold Price (US$/oz)	$715	$1,200	$1,350	$1,450	$1,750	$1,950
(20.0%)		$555	$793	$952	$1,428	$1,745
(10.0%)		$437	$675	$834	$1,310	$1,627
--		$318	$556	$715	$1,191	$1,508
10.0%		$200	$438	$596	$1,072	$1,390
20.0%		$81	$319	$478	$954	$1,271

Opex \ Gold Price (US$/oz)	$1,200	$1,350	$1,450	$1,750	$1,950
(20.0%)	26.2%	35.2%	41.3%	60.2%	73.1%
(10.0%)	21.5%	30.3%	36.3%	55.0%	67.9%
--	16.9%	25.6%	31.5%	49.9%	62.6%
10.0%	12.4%	21.0%	26.8%	44.8%	57.4%
20.0%	8.0%	16.5%	22.2%	39.8%	52.3%

Pre-Tax IRR Sensitivity To Capex

Total Capex \ Gold Price (US$/oz)	$715	$1,200	$1,350	$1,450	$1,750	$1,950
(20.0%)		$397	$636	$794	$1,270	$1,588
(10.0%)		$358	$596	$755	$1,231	$1,548
--		$318	$556	$715	$1,191	$1,508
10.0%		$279	$517	$675	$1,151	$1,469
20.0%		$239	$477	$636	$1,112	$1,429

Total Capex \ Gold Price (US$/oz)	$1,200	$1,350	$1,450	$1,750	$1,950
(20.0%)	22.7%	33.6%	41.2%	64.9%	81.3%
(10.0%)	19.5%	29.2%	35.8%	56.5%	70.9%
--	16.9%	25.6%	31.5%	49.9%	62.6%
10.0%	14.7%	22.7%	28.0%	44.6%	56.0%
20.0%	12.8%	20.2%	25.1%	40.2%	50.5%

Pre-Tax IRR Sensitivity To Recovery Mill

Recovery Mill \ Gold Price (US$/oz)	$715	$1,200	$1,350	$1,450	$1,750	$1,950
91.7%		$278	$511	$666	$1,132	$1,442
92.7%		$298	$534	$690	$1,161	$1,475
93.7%		$318	$556	$715	$1,191	$1,508
94.7%		$339	$579	$740	$1,221	$1,541
95.7%		$359	$602	$764	$1,250	$1,574

Recovery Mill \ Gold Price (US$/oz)	$1,200	$1,350	$1,450	$1,750	$1,950
91.7%	15.4%	23.9%	29.7%	47.5%	60.0%
92.7%	16.1%	24.8%	30.6%	48.7%	61.3%
93.7%	16.9%	25.6%	31.5%	49.9%	62.6%
94.7%	17.6%	26.4%	32.4%	51.1%	64.0%
95.7%	18.4%	27.3%	33.3%	52.2%	65.3%

Pre-Tax IRR Sensitivity To Head Grade

Head Grade \ Gold Price (US$/oz)	$715	$1,200	$1,350	$1,450	$1,750	$1,950
(10.0%)		$128	$342	$485	$914	$1,199
(5.0%)		$223	$449	$600	$1,052	$1,354
--		$318	$556	$715	$1,191	$1,508
5.0%		$413	$663	$830	$1,330	$1,663
10.0%		$508	$770	$945	$1,468	$1,817

Head Grade \ Gold Price (US$/oz)	$1,200	$1,350	$1,450	$1,750	$1,950
(10.0%)	9.9%	17.8%	23.0%	39.0%	50.2%
(5.0%)	13.4%	21.7%	27.2%	44.4%	56.4%
--	16.9%	25.6%	31.5%	49.9%	62.6%
5.0%	20.3%	29.6%	35.8%	55.4%	68.9%
10.0%	23.8%	33.6%	40.2%	61.0%	75.3%

Table 22-5: Post-Tax Sensitivity Analysis

Post-Tax NPV Sensitivity To Discount Rate

Discount Rate	Gold Price (US$/oz)					
	$423	$1,200	$1,350	$1,450	$1,750	$1,950
1.0%		$341	$541	$669	$1,052	$1,303
3.0%		$248	$421	$532	$860	$1,075
5.0%		$174	$327	$423	$707	$892
8.0%		$91	$218	$297	$530	$681
10.0%		$49	$163	$233	$438	$572

Post-Tax IRR Sensitivity To Discount Rate

Discount Rate	Gold Price (US$/oz)					
		$1,200	$1,350	$1,450	$1,750	$1,950
1.0%		13.1%	20.5%	25.2%	38.5%	47.2%
3.0%		13.1%	20.5%	25.2%	38.5%	47.2%
5.0%		13.1%	20.5%	25.2%	38.5%	47.2%
8.0%		13.1%	20.5%	25.2%	38.5%	47.2%
10.0%		13.1%	20.5%	25.2%	38.5%	47.2%

Post-Tax IRR Sensitivity To FX

FX	Gold Price (US$/oz)					
	$423	$1,200	$1,350	$1,450	$1,750	$1,950
(20.0%)		($89)	$44	$132	$375	$528
(10.0%)		$44	$190	$283	$542	$711
--		$174	$327	$423	$707	$892
10.0%		$297	$456	$561	$869	$1,072
20.0%		$413	$584	$697	$1,031	$1,250

Post-Tax IRR Sensitivity To FX

FX	Gold Price (US$/oz)					
		$1,200	$1,350	$1,450	$1,750	$1,950
(20.0%)		1.2%	7.0%	11.0%	22.9%	30.2%
(10.0%)		7.0%	13.8%	18.3%	30.8%	38.7%
--		13.1%	20.5%	25.2%	38.5%	47.2%
10.0%		19.1%	26.8%	31.7%	46.2%	55.5%
20.0%		24.7%	32.8%	38.1%	53.6%	63.7%

Post-Tax IRR Sensitivity To Opex

Opex	Gold Price (US$/oz)					
	$423	$1,200	$1,350	$1,450	$1,750	$1,950
(20.0%)		$326	$469	$564	$842	$1,025
(10.0%)		$253	$398	$494	$775	$959
--		$174	$327	$423	$707	$892
10.0%		$93	$252	$351	$637	$825
20.0%		$8	$173	$277	$567	$756

Post-Tax IRR Sensitivity To Opex

Opex	Gold Price (US$/oz)					
		$1,200	$1,350	$1,450	$1,750	$1,950
(20.0%)		20.9%	27.9%	32.4%	45.5%	53.9%
(10.0%)		17.1%	24.2%	28.8%	42.1%	50.6%
--		13.1%	20.5%	25.2%	38.5%	47.2%
10.0%		9.1%	16.7%	21.5%	35.0%	43.8%
20.0%		5.3%	12.7%	17.7%	31.5%	40.3%

Post-Tax IRR Sensitivity To Capex

Total Capex	Gold Price (US$/oz)					
	$423	$1,200	$1,350	$1,450	$1,750	$1,950
(20.0%)		$254	$406	$502	$786	$972
(10.0%)		$214	$366	$462	$746	$932
--		$174	$327	$423	$707	$892
10.0%		$135	$287	$383	$667	$853
20.0%		$95	$247	$344	$628	$813

Post-Tax IRR Sensitivity To Capex

Total Capex	Gold Price (US$/oz)					
		$1,200	$1,350	$1,450	$1,750	$1,950
(20.0%)		19.2%	28.8%	34.9%	52.3%	63.5%
(10.0%)		15.8%	24.2%	29.4%	44.6%	54.4%
--		13.1%	20.5%	25.2%	38.5%	47.2%
10.0%		10.8%	17.5%	21.7%	33.7%	41.5%
20.0%		8.8%	15.0%	18.8%	29.8%	36.8%

Post-Tax IRR Sensitivity To Recovery Mill

Recovery Mill	Gold Price (US$/oz)					
	$423	$1,200	$1,350	$1,450	$1,750	$1,950
91.7%		$147	$299	$393	$672	$854
92.7%		$161	$313	$408	$689	$873
93.7%		$174	$327	$423	$707	$892
94.7%		$188	$341	$438	$724	$912
95.7%		$201	$355	$452	$742	$931

Post-Tax IRR Sensitivity To Recovery Mill

Recovery Mill	Gold Price (US$/oz)					
		$1,200	$1,350	$1,450	$1,750	$1,950
91.7%		11.8%	19.1%	23.7%	36.9%	45.5%
92.7%		12.4%	19.8%	24.4%	37.7%	46.3%
93.7%		13.1%	20.5%	25.2%	38.5%	47.2%
94.7%		13.7%	21.2%	25.9%	39.4%	48.1%
95.7%		14.4%	21.9%	26.6%	40.2%	49.0%

Post-Tax IRR Sensitivity To Head Grade

Head Grade	Gold Price (US$/oz)					
	$423	$1,200	$1,350	$1,450	$1,750	$1,950
(10.0%)		$44	$190	$283	$542	$712
(5.0%)		$111	$260	$353	$625	$802
--		$174	$327	$423	$707	$892
5.0%		$237	$392	$492	$788	$982
10.0%		$297	$456	$560	$869	$1,072

Post-Tax IRR Sensitivity To Head Grade

Head Grade	Gold Price (US$/oz)					
		$1,200	$1,350	$1,450	$1,750	$1,950
(10.0%)		7.0%	13.8%	18.4%	30.8%	38.8%
(5.0%)		10.1%	17.2%	21.8%	34.7%	43.1%
--		13.1%	20.5%	25.2%	38.5%	47.2%
5.0%		16.1%	23.7%	28.5%	42.4%	51.4%
10.0%		19.1%	26.8%	31.7%	46.2%	55.5%

23 Adjacent Properties

O3 Mining controls additional claims and concessions adjacent to or near the Marban Project including the “Malartic” property, which hosts the Marban Project property, the “Alpha” property and the “East Cadillac” property (Figure 23-1). These properties are not the subject of this report.

Various exploration properties at different stages of exploration and development, which are controlled by various companies, occur within the vicinity of the Marban Project property (see Figure 23-1 and Figure 7.1 for the location of gold deposits in the vicinity of the project area).

Figure 23-1: Adjacent Properties

Source: O3 Mining, 2020

24 Other Relevant Data & Information

There is no other relevant data or information.

25 Interpretation & 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 Project 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 9 million ounces of gold. The Marban Project property is located in line with the past producing Kiena and Camflo mines (production of 1.70 and 1.88 million ounces of gold respectively).

The Marban Project property includes three historic mines (Kierens, Norlartic and Marban) that produced almost 600,000 ounces of gold. The geologic environment of the Marban Project property is similar to other gold-producing areas of the Abitibi greenstone belt.

Gold mineralization at the Marban deposit is also quite unusual in that it has an east-west strike and moderate dips, unlike the more north-westerly trending and steeply dipping mineralization that is typical of other mineralized areas of the property. Gold at the Marban deposit occurs along a series of relatively thin but through going shear zones that lie primarily within a package of basaltic flows. The mineralized shears are deformed into folds that are characterized by subhorizontal fold axes.

A zone of relatively tight folding occurs within an elevation range of between 150 and 250 metres below the surface throughout most of the deposit, where much of the higher-grade mineralization (as well as historic mining) is concentrated. The mineralized shears extend updip beyond this subhorizontal zone of tight folding to the bedrock surface and downdip through to the limits of drilling. Mineralization at Kierens and Norlartic occurs within the northwest-trending Norbenite shear zone and is often closely associated with dikes and related shear zones, quartz veinlets, and sulfides. The dikes, shears, and related mineralization have steep (Norlartic) to subvertical (Kierens) dips to the north.

Significant gold mineralization has been identified at several areas on the property, and a number of styles of mineralization are evident. Gold resources have been estimated at three deposits: Marban, Kierens, and Norlartic.

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 Project. Historic data that has been used for the resource estimate has been validated through extensive statistical studies.

25.4 Metallurgical Testwork

Metallurgical testwork and associated analytical procedures were appropriate to the mineralisation type, appropriate to establish 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 2012 testwork results on composite samples representative of the Marban deposit, doré can be produced at a high recovery of gold.

25.5 Mineral Resource Estimates

The mineral resource estimate has been done using CIM standards and 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 Mine Plan

Reasonable open pit mine plans, mine production schedules, and mine capital and operating costs have been developed for the Marban project PEA.

Pit layouts and mine operations are typical of other regional open pit gold operations, and the unit operations within the developed mine operating plan are proven to be suitable for these other operations.

The mine plan and estimated mine capital and operations costs are reasonable at a scoping level of engineering and support the cash flow model and financials developed for the PEA.

25.7 Recovery Plant

The plant will process material at a rate of 4.0 Mt/a with an average head grade of 0.97 g/t Au to produce doré.

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

Access to the Marban Project is via Route 117. The Project access road is a short all-season gravel road that connects to Route 117 approximately 15 km east of Val-d'Or. Within the site, heavy equipment roads will connect the main pit and waste rock pit to the main facilities and processing areas. Secondary roads will connect the plant to the crusher.

Infrastructure to support the Marban Project will consist of site civil work, site facilities/building, a water system, and site electrical. Site civil work includes designs for the following infrastructure: light vehicle and heavy equipment roads; overburden stripping and stockpiling; mine facility platforms and process facility platforms; contact water pond; TSF; waste storage facilities; and a high voltage substation platform.

The mine facilities will include the administration offices, truckshop and warehouse, tire repair shop, mine workshop, mine dry, and miscellaneous facilities. The process facilities will include the process plant, crusher facility, process plant workshop and assay laboratory. Both the mine facilities and process facilities will be serviced with potable water, fire water, compressed air, power, diesel, communication, and sanitary systems.

The project power will come from the existing power grid.

There will be one main waste storage area, centrally located to the west and south of the mine pits.

The TSF has a preliminary design to accommodate 60.4 Mt (~41.3 Mm^3) of tailings solids that will be produced over a 15 year mine life, based on an annual throughput of 4 Mtpa (11,000 tpd) for the mill. The first seven years of tailings production (approximately 19.3 Mm^3) will be stored in a surface TSF with the remaining 8 years of tailings production (approximately 22.0 Mm^3) stored in the mined out Kierens Pit #4. Water will be reclaimed from the Water Management Pond for re-use in the mill. Water from the TSF will be collected in the water management pond. Excess water in the water management pond will be treated in an effluent treatment plant and discharged to lac Montigny.

In order to allow mining of the Kierens and Norlartic (KN) pits, the Kierens Creek must be diverted. A diversion channel is planned to divert Kierens Creek from its current course along an alignment to the north of the planned KN pits. The channel is designed for a probable maximum flood (PMF) event and must divert runoff from an approximately 22 km^2 of drainage in the upstream catchment area. Water management for the site will be through a combination diversion / collection ditches and ponds.

Pits will be dewatered with conventional dewatering equipment (submersible pumps place in pit bottoms or in underground workings). Preliminary pit inflows of 4,000 m^3 per day are estimated for when the pits will be at their deepest. This is based on the assumptions that there are no major pit inflows from the esker running below the Kierens stream and that the rock mass is not hydraulically conductive. Pit water will be pumped to collection ponds adjacent to the pits, where it will be managed as per the overall site water management plan.

25.9 Environmental, Permitting and Social Considerations

Several environmental baseline studies have been completed, mostly in 2016. As the Project continues to be defined, and in regards 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. Principal impacts anticipated relating to wetlands, fish habitat, air quality, noise/vibrations and water resources. No specific inordinate environmental risk to project development was identified. Although they are some sensitive elements in the proposed layout surroundings, optimization could be made to eliminate or reduce the effect on these components. Note that consultation with stakeholders, specifically First Nations communities, may highlight additional impacts.

Based on the static test results and the new MELCC 2020 guidelines for ore and mine waste characterization, waste rock and ore are classified as leachable but don't have the potential for acidic drainage. Therefore, the management facilities have to comply with Directive 019 groundwater protection measure. A waste rock stockpile and a co-disposal facility for tailings and waste rock management are planned for the Project.

The Marban 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.

No engagement activities were initiated. Both provincial and federal governments recommend that project initiators engage as soon as possible in a process of information and consultation with local and indigenous communities.

Closure costs for the Marban Project are evaluated at approximately $12.2M, including direct and indirect costs, and 20% contingency. A cost-benefit analysis for backfilling the pit will have to be included in the closure plan.

25.10 Markets and Contracts

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. 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.

25.11 Capital Cost Estimates

The capital cost estimate is presented at a ±50% accuracy, using a base date of Q2, 2020, and an exchange rate assumption of US$0.74:C$1.00.

Capital costs are estimated at $256M of initial capital, $189 M of sustaining capital, for an overall capital cost estimate of $455M, including closure costs.

25.12 Operating Cost Estimates

The operating cost estimate is presented at a ±50% accuracy, using a base date of Q2, 2020, and an exchange rate assumption of US$0.74:C$1.00.

Operating costs are estimated at $112 M/a, or $28.20/t processed.

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 based on a 5% discount rate.

The analysis uses the following key inputs:

- Construction period of one year;

- Mine life of 15.2 years;
- Base case gold price of US$1,450/oz Au
- 100% ownership with 1.5% NSR
- The mine is scheduled to deliver 60.4 Mt of mill feed grading 0.97 g/t Au.

The pre-tax net present value discounted at 5% (NPV5%) is C$715M, the internal rate of return IRR is 31.5%, and payback is 3.7 years. On a post-tax basis, the NPV5% is C$423 M, the IRR is 25.2%, and the payback period is 4.0 years.

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

25.14 Risks and Opportunities

25.14.1 Risks

25.14.1.1 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
- Historic underground openings location

25.14.1.2 Mining

Risks to the PEA defined are mill feed quantities gold grades and associated open pit strip ratio include changes to the following factors and assumptions:

- Metal Prices
- Interpretations of Mineralization geometry and continuity in mineralized zones
- Exact dimensions of underground voids created by historic mining
- Geotechnical and hydrogeological assumptions
- Ability of the mining operation to meet the annual production rate and anticipated grade control standards
- Operating cost assumptions
- Mine operation and process plant recoveries

- Ability to meet and maintain permitting and environmental licence conditions, and the ability to maintain the social license to operate. Specific concern exists regarding the timing to permit the diversion of existing water flow above the KN deposits.

25.14.1.3 Processing

Cyanide detoxification reagent consumption rates are assumed based on benchmarks due to lack of testwork.

Slurry rheology is assumed based on benchmarks due to lack of testwork

25.14.2 Environment and Permitting

Table 25-1: Environmental Risk Assessment

Risk Description and Potential Impact		Mitigation Approach
1.	Project delayed due to changes in regulations/government representatives as a result of elections: Project delayed.	Stay alert to the changes in regulatory regimes.
2.	Inadequate design prompts request for major modifications from one or both levels of government: Project delayed, increased design costs.	Include technologies and practices widely accepted in the mining industry. Share project information with governments early during the project development. Include technologies and practices well accepted in the mining industry.
3.	Project cannot proceed if not accepted by local communities due to lack of communication: Project delayed.	Keep regular contact with stakeholders and develop and implement a consultation plan.
4.	Authorizations and permits from both levels of government could take longer than expected: Project delayed.	Develop and maintain a realistic schedule, keep close contact with the authorities throughout the process.
5.	Encroachment in fish habitat by mine waste disposal: Alternative assessment (Schedule 2) – Federal requirement leading to major delay (1-year minimum).	Proceed with fish habitat validation in the TSF and WMP sector. Confirm with federal authorities that using the pits for waste rock disposal after the Kierens creek diversion will not trigger Schedule 2.
6.	Only static tests were conducted on waste rock and ore samples, which may underestimate/overestimate the metal leaching and acid generation potential: Impact on management and closure approaches.	Proceed with kinetic tests. Reassess geochemical behaviour at every stage of the Project.
7.	At this stage, a limited number of ore samples (4) have been analyzed for geochemical characterization, and these are not sufficient to be representative of the overall mineralized zone: Impact on management.	Continue the geochemical characterization on additional ore samples to ensure representativeness.
8.	No tailings geochemical properties available: Impact on management.	Carry out a geochemical characterization on representative tailings samples.

Risk Description and Potential Impact	Mitigation Approach
9. Weathering and metal leaching rates may be underestimated during geochemical characterization; metals may also behave in unexpected ways: Impact on planned water treatment approaches.	Reassess geochemical behaviour at every stage of the Project.
10. Delay of mining lease issue due to delays in getting the closure plan approval: Project delayed.	Develop a detailed closure plan during feasibility stage with proper mine layout and sufficient geochemistry data on ore, waste rock and tailings.
11. Some soils within the proposed layout might be contaminated. Increased costs.	Evaluate the need for additional soil characterization on potentially contaminated lands to evaluate the environmental liability.

25.14.3 Opportunities

25.14.3.1 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.

25.14.3.2 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.

Infill drilling at each of the deposits could upgrade the Classification from Inferred resources to provide additional Measured and Indicated resources.

Continued geologic modelling and adhering to the QAQC procedures outlined herein will increase both the extent and the confidence in the resource modelling.

25.14.3.3 Mining

With detailed metallurgical testwork information on lithologies and zones, the mining sequence may be altered to provide higher value initially.

There is potential for improved slope design, when additional geotechnical data such as waste rock strength are available from drill testing.

25.14.3.4 Environment and Permitting

Table 25-2: Environmental Opportunity Assessment

Opportunity Explanation		Benefit
1.	Initiating geochemical tests on tailings and pursuing kinetic tests on waste rock and ore would provide a better understanding of metal leaching and acid generation potential.	Understanding of mine waste behaviour under conditions that are relevant to natural environments would help define appropriate management and restoration options and lower the closure costs.
2.	Development of a water quality model will improve the understanding of constituents that need to be treated, as well as their expected concentrations.	The model outcome would support adequate water treatment requirements.
3.	Making more land available outside the existing O3 Mining claims, may prevent encroaching fish habitat and avoid or minimize impact on wetlands.	Reduction in compensation costs and permitting effort.
4.	Multiplying engagement activities and consultation effort to keep the stakeholders, First Nations and local community involved in the Project development.	Facilitate social acceptability.

25.15 Interpretation and Conclusions

The total Measured and Indicated Mineral Resource is estimated at 54.1 Mt at a grade of 1.10 g/t Au for a total of 1.9 Moz. Additional Inferred Mineral Resources represent 24.2% of the total ounces when combined with the Measured and Indicated Mineral Resource.

Based on the assumptions and parameters presented in this Report, the PEA shows positive economics (i.e. C$423 M post-tax NPV(5%) and 25.2% post-tax IRR). The PEA supports that additional detailed studies are warranted.

26 Recommendations

26.1 Introduction

The recommended work is divided into two main initiatives. The first initiative will consist of drilling; additional metallurgical testwork; materials handling tests; mine geotechnical data collection, data reviews in support of geotechnical and hydrological assumptions; additional hydrological data gathering; water treatment testwork; review of cost assumptions for grade control; additional mine studies, reviews of available climate data; collection of additional climate-related information, and geotechnical data collection in support of infrastructure locations and designs, and data collection on potential borrow pit sources.

The second initiative will consist of project environmental, permitting, and social de-risking activities.

26.2 Drilling

The present PEA considers production from the Marban, Norlartic (South and North), Kierens, North-North and Gold Hawk deposits. The extensions of theses ore deposits offer significant opportunity to grow the mineral resource base, where many significant historical intercepts remain open and warrant follow-up drilling. For example, the high-grade gold veins at Gold Hawk have been outlined by many different drilling programs since the 1980s. Drilling at 30-metre spacing and to a vertical depth of 400 metres identified three veins or vein systems named Vein #1, #2, and #3, remains open at depth.

Additionally, the Marbenite and Norbenite shears host several other mineralized zones geologically similar to the typical Malartic District vein type that offer potential to increase the resource, namely Orion 8, MK, Malartic Hygrade, Malartic H and Malartic NE. Past production (U/G) at Malartic Hygrade was 28,000t at 19.6g/t Au (Sigeom). There is a non 43-101 compliant resource at Orion 8 of 205,000t at 8.4g/t Au (Sigeom). Geophysical data suggests a 1.1 kilometre stretch of the Norbenite shear, which hosts the North Zone mineralization, exists at Marban NE. Many significant historical intercepts in these zones, including high-grade, remain open and warrant follow up drilling. All of these zones are located within 3 kilometres of the open pits considered in the present PEA.

Table 26-1 below summarizes the expected drill expenditures for the next two phases of drilling.

Table 26-1: Drilling Budget

Phase	Description	Metreage (m)	Budget (C$M)
1	Exploration of known mineralization	40,000	5.6
2	Infill drilling to upgrade Inferred to Indicated	20,000	2.8

26.3 Sampling and QAQC

Prior to future reporting of higher significance, it is recommended that:

- Re-assaying of pre-1980 pulps where available should be done in order to provide significant additional data at lower cost than drilling to upgrade the Classification.

- Commercially available and certified blank material be sourced and used for QAQC in future drilling.
- Re-assays of the 10 samples before and after the failed blanks be run with newly acquired blank material inserted in the stream.
- All future drilling and QAQC to include CRMs of lower grades, such as 0.15 g/t, 0.25 and 0.35 g/t Au.
- All future QAQC employ the definition of a failed CRM at the +/- 3 SD level, or two consecutive samples in the warning zone of +/-2 SD.
- Re-assays are accomplished on failed samples at the +/- 3 SD level, to include the 10 samples before and after the fail, with particular attention to Marban assays in 2006 and 2007. Insertion of newly acquired CRM material is advisable.
- Duplicate core samples in Kierens-Norlartic be resampled to 5% so that the possibility that the original core assays were selectively sampled can be excluded.
- The Kierens-Norlartic QAQC database be augmented with external laboratory duplicates from drilling after 2007.
- Additional specific gravity measurements in all lithologies should be done at both deposit areas to increase the confidence of the SG.

26.4 Mining

The following recommendations are made with regards to advancing the mine engineering of the Marban project to a Pre-Feasibility Study:

- Targeted open pit geotechnical drilling and detailed geotechnical logging of some exploration drillholes, review of the geological models and review of historical geotechnical and underground geological mapping. Holes should be drilled both parallel and perpendicular to foliation to eliminate blind zones in the structural data. The following is recommended:
    - Marban deposit: 6 drillholes, approximately 300 m – 400 m length.
    - Norlartic deposit: 4 drillholes, approximately 150 m – 200 m length.
    - Kierens deposit: 4 drillholes, approximately 100 m length.
- Gather structural fabric data (foliation and major joint sets orientation) to assess variability. Using the structural fabric data, limit equilibrium analyses of potential planar, wedge and toppling failures should be conducted to determine bench, inter-ramp and overall slope design.
- Lithological and/or alteration interpretation to identify schistosic zones (geotechnically poorer ground). Develop a 3D geotechnical interpretation of the Talc-Chlorite-Schist (TCS) bands and develop a more precise pit slope domaining relative to the TCS unit.

- Laboratory testing for intact rock strength (unconfined compressive strength tests and indirect tensile strength tests) and for discontinuity strength (direct shear tests). It is recommended that the ISRM intact rock strength be logged in exploration holes.

- Crown pillar analysis for open pit mining over historic underground openings. Specific stability assessments should be done where historical openings are planned to be located behind interim or final pit walls or below pit floor.

- Overburden investigation for pit slope assessment.

- Packer testing should be conducted to determine pit hydrogeology, hydraulic conductivity and refine pit water inflow estimates.

- Further hydrogeological and hydrological characterization are required in the Kierens-Norlartic area where the pits will be mined beneath the Kierens stream.

- Condemnation drilling of the footprints identified for the waste rock and overburden storage facilities, as well as site infrastructure.

- Updates to designs of open pits, stockpiles, and mine haul roads incorporating results from all other recommended work programs.

- Mine operational trade-off studies examining contractor vs. owner equipment fleet, lease vs. purchase equipment fleet, cost comparisons of various equipment class sizes, and utilization of electrical driven mine equipment over diesel driven units.

A budget of $1.5M is estimated for the above work programs and studies.

26.5 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 leach kinetics and contaminant correlations.

Additional comminution tests (e.g. crushing work index, rod work index, SMC 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.

An extended gravity-recoverable gold test should be conducted on a master composite sample to confirm the PEA flowsheet.

A tailings cyanide detoxification test is recommended. A tailings dewatering test and a concentrate dewatering test are recommended. Rheology of the tailings slurry stream should be conducted on the combined composite sample, without any prior treatment that would affect the rheology.

26.6 Material Handling

Material handling test work is recommended for design of bins, chutes, conveyors and stockpile drawdown

26.7 Hydrological

Hydrogeological testing (packer testing, profile tracer testing) and instrumentation (i.e. piezometers) should be installed in select holes to provide basic data for groundwater modelling and excavation dewatering/depressurization simulations.

26.8 Water Treatment

After the site-wide water balanced has been further evaluated and ML/ARD parameters are better understood, water treatment testwork should be conducted to confirm that impurities can be removed from water prior to discharge to the environment.

26.9 Infrastructure Geotechnical, Construction Borrow Materials, and Hydrological

Field mapping, geotechnical sample collection (boreholes, tests pits) and laboratory studies should be conducted to identify borrow material sources for construction activities, and provide information for support of the WRS, OBS, plant site, ancillary facilities locations and the TSF design.

26.10 Tailings Storage Facility

The following activities are recommended to support the design of the TSF for the PFS:

- Fish sampling in the proposed tailings and water management areas should be conducted to confirm fish presence/absence in the waterbodies of interest that may be impacted by the proposed development.
- Condemnation drilling for the selected tailings management site to verify the absence of mineralization.
- Geotechnical site investigations at the preferred site to characterize the foundation conditions and potential borrow sources.
- In-situ permeability tests of the overburden soils and bedrock.
- Tailings assessment to help characterize the geotechnical properties in order to understand the deposition characteristics.
- Refinement and further study of the proposed TSF embankment and water management pond dam alignment and configuration.
- Further evaluation of the water management pond dam requirements such as allowable seepage gradients and management of seepage water.
- Formal dam breach analysis along with dam hazard classification in accordance with Canada Dam association (CDA, 2013) Dam Safety Guidelines.

26.11 Environment and Permitting

In the next stages of the project, it is recommended to continue the environmental baseline studies and environmental assessment as required to better define the environmental and social impacts. Mainly, the following activities should be undertaken:

- Undertake environmental baseline studies to cover the TSF and WMP sector;
- Complete environmental baseline studies of the main sector;
- Initiate public and First Nations engagement activities to ensure that their expectations and concerns are addressed throughout the Project development;
- Initiate government involvement to ensure that their expectations and concerns are addressed throughout the Project development;
- Initiate Kierens Creek diversion conceptual engineering;
- Initiate provincial and federal environmental assessment processes;
- Proceed with kinetic testing on sufficient samples of waste rock and ore;
- Initiate an exhaustive geochemical characterization on tailings as soon as samples are made available

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