Technical Report

Lamaque Project

Québec, Canada

Project Location:

Bourlamaque Township, Province of Québec, Canada

(NTS: 32C/04)

(UTM: 293960E, 5329260N)

(NAD 83, Zone 18)

Effective Date: March 21st, 2018
Signature Date: March 29th, 2018

Qualified Person	Company
Mr. Colm Keogh, P.Eng.	Eldorado Gold Corporation
Mr. Jacques Simoneau, P.Geo.	Eldorado Gold Corporation
Dr. Stephen Juras, Ph.D., P.Geo.	Eldorado Gold Corporation
Ms. Marianne Utiger, P.Eng.	WSP Canada inc.
Mr. François Chabot, P.Eng.	Eldorado Gold Corporation.

TABLE OF CONTENTS

LIST OF FIGURES

LIST OF TABLES

GLOSSARY

Units of Measure

Amphibole	Amf
Annum (year)	a
Billion	B
Biotite	Bt
Centimeter	cm
Cubic centimeter	cm^3
Cubic meter	m^3
Day	d
Days per year (annum)	d/a
Degree	°
Degrees Celsius	°C
Dollar (American)	US$
Dollar (Canadian)	CAN$
Euro	€
Feldspar	Fp
Gallon	gal
Gram	g
Grams per tonne	g/t
Greater than	>
Hectare (10,000 m^2)	ha
Horse Power	hp
Hour	h
Kilo (thousand)	k
Kilogram	kg
Kilograms per cubic meter	kg/m^3
Kilograms per hour	kg/h
Kilograms per square meter	kg/m^2
Kilograms per tonne	kg/t
Kilometer	km
Kilometers per hour	km/h
Kilopascal	kPa
Kilopascal (absolute)	kPa(a)
Kilopascal (gauge)	kPa(g)
Kilotonne	kt
Kilovolt	kV
Kilowatt hour	kWh
Kilowatt hours per year	kWh/a
Kilowatt	kW
Less than	<

Litre	L
Megawatt	MW
Meter	m
Meter above Sea Level	masl
Metric ton (tonne)	t
Microns	µm
Milligram	mg
Milligrams per litre	mg/L
Millilitre	mL
Millimeter	mm
Million cubic meters	Mm^3
Million ounces	Moz
Million tonnes per Annum	Mtpa
Million tonnes	Mt
Million	M
Minute (time)	min
Month	mo
Newton	N
Ounce	oz
Parts per million	ppm
Pascal	Pa
Percent	%
Percent by Weight	wt%
Pound	lb
Quartz	Qz
Second (time)	sec
Specific gravity	SG
Square centimeter	cm^2
Square kilometer	km^2
Square meter	m^2
Thousand tonnes	kt
Three Dimensional	3D
Tonnes per day	t/d or tpd
Tonnes per hour	tph
Tonnes per year	tpa
Volt	V
Watt	W
Week	wk
Weight/weight	w/w

Abbreviations and Acronyms

Acid Base Accounting	ABA
Acid Neutralizing Capacity	ANC
Acidity or Alkalinity	pH

Aluminum	Al
Ammonium Nitrate/Fuel Oil	ANFO
Analytical Detection Limit	ADL
Antimony	Sb
Argon	Ar
Arsenic	As
Association for the Advancement of Cost Engineering	AACE
Atomic Adsorption	AA
Backscattered Electron Imaging	BSE
Barium	Ba
Before Christ	BC
Biochemical Oxygen Demand	BOD
Cadmium	Cd
Carbon-in-leach	CIL
Canadian Institute of Mining, Metallurgy, and Petroleum	CIM
Closure Capping Stockpile	CCS
Coefficient of Variance	CV
Counter-current decantation	CCD
Copper	Cu
Cyanide Amenability	CNA
Distributed Control System	DCS
East	E
Effective Grinding Length	EGL
Eighty percent (80%) passing particle size, feed	F80
Eighty percent (80%) passing particle size, product	P80
Eldorado Gold Corporation	Eldorado
Engineering, Procurement, Construction Management	EPCM
Environmental Impact Assessment	EIA
Environmental Impact Statement	EIS
Environmental Management Plan	EMP
European Goldfields Limited	EGU
European Union	EU
Feasibility Study	FS
Flocculant	FLOC
Fresh Air Raise	FAR
Front End Loader	FEL
Geological Strength Index	GSI
Ground-Engaging Tools	GET
Gold	Au
Gold Equivalent	Au_Equiv
Gravity Recoverable Gold	GRG
Hellas Gold	Hellas
High Density Polyethylene	HDPE
High Grade	HG

In Pit Tailings Management Facility	IPTMF
Inductively Coupled Plasma	ICP
Inflow Design Flood	IDF
Inner Diameter	ID
Integrated Waste Management Facility	IWMF
Intermediate Bulk Container	IBC
Internal Return Raise	IRAR
Internal Rate of Return	IRR
International Organization for Standardization	ISO
Iron	Fe
Joint Management Decision	JMD
KL	Karatza Lakkos
Lead	Pb
Lerchs-Grossman	L-G
Life-of-mine	LOM
Liquefied Propane Gas	LPG
Load Haul Dumps	LHD
Low Grade	LG
Manganese	Mn
Measured & Indicated	M&I
Million Years	Ma
National Instrument 43-101	NI 43-101
Natural Gas	NG
Nearest Neighbour	NN
Net Present Value	NPV
Net Smelter Return	NSR
Nickel	Ni
North	N
North East	NE
North Waste Rock Dump	NWRD
North West	NW
Ordinary Kriging	OK
Outer Diameter	OD
Oxide Ore Stockpile	OOS
Potassium	K
Potassium Amyl Xanthate	PAX
Prefeasibility Study	PFS
Pressure Oxidation	POX
Probability Assisted Constrained Kriging	PACK
Programmable Logic Controllers	PLCs
Qualified Person(s)	QP(s)
Quality assurance	QA
Quality control	QC
Remote Mining Technology	RMT

Return Air Raise	RAR
Reverse Circulation	RC
Rock Mass Rating	RMR
Rock Quality Designation	RQD
Run of Mine	ROM
Scanning Electron Microscope	SEM
Secondary Contact Water Pond	SCWP
Selective Mining Unit	SMU
Selenium	Se
Semi-autogenous Grinding	SAG
Silicon	Si
Silver	Ag
Site Wide Water Balance	SWWB
Site Wide Water Management	SWWM
South	S
South East	SE
South Water Management Pond	SWMP
South West	SW
Sale of Equipment	SOE
Sale of Gas	SOG
Specific Gravity	SG
Standard Reference Material	SRM
Static Secondary Ion Mass Spectrometry	SSIMS
Strategic Environmental Assessment	SEA
Strontium	Sn
Sub Level Caving	SLC
Sub Level Open Stoping	SLOS
Sulphide	S^{2-}
Sulphuric Acid	H_2SO_4
Tailings Management Facility	TMF
Technical Study	TS
Tellurium	Te
Total dissolved Solids	TDS
Total Suspended Solids	TSS
TVX Gold Inc.	TVX
Underground	UG
Universal Transverse Mercador	UTM
Uranium	U
Value Added Tax	VAT
Variable Speed Drive	VSD
Waste Material Transfer Area	WMTA
Water Management Pond	WMP
Water Treatment Plant	WTP
West	W

Work Breakdown Structure	WBS
Yitrium	Y
Ytterbium	Yb
Zinc	Zn

SECTION • 1 EXECUTIVE SUMMARY

1.1 INTRODUCTION

Eldorado Gold Corporation (Eldorado), an international gold mining company based in Vancouver, British Columbia, owns the Lamaque Project in Quebec, Canada. Eldorado and InnovExplo have prepared this technical report to report the results of a Preliminary Feasibility Study in support of declaring maiden mineral reserves for the Lamaque Project (the "Project").

Geological information and data for this report were obtained from the Lamaque Project. Metallurgical tests were completed by third party laboratories to support calculations and design of the proposed process flowsheet. The underground mine designs and mining methods, tailings management and water management were designed from first principles to a pre-feasibility level.

When preparing reserves for any of its projects, Eldorado uses a consistent prevailing gold price methodology that is in line with the 2015 CIM Guidance on Commodity Pricing used in Resource and Reserve Estimation and Reporting. These are the lesser of the three-year moving average and the current spot price. These were set as of September 2017 for Eldorado's current mineral reserve work. For gold, these are US$1,200/oz Au. All cut-off grade determinations, mine designs and economic tests of economic extraction used these pricing for the Lamaque Project and the mineral reserves work discussed in this technical report. In order to demonstrate the potential economics of a project, Eldorado may elect to use metal pricing closer to the current prevailing spot price and then provide some sensitivity around this price (for the Lamaque Project, metal prices used for this evaluation were US$1,300/oz Au). This analysis generally provides a better 'snapshot' of the project value at prevailing prices rather than limiting it to reserve prices that might vary somewhat from prevailing spot prices. Eldorado stresses that only material that satisfies the mineral reserve criteria is subjected to further economic assessments at varied metal pricing.

Third party experts ("Experts") have supplied some information that was used for the development of the study. The qualified persons have reasonable confidence in the information provided by the following third party consultants, including InnovExplo (for report coordination and peer review), Wood Plc. of Dorval, Quebec (for tailings management), BBA Inc of Montreal, Quebec (for project management), WSP of Montreal and Val D'or, Quebec (for metallurgical testwork and process plant capital and operating costs) and Golder Associates Ltd. of Mississauga, Ontario and Val d'Or, Quebec (for mine geotechnical and rock dump design and parameters).

1.2 CONTRIBUTORS AND QUALIFIED PERSONS

The qualified persons (QPs) responsible for preparing this technical report as defined in National Instrument 43-101 (NI 43-101), Standards of Disclosure for Mineral Projects and in compliance with 43-101F1 (the "Technical Report") are:

- Mr. Colm Keogh, P.Eng., Eldorado Gold author of items 2, 3, 15, 16, 18 to 22 and 24; co-author of items 1 and 25 to 27;
- Mr. Jacques Simoneau, P.Geo., Eldorado Gold author of items 4 to 11 and 23; co-author of items 1 and 25 to 27;

- Dr. Stephen Juras, P. Geo., Eldorado Gold author of items 12 and 14; co-author of items 1 and 25 to 27;
- Ms. Marianne Utiger, P.Eng., WSP Canada Inc. author of items 13 and 17; co-author of items 1 and 25 to 27;
- Mr. François Chabot, P.Eng., Eldorado Gold author of item 20; co-author of 1 and 25 to 27.

1.3 PROPERTY DESCRIPTION AND OWNERSHIP

The Lamaque Project is situated near the city of Val-d'Or in the province of Québec, Canada, approximately 550 km northwest of Montréal. The coordinates for the approximate center of the host of the mineral reserves, the Triangle deposit, are latitude 48°4'38'' N and longitude 77°44'4'' W. According to the Canadian National Topographical System (NTS), the project is situated on map sheets 32C/04 and 32C/03, between UTM coordinates 295,700mE and 296,900mE, and between 5,328,200mN and 5,329,350mN (NAD83 projection, Zone 18N). Figure 1.1 shows the location of the Lamaque Project.

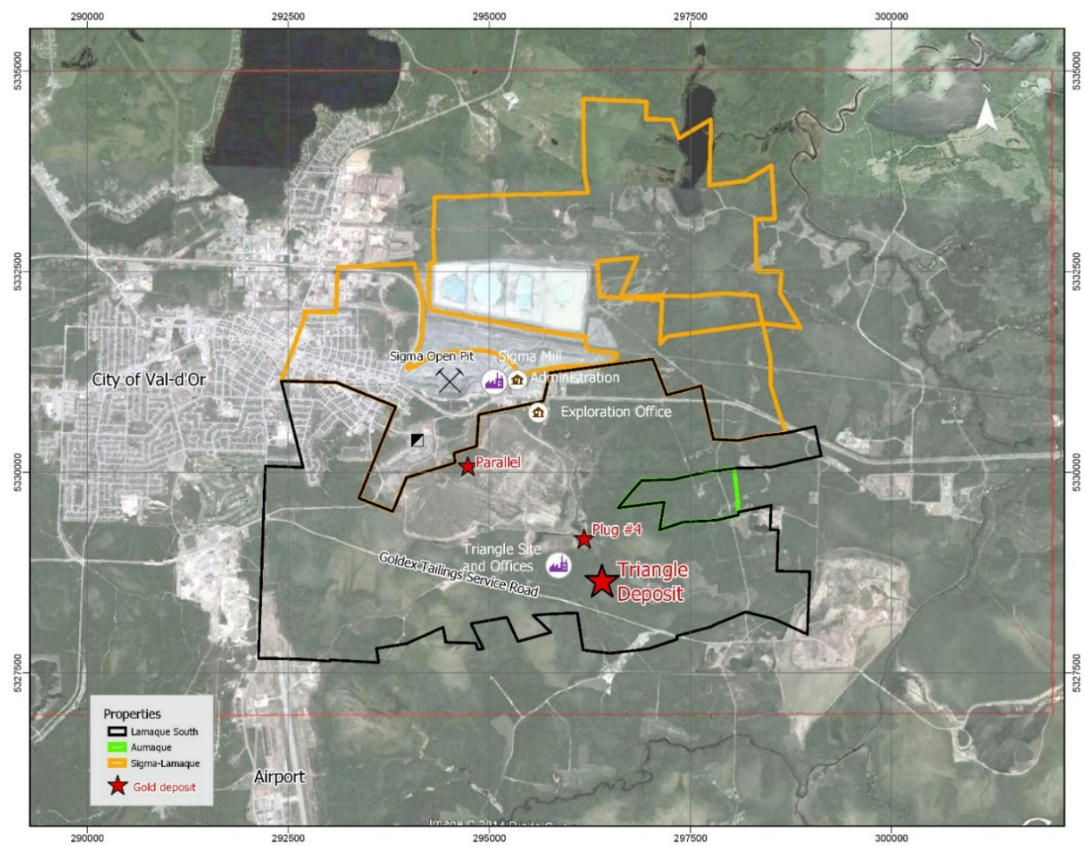


Figure 1.1 Location of the Lamaque Project with respect to the city of Val-d'Or

The Triangle deposit is included within the land package consisting of the Lamaque property, which covers 3,661.76 ha and comprises ten (10) mining concessions (2,325.09 ha) and 66 claims (1,336.67 ha), all of which are in good standing at the time of this report. The Lamaque property represents the amalgamation of three (3) separate but contiguous properties: Lamaque South, Sigma-Lamaque and Aumaque (Figure 1.1).

The mining concessions and exploration claims from the Lamaque South Property, which includes the Triangle Project are registered under the Integra Gold Corp. name, while the mining concessions and exploration claims from the Sigma-Lamaque and Aumaque properties are registered under the Or Intégra (Québec) Inc. name. Or Intégra (Québec) Inc. is a wholly owned subsidiary of Integra Gold Corp. In July of 2017, Eldorado Gold purchased all outstanding shares of Integra Gold, and therefore is the sole owner of the Lamaque South, Sigma-Lamaque and Aumaque properties.

1.4 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

The Lamaque Project lies to the southeast of the Val-d'Or urban center. The property is accessible via a public gravel road from the Provincial Highway 117 before eastern city limits or via 7e Street and the Chemin Des Gravieres Street in southern city limits. The city of Val-d'Or has a humid continental climate that closely borders on a subarctic climate. Winters are cold and snowy, and summers are warm and damp.

All requirements, including a quality supply of hydro-electric power to support a mining operation, are available in Val-d’Or, and there is an ample supply of water on or near the Lamaque Project to supply a mining operation. Also available is a local skilled labor force with experienced mining and technical personnel. A number of mining and mineral exploration companies have offices located in the area

The Abitibi region has a typical Canadian Shield-type terrain characterized by low local relief with occasional hills and abundant lakes. The mine site is bordered to the north by a large unpopulated wooded area, a portion of which is currently used for tailings and waste rock disposal.

1.5 GEOLOGY AND MINERALIZATION

The Lamaque Project is located in the southeastern Abitibi Subprovince of the Archean Superior Province in the Canadian Shield. More precisely, it is located in the Val-d’Or mining camp, northwestern Quebec. The region has several large-scale strike faults and/or shear zones, trending W to WNW and dipping steeply to the north. The Cadillac Tectonic Zone (“CTZ”) is one of the most prolific structures in terms of gold mineralization. The CTZ is important not only for its metallogenic wealth, but also for its geodynamic models and juxtaposition of varied lithologic assemblages along its subsidiary faults. The E-W and WNW sections of the fault reflect a deep asymmetry in the Abitibi Subprovince, a feature that influenced the styles and episodes of gold mineralization.

The Triangle deposit is hosted by volcaniclastic rocks and minor volcanic flows of the Val-d’Or Formation, which have been intruded by numerous intrusions, dykes and sills of compositions varying from mafic to intermediate. It is centered around a sub-circular shaped porphyritic diorite intrusion termed The Triangle Plug. The intrusion is very similar to the Lamaque Main Plug, which host the historical Lamaque Mine, in terms of its composition, shape, relationship with the surrounding lithologies and style of mineralization. Both intrusions consist of two type of porphyritic diorite. One is a mafic diorite rich in amphibole (35-40%) in its matrix and high in K; the other is a more felsic diorite with less than 20% mafic minerals, and the matrix consist of more biotite than amphibole.

Gold mineralization at Triangle occurs within quartz-tourmaline-carbonate and pyritic gold veins in the Triangle Plug and in the adjacent massive mafic lapilli-blocks tuffs. The veins are localized in a series of shear zones and fractures that cut both units. The thickest and most continuous veins are localized with E-W striking ductile-brittle reverse shear zones dipping 50-70° south. Veins also occur as extensional shear vein splays dipping 20-45° south as well as subhorizontal extension veins. The latter generally are not volumetrically significant. Gold also occurs in sericite-carbonate-pyrite alteration selvages along the veins, which are commonly foliated to schistose within fault zones.

The Plug No. 4 deposit is located 550m north of the Triangle deposit. No. 4 Plug is a fine- to medium-grained magnetic gabbro intrusion measuring roughly 100 to 150 m in diameter (Fig.7.6). A series of E-W striking reverse shear zones dipping between 45° and 75° to the south cuts all units at Plug No. 4. The shear zones are spaced roughly 25 to 50 m apart and have been identified to a depth of more than 1000m from surface in the gabbro. Gold mineralization at Plug No. 4 is found in quartz-tourmaline-carbonate-pyrite veins. Sub-horizontal extensional veins also occur in this deposit as vein arrays or clusters that extend for tens of meters down the center core of the gabbro intrusion. At Plug No. 4 these vein clusters can carry significant but quite erratic gold concentration.

The Parallel Deposit is located 650 m northwest of the No. 3 Mine and 900 m east-southeast of Lamaque Mine main shaft. Gold mineralization at Parallel is hosted within fine- to medium-grained porphyritic diorite, similar to the host rock at the Sigma Mine. The ore zones at Parallel occur as sub-horizontal extension veins at shallow depths (70-200 m) and as east-west striking shear veins dipping approximately 30° south at deeper levels. The mineralized veins consist of quartz and carbonate with lesser amounts of tourmaline, chlorite and sericite.

1.6 DRILLING, SAMPLING METHOD, APPROACH AND ANALYSES

Diamond-drill holes are the principal source for geologic, grade and metallurgical data on the deposits comprising the Lamaque Project. Drilling totals on the Triangle, Parallel and Plug #4 deposits amount to 899 drill holes totaling some 416,034 m. The majority have been completed since 2015, when Integra / Eldorado Gold took over the responsibilities for planning, core logging, interpretation and supervision and data validation of the various diamond drill campaigns.

Drilling was done by wireline method with N-size (NQ, 47.6 mm nominal core diameter) equipment using up to nine (9) drill rigs. Drillers placed core into wooden core boxes with each box holding about 4.5 m of NQ core. Geology and geotechnical data were collected from the core and core was photographed before sampling. Standard logging and sampling conventions were used to capture information from the drill core.

Sample intervals were marked up on the drillcores by the logging geologist. All vein and shear zone occurrences were sampled with additional “bracket sampling” into unmineralizated host rock on both sides of the veins or shear zones. Typically about 40% of a hole was sampled. The core was cut at the Company's core shack facility in Val-d'Or, Québec. For security and quality control, diamond drill core samples were catalogued on sample shipment memos, which were completed at the time the samples were being packed for shipment. Standards, duplicates and blanks were inserted into the sample stream by Eldorado staff.

Sample preparation procedures for routine fire assaying are to initially crush the entire sample to 10 mesh. A 250 g subsample is split by a riffle unit and pulverized to 85% minus 200 mesh. This subsample is sent for assay where a 30 g subsample is taken and fire-assayed with an atomic absorption (AA) spectrometry finish. Any values greater than or equal to 5 ppm Au were reassayed by fire assay using a gravimetric finish. The sample batches contained QA/QC samples comprising standard reference materials (SRMs), duplicates and blanks. These were inserted at a general rate of 1 in 20, 1 in 50 and 1 in 20, respectively. All material used for blank samples consisted of locally-sourced barren limestone.

It is in the Eldorado opinion the QA/QC results demonstrate that the Lamaque Project database for assays obtained from 2015 to 2017 is sufficiently accurate and precise for resource estimation.

Checks to the entire drillhole database were undertaken. Comparisons of the digital database were made to original assay certificates and survey data. Any discrepancies found were corrected and incorporated into the current resource database. Eldorado concluded that the data supporting the Lamaque Project resource work is sufficiently free of error to be adequate for estimation.

1.7 METALLURGICAL TESTING

The metallurgical test work in support of the Preliminary Feasibility Study was conducted on newly created samples at Bureau Veritas ("BV"). The samples mainly comprised C2 and C4 coarse assay rejects zone composites. A Master composite consisting of composite samples from C1, C2, C4 and C5 was also made. C2 and C4 core samples were used for grinding testwork. Variability tests were conducted on ten samples, one from the C1 zone, four (4) from the C2 zone, four (4) from the C4 zone and one for the C5 zone.

Tests included chemical analyses, comminution testwork, gravity concentration E GRG tests, multiple whole ore cyanidation tests, carbon gold loading testwork, cyanide destruction tests as wells as thickening, rheology and filtration testwork

- The Bond rod mill work index, Bond ball mill work index and Abrasion index were measured and ranged from 15.8 to 18.1 kWh/t, 11.2 to 13.5 kWh/t, and 0.123 to 0.206 g respectively.
- Extended Gravity Recoverable Gold (E-GRG) tests resulted in cumulative gold recoveries after 3 passes ranging between 37 and 56%, with the C2 composite presenting the lowest gold recovery. Tests on the Camflo composite showed that gravity gold content can be quite variable.
- Whole ore cyanidation tests showed that high pH (>11.7) considerably increases recovery and reduces cyanide consumption. Addition of 1.5 kg/t lead nitrate increased recovery even further for the C2 composite.

- A finer grind resulted in higher recovery, with about a 2% increase when going from 80% passing 75 to 50 µm at 48-56 h retention times.
- Lower cyanide concentrations resulted in slower kinetics and reduced cyanide consumption. The effect on final recovery was however not clear from the tests.
- The impact of high dissolved oxygen concentrations (>20 mg/L) compared to those achievable with air (9-10 mg/L) was not conclusive due to the difficulty of controlling the pH to the same value from one test to the other.
- No particular variability sample stood out with results considerably different from the composite ore samples. C2 Upper performed a little better than C2 Mid and C2 Lower. C1 showed very good recoveries. C5 recoveries were similar to those of the C2 zone.
- Results from the same variability tests on the 3-day Camflo rod mill feed composites showed that ore processed during the first Camflo bulk sample campaign is not representative of the C2 zone as a whole.
- Cyanide destruction tests showed that residual cyanide concentrations of < 1.0 mg/L CN_{WAD} are achievable using 4.5 g SO_2/g CN_{WAD}, around 1.5 g CaO/ g CN_{WAD} and around 0.04 kg/t Cu.
- Thickening tests showed that the slurries thickened easily with about 20 g/t flocculant consumption and hydraulic rates ranging from 3.22 to 3.68 m^3/m^2/h.
- Vacuum filtration tests on thickened detoxed tailings achieved moisture contents from 17.6 to 18.5% for filtration rates of 1200 to 1600 dry kg/m^2/h.
- Pressure filtration on the same tailings reduced moisture content to about 10% and would require around 0.78 m3 chamber volume per dry tonne.

1.8 MINERAL RESOURCE ESTIMATES

The Mineral Resource estimate for the Lamaque deposits (Triangle, Parallel and Plug #4) used data predominantly from surface diamond drillholes. The resource estimates were made from 3D block models created by utilizing commercial geological modelling and mine planning software. The block model cell size is 5 m east by 5 m north by 5 m high. The assays were composited into 1 m fixed length downhole composites.

At Triangle, six main shear zones were modelled: C1, C2, C3, C4, C5 and C7. Of these, C2 and C4 contain the bulk of the gold mineralization. Except for C7, each of the main shear zones has associated mineralized splay zones with the splays related to C1, C3 and C4 being the most significant with respect to gold mineralization. The most significant mineralization at Parallel is found in moderately-dipping, hybrid shear/extensional zones to the footwall of a non-mineralized higher-order steep structure. At Plug 4, there is a distinct set of stacked steep shear zones having similar orientations to those at Triangle. A feature that is unique to Plug 4 is the occurrence of sub-horizontal vein arrays associated with these steep mineralized shears. The flat zones are captured in a single set of solids representing spatially coherent zones above 0.5 g/t Au.

Gold grades in the Triangle deposit are highest in C4 and C5 shear zones, followed by C2 and the C4 and C5 Splay veins. Coefficients of variation (CVs) for uncapped data are highest in the upper Triangle shears of C1 through C3. Variation due to extreme grades are notably less in veins and shears associated with C4 and deeper zones at Triangle. Parallel is a small high grade deposit akin to the deeper Triangle shears in gold distribution. Plug #4 is a distinctly lower grade deposit displaying a highly variable gold distribution.

Extreme grade analysis on the Triangle mineralized zones showed that a capping strategy should aim to remove about 12 percent of the gold metal in the estimate. This was achieved by implementing an 80 g/t Au cap to the assay data prior to compositing. For the Parallel data, a similar reduction was met by applying a 60 g/t Au cap grade. Plug #4 extreme grade analysis indicated that 20 percent of the gold metal needed to be removed which was achieved by using a 60 g/t Au cap grade on the assay data.

Grade modelling consisted of interpolation by ordinary kriging (OK) and Inverse Distance Weighting (ID) to the second power (a few domains in Parallel and Plug #4 used third power ID). ID interpolation was use for zones that had limited data (individual splays at Triangle, C1 and C7 at Triangle, all Parallel and Plug #4 domains). Nearest-neighbour (NN) grades were also interpolated for validation purposes but where interpolated using the longer length composite data set. Blocks and composites were matched on mineralized zone or domain. The resultant grade models were validated visually and checked for biases in the global estimate and local trends; none were observed.

1.8.1 Mineral Resource Classification

The mineral resources of the Triangle, Parallel and Plug #4 deposits were classified using logic consistent with the CIM Definition Standards for Mineral Resources and Mineral Reserves referred to in National Instrument 43-101. The mineralization of the project satisfies sufficient criteria to be classified into measured, indicated, and inferred mineral resource categories.

Inspection of the Triangle model and drillhole data on plans and cross-sections, combined with spatial statistical work and investigation of confidence limits in predicting planned annual and quarterly production, contributed to the setup of various distance to nearest composite protocols to help guide the assignment of blocks into measured or indicated mineral resource categories. Reasonable grade and geologic continuity is demonstrated over most of the C2 and C4 zones in the Triangle deposit, where the average distance of the samples to a block center interpolated by samples from at least two drill holes, is up to 30 m. Blocks that met these criteria were classified as indicated mineral resources. Indicated resource blocks within the area covered by underground development and infill diamond drilling in C2 were re-classified as measured mineral resources. All remaining model blocks containing a gold grade estimate were assigned as inferred mineral resources.

For the other two deposits, similar protocols were used. At Parallel, Indicated and Inferred mineral resource classification followed the logic used at Triangle. Plug #4 model used an average distance of 25 m to a block center interpolated by samples from at least two drill holes to be classified as Indicated mineral resources. No measured mineral resource were declared at Parallel or Plug #4.

1.8.2 Mineral Resource Summary

The Mineral Resources for the Lamaque Project, as of 31 December 2017, are shown in Table 1.1. The mineral resources are reported at a 2.5 g/t gold cut-off grade.

Table 1.1 Lamaque Mineral Resources, as of December 31, 2017

Deposit Name	Categories	Tonnes (x 1,000)	Grade Au (g/t)	Contained Au (oz x 1,000)
Triangle	Measured	132	10.4	44
	Indicated	3,582	8.84	1,018
	Measured + Indicated	3,714	8.9	1,062
	Inferred	4,648	7.42	1,109
Parallel	Measured	-	-	-
	Indicated	221	9.92	70
	Measured + Indicated	221	9.92	70
	Inferred	206	8.7	57
Plug #4	Measured	-	-	-
	Indicated	762	5.84	143
	Measured + Indicated	762	5.84	143
	Inferred	514	5.56	92
Total Resources	Measured	132	10.4	44
	Indicated	4,565	8.39	1,231
	Measured + Indicated	4,697	8.45	1,275
	Inferred	5,368	7.29	1,258

1.9 MINERAL RESERVE ESTIMATES

The Mineral Reserve estimate for the Lamaque Project is based on Measured and Indicated Mineral Resources for the Triangle Deposit upon which the mining plan and economical study have demonstrated economical extraction. Mineral reserves are reported using a gold price of US$ 1,200/oz and an exchange rate of 1.30CAN$/US$. An estimated cut-off grade of 3.5 g/t Au was applied at stope scale for the discrimination of material to be retained in reserves. All stopes falling below the cut-off grade were taken out of the mine pan.

The Eldorado mineral resources model served as the basis for estimating mineable tonnage and metal content in the mine plan. Using Autocad and Promine, stopes shapes were cut from the 3-D wireframes, thus representative of undiluted tonnage. The following modifying factors were then applied to the stope shape physical attributes to determine the reserve tonnes and grades:

- Only considered Measured and Indicated resource classified model blocks
- Long-hole extraction for parts of the orebody dipping steeper than 42°;
- Room-and-pillar extraction for portions of the orebody dipping less than 42°.
- External dilution of 0.75 m over break for long hole stopes except for those planned in C4 where overbreak was set to 1.0 m to reflect greater depth and less favorable geotechnical conditions. The overbreak material is assumed to be barren (0g/t).

- Additional 5% backfill dilution was applied to the long hole stopes
- Mining recovery of 95% for long hole stopes and 85% for room and pillar areas.

The mineral reserves for the Lamaque Project, as of 31 December 2017, are shown in Table 1.2. The mineral reserves are reported at a 3.5 g/t gold cut-off grade.

Table 1.2 – Lamaque Project Mineral Reserves, as of December 31, 2017

Category	Tonnes (x1,000)	Grade (g/t Au)	Contained Au (oz x 1,000)
Proven	111	8.78	31
Probable	3,698	7.25	862
Proven + Probable	3,809	7.30	893

1.10 MINING METHODS

The present mine plan is based entirely on the proven and probable reserves of 3.8 million fully diluted and fully recovered tonnes described in the previous section. The Triangle Zone will be the only area of extraction of this PFS representing the most advanced part of the property.

In July 2016, Integra contracted Promec Mining to develop a ramp and access drifts to reach the Triangle Zone for the purpose of extracting a 60,000-tonne bulk sample. As of February 28th 2018, more than 5,810 metres of lateral development, in waste and ore, and more than 340 meters of raises have been excavated following the mine plan detailed herein.

From the early phases of the project it was recognized that the reserves in Triangle would be optimally recovered through a haulage ramp system to surface, as opposed to a vertical shaft which could not be economically justified at this time. The ramps will be used to haul ore and waste and provide access for personnel, equipment, materials and services, and form part of the exhaust air circuit.

The Triangle long-hole stopes are designed within four sub-parallel major zones dipping 50-70° to the south. Approximately 95% of all reserves are located within these zones; the remainder are located in low-dipping (20-45°) splays off the main shears (C-splays). Each of the C-zones (C2 West, C1-C2 East and C4) will be serviced by a dedicated haulage ramp. A total of 36 levels are required to extract the reserves over the vertical extent of the orebody. The total lateral development requirements are estimated at 54,000 meters over the LOM. Sector C2 West and C1-C2 East are connected via level 184 and 148. Sector C2 West and C4 are tied together at level 328 (Figure 1.2).

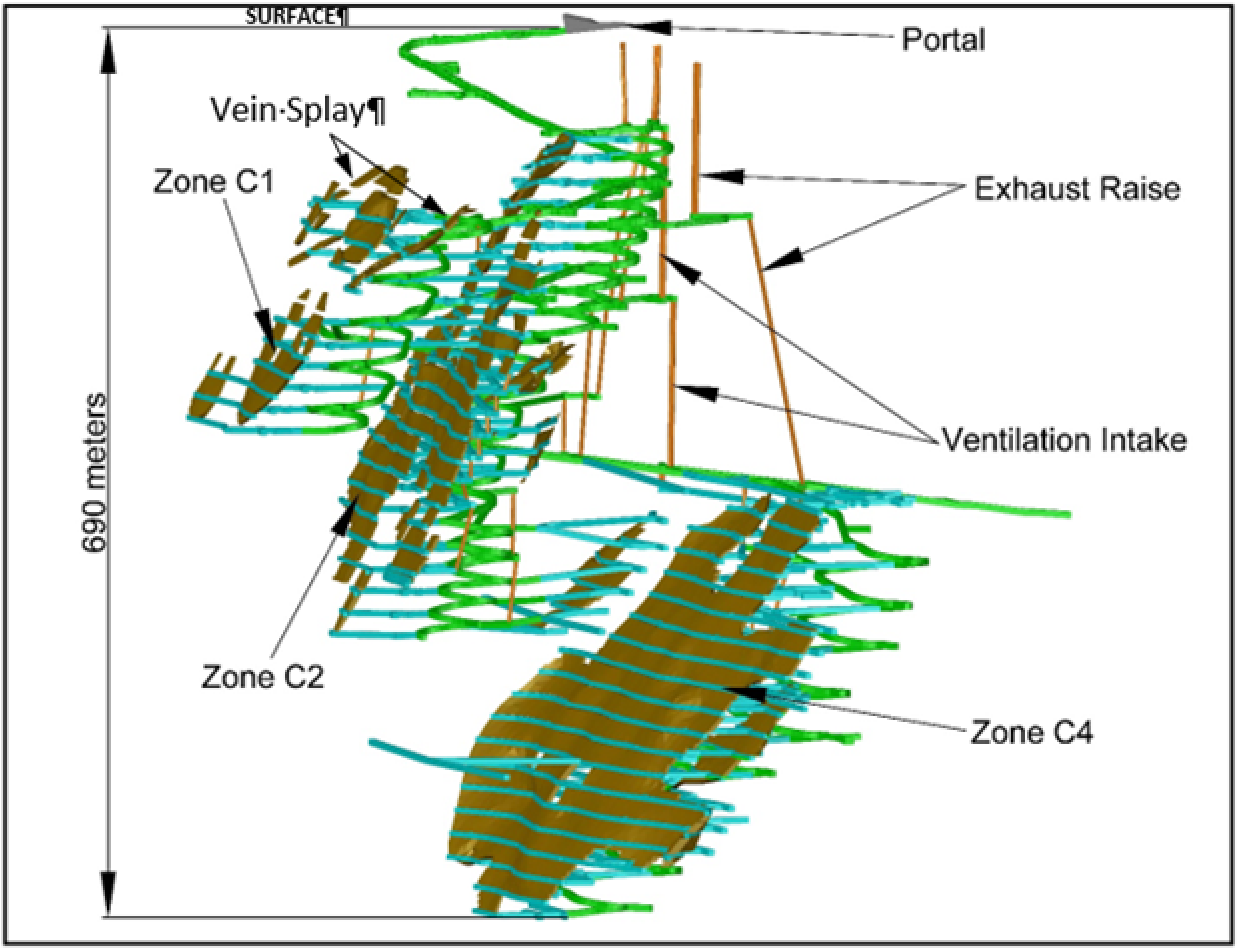


Figure 1.2 – Isometric view of the Triangle deposit and related development

The dimensions of the main ramp were determined according to required clearances for the selected mobile equipment, considering ventilation requirements during development and production. It was determined that a 5.1 m wide by 5.5 m high profile would be suitable for a 45 t haul truck, although it would be possible to use 60t trucks upon removal of the ventilation ducts. In general, all the ramps are designed to be driven at a 15% gradient while minimizing intersections with major structures and optimized to reduce development requirements to access each production area. The main ramp is fully serviced for ventilation, power (compressed air and electricity), communications, industrial water and dewatering.

Remuck bays are laid along the main ramp typically at 125 m intervals (to a maximum of 150 m) and will eventually be converted to store equipment and materials, sumps, refuge stations or explosives magazines. Level access crosscuts and haulage drifts are planned to be developed off the ramp at a 5.0 m by 5.0 m profile. All infrastructure development, which would not be used for access, i.e., remucks, ventilation drifts and sumps, were designed with a 5.0 m by 5.0 m profile.

A longitudinal retreat long-hole mining method was selected for the Triangle deposit based on the orebody characteristics, including grade, dip, continuity, thickness, etc. Although some part of the

C-zones have hanging walls dipping shallowly (as low as 43 degrees), the approach in this study has been to promote the application of long-hole mining where possible.

Sill pillars will be temporarily left in place and will serve to increase the number of production areas in operation at the same time. The sill pillars will be recovered as the mining progresses upwards to the sill level. 3D numerical modelling results indicate that 18 m vertical height sill pillars are considered adequate for both the C2 and C4 zones.

Rockfill will be used to backfill the stopes. Since a longitudinal retreat mining sequence will be used, only about 1/3 to 1/2 of the volume is planned to be filled with cemented rockfill (CRF) and the rest of the void will be filled with non-cemented (or unconsolidated) rockfill. There will be a minimum of 2 m of CRF from the stope wall.

The ore extracted from the bulk sample program and the ore produced in Q1 of 2018 to Q4 of 2018 will be transported and processed under custom milling contracts. During this time the Sigma mill will be refurbished and upgraded to process Triangle ore production as of Q1 2019.

The overall life of mine based on Proven and Probable reserves is estimated at approximately 8 years which includes one (1) year of pre-production.

1.11 PROCESS PLANT AND RECOVERY METHODS

This PFS assumes the Sigma Mill will process an average 575,000 tonnes per year, with toll milling at the Camflo and Westwood mills in 2018, Sigma mill start-up in Q4 2018 and processing of about 440,000 tonnes in 2019.

The Sigma Mill is composed of crushing, grinding, gravity concentration, leach and carbon-in-pulp (CIP) circuits, as well as elution, carbon regeneration and refinery areas. The crushing circuit includes a jaw crusher, cone crusher and a screen which needs to be replaced. The grinding circuit is composed of a primary rod mill and two ball mills. The circuit will be rearranged to use one ball mill as the secondary grinding step and the other as a tertiary grinding step, with each mill in closed circuit with its own cyclone, to be able to achieve a 50 µm grind size.

In 2019, installation of a new gravity concentration circuit is planned. This circuit will include two new vibrating screens, two new gravity concentrators and an intensive cyanidation skid with its dedicated electrolysis cell to be installed in the refinery. In the meantime, for the 2018 start-up one of the existing gravity concentrators will be refurbished and the existing shaking table will be used. For the time being, a new building extension to the north of the grinding building is planned for the installation of the new gravity circuit.

A pre-aeration tank is planned to be added before the leach circuit in 2019. The five leach tanks provide 52 hours of retention time before the slurry goes on to the 7 tank CIP circuit, four of which need to be replaced, for a combined leach and CIP retention time of 66 hours.

In 2019, installation of a new cyanide recovery thickener is planned. By then the paste plant will have started operation (planned for Q3 2019). Paste plant thickener overflow will be mixed with the CIP tailings prior to feeding the cyanide recovery thickener to enable cyanide recovery to the overflow. The underflow from the thickener will go on to cyanide destruction. Prior to installation of the cyanide recovery thickener, CIP tailings will go directly to cyanide destruction where sodium metabisulfite will be used as SO2 source to reduce cyanide concentrations to environmentally acceptable limits.

A few reagent systems will be added to the plant, namely lead nitrate, milk of lime and sodium metabisulfite. Copper sulphate tanks will be moved and equipped with new pumps. The lead nitrate system will be installed in the former SAG building whereas milk of lime, sodium metabisulfite and copper sulphate systems will be installed in new building extensions to the south of the plant. In order to prepare the plant for potential future compliance to the International Cyanide Management Code, the cyanide tank will also be moved to a new building extension.

New low pressure air compressors will also be installed to feed the pre-aeration, leach, CIP and cyanide destruction tanks. For this the existing compressor building will be extended on each side.

Table 1.3 presents the expected gold recoveries achievable in the Sigma mill for each zone, estimated based on testwork results obtained thus far.

Table 1.3 Expected Metallurgical Recoveries for the Triangle Deposit

Zone	Expected Gold Recovery (%)
C2	93.8
C4	95.3

The Sigma mill is expected to hire 42.5 workers. Average grinding power consumption has been estimate at 22.4 kWh/t and average total power at 56 kWh/t. Water consumption from the Sigma underground mine has been estimated at 27 m^3/h. Lead nitrate will be used for processing of the C2 zone ore but its use is not planned for the C4 zone ore, as it did not increase recovery for this zone.

In addition to the mill modifications and equipment replacements, almost all major equipment within the plant require some degree of refurbishing. This includes crushers, conveyors and feeders, grinding mills, the existing cyclone, screens, agitators, pumps and the regeneration kiln. Plant general infrastructure, such as piping systems, electrical system, instrumentation and control systems, the plant building itself, building ventilation, dust collection, spill containment areas and safety shower circuit, also need upgrades to meet current standards. These refurbishment works represent more than two thirds of the mill related capital costs.

1.12 PROJECT INFRASTRUCTURE

1.12.1 Triandle Mine

The Triangle mine site consists of the following buildings are built as part of the current mine surface infrastructure:

- A garage with 6 working bays, a warehouse, a compressor room and offices to serves maintenance and procurement teams;
- A temporary office for the mine department made of prefab modules;

- A temporary office for technical services and administration made of prefab modules;
- A temporary office for health and safety, training made of prefab modules;
- A mine rescue local made of prefab modules;
- A temporary 400 person dry facility built with prefab modules.

The following buildings are be built and put into service by the end of 2018:

- A fabric building to serve as cold storage.
- A building constructed next to the main ventilation raise to serve as the permanent location for the underground service compressors.
- A diamond drill core logging facility.

The following building shall be built and put into service by the end of 2020:

- A permanent administration. Technical and operational services two-story building. This facility will include a 400 person dry facility. The current dry facility will be kept in service to cater to third party contractors working at the mine site.

Waste rock stockpiles will serve as permanent storage for the waste rock extracted from the underground mine. The Triangle waste rock pile is located close to the portal to limit transport distance. The waste stockpile has been built and permitted to accept waste rock. This stockpile's footprint covers an area of 52,000 m^2 for a capacity of 400,000 m^3. In 2019, this stockpile will be enlarged to a footprint covering 83,000 m^2 for a final capacity of 760 000 m^3.

1.12.2 Paste Plant

Sigma mill tailings will be dewatered, mixed with cement and disposed in the open pit in the pastefill plant new facility.

Laboratory testwork programs were carried out on tailings bulk sample to determine the amenability to liquid-solid separation and to determine a preliminary paste recipe. Paste mix recipe determination is still on-going.

The tailings is amenable for thickening process. Vacuum filtration results shows high variability. The conservative results were used for preliminary equipment selection. Optimization work must be conducted.

Preliminary paste mix results and assumptions based on similar project were made to estimate binder content, slump, strength, pipeline pressure loss for the disposition of the tailings in the open pit. These parameters will be confirmed during next engineering stage, when paste testwork program will be completed.

The Paste fill plant is composed of thickening, filtration, mixing and paste pumping areas. The plant is design to take 100% of Sigma mill talings. This availability will be achieved by installing a second production line. Each line is equipped with a vacuum disc filter, paste mixer and positive displacement pump

1.12.3 Tailings Impoundment Area

For most of the tailings (89.5%) produced over the life of mine, the management approach selected is under the form of cemented paste backfill discharged end pipe in the former Sigma open-pit. The pit offers a capacity of approximately 13.5 million m^3 close to the process plant, where the current life of mine plans requirement for 2.6 million m^3 at a final elevation of 259 m. The selected approach avoid building a new tailings surface facility which is seen as a positive asset to the project in term of permitting and social acceptability.

A layer of crushed rock will be installed (a pervious surround, e.g. MEND 2015) to surround the tailings and act as a drain for run-off. During operations, the purpose of pervious surround is to divert run-off from becoming contact water. During post-closure, i.e. when the Sigma pit will be flooded, the purpose of pervious surround would be to limit groundwater contact with the tailings mass. Once flooded, a spillway built at elevation 306.5 m will control the overflowing water from the northwest side of the open pit to the channel along Highway 117.

1.13 ENVIRONMENT AND PERMITTING

Federal Regulations and Permitting: The Canadian Environmental Assessment Agency advised that the Lamaque South Project is considered as an extension of the Sigma-Lamaque Mine and Mill Complex. Given that the extension represents an increase of less than 50% in the area of the mine operations, it is not subject to a federal environmental assessment.

An authorization pursuant to section 35(2) of the amended Fisheries Act for work that may result in serious harm to fish that are part of a commercial, recreational or Aboriginal fishery, or to fish that support such a fishery, will not be required for this Project.

Federal regulations regarding Mining Effluent applies. It regulates quality of the final effluent and include provision on environmental surroundings monitoring.

Provincial Regulations and Permitting: The *Environment Quality Act* (EQA) of Québec is divided into two chapters. Chapter I sets out provisions of general application while Chapter II outlines the particular provisions for the region covered by the James Bay and Northern Québec Agreement. The Lamaque Project is located outside the territory to which the relevant EA regime of the JBNQA territory applies. Only Chapter I applies to the Project.

The main articles of Chapter I of the EQA that relate to obtaining environmental certificates of authorization or permits are articles 22 (general case), article 31.1 (environmental impact assessment studies), article 32 (drinking water and domestic wastewater) and article 48 (atmospheric emissions). Now that the project encompasses the Sigma-Lamaque Complex, the issuer will also be subject to a de-pollution attestation under article 31.11 of the EQA.

The Project is not subject to provincial EA, since the proposed production is less than 2,000 tpd. The Project is subject to section 22 of the EQA for the operation of a mine. Mining operation authorization under section 22 is presently issued as the mine water treatment system.

In addition, the Company is planning to obtain other permits, authorizations, approvals and leases from both the MERN (Ministère de l’Énergie et des Ressources naturelles) and the MDDELCC for various components of the overall project development work, as required. These applications will be submitted as part of the ongoing process of developing the site and should therefore not impact the project's critical path schedule.

Reclamation costs for the Lamaque Project were evaluated at $C1.9 M. The first payment of 50% is secured as required. Both 25% payments are due in 2019 and 2020.

Mining lease regarding the Triangle project was issued in March 2018.

In May 2015, a monitoring committee was formed to keep the Company's stakeholders informed about the Lamaque Project. Quarterly meetings are organized to provide updates on the status of the Project to the committee members. All proceedings are put on the Company's website. The purpose of such a monitoring committee, which is required under section 101.0.3 of the Québec Mining Act, is to develop the involvement of the local community in mining projects. The committee has representatives from the municipal sector, the economic sector, the public and the Nation Anishnabe de Lac-Simon.

1.14 CAPITAL AND OPERATING COSTS

1.14.1 Capital Costs

The pre-production costs are estimated at $CAD 158 M, net of production revenue received from Q2 year 2018 to Q2 year 2019 of the preproduction period ($CAD 104 M).

Pre-production capital costs include completion of surface infrastructure at Triangle (site preparation, roads, electric and water lines, buildings); refurbishment of existing office buildings at the Sigma site; refurbishment of Sigma mill and tailings facility; mobile equipment; development and capitalized operating costs; and owner costs (closure plan costs in line with required financial guarantees, company staff and indirect costs).

Pre-production capital costs are low given that there is an existing processing plant and tailings facility, major infrastructure at Triangle is already in place, and underground development has reached Level 202.

Pre-production is anticipated to take place over a 15-month period with most of capital expenditures allocated to ramp development, mill refurbishment, tailings expansion and development of mineralized zones.

Contingencies are included in the cost shown in Table 1.4. For sustaining capital development, a growth factor of 10% is included in the scheduled meters. A 10% contingency has been applied on Triangle and Sigma infrastructure. Contingencies of 18% for mill refurbishment at Sigma, and 15% for the new paste plant at Sigma were applied.

During the 15-month pre-production period, it is anticipated that 62,900 ounces of gold will be produced, providing revenue of $CAD 104.2M. The pre-production revenue will be used to offset pre-production capital expenditures.

Table 1.4 – Capital Cost Estimate ($CAD M)

Description	Construction	Sustaining	Total	% Contingency
Sustaining Development	22.8	71.4	94.2	10%
Equipment	14.1	42.0	56.1	0%
Maintenance	8.2	31.5	39.8	0%
Underground Construction	1.6	4.5	6.1	0%
Underground Ancillary Equipment	4.6	6.4	11.0	0%
Triangle Constructions and Ancillaries	29.2	18.1	47.3	10%
Sigma Mill	56.1	8.8	64.9	18%
Sigma Paste Plant	9.7	21.7	31.4	15%
Tailing Residue Area and Other Sigma site Costs	7.3	12.2	19.5	10%
Mine Restoration Costs	2.1	3.4	5.5	10%
Royalties Buy Out	3.0	0.0	3.0	0%
Salvage Value	0.0	-14.5	-14.5	100%
Total	**158.8**	**205.5**	**364.3**	

1.14.2 Operating Costs

Operating costs are estimated in Q1 2018 Canadian dollars with no allowance for escalation. The total operating cost and average unit operating costs are summarized in Table 1.5. The overall unit operating cost is $CAD 152.52 per tonnes of milled ore.

Operating costs are summarized below for the production period.

Table 1.5 – Summary of total operating costs

Description	Total Cost (C$M)	Unit cost ($CAD) ($/t)
Mining	324.1	85.10
Milling (including ore transport and refining)	156.1	40.97
G&A	100.8	26.45
Total	**580.9**	**152.52**

Notes: Operating costs are shown life of mine (including pre-production operating costs).

1.15 FINANCIAL ANALYSIS

The economic analysis for the Project based on US$1,300/oz Au indicates an after-tax net present value (NPV) of US$211 M, using a discount rate of 5%. An internal rate of return (IRR) of 35% on an after-tax basis is achieved. Payback of the initial capital is achieved in 3.3 years from the start of production.

At the mineral reserve metals price of US$1,200/oz Au, the Project shows positive economics. The after tax IRR is 25.9 % and the NPV is estimated to be US$150.3 M using the 5% discount rate, with a calculated payback period of 3.9 years.

The economic model was subjected to a sensitivity analysis to determine the effects of changing metal prices and capital and operating expenditures on the Project financial returns. This analysis showed that the project economics are robust, and are most sensitive to metal prices.

In 2013 the province of Quebec introduced a mining tax based on the operator's profitability. Profitability is calculated by dividing the operator's annual profit by the gross value of the operator's annual output. A progressive rate structure is then applied based on the operator's profit margin. The Company expects that through the use of intercompany financing and existing tax losses that the Company will not pay any current income tax on the profits from the project.

The Lamaque Property is made up of several individual mining concessions that are subject to a 2% royalty in favour of various parties. Each of the royalty agreements provide the Company with a buyout provision to reduce the royalty rate to 1%. For purposes of the economic analysis, a buyout amount of C$3.0 million has been included in the capital costs of the project and a royalty rate of 1% has been applied to all commercial ounces.

The resulting main parameters and cash flow analysis are presented in Table 1.6.

Table 1.6 – Cash Flow Analysis Summary

Parameter	Units	Results
Peak Milling Rate	ktpa	600
Mine Life	year	7
Average Grade	g/t	7.30
Average Recovery Rate	%	94.5%
Average Annual Gold Production	Koz	117
Peak Gold Production	Koz	135
Average Cash Costs	US$/oz	516
Average All-in Sustaining Costs	US$/oz	717
Estimated Capital Expenditure	US$ M	284
Initial Capital Costs (to commercial production)	US$ M	122
Capitalized Operating Costs (pre commercial production)	US$ M	57
Proceeds from Pre-Commercial Gold Sales	US$ M	(80)
Sustaining Capital	US$ M	162
Gold Price Assumption Used in Financial Analysis	US$/oz	1,300
NPV-5% (after tax)	US$ M	211
IRR (after tax)	%	35%
Payback Period	year	3.3

Notes: • *Total Cash Cost: Direct operating cost, refining and royalties*
• *All-in Sustaining Cost: Direct operating cost, refining, royalties, sustaining capital, reclamation and salvage*

1.16 OTHER RELEVANT DATA AND INFORMATION

1.16.1 Life of Asset Strategy

Eldorado Gold Corporation endeavours to maximise the value of the Lamaque project by adding to its existing reserve base, thereby extending mine life and potentially increasing the rate of gold production.

The focus at Lamaque since the acquisition in July 2017 has been on infill drilling the upper portion of the Triangle deposit to quantify and declare the maiden reserves. Eldorado Gold Corporation will continue to infill drill resources to add to its existing reserve base. 58,500 meters of infill drilling are planned for 2018, and a provision to maintain this level of infill drilling in subsequent years have been factored into the operational cost model.

The Lamaque property hosts significant Indicated resources in the Plug 4 and Parallel zones that have not been incorporated into the current reserves. Requirements for the recovery of these zones will be evaluated in detail over the course of 2018, alongside the evaluation of an option to develop an underground ramp to haul ore from Triangle to the Sigma mill, which would pass in close proximity to the Plug 4 and Parallel zones. The underground ramp is expected to reduce ore transport costs, improve access, and could serve as an excellent exploration drill platform for the Plug 4 and Parallel zones as well as new targets along its route.

1.16.2 Project Risks and Opprtunities

This sub-section highlights the main project risks and opportunities.

Project Risks

- The Company received the Certificate of Authorization, closure plan and the mining lease for the Triangle deposit in the first quarter 2018. The only outstanding permit is for ore processing at the Sigma mill, which is expected to be received during the third quarter 2018;
- The thickness and continuity of each vein may vary over the expected distances;
- Encountering more water infiltration than expected in current plan;
- High pH operations, which could lead to a higher acid washing of the carbon. Further testing during the detailed engineering phase will allow to properly mitigate this situation.

Project Opportunities

- The Company is evaluating an option to build an underground ramp to haul ore from Triangle to the Sigma mill, while passing through the Plug 4 and Parallel ore zones. The underground ramp is expected toreduce ore transport costs, improve access, and could serve as an excellent exploration drill platform for the Plug 4 and Parallel deposits as well as new targets along its route;
- Increasing automation in operations could yield opportunities in lower operating costs and increases in production;
- More tests without pre-aeration should be conducted and the PEA trade-off study for inclusion or not of a pre-aeration tank redone as the difference in cyanide consumption no longer justifies the addition of a pre-aeration tank. The addition of the tank would need to be justified based on added recovery, and potentially on beneficial effect on cyanide destruction
- Opportunity for further extension of the known ore zones in Triangle. Also opportunity to add other ore zones (Plug 4, Parallel etc) that have known resources on them but are not included in the current mine plan.

1.17 INTERPRETATION AND CONCLUSION

The results of this PFS demonstrate that the Lamaque Mine Project warrants development due to its positive, robust economics. To date the qualified persons are not aware of any fatal flaws on the Lamaque project and the results are considered sufficiently reliable to guide Eldorado management in a decision to further advance the Project. It is concluded that the work completed in the prefeasibility study indicate that the exploration information, mineral resource and mineral reserve estimates and Project economics are sufficiently defined to indicate the Project is technically and economically viable.

1.18 RECOMMENDATIONS

Due to the positive results of the 2017 underground bulk sampling program, completion of permitting for Lamaque production and PFS robust economics, it is recommended to expediently advance the Lamaque Project to pre-production followed by production including the Sigma mill refurbishment.

Below is a description of the recommended steps in the continued advancement of the Lamaque Project. Table 1.7 summarizes each item and its estimated cost.

Table 1.7 – Proposed Work Program and Budget

Item		Cost (CAD$)
1	26.1.Undertake an economical study to re-evaluate the potential of the Parallel Zone resources	60,000
2	Undertake a trade-off studies for alternate method of ore transportation, mining method and backfill method	150,000
3	Continue exploration on the Triangle Mine Project with 34,000m.	5,000,000
4	Undertake technical study to evaluate underground ramp from Sigma to Triangle to serve as access to Parallel zone, exploration drilling base and ore transportation between Triangle and Sigma.	150,000
5	Complete permitting works for Sigma mill and tailing management	500,000
Total		**$5,860,000**

SECTION • 2 INTRODUCTION

In 2017, at the request of François Chabot , P.Eng., Directeur ingénierie et operations, a group of experts (“Experts”) from WSP Canada inc. (“WSP”), InnovExplo Inc. (“InnovExplo”), Wood plc (“Wood”), Golder and ass. (“Golder”), Howden Simsmart, BBA inc. (“BBA”) were retained by Eldorado Gold Corp. (“Eldorado”, the “issuer” or the “Company”) to collaborate to a Technical Report (the “Report”) to present and support the results of a Preliminary Feasibility Study for the Lamaque Project (the “Project”) .

This report has been prepared in accordance with Canadian Securities Administrators’ National Instrument 43-101 Respecting Standards of Disclosure for Mineral Projects (“NI 43-101”) and its related Form 43 101F1. InnovExplo is an independent mining and exploration consulting firm based in Val-d’Or (Québec).

Eldorado is an international gold mining company based in Vancouver, British Columbia.

When preparing reserves for any of its projects, Eldorado uses a consistent prevailing gold price methodology that is in line with the 2015 CIM Guidance on Commodity Pricing used in Resource and Reserve Estimation and Reporting. These are the lesser of the three-year moving average and the current spot price. These were set as of September 2017 for Eldorado’s current mineral reserve work. For gold, these is US$1,200/oz Au. All cut-off grade determinations, mine designs and economic tests of economic extraction used these pricing for the Lamaque Project and the mineral reserves work discussed in this technical report. In order to demonstrate the potential economics of a project, Eldorado may elect to use metal pricing closer to the current prevailing spot price and then provide some sensitivity around this price (for the Lamaque Project, metal prices used for this evaluation were US$1,300/oz Au). This analysis (in Section 22 of this report) generally provides a better ‘snapshot’ of the project value at prevailing prices rather than limiting it to reserve prices, that might vary somewhat from prevailing spot prices. Eldorado stresses that only material that satisfies the mineral reserve criteria is subjected to further economic assessments at varied metal pricing.

The Lamaque Project in Val-d’Or, Québec (Canada), is the amalgamation of the Lamaque South Property and the Sigma-Lamaque mine and mill facilities (the “Sigma-Lamaque Complex”).

The company Or Eldorado is 100% owner of the Sigma-Lamaque Complex. The Sigma-Lamaque Complex is adjacent to the Lamaque South Property, and comprises three separate properties and certain facilities and infrastructure, as follows:

- The Sigma-Lamaque Property;
- The Aumaque–Union Gold–Audet Property;
- The Sigma II Property;
- The Sigma-Lamaque milling facility;
- The Sigma tailings facility;
- The mining infrastructure of the former Sigma and Lamaque mines.

2.1 PRINCIPAL SOURCES OF INFORMATION

Eldorado and the Experts believes the information used to prepare the Technical Report and to formulate its conclusions and recommendations is valid and appropriate considering the status of the project and the purpose for which the report is prepared. The authors, by virtue of their technical review of the project, affirm that the work program and recommendations presented in the report are in accordance with NI 43-101 and CIM Definition Standards for Mineral Resources and Mineral Reserves.

The external to Eldorado Experts and QP do not have, nor have they previously had, any material interest in Eldorado or its related entities. The relationship with the issuer is solely a professional association between the issuer and the independent consultants. The Technical Report was prepared in return for fees based upon agreed commercial rates, and the payment of these fees is in no way contingent on the results of the Technical Report.

2.2 QUALIFIED PERSONS AND INSPECTION ON THE PROJECT

The Technical Report was assembled by InnovExplo. The qualified persons (QPs) for the Technical Report are:

- Mr. Colm Keogh, P.Eng., Eldorado Gold author of items 2, 3, 15, 16, 18 to 22 and 24; co-author of items 1 and 25 to 27;
- Mr. Jacques Simoneau, P.Geo., Eldorado Gold author of items 4 to 11 and 23; co-author of items 1 and 25 to 27;
- Dr. Stephen Juras, P. Geo., Eldorado Gold. author of items 12 and 14; co-author of items 1 and 25 to 27;
- Ms. Marianne Utiger, P.Eng., WSP Canada inc. author of items 13 and 17; co-author of items 1 and 25 to 27;
- Mr. François Chabot, P.Eng., Eldorado Gold author of item 20; co-author of 1 and 25 to 27.

2.3 SITE VISITS

Mr. Colm Keogh, of Eldorado, visited the Lamaque Project on numerous occasions with the most recent visit on February 27-28, 2018.

Mr. Jacques Simoneau, from Eldorado, works full time on the Lamaque Project, including the Lamaque Project, since February 2015.

Dr. Stephen Juras, from Eldorado, visited the Lamaque Project on numerous occasions with the most recent visit occurring on January 23-25, 2018.

Ms. Marianne Utiger, of WSP, visited the Sigma Mill on January 25, 2018 accompanied by Gabriel Belley of WSP. Several other WSP employees visited the Sigma plant on a weekly or almost daily basis since January 2018.

Mr. François Chabot, from Eldorado, works full time on the Lamaque Project, including the Lamaque Project, since 2013.

2.4 EFFECTIVE DATE

The effective date of this Technical Report is March 21, 2018.

2.5 ABBREVIATIONS, UNITS AND CURRENCIES

All currency amounts are stated in Canadian Dollars ($, CA$, CAD) or US dollars (US$, USD). Quantities are stated in metric units, as per standard Canadian and international practice, including metric tonnes (tonnes, t) and kilograms (kg) for weight, kilometres (km) or metres (m) for distance, hectares (ha) for area, percentage (%) for copper and nickel grades, and gram per metric ton (g/t) for gold, platinum and palladium grades.

SECTION • 3 RELIANCE ON OTHER EXPERTS

Eldorado prepared this document with input from the Lamaque mine staff and other well qualified individuals and third party experts. The qualified persons did not rely on a report, opinion or statement of another expert who is not a qualified person, concerning legal, political, environmental or tax matters relevant to the technical report.

SECTION • 4 PROPERTY DESCRIPTION AND LOCATION

4.1 LOCATION

The Lamaque Project is situated on the southern part of Eldorado's Lamaque properties near the city of Val-d'Or in the province of Québec, Canada, approximately 550 km northwest of Montréal (Fig. 4.1). The coordinates for the approximate centre of the Triangle Deposit which is one of the deposits which form part of the Lamaque Project are latitude 48°4'38'' N and longitude 77°44'4'' W. According to the Canadian National Topographical System (NTS), the project is situated on map sheets 32C/04 and 32C/03, between UTM coordinates 295,700mE and 296,900mE, and between 5,328,200mN and 5,329,350mN (NAD83 projection, Zone 18N). Figures 4.1 and 4.2 show the location of the Lamaque Project.

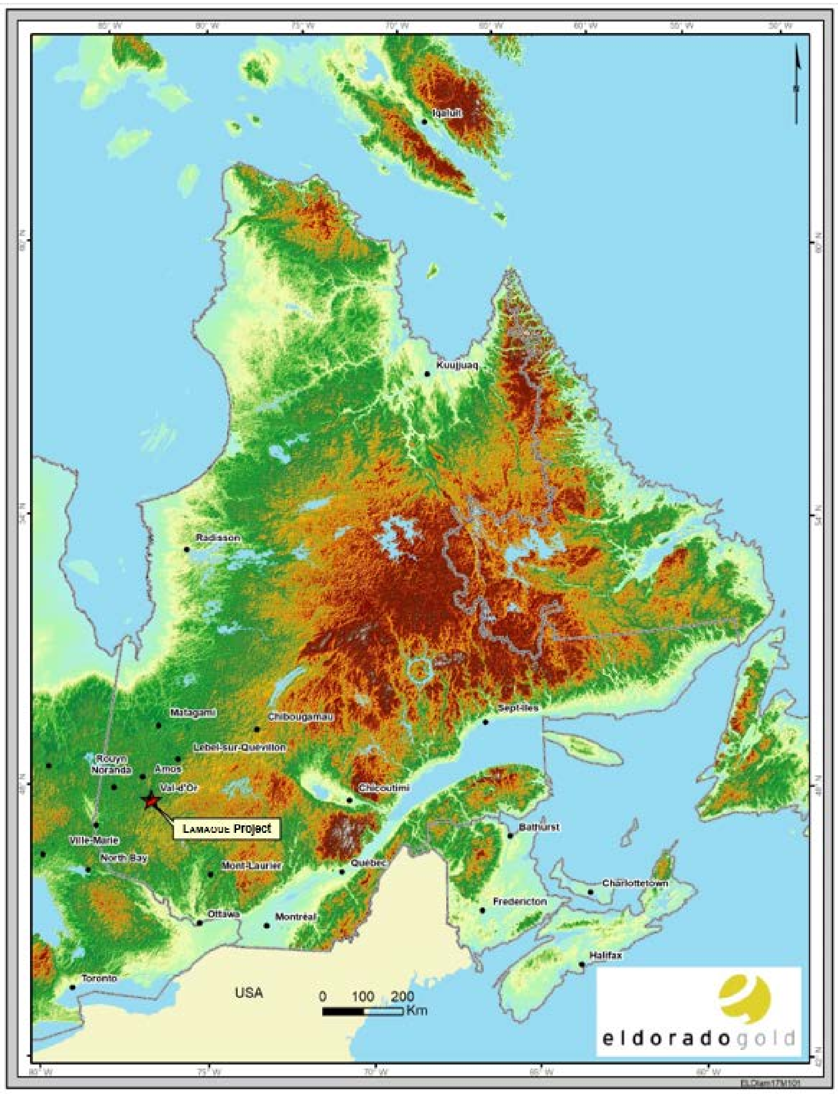


Figure 4.1 – Location of the Lamaque Project in the province of Quebec

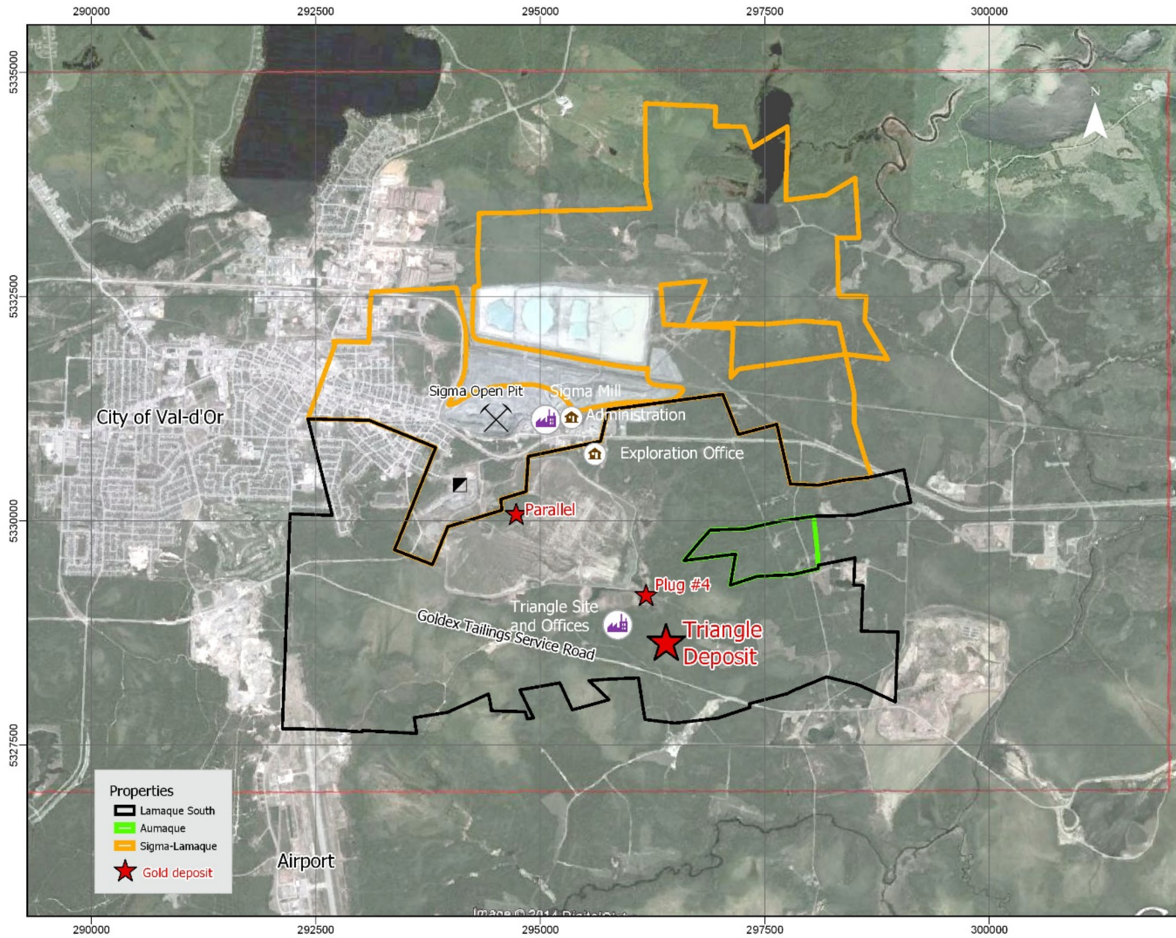


Figure 4.2 – Location of the Lamaque Project with respect to the city of Val-d'Or.

4.2 PROPERTY DESCRIPTION

4.2.1 Land Tenure

The Government of Québec recognizes thirteen (13) types of land registration for mining and exploration. The Lamaque Project consists of map designated claims (CDC) and mining concessions (CM).

The Lamaque Project is included within the land package consisting of the Lamaque property, which covers 3,661.76 ha and comprises ten (10) mining concessions (2,325.09 ha) and 66 claims (1,336.67 ha), all of which are in good standing at the time of this report. The Lamaque property, or group of properties, represents the amalgamation of three (3) separate but contiguous properties: Lamaque South, Sigma-Lamaque and Aumaque (Fig. 4.2). Details of each property are provided below. A complete list of the mining concecessions and mineral claims is shown in Table 4.1. The mining concessions and exploration claims from the Lamaque South Property, which includes the Triangle deposit are registered under the Integra Gold Corp. name, while the mining concessions and exploration claims from the Sigma-Lamaque and Aumaque properties are registered under the

Or Intégra (Québec) Inc. name. Or Intégra (Québec) Inc. is a wholly owned subsidiary of Integra Gold Corp. In July of 2017, Eldorado Gold purchased all outstanding shares of Integra Gold, and therefor is the sole owner of the Lamaque South, Sigma-Lamaque and Aumaque properties.

Lamaque South Property

The Lamaque South Property comprises four (4) contiguous mining concessions (375, 380, 264PTA and 314PTA), and thirty (35) map designated claims (CDC), covering a total of 1,861.11 ha (see Fig. 4.3 and Table 4.1). The titles of the property are registered under the name Integra Gold Corp and are all in good standing. The Triangle deposit, which is the the subject this report, is found within the Lamaque South property.

Sigma-Lamaque Property

The Sigma-Lamaque Property covers 1,733.98 ha and comprises the historical Sigma open pit and underground mine and all associated infrastructures, including a 2,200 tpd capacity mill, a tailings dam, workshops, offices and other ancillary buildings. The property also includes the historical Lamaque underground mine. The underground workings of the Lamaque mine comprise levels 1 to 36 (1,100 m) at a vertical spacing of 30 m, whereas those of the Sigma mine comprise levels 1 to 40 (1,850 m) at variable vertical spacings. The Sigma-Lamaque Property consists of five mining concessions 272, 264PTB, 264PTC, 314 and 318 (total of 979.07 ha) and thirty-one (31) claims (754.91 ha) (see Fig. 4.3 and Table 4.1). The titles are registered under the name Or Intégra (Québec) Inc.

One of the previous operators, Teck Cominco Ltd, granted rights to part of the surface area in the centre of the Sigma-Lamaque Property to the city of Val-d'Or for use as a mining museum, which opened in 1996 (see Fig. 4.4). The museum is managed by the non-profit organization Corporation du village minier de Bourlamaque (La Cité de l'Or). The area includes the headframes, the surviving mine buildings and 100 m of decline into the Lamaque No. 2 mine.

More recently, part of the Sigma-Lamaque Property was developed as an open pit gold mine, operated by Century Mining. The mining concessions in question carry unconditional rights to the surface and rights to mine minerals from the underlying bedrock.

Aumaque Property

The Aumaque Property, is contiguous with the eastern boundaries of the Sigma-Lamaque and Lamaque South properties in Bourlamaque Township. The land package (see Fig. 4.3 and Table 4.1) consists of one (1) mining concession (CM 270). The titles of the property are registered under the name Or Intégra (Québec) Inc.

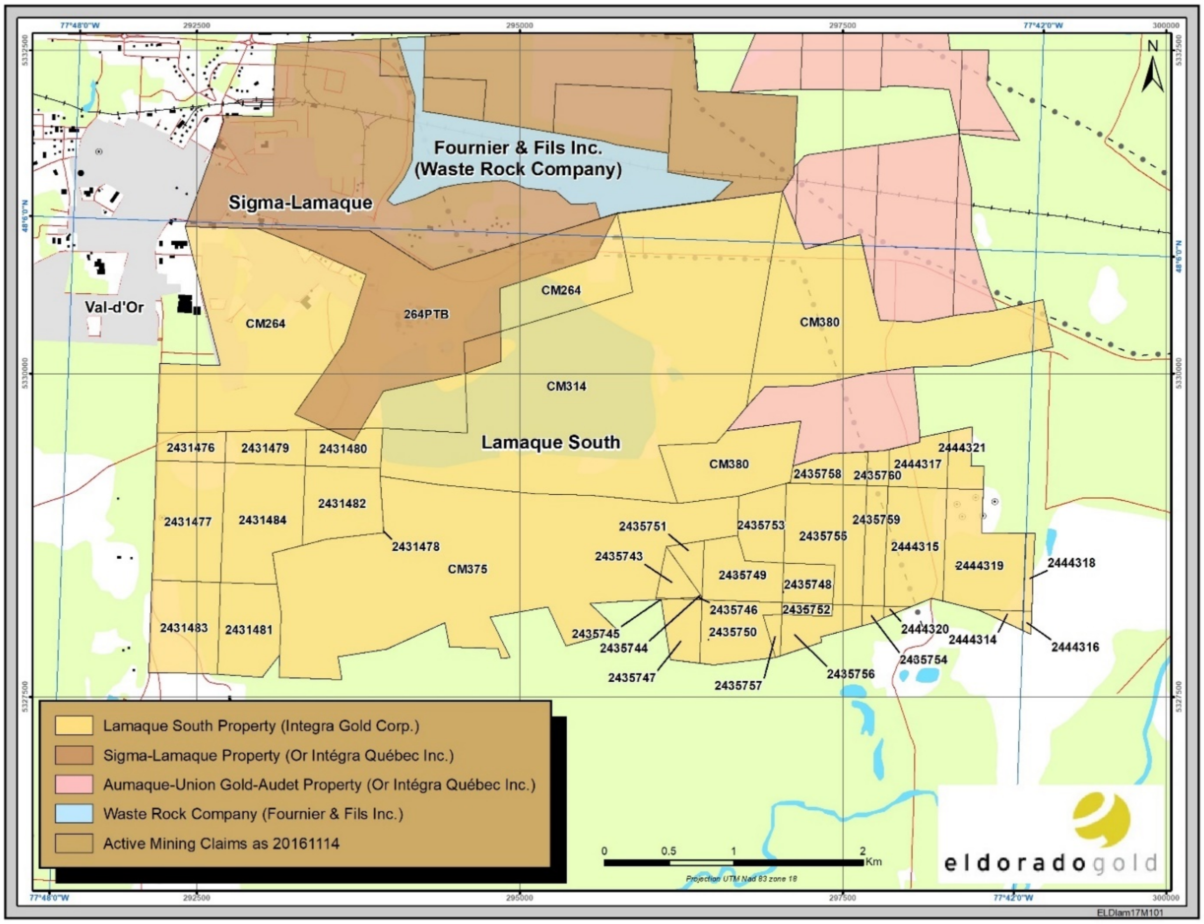


Figure 4.3 – Claim map of the Lamaque Project near Val-d'Or, Québec, Canada. Source: GESTIM (MERN), Government of Québec (as at February 10, 2018)

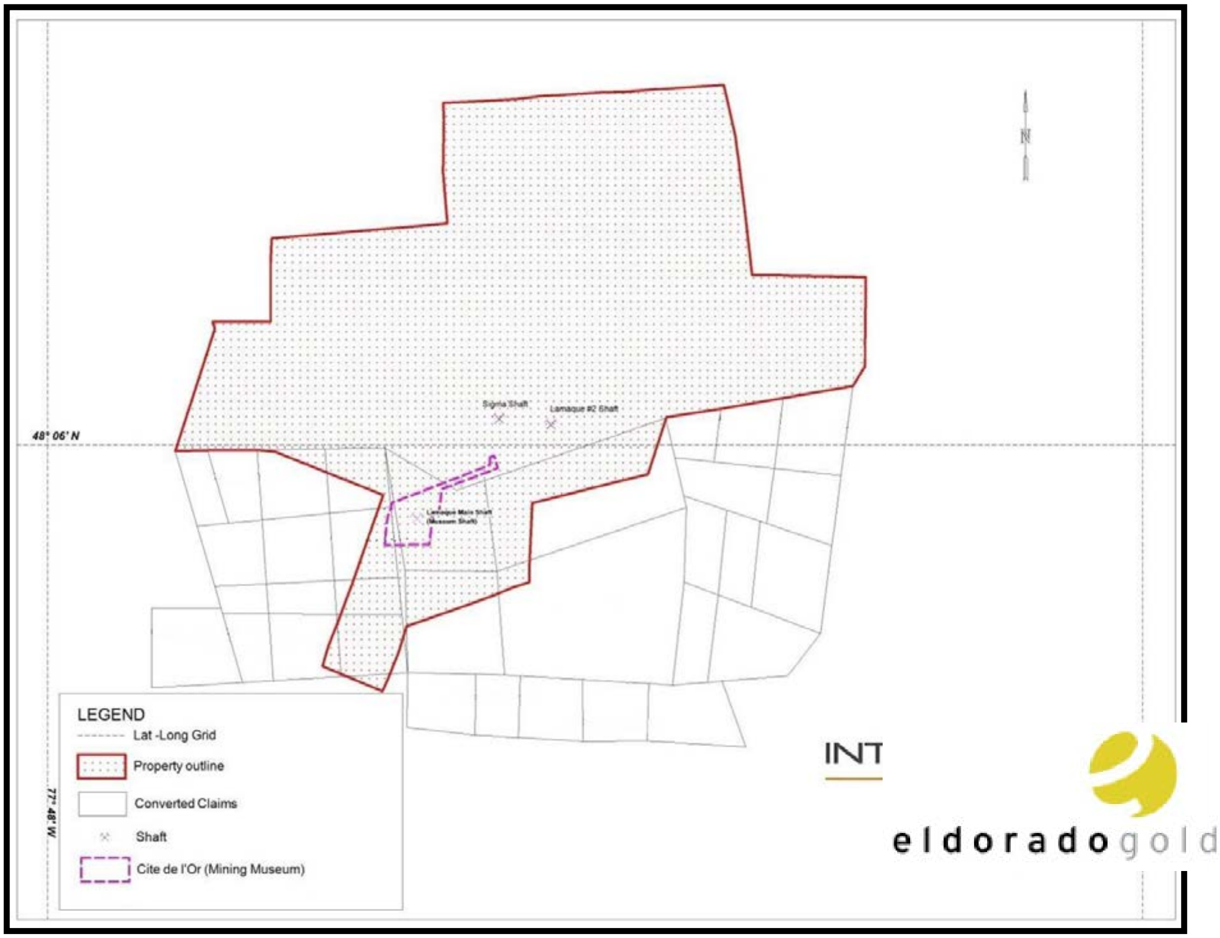


Figure 4.4 –Cité de l'Or mining museum boundary within the Sigma-Lamaque Property. Source: Century Mining (May 13, 2013)

Table 4.1 –Claims and mining concessions composing the Lamaque Project

Property	Type	Title No.	Hectares	Status	Expiry	Company
Aumaque	CM	270	66.67	Active	2019-01-31	Or Intégra (Québec) Inc.
SUB-TOTAL		1 Mining Concession	66.67			
Lamaque South	CDC	2431476	12.00	Active	2019-04-30	Integra Gold Corp.
Lamaque South	CDC	2431477	49.26	Active	2019-04-30	Integra Gold Corp.
Lamaque South	CDC	2431478	0.01	Active	2019-04-30	Integra Gold Corp.
Lamaque South	CDC	2431479	16.07	Active	2019-04-30	Integra Gold Corp.
Lamaque South	CDC	2431480	17.20	Active	2019-04-30	Integra Gold Corp.
Lamaque South	CDC	2431481	32.62	Active	2019-04-30	Integra Gold Corp.
Lamaque South	CDC	2431482	32.98	Active	2019-04-30	Integra Gold Corp.
Lamaque South	CDC	2431483	37.93	Active	2019-04-30	Integra Gold Corp.
Lamaque South	CDC	2431484	51.80	Active	2019-04-30	Integra Gold Corp.
Lamaque South	CDC	2435743	7.22	Active	2019-07-27	Integra Gold Corp.
Lamaque South	CDC	2435744	0.01	Active	2019-07-27	Integra Gold Corp.
Lamaque South	CDC	2435745	0.10	Active	2019-07-27	Integra Gold Corp.
Lamaque South	CDC	2435746	0.22	Active	2019-01-24	Integra Gold Corp.
Lamaque South	CDC	2435747	13.04	Active	2019-01-24	Integra Gold Corp.
Lamaque South	CDC	2435748	12.03	Active	2019-01-24	Integra Gold Corp.
Lamaque South	CDC	2435749	24.30	Active	2019-01-24	Integra Gold Corp.
Lamaque South	CDC	2435750	25.97	Active	2019-01-24	Integra Gold Corp.
Lamaque South	CDC	2435751	6.34	Active	2019-01-24	Integra Gold Corp.
Lamaque South	CDC	2435752	3.57	Active	2019-01-24	Integra Gold Corp.
Lamaque South	CDC	2435753	19.74	Active	2019-09-15	Integra Gold Corp.
Lamaque South	CDC	2435754	2.19	Active	2019-09-15	Integra Gold Corp.
Lamaque South	CDC	2435755	45.51	Active	2019-09-15	Integra Gold Corp.
Lamaque South	CDC	2435756	13.91	Active	2019-09-15	Integra Gold Corp.
Lamaque South	CDC	2435757	3.01	Active	2019-09-15	Integra Gold Corp.
Lamaque South	CDC	2435758	12.58	Active	2019-09-15	Integra Gold Corp.
Lamaque South	CDC	2435759	15.43	Active	2019-09-15	Integra Gold Corp.
Lamaque South	CDC	2435760	4.55	Active	2019-09-15	Integra Gold Corp.
Lamaque South	CDC	2444314	2.61	Active	2018-06-30	Integra Gold Corp.
Lamaque South	CDC	2444315	40.95	Active	2018-06-30	Integra Gold Corp.
Lamaque South	CDC	2444316	1.10	Active	2018-06-30	Integra Gold Corp.
Lamaque South	CDC	2444317	16.57	Active	2018-06-30	Integra Gold Corp.
Lamaque South	CDC	2444318	4.17	Active	2018-06-30	Integra Gold Corp.
Lamaque South	CDC	2444319	45.68	Active	2018-06-30	Integra Gold Corp.

Property	Type	Title No.	Hectares	Status	Expiry	Company
Lamaque South	CDC	2444320	0.92	Active	2018-06-30	Integra Gold Corp.
Lamaque South	CDC	2444321	10.17	Active	2018-06-30	Integra Gold Corp.
SUB-TOTAL		35 Claims	581.76			
Lamaque South	CM	264PTA	357.19	Active	2019-01-31	Integra Gold Corp.
Lamaque South	CM	314PTA	401.11	Active	2019-01-31	Integra Gold Corp.
Lamaque South	CM	375	325.50	Active	2019-01-31	Integra Gold Corp.
Lamaque South	CM	380	195.55	Active	2019-01-31	Integra Gold Corp.
SUB-TOTAL		4 Mining Concessions	1279.35			
Sigma-Lamaque	CDC	2431221	57.50	Active	2019-05-15	Or Intégra (Québec) Inc.
Sigma-Lamaque	CDC	2431222	0.28	Active	2019-05-15	Or Intégra (Québec) Inc.
Sigma-Lamaque	CDC	2431223	19.50	Active	2019-05-15	Or Intégra (Québec) Inc.
Sigma-Lamaque	CDC	2431224	16.33	Active	2019-05-15	Or Intégra (Québec) Inc.
Sigma-Lamaque	CDC	2431225	16.08	Active	2019-05-15	Or Intégra (Québec) Inc.
Sigma-Lamaque	CDC	2431226	1.70	Active	2019-05-15	Or Intégra (Québec) Inc.
Sigma-Lamaque	CDC	2431227	2.31	Active	2019-05-15	Or Intégra (Québec) Inc.
Sigma-Lamaque	CDC	2431228	0.31	Active	2019-05-15	Or Intégra (Québec) Inc.
Sigma-Lamaque	CDC	2431229	8.07	Active	2019-05-15	Or Intégra (Québec) Inc.
Sigma-Lamaque	CDC	2431230	0.02	Active	2019-05-15	Or Intégra (Québec) Inc.
Sigma-Lamaque	CDC	2431231	0.25	Active	2019-05-15	Or Intégra (Québec) Inc.
Sigma-Lamaque	CDC	2431232	40.61	Active	2019-05-15	Or Intégra (Québec) Inc.
Sigma-Lamaque	CDC	2431233	43.45	Active	2019-05-15	Or Intégra (Québec) Inc.
Sigma-Lamaque	CDC	2431234	2.52	Active	2019-05-15	Or Intégra (Québec) Inc.
Sigma-Lamaque	CDC	2431235	52.12	Active	2019-05-15	Or Intégra (Québec) Inc.
Sigma-Lamaque	CDC	2431236	46.82	Active	2019-05-15	Or Intégra (Québec) Inc.
Sigma-Lamaque	CDC	2431237	18.07	Active	2019-05-15	Or Intégra (Québec) Inc.
Sigma-Lamaque	CDC	2431238	34.27	Active	2019-05-15	Or Intégra (Québec) Inc.
Sigma-Lamaque	CDC	2431239	36.18	Active	2019-05-15	Or Intégra (Québec) Inc.
Sigma-Lamaque	CDC	2431240	31.73	Active	2019-05-15	Or Intégra (Québec) Inc.
Sigma-Lamaque	CDC	2431241	37.16	Active	2019-05-15	Or Intégra (Québec) Inc.
Sigma-Lamaque	CDC	2431242	49.99	Active	2019-05-15	Or Intégra (Québec) Inc.
Sigma-Lamaque	CDC	2431243	25.69	Active	2019-05-15	Or Intégra (Québec) Inc.
Sigma-Lamaque	CDC	2431244	34.11	Active	2019-05-15	Or Intégra (Québec) Inc.
Sigma-Lamaque	CDC	2431245	13.18	Active	2019-05-15	Or Intégra (Québec) Inc.
Sigma-Lamaque	CDC	2431246	44.42	Active	2019-05-15	Or Intégra (Québec) Inc.
Sigma-Lamaque	CDC	2431247	33.91	Active	2019-05-15	Or Intégra (Québec) Inc.
Sigma-Lamaque	CDC	2431248	16.18	Active	2019-05-15	Or Intégra (Québec) Inc.

Property	Type	Title No.	Hectares	Status	Expiry	Company
Sigma-Lamaque	CDC	2431249	12.39	Active	2019-05-15	Or Intégra (Québec) Inc.
Sigma-Lamaque	CDC	2431250	55.49	Active	2019-05-15	Or Intégra (Québec) Inc.
Sigma-Lamaque	CDC	2431251	4.27	Active	2019-05-15	Or Intégra (Québec) Inc.
SUB-TOTAL		31 Claims	754.91			
Sigma-Lamaque	CM	264PTB	279.38	Active	2019-01-31	Or Intégra (Québec) Inc.
Sigma-Lamaque	CM	264PTC	82.69	Active	2019-01-31	Or Intégra (Québec) Inc.
Sigma-Lamaque	CM	272PTA	311.92	Active	2019-01-31	Or Intégra (Québec) Inc.
Sigma-Lamaque	CM	314PTB	126.97	Active	2019-01-31	Or Intégra (Québec) Inc.
Sigma-Lamaque	CM	318PTA	178.11	Active	2019-01-31	Or Intégra (Québec) Inc.
SUB-TOTAL		5 Mining Concessions	979.07			
TOTAL			3661.76			

4.2.2 Royalties

4.2.2.1 Royalty on the Sigma-Lamaque Property

When Century Mining purchased the Lamaque part of the Sigma-Lamaque Property from Sigma–Lamaque Limited Partnership, the property was subject to a sliding scale net smelter return ("NSR") royalty owned by Société Québecoise d'Exploration Minière (SOQUEM), a Québec government corporation. This royalty was subsequently bought out for CA$2 million.

4.2.2.2 Royalty and agreement on the Lamaque South Property

The Lamaque South property has been the subject of several agreements over the years involving different companies. Although all of the claims and mining concessions of the property are 100% owned by Eldorado Gold through their acquisition of Integra Gold, several of these past agreements included royalties to various companies. The following text is a summary of these agreements. Table 4.2 summarizes the current royalties on the project and figure 4.5 shows the location of where these royalties apply on the map.

Following the closure of the Lamaque mine, Teck Corporation entered into joint venture agreements with Golden Pond Resources Ltd ("Golden Pond") and Tundra Gold Mines Inc. ("Tundra") to explore a portion of the former Lamaque Mine Property in 1985. The Golden Pond JV and some of the Tundra JV covered most of the ground now owned by Integra.

In June 2003, Kalahari Resources Inc. ("Kalahari"; predecessor company to Integra Gold) entered into an option agreement with Teck Cominco Ltd (Teck) to earn a 50% to 53% interest in claims covering approximately 1,244 hectares, which it called the Lamaque Project. A 2% NSR royalty is payable on the property to Teck, of which half (1%) may be purchased for CA$2 million within one year of commercial production. On September 22, 2009, Kalahari entered into an option agreement with Alexandria Minerals Corp. ("Alexandria") to earn a 100% interest in the former Roc d'Or East Extension Property. Over a 3-year period, Kalahari fulfilled this agreement by paying CA$25,000 in cash and issuing 500,000 shares to Alexandria. A 2% NSR is payable to Alexandria on the titles of this historical property, of which half (1%) may be purchased for CA$1 million. This claim group is adjacent to the issuer's 100% owned Roc d'Or East claims, which became part of its Lamaque South Property.

In October 2009, Kalahari entered into separate agreements with Tundra and Golden Pond (Kalahari's joint venture partners on what was then known as the Lamaque Property) to purchase their interests and consolidate a 100% ownership of the Lamaque Property. The objective was to initiate more advanced exploration. The agreements required the issuance of 9,593,128 shares to Tundra and 2,902,861 shares to Golden Pond

In December 2010, Kalahari changed its name to Integra Gold Corp. In addition to retaining its 100% ownership of the Lamaque Property, Integra also acquired 100% of Teck's interest in the adjacent Roc d'Or East and Roc d'Or West claims. There is a 2% NSR payable to Teck on all former Teck claims, of which half (1%) may be purchased for CA$2 million at any time within one year of commercial production.

In December 2010, Integra acquired an option to earn a 100% interest in the Bourlamaque Property (2 claims; 16 hectares) in Bourlamaque Township, adjacent to the Lamaque Property. Consideration for the property acquisition consisted of CA$3,500 cash and 10,000 shares. In addition to fulfilling these terms, Integra also purchased the entire NSR for CA$5,000 on April 30, 2013. Therefore, there is no outstanding royalty on the Bourlamaque Property.

In June 2011, Integra entered into an option agreement to acquire a 100% interest in the MacGregor Property located in Bourlamaque Township, adjacent to the Bourlemaque Property. The claims are subject to a 2% NSR, 0.6% of which is payable to Jean Robert, 0.6% to Les Explorations Carat and the remaining 0.8% to Albert Audet. One-half (1%) of this NSR may be purchased for CA$500,000.

In January 2012, Integra entered into an option to acquire a 100% interest in the Donald Property located in Bourlamaque Township, adjacent to the Lamaque South Property. The claims were subject to a 3% gross metal royalty payable to Les Entreprises Minière Globex Inc., one-third (1%) of which may be purchased for CA$750,000.

In October 2015, Osisko Gold Royalties Ltd entered into a definitive agreement to acquire a 2% NSR royalty held by Teck on the Lamaque Property for a cash consideration.

In October 2015, Sandstorm Gold Ltd acquired the 2% NSR royalty held by Alexandria on the Roc d'Or East Extension claims owned by Integra.

4.2.3 Exploration Permit

No permit is needed if mapping, sampling and geophysical surveys are to be conducted on the mineral claims, as long as there is no disturbance of the natural environment.

A regular "Permis d'Intervention en Forêt" must be obtained from the Ministère de Forêts, Faune et Parcs (MFFP in order to conduct drilling, trenching, stripping or any other surface disturbance on the property. These permits need to be obtained each time a surface disturbance is contemplated. To obtain the permits, the claim holder must indicate the location and type of work that will be conducted on the application. The permits can usually be obtained within two weeks.

Table 4.2 – Royalties summary table

Property	Owner	Royalty type	%	Nsr company	By-back clause	Amount	Comment
Sigma-Lamaque	Or Integra (Québec) Inc.	none					
Aumaque	Or Integra (Québec) Inc.	none					
Lamaque Roc d'Or West Roc d'Or East	Integra Gold Corp.	NSR	2%	Osisko Royalties	1%	1M$	Osisko acquired NSR from Teck in 2015
Roc d'Or East Extension (CL 3691171)	Integra Gold Corp.	NSR	2%	Sandstorm	1%	1M$	Triangle Claim. Sandstorm acquired NSR from Alexandria in 2015
Bourlamaque	Integra Gold Corp.	none					
Donald	Integra Gold Corp.	GMR	3%	Globex	1%	750K$	
McGregor	Integra Gold Corp.	NSR	2%	Jean Robert (0.6%) Les Explorations Carat (0.6%) Albert Audet (0.8%)	1%	500K$	

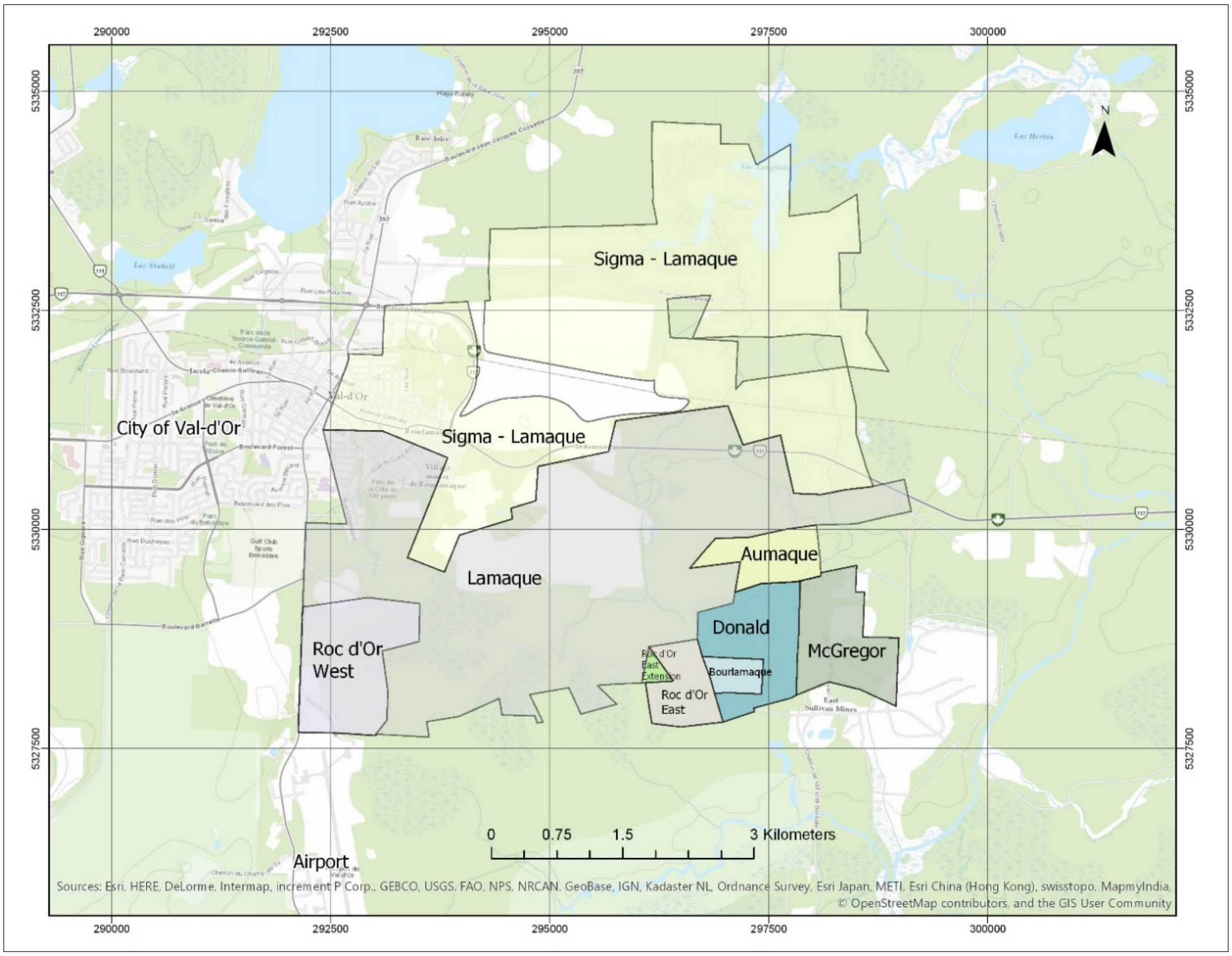


Figure 4.5 – location of historical property in relation to royalties.

4.2.4 Location of Mineralization

All mineralized zones or areas that the issuer plans to explore and potential exploitation of gold deposits that are the subject of this report are located within the boundaries of the three properties consisting of the Lamaque Project.

4.2.5 Comments

To the extent known, there are no significant factors or risks besides those discussed in this report that may affect the issuer's right or ability to perform work on the Lamaque Project.

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

5.1 ACCESSIBILITY

The Lamaque Project lies to the southeast of the Val-d'Or urban center (Fig. 5.1). The property is accessible via a public gravel road from the Provincial Highway 117 before eastern city limits or via 7e Street and the Chemin Des Gravieres street in southern city limits (Fig. 5.1).

The Val-d'Or airport is located at the southern edge of the property through the 7e street and has regularly scheduled flights to and from Montréal. Val-d'Or is a six-hour drive north from Montréal and has daily bus service between Montréal and other cities in the Abitibi region.

Canadian National Railroad (CN) operates a feeder line that runs through Senneterre and Amos, connecting to the North American rail system eastward through Montréal and westward through the Ontario Northland Railway. A CN branch line runs through Val-d'Or and crosses the Lamaque Project. Passenger rail service is offered by VIA Rail from Montréal to Senneterre (65 km northeast of Val-d'Or) on Monday, Wednesday and Friday, and from Senneterre to Montréal on Tuesday, Thursday and Sunday.

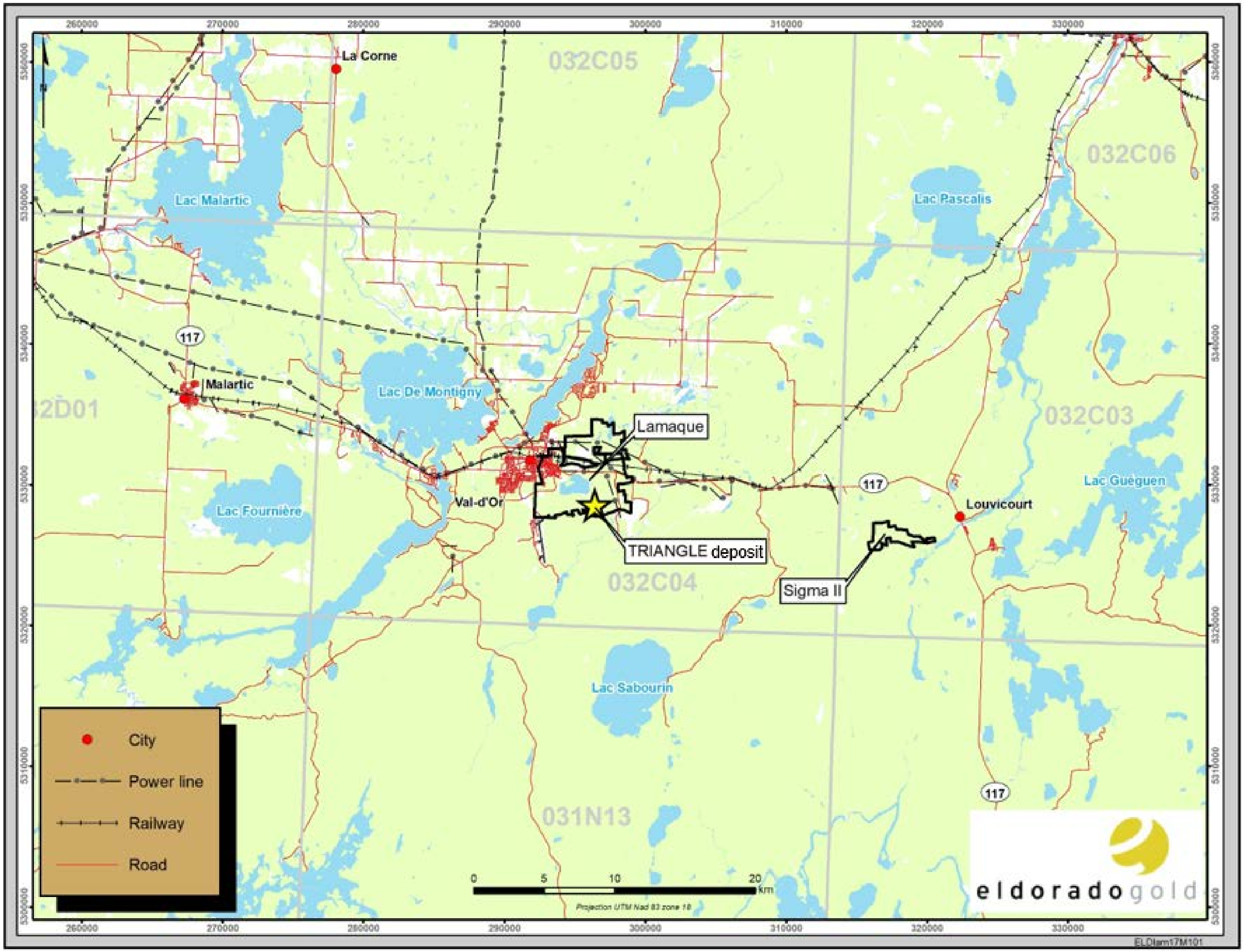


Figure 5.1 – Access and waterways of the Lamaque Project and surrounding region

Figure 5.2 – Location and access of the Lamaque Project, Bourlamaque Township, Province of Québec

5.2 CLIMATE

The city of Val-d'Or has a humid continental climate that closely borders on a subarctic climate. Winters are cold and snowy, and summers are warm and damp. Based on Environment Canada statistics from 1971 to 2000, the region is characterized by a mean daily temperature of +1° C (Table 5.1). The lowest recorded temperature was -43.9° C and the highest recorded temperature was +36.1°C. The average high in July was +23.4° C. and the average low in January was -23.5° C. In winters, temperatures can drop to below -30° C for extended periods, and extreme temperatures below -40° C can occur from December through March.

Table 5.1 – Annual temperature data for the period 1971 to 2000

Parameters	Jan.	Feb.	March	April	May	June	July	Aug.	Sept.	Oct.	Nov.	Dec	Annual
Maximum (°C)	-10.9	-8.6	-1.5	6.6	16.1	21	23.4	21.7	15.5	8.5	0.1	-7.6	7
Minimum (°C)	-23.5	-21.9	-14.6	-5	2.7	7.8	11	9.7	4.6	-0.5	-8.2	-18.9	-4.7
Mean (°C)	-17.2	-15.3	-8.1	0.8	9.4	14.4	17.2	15.8	10.1	4	-4.1	-13.3	1

Note: Table supplied by Century Mining Corporation for the August 2011 Technical Report (Lewis et al., 2011).

The mean annual precipitation over the same 30-year period was approximately 909 mm, broken down as 631 mm of rain and 296 cm of snow (Table 5.2). The heaviest rainfall occurs between May and October. September receives the most rain, with a total of 99.8 mm. Snow generally falls from October to May, with reliable snow cover from November to April. The heaviest snowfall occurs during December and January, with means of 61 and 56 cm, respectively.

Evaporation is highest during the summer months and virtually nil during the winter. The annual mean evaporation and evapo-transpiration rates are 541 mm and 489 mm, respectively.

Table 5.2 – Annual precipitation data for the period 1971 to 2000

Parameters	Jan.	Feb.	March	April	May	June	July	Aug.	Sept.	Oct.	Nov.	Dec.	Annual
Rain (mm)	5.5	3.4	20.1	35.8	75	92.4	95.4	93.2	99.8	72.2	34.1	8.3	631
Snow (cm)	56	40.8	48.6	29.2	2.5	0.3	0	0	1.9	14.6	45.5	61	296
Total precipitation (mm)	56	40.5	65.2	66	77.7	92.7	95.4	93.2	101.9	86.6	76.2	62.5	909

Note: Table supplied by Century Mining Corporation for the August 2011 Technical Report (Lewis et al., 2011)

Winds are generally light. During storm events, sustained winds have been recorded at speeds ranging from 48 to 63 km/hr, with gusts up to 119 km/hr. Winter storms with snow accumulation of up to 45 cm have been recorded in recent years, although they are rare. Between August and January, a southerly wind is dominant, whereas a northwesterly wind is more common between February and July.

5.3 LOCAL RESOURCES AND INFRASTRUCTURES

Val-d'Or was founded in the 1930s and has a long and rich mining heritage. Val-d'Or, with a current population of approximately 32,781 persons in 2014 (Statistics Canada), is a modern city and one of the largest communities in the Abitibi region of Québec. Both the Lamaque and Sigma mines are located within the municipality of Val-d'Or. Historically, these two mines were the largest producers in the area. Val-d'Or has been a mining service center since its inception.

All requirements, including a quality supply of hydro-electric power to support a mining operation, are available in Val-d'Or, and there is an ample supply of water on or near the Lamaque Project to supply a mining operation (See section 18). Also available is a local skilled labor force with experienced mining and technical personnel. A number of mining and mineral exploration companies have offices located in the area. Local resources include the following:

- Assayers – commercial laboratories;
- CANMET – federal government underground mining research office;
- Civil construction companies;
- Diamond drilling – multiple contractors;
- Engineering firms;
- Freight forwarding;
- Geology – consultants;
- Geophysics – contractors;
- Land surveyors;
- Mining contractors;
- Mining industrial suppliers – including diesel engines, explosives suppliers, machine shops, mechanics, electrical and cable, electronics, tires.

5.4 PHYSIOGRAPHY

The Abitibi region has a typical Canadian Shield-type terrain characterized by low local relief with occasional hills and abundant lakes. The average topographic elevation is approximately 300 m above sea level (masl) and generally varies less than 100 m. Large areas are dominated by swamps and ponds. Local flora in the area are predominantly spruce, pine, fir and larch, with a much smaller percentage of deciduous trees, such as birch and poplar.

The mine site is bordered to the north by a large unpopulated wooded area, a portion of which is currently used for tailings and waste rock disposal. A large swamp partially covers parts of the property, while spruce forest and mixed deciduous and coniferous forest cover the eastern, western and southern extremities. The elevation difference at the project rarely exceeds 50 m, except where eskers and glacial deposits are found. The property is at an elevation of about 320 masl.

The old tailings retention area from the Lamaque mine covers a large part of the central part of the property. This tailings area is generally populated with herbaceous growth, grasses and areas of small trees planted by previous operators. Spruce forest and mixed deciduous and coniferous forest cover much of the rest of the property.

SECTION • 6 HISTORY

This item is modified excerpt from the technical report by Poirier et al. (2015a). The history of the Sigma and Lamaque mines in that report was based largely on the technical report of Lewis et al. (2011). It has been updated where applicable.

Gold was first discovered in the Val-d'Or area in 1923 by R.C. Clark on what later became the Lamaque Property. The gold was contained in a small quartz vein within a narrow shear zone, and the pocket of coarse gold was removed in a single blast from the otherwise barren vein. Intensive prospecting by trenching under George Kruse resulted in the discovery of the No. 3 vein in 1924. The No. 1 vein was also stripped and trenched, but did not carry significant gold.

6.1 HISTORY OF THE MINES

6.1.1 Lamaque Mine (Lamaque Gold Mines Ltd)

In 1924, William Read took an option on R.C. Clark's claims. In November 1928, in partnership with Hector Authier, the company Read-Authier Mines Ltd ("Read-Authier") was formed to acquire property in Bourlamaque Township. In 1929, Read-Authier drilled 19 core holes for a total of 2,143 m to test the veins along strike and at depth. Results were encouraging but inconclusive. In the late summer of 1932, Major MacMillan optioned Read-Authier's southern claim group for Teck-Hughes Gold Mines Ltd ("Teck-Hughes") and drilled five (5) holes totalling 520 m to check the previous results. Teck-Hughes subsequently exercised its option and formed Lamaque Gold Mines Ltd ("Lamaque Gold") in December 1932. Lamaque Gold took over the original discovery and a number of adjoining claims, with Read-Authier retaining a 30% interest in the original claims.

Shaft sinking started in January 1933, followed by lateral development and mill construction. The mine officially started up in March 1933 with an "ore reserve" of 67,580 metric tons at 10.62 g/t Au. Development was accelerated on March 3, 1933, when U.S. President Franklin D. Roosevelt devalued the U.S. dollar and the official gold price jumped from US$20.67 to US$35.00 an ounce. Sufficient ore was subsequently developed to justify a mill, with construction starting in the summer of 1934. Later, shafts were sunk adjacent to the Main (or No. 1) mine, including the No. 2, 3, 4, 5, 6 and 7 shafts, and the East and West mine areas were developed.

The No. 2 mine, was developed in 1950–1951 approximately 1,158.24 m (3,800 ft) northeast of the Main mine area (not to be confused with the No. 2 Shaft located on the Lamaque South Property), both close to and along the extension of Sigma mine structures. Nine levels were developed to a depth of 410.6 m (1,347 ft). Production from the No. 2 mine ceased on November 30, 1955.

Gold production commenced at the Lamaque mine in April 1935, with an initial mill capacity of 250 short tons per day. Mill capacity was increased to 500 tons per day by November 1935, and to 1,200 tons per day by December 1937. During World War II, the mill operated at reduced tonnage due to the war effort. In 1951, the mill capacity was raised to 1,500 tons per day, and in 1953 it increased again to 2,100 tons per day. Production was cut back to 1,800 tons per day in 1972. Operations ceased in June, 1985 and the mill was demolished in 1992.

The Lamaque No. 3 mine was developed in 1961. It was developed to a depth of 223.7 m (734 ft) and included the No.3, No. 4 and No. 5 plugs. Production ceased at the No. 3 mine in 1967.

From 1952 through to 1985, the Lamaque mine was the largest gold producer in Québec. In 1985, the Lamaque mine closed.

The principal mining area of the property was acquired by Placer Dome in November 1993, and limited production took place in the No. 2 mine area (these production figures are included in the Sigma mine data in Table 6.2). The surface rights were acquired by Placer Dome in October 1999. No mining or underground development was conducted between 1999 and 2010.

In September 2004, Century Mining Corporation Inc. ("Century Mining") purchased the Sigma and Lamaque mines, and the company re-opened the Lamaque mine in 2010 (see Sigma-Lamaque section below). The mine was shut down in May 2012 and is currently on care and maintenance.

At its peak of production, the Lamaque mine employed 215 persons underground from a total payroll of 385. Total production figures for the principal mining areas at the Lamaque mine are shown in Table 6.1.

Table 6.1 – Total production figures for the principal mining areas of the Lamaque mine (1935 to 1985)

Mining Area	Tonnes Milled	Gold Grade (g/t)	Ounces Produced
Main Plug	18,166,848	6.34	3,695,194
East Plug	2,721,397	3.94	343,827
West Plug	1,491,952	4.56	219,014
No. 2 Mine	1,482,775	4.97	237,596
No. 3 Mine	318,560	6.30	58,536
Total Production	24,151,963	5.86	4,554,167

6.1.2 Sigma Mine (Sigma Mines, Placer Dome, McWatters)

In the summer of 1933, Read-Authier sent consulting engineer Herber Bambick to inspect its north claim group. At the time, the area was accessible by water from Amos (93 km) or by sleigh from in winter the CN railway at Barraute (61 km). In October 1933, Bambick discovered a vein from which encouraging results were obtained after conducting a trenching program with a roughly 90 m strike. This was followed by a diamond drilling program.

Dome Mines Ltd ("Dome Mines"; later Placer Dome Inc. (1987)) was invited to examine the property that same year. James B. Redpath, a recent mining engineering graduate from McGill University, checked the sampling results, and an agreement to purchase was negotiated by J.G. McCrea in February 1934. Read-Authier retained a 40% interest. Sigma Mines Ltd was incorporated in April 1934, and reincorporated in 1937 as Sigma Mines (Québec) Ltd ("Sigma Mines").

By the end of 1934, a mining camp had been erected with accommodation for 50 men. A diamond drilling program totalling 3,350 m had been completed, revealing a mineralized zone 365 m long and 75 m deep. A second parallel zone of mineralization was discovered 60 m to the north. An inclined shaft (No. 1 shaft) was sunk at 65° on the southern zone to a depth of 80 m. During the first year, 1,632 m of underground development partially opened up the two zones, revealing excellent grades and widths.

In 1935, the No. 2 vertical shaft was sunk to a depth of 300 m. Exploration identified irregularly distributed gold in 7 zones. In 1936, further diamond drilling confirmed the continuity of the mineralization down to 300 m. In June, 1936, construction started on a 300 ton per day cyanide plant that could be expanded to 500 tons per day. The mill was expanded to full capacity the following year, and by 1938 the mill was operating at 650 tons per day. In 1938, the No. 2 shaft was deepened to 610 m. The mill capacity continued to expand, such that by late 1939, it reached 750 tons per day. In 1940, the capacity was increased to 1,000 tons per day and by 1942 the plant was operating at 1,100 tons per day.

In 1938, Read-Authier was dissolved with shareholders receiving 38 shares of Lamaque Gold, 21 shares of Sigma Mines and 100 shares of Union Mining Corporation (Union Mining) for each 100 Read-Authier shares held.

During World War II, supply and labour shortages reduced production to 800 tons per day for the duration of the war. During this period, mining of the more labour-intensive high-grade flat veins was suspended in favour of the higher volume but lower grade steep veins and dykes. Mining operations returned to pre-war levels by 1948.

In 1952, the sinking of the No. 2 shaft reached its final depth of 1,018 m at the 25th level. In 1958, sinking began on the No. 3 shaft from the 22nd level. By 1960, drifting on the new 30th level indicated that mineralized shoots contained grades comparable to the upper part of the mine. In 1972, the No. 3 shaft reached its final depth of 1,817 m below surface, 53 m below the 40th level.

On August 15, 1971, U.S. President Richard Nixon ended the gold exchange standard and the gold price was allowed to float. Between August 1972 and May 1974, mill capacity was expanded to 1,460 tons per day, and further expanded to 2,200 tons per day in 1995.

In September 1997, Placer Dome Inc. (formerly Dome Mines) sold the Sigma mine to McWatters Mining Inc. ("McWatters"). In 1998, McWatters reduced underground production. The next year, the company reduced it by another 500 metric tons per day (tpd). In July 1999, McWatters closed the underground mine just 22 months after it took over operations. Although McWatters' underground production records appear to be incomplete, it is estimated that 350,000 metric tons were mined from the underground operations under McWatters' tenure.

In 1998, a small open pit was developed behind the Lamaque shaft. In 1999 and 2000, limited open pit operations extracted roughly 377,000 tonnes of ore with an average grade of 2.73 g/t Au, which was processed in the Sigma concentrator. The mill was expanded to 3,000 tpd in 2000 and to 5,000 tpd in 2002. The development of a larger open pit started in November 2002, with ore processing beginning in early 2003. The McWatters open pit operation never reached commercial production (defined as 60% of design capacity for a period of 90 consecutive days). All the McWatters mining operations were shut down in October 2003, and McWatters was placed into bankruptcy.

6.1.3 Lamaque and Sigma Mines (Century Mining)

Century Mining purchased the Sigma and Lamaque mines in September 2004 and re-started the Sigma open pit mine. The open pit was closed in the fall of 2007 and production commenced from underground. In July, 2008, underground production was suspended and the mine was put on care and maintenance due to economic and financial considerations.

In 2010, the Lamaque mine was re-opened. Production was sourced mainly from the narrow, horizontally lying, flat veins. Mining and development used trackless methods, and a low profile fleet was acquired for mining in the flats. Due to the undulating and thin nature of the flat veins, significant dilution was encountered during stoping. Development and some limited production took place in the Bédard Dyke area. In the North Wall area, a contractor developed access and infrastructure for future vertical stoping areas. These stoping areas were designed to encompass the majority of the near and medium term production sources. Access for all three areas was gained via portals and declines developed from within the old Sigma open pit.

Table 6.2 summarizes the total historical production from the Lamaque and Sigma mines to the end of May 2012.

Table 6.2 – Total production from the Sigma and Lamaque mines to end of May 2012 (Lewis et al., 2011).

Mine Operator	Operating Period	Production Figures		
		Tonnes	Grade (g/t)	Oz
Lamaque Gold	1935 to 1985	24,151,963	5.9	4,554,167
Sigma Mines*	1937 to 1997	23,898,243	5.8	4,456,420
McWatters	1997 to 2003	3,724,000	2.2	263,405
Century Mining	2004	0	0	-
	2005	1,112,746	1.6	57,241
	2006	1,415,530	1.6	72,817
	2007	1,155,937	1.5	55,747
	2008	46,719	3.2	4,807
	2009	0	0	0
	2010	157,561	2.9	14,691
	2011	176,918	2.57	14,618
	2012	73,570	2.1	4,967
Total		55,913,187	5.3	9,498,880

* Includes limited production from the Lamaque No. 2 mine area after Placer Dome purchased a portion of the Lamaque Property in 1993.

6.2 LAMAQUE SOUTH PROPERTY

In order to explore a portion of the historical Lamaque Property, Teck entered into joint venture agreements with Golden Pond Resources Ltd ("Golden Pond") and Tundra Gold Mines Inc. in 1985. The Golden Pond JV and some of the Tundra JV covered most of the ground now owned 100% by

Integra (excluding the Lamaque mine area under the Tundra JV), but in addition included two small claims at the southern limit of the Villemaque Block (claims previously identified as 422883-2 and 421475-2). The Tundra JV also included two non-contiguous parcels: the first parcel of land centred on the No. 5 Plug, and the second parcel centred on the No. 4 Plug. Teck was the operator for both the Golden Pond and Tundra JV programs.

In December 1988, Tundra signed an agreement with Teck to acquire a 100% interest in all of Teck's assets at Lamaque. The assets to be acquired included the Main mine property, all surface structures including the mill, surface and underground equipment, and Teck's interest in the Tundra, Golden Pond and Roc d'Or Mines agreements. The purchase price for the assets was $8 million. Tundra was also required to complete an exploration program and sink an exploration shaft to 304.8 m (1,000 ft) on the No. 4 Plug. Preliminary work was initiated to meet the obligations of the agreement, but a downturn in the industry made funding difficult and the 1988 option was never exercised, leaving Teck with a 100% interest in the Main mine and mill area, which was eventually optioned and purchased by Placer Dome Inc. Subsequently, Tundra's and Golden Pond's interest in the Tundra and Golden Pond JV properties was diluted to 50% due to non-payment of their respective portions of lease rentals, assessment filings and taxes.

No exploration was conducted on the Tundra and Golden Pond JV properties between 1990 and 2003, when Kalahari Resources Ltd ("Kalahari"; now Integra Gold) and Teck Cominco signed an agreement providing Kalahari the option to earn Teck's interest in the JV properties. In 2009, Kalahari purchased the remaining Tundra and Golden Pond interests in the properties through a share swap. Kalahari changed its name to Integra in December 2010 and now owns 100% of the property.

During the period between January 2003 and December 2014, exploration work was completed on the Lamaque South Property, mainly via drilling campaigns. Over 156,248 metres of drilling was completed, mainly on various geophysical targets and the following zones: Fortune, Parallel, Triangle, South Triangle, No. 6 Vein, No. 4 Plug, No. 5 Plug, Sigma Vein Extension, Mylamaque and Sixteen Zone. The various drilling programs, and their results, have been discussed in detail in previous NI 43-101 technical reports, all of which were filed on SEDAR website (Risto et al., 2004; Beauregard et al., 2011; 2012; 2013; 2014, Poirier et al., 2014; Poirier et al., 2015a; Poirier et al., 2015b).

The drilling during that time was completed by Orbit-Garant Drilling from Val-d'Or, Québec. Analyses were completed by Bourlamaque Assay Laboratory and ALS Canada in Val-d'Or. Exploration work from 2003 to 2008 was supervised by Don Cross and Terrence Coyle; and from 2009 to the end of 2014, Geologica Groupe-Conseil Inc. ("Geologica") was responsible for exploration guidance, geoscientific compilation, drill hole planning, supervision, logging and data validation, as well as the geological interpretation of mineralized zones on cross-sections and longitudinal sections.

In 2009, Geologica was responsible for establishing QA/QC sampling protocols, and these protocols were followed for all drilling campaigns. Each drill hole had duplicates, blanks and standards.

In 2009, Geologica began a re-sampling program of diamond drill core from the 2003–2008 campaigns. This re-sampling program included the addition of QA/QC samples. The total in 2009 was 1,654 samples from 121 holes, and 319 QA/QC samples.

6.3 DRILLING ON THE SIGMA-LAMAQUE PROJECT

6.3.1 Pre-2004 (previous operators)

Exploration in the Lamaque and Sigma mines before 2004 was conducted using EX core for short underground diamond drill holes and AX and AQ for longer underground and surface diamond drill holes. Previous operators had drilled approximately 40,000 diamond holes at Lamaque from both surface and underground during the 1970s (Burns and Mark, 2009). Previous geological teams considered the small core size to be adequate for sampling the deposit. Rock strength was high, estimated to range from 25,000 to 50,000 psi.

Core recovery in all environments on the property was generally 100%. Core loss was very rare as highly stressed ground that could contribute to core grinding was absent, and faults and shears rarely contained broken ground (Burns and Mark, 2009).

6.3.2 Century Mining (2004–2012)

During the period from 2004 to 2012, Century Mining completed a limited amount of surface diamond drilling on the property.

Between 2010 and 2012, Century Mining conducted an exploration and definition drilling program to delineate and characterize mineralization in the Bédard Dyke. The objective of the drilling program was to test the extent of the dyke, as well as the gold grade of the mineralization contained within it. The drilling program was conducted originally from surface and later from underground, once access was available. The surface diamond drilling was initiated to test the overall extent of the Bédard Dyke and the general grade of the mineralization, while the underground drilling program further delineated the extent of the Bédard Dyke contacts and the internal structure of the mineralization. A total of 22 surface and 54 underground drill holes totalling 3,747 m (12,292 ft) and 2,647 m (8,684 ft), respectively, were drilled.

The surface drilling program on the Bédard Dyke extended the mineralized zone and confirmed the spatial location of the dyke for mine planning and future development. The drilling indicated that the Bédard Dyke has a minimum strike length of 210 m (689 ft) in the southwest direction, and dips approximately 80° south. The dyke pinches, swells and bifurcates along both its strike and down-dip directions. The dyke’s width varies from approximately 2 m (6.6 ft) at its western end to 10 m (32.8 ft) on the eastern end near the Sigma open pit.

During the period from 2006 to 2012, underground definition drilling was conducted by fanning holes along horizontal or vertical sections. The spacing between the fans ranged between 50 and 100 m (150 to 300 ft), determined by both the local structural complexity of the mineralization and accessibility (Figure 6.1). The underground definition drilling was conducted in the North Wall, the Sigma mine, the Bédard Dyke and the Lamaque mine.

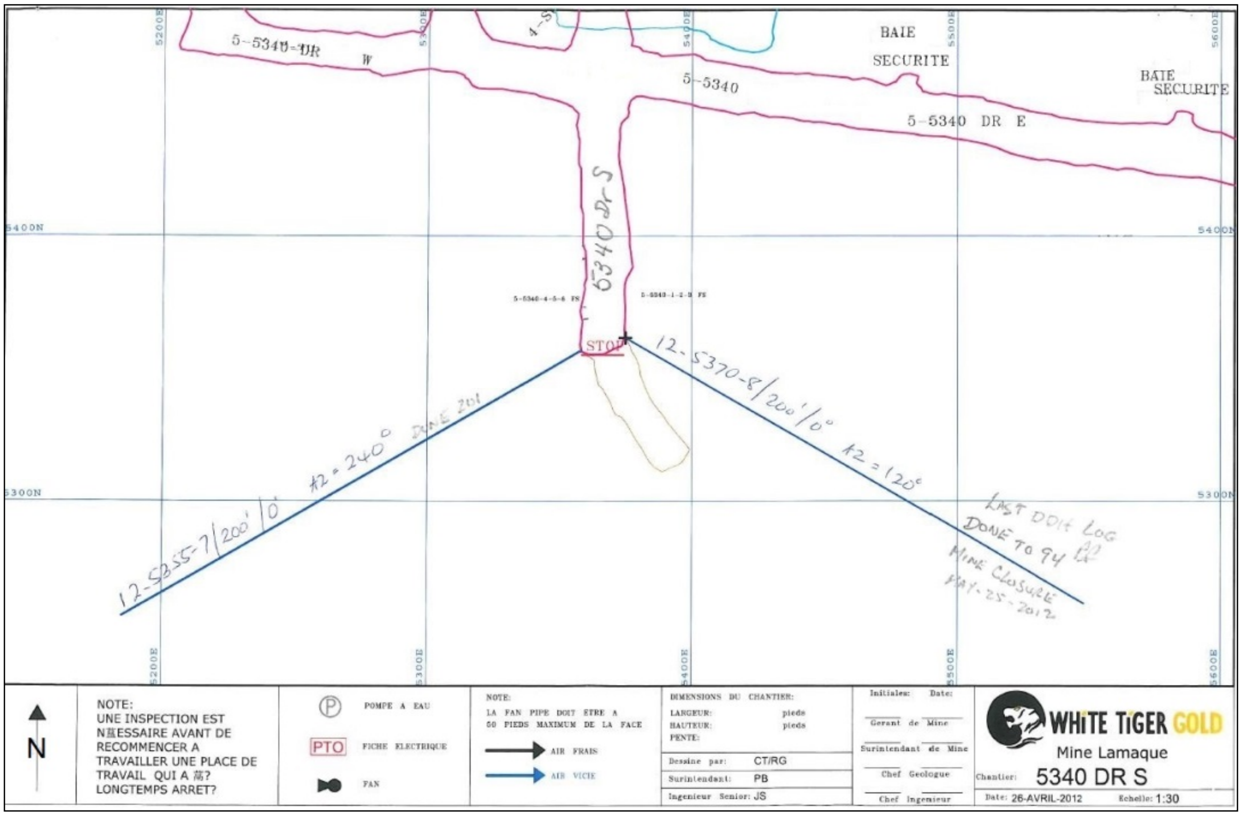


Figure 6.1 – Level plan of drift 5-5340, Lamaque mine, and underground drill holes.

Figure provided in May 2013 by Paul Bédard, Chief Geologist, Century Mining.

SECTION • 7 GEOLOGICAL SETTING AND LOCATION

7.1 THE ABITIBI GREENSTONE BELT

The Lamaque Project is located in the southeastern Abitibi Greenstone Belt of the Archean Superior Province in the Canadian Shield. The Abitibi Greenstone Belt has been historically subdivided into northern and southern volcanic zones defined using stratigraphic and structural criteria (Dimroth et al., 1982; Ludden et al., 1986; Chown et al., 1992), based mainly on an allochthonous greenstone belt model (i.e., interpreting the belt as a collage of unrelated fragments). The first geochronologically constrained stratigraphic and/or lithotectonic map (Figure 7.1), interpreted by Thurston et al. (2008), includes the entire known span of the Abitibi Greenstone Belt (i.e., from the western Kapuskasing Structural Zone to the eastern Grenville Province). Thurston et al. (2008) described the Abitibi Greenstone Belt as composed mainly of volcanic units that were unconformably overlain by large Timiskaming-style sedimentary assemblages. More recent mapping surveys and new geochronological data also support an autochthonous origin for the Abitibi Greenstone Belt.

Generally, the Abitibi Greenstone Belt comprises east-trending synclines containing volcanic rocks and intervening domes cored by synvolcanic and/or syntectonic plutonic rocks (gabbro-diorite, tonalite and granite), separated by east-trending turbiditic wacke bands (MERQ-OGS, 1984; Ayer et al., 2002a; Daigneault et al., 2004; Goutier and Melançon, 2007). The volcanic and sedimentary strata usually dip vertically and are separated by steep east-striking faults. Some of these faults, such as the Porcupine-Destor Fault, display evidence of overprinting deformation events, including early thrusting and later strike-slip and extension events (Goutier, 1997; Benn and Peschler, 2005; Bateman et al., 2008). Two ages of unconformable successor basins are observed: widely distributed fine-grained clastic rocks in early Porcupine-style basins, followed by Timiskaming-style basins composed of coarser clastic sediments and minor volcanic rocks, largely proximal to major strike-slip faults such as the Porcupine-Destor and Larder Lake–Cadillac fault zones and other similar regional faults in the northern Abitibi Greenstone Belt (Ayer et al., 2002a; Goutier and Melançon, 2007). The Abitibi Greenstone Belt is intruded by numerous late-tectonic plutons composed mainly of syenite, gabbro and granite, with lesser lamprophyre and carbonatite dykes. Commonly, the metamorphic grade in the Abitibi Greenstone Belt varies from greenschist to subgreenschist facies (Jolly, 1978; Powell et al., 1993; Dimroth et al., 1983b; Benn et al., 1994), except in the vicinity of most plutons where the metamorphic grade corresponds mainly to the amphibolite facies (Jolly, 1978).

7.2 NEW ABITIBI GREENSTONE BELT SUBDIVISIONS

Abitibi Greenstone Belt subdivisions were redefined using new mapping and geochronology data from the Ontario Geological Survey and Géologie Québec. The following section presents an overview of these new subdivisions, mostly abridged from Thurston et al. (2008) and references therein.

Seven (7) discrete volcanic stratigraphic episodes define the new Abitibi Greenstone Belt subdivisions based on numerous U-Pb zircon age groupings. The new U-Pb zircon ages clearly show timing similarities for volcanic episodes and plutonic activity between the northern and southern portions of the Abitibi Greenstone Belt, as indicated in Figure 7.1. These seven volcanic episodes (Figure 7.1) are listed below, from oldest to youngest:

- Volcanic episode 1 (pre-2750 Ma);
- Pacaud Assemblage (2750–2735 Ma);
- Deloro Assemblage (2734–2724 Ma);
- Stoughton-Roquemaure Assemblage (2723–2720 Ma);
- Kidd-Munro Assemblage (2719–2711 Ma);
- Tisdale Assemblage (2710–2704 Ma);
- Blake River Assemblage (2704–2695 Ma).

The Abitibi Greenstone Belt successor basins are of two types: 1) laterally extensive turbidite-dominated basins corresponding to the Porcupine Assemblage (Ayer et al., 2002a); and 2) later and aerially more restricted alluvial-fluvial or Timiskaming-style basins (Thurston and Chivers, 1990).

The geographic boundary (Figure 7.1) between the northern and southern parts of the Abitibi Greenstone Belt has no tectonic significance but is similar to the limits between the internal and external zones of Dimroth et al. (1982) and those between the Central Granite-Gneiss and Southern Volcanic zones of Ludden et al. (1986). The boundary between the northern and southern parts passes south of the wackes of the Chicobi and Scapa groups, with a maximum depositional age of 2698.8 ± 2.4 Ma (Ayer et al., 1998, 2002b).

The Abitibi Subprovince is bounded to the south by the Larder Lake–Cadillac Fault Zone, a major crustal structure that separates the Abitibi and Pontiac subprovinces (Chown et al., 1992; Mueller et al., 1996a; Daigneault et al., 2002, Thurston et al., 2008) (Figure 7.1). The northern boundary is formed by the Opatica Subprovince (Figure 7.1), a complex plutonic-gneiss belt composed of strongly deformed and locally migmatized tonalitic gneisses and granitoid rocks formed between 2800 and 2702 Ma (Sawyer and Benn, 1993; Davis et al. 1995). It is mainly

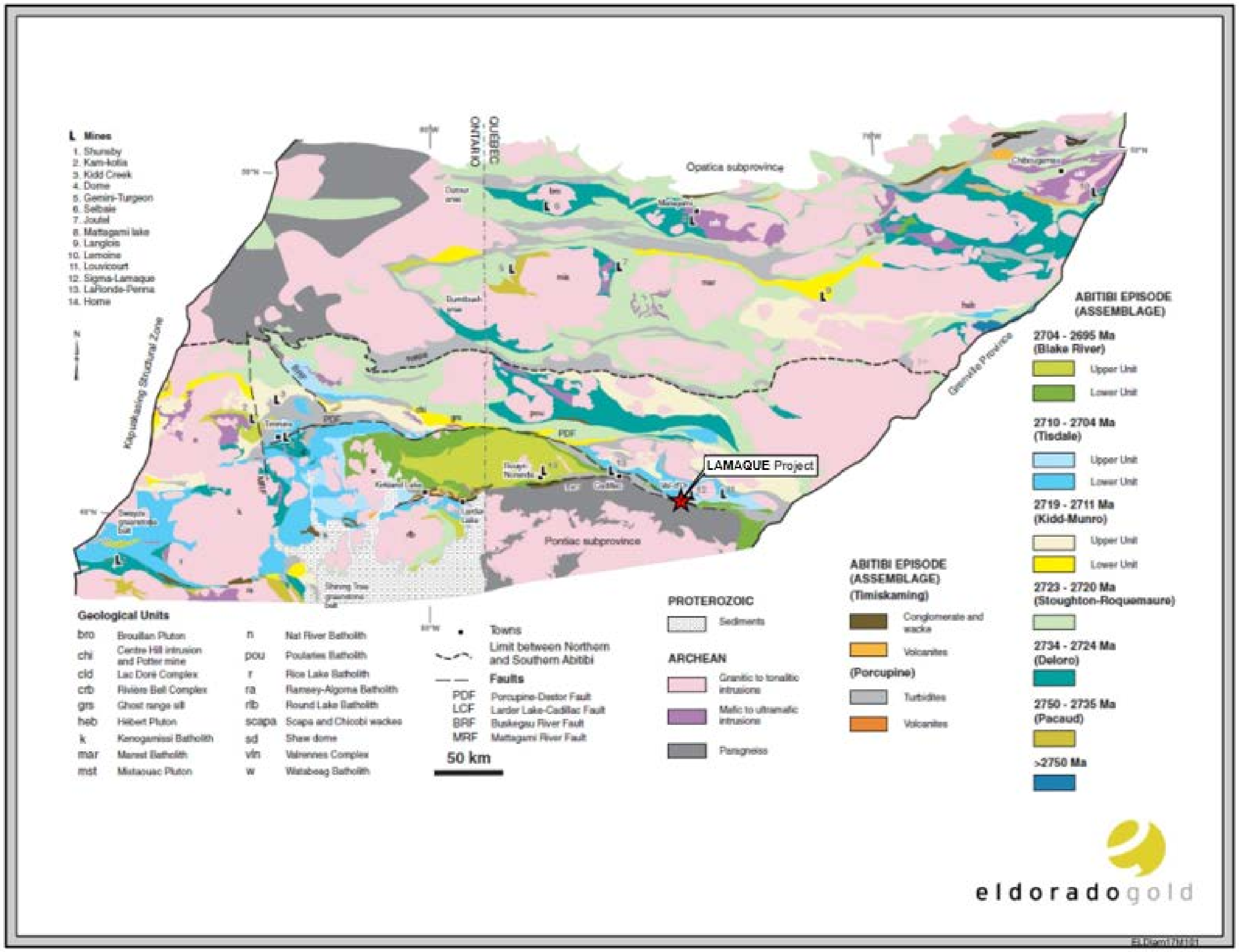


Figure 7.1 – Geology of the Abitibi Greenstone Belt based on Ayer et al. (2005) and Goutier and Melançon (2007). Figure modified from Thurston et al. (2008).

7.3 REGIONAL GEOLOGY

The Lamaque Project is located in the Val-d'Or mining camp. The camp geology is described below using information compiled from the following publications: Gunning and Ambrose (1940), Norman (1947), Latulippe (1966), Dimroth et al. (1982, 1983a, 1983b), Imreh (1976, 1984), Desrochers et al. (1993), Desrochers and Hubert (1996), Pilote et al. (1997, 1998a, 1998b, 1999, 2000, 2015a, 2015b, 2015c), Scott et al. (2002) and Scott (2005).

The region can be divided into two stratigraphic groups based on regional tectonics and volcano-sedimentary stratigraphy (Pilote et al., 1999): the basal Malartic Group comprising the La Motte-Vassan, Dubuisson and Jacola formations, and the upper Louvicourt Group comprising the Val-d'Or and Héva formations (Figure 7.2). These two groups are bounded to the south by rocks of the Pontiac Subprovince and the Piché Group.

The Malartic Group comprises mainly ocean floor komatiite and tholeiitic basalt flows and sills, with minor sedimentary rocks, which are interpreted to have formed in an extensional environment related to mantle plumes. The Louvicourt Group is composed mainly of mafic to felsic volcanic rocks that formed in a subduction-related deep marine volcanic arc.

7.3.1 Stratigraphy

The Val-d'Or mining camp is situated in the eastern segment of the southern part of the Abitibi Subprovince at the boundary with the Pontiac Subprovince. In this region, the Cadillac Tectonic Zone (CTZ) marks the separation between these two subprovinces. From south to north, this region is underlain by the lithologies of the Pontiac Group (PO), the Piché Group (PG), the Héva Formation (HF), the Val-d'Or Formation (VDF), the Jacola Formation (JF), the Dubuisson Formation (DF) and the La Motte-Vassan Formation (LVF).

7.3.1.1 Pontiac Subprovince (PO)

The PO covers the area to the south of the CTZ. It is lithologically homogeneous in this region and dominated by sandstones (60%) and shales (40%). Small mafic tuff bands occur locally but constitute less than 1% of the rock sequence. In outcrop, the rocks of the PO exhibit a pale brown color for the sandstone and darker brown for the mudstone. Tuffs stand out from other lithologies by their greenish color and porous appearance.

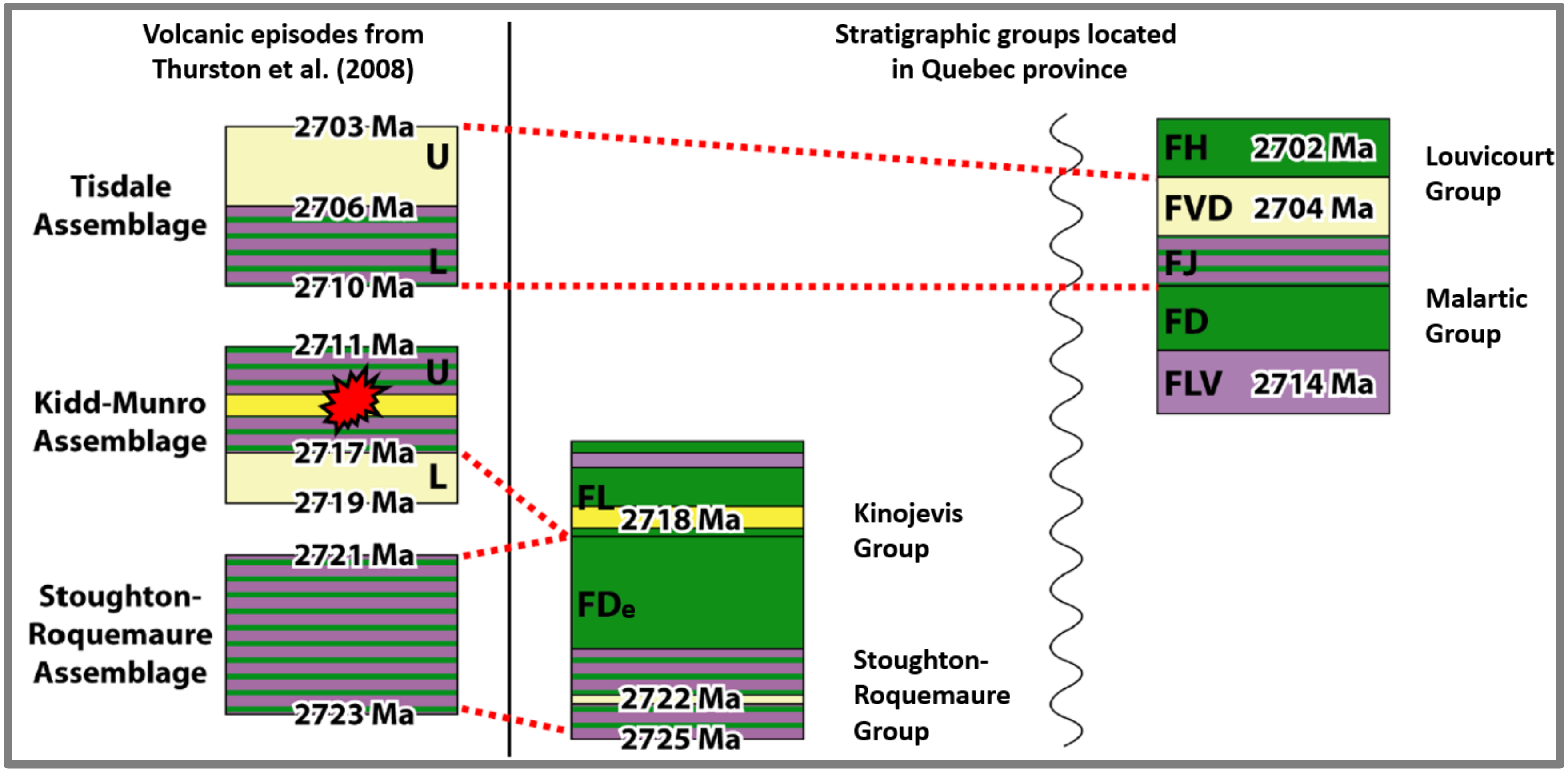


Figure 7.2 – Tisdale and Kidd-Munro assemblages and stratigraphic groups in the province of Québec (Adapted and modified from Pilote et al., 2014).
Abbreviations are based on French names and thus different from those used in the text: FH = Héva Formation, FVD = Val-d'Or, Formation, FJ = Jacola Formation, FD = Dubuisson Formation, FLV = La Motte–Vassan Formation, FDe = Deguisier Formation, FL = Lanaudière Formation.

The level of deformation is variable. South of the CTZ, bedding and primary textures are commonly preserved. Elsewhere, in the more deformed sectors, these sedimentary rocks show a tectonic banding that is superimposed on the original stratigraphic layering (S0).

7.3.1.2 Piché Group (PG)

Outcrops of the PG (Latulippe, 1976) are rare in the region. The position of the rocks of this group generally coincides with the location of the CTZ, leading many researchers to describe the group as a band of talc-chlorite or talc-chlorite-carbonate schist (e.g., Gunning and Ambrose, 1940). Recently, however, it has been shown that the distribution and intensity of deformation is heterogeneous within the CTZ. Primary textures have been preserved in areas where deformation is less intense. These less deformed rocks are typically discontinuous and encompassed by schist. In underground workings and drill core, these pockets are of basaltic and komatiitic composition (Sansfaçon and Hubert, 1990), whereas at surface, they are essentially ultramafic and exhibit cumulate textures. Spinifex textures are locally preserved.

In 2013, an age of 2709.5 ± 2 Ma was obtained for a tonalite dyke that cuts the ultramafic units of the Buckshot pit near the Canadian Malartic deposit, constraining the minimum age for the PG..

7.3.1.3 Héva Formation (HF)

The HF (2702 ± 2 Ma) is 2 to 5 km thick. It is located between the CTZ and the VDF. The HF represents a separate volcanic cycle from that of the VDF, comprising volcaniclastic rocks, pyroclastic rocks, and dykes and sills of gabbroic to dacitic composition. Volcaniclastic rocks are characterized by coarse or fine tuff horizons with millimetre-scale laminations, intruded by gabbro and dacite. Disruptions in the volcaniclastic beds and peperite textures indicate that the dykes and sills were injected into unconsolidated sediments. In most cases, the interaction between magma and sediment formed complex structures of pseudo-pillows in the magma rather than true peperite. The volume and styles of the gabbro and dacite intrusions suggest a proximal position relative to the volcanic centre.

7.3.1.4 Val-d'Or Formation (VDF)

The VDF (2704 ± 2 Ma) is 1 to 3 km thick and comprises submarine volcaniclastic deposits formed by autoclastic and/or pyroclastic mechanisms. These deposits include 1 to 20 m thick layers of brecciated and pillowed andesite flows with feldspar and hornblende porphyries, intercalated with volcaniclastic beds 5 to 40 m thick. The pillows exhibit a variety of forms, from strongly amoeboid to lobed. Lobed pillows are 1 to 10 m long and 0.5 to 1.5 m high and have a vesicularity index of 5% to 40%. The volcaniclastic beds are composed of lapilli tuff, lapilli and block tuffs, and to a lesser extent, fine to coarse tuffs.

7.3.1.5 Jacola Formation (JF)

The JF (2706 ± 2) lies north of the VDF. It consists of a cyclic package comprising, from bottom to top, komatiitic flows, basalts and andesitic volcaniclastic rocks. The sequences may be complete or truncated. Komatiitic lavas are observed in the form of massive flows with local spinifex textures. Basaltic flows are massive, pillowed and sometimes in the form of flow breccias. Magnesian basalts are also present in small amounts. They are easily identified by their characteristic pale grey color.

7.3.1.6 Dubuisson Formation (DF)

The DF (2708 ± 2 Ma) consists mainly of pillowed and massive basalt with various interbedded komatiitic flows (Imreh, 1980). Ultramafic and mafic flows are similar to those described in the LVF (see below), but in different proportions.

7.3.1.7 La Motte–Vassan Formation (LVF)

The LVF crops out on the north side of Lac De Montigny and has variable apparent thickness, up to a maximum of 6 km. The LVF consists of komatiites, tholeiitic basalts and magnesian basalts. The base of the sequence is mostly represented by komatiites with some minor intercalated basalt. However, a decrease in the proportion of komatiites is observed toward the top of the sequence (Imreh, 1984). Komatiites are mainly found in two morphofacies: 1) classic sheet flow with spinifex textures or tube-shaped flows, and 2) mega-pillows. The basalt flows are usually massive or pillowed; more rarely, they are brecciated (Imreh 1980). The age of the LVF (2714 ± 2 Ma) suggests it may be contemporaneous with the upper part of the Kidd-Munro Assemblage (Figure 7.2).

7.3.2 Intrusive rocks

The initial volcanic and structural architecture is cut and disrupted by several intrusive events (summarized from Pilote et al., 2000), mainly: the synvolcanic Bourlamaque Pluton (2700 ± 1Ma), pre to early-tectonic dykes and stocks as the Snow Shoe and the East Sullivan suites (respectively dated at 2694 ± 3 Ma and 2684 ± 1 Ma) and the syn to post-tectonic Preissac-La Corne Pluton (2680–2642 Ma).

7.3.3 Structural fabrics

Pilote et al. (2015c) established the nomenclature for the various structural elements in the region, as described below.

The oldest regional schistosity S1 is normally subparallel to bedding, S0. The overall S1 trend is NW-SE with moderate to steep dips to the north. S1 contains the primary stretching lineation L1. In the southwestern part of the region, S0 and S1 are jointly folded into “Z” folds, with an average axial plane of N095°/85° marked by the regional schistosity S2. The axes of these folds, F1-F2, are parallel to the plunges shown by the L1 stretching lineation contained in S1.

A late S3 cleavage is the product of kinking and irregular conical chevron folds in highly altered units showing a strong pre-existing anisotropy. Those fold are commonly intrafolial to S2 and their envelopes measure under the meter. Intersection lineation on S2 mostly plunges moderately to steeply westward.

7.3.3.1 Large-scale fault zones

The region has several large-scale strike faults and/or shear zones, striking W to WNW and dipping steeply to the north. They are, from south to north: the Cadillac Tectonic Zone (CTZ), the Parfouru Fault (PF), the Marbenite Fault (MF), the Norbenite Fault (NF), the K Shear Zone (KSZ) and the Rivière Héva Fault (RHF). The descriptions below are presented in the same south–to-north order.

All of these major structures contain dykes or stocks of monzonitic or tonalitic composition with highly variable ages (pre-, syn- or post-tectonic) that are spatially associated with several gold mines (Norlartic, Marban, Kiena, Sullivan, Goldex, Siscoe, Joubi, Sigma and Lamaque). The observed diversity in the styles and ages of gold mineralization related to these large-scale strike faults and/or shear zones demonstrates that several distinct episodes of mineralization occurred.

7.3.3.2 Cadillac Tectonic Zone (CTZ)

The CTZ (Figure 7.3-7.4) is one of the most controls on gold mineralization in the Abitibi Greenstone Belt. The CTZ is important not only for its metallogenic wealth, but also for its geodynamic significance and for the juxtaposition of varied lithologic assemblages along its subsidiary faults. The E-W and WNW striking sections of the fault reflect a deep asymmetry in the Abitibi Subprovince, a feature that influenced the styles and episodes of gold mineralization.

The CTZ has long been known to be associated with talc-chlorite-serpentine schists that have now been assigned to the Piché Group (PG). The CTZ is 200 to 1000 m wide, consisting of anastomosing fault strands that isolate distinct lithological wedges displaying variable degrees of deformation.

Numerous intrusions of various shapes, sizes, compositions and ages are also found along the CTZ. Calc-alkaline intrusions were injected between 2690 and 2680 Ma, whereas younger alkaline intrusions were emplaced between 2680 and 2670 Ma. These features reveal the role of the fault as a conduit for both magmas and hydrothermal fluids, and also demonstrate its long-lived deep crustal nature. In the Val d’Or region, the CTZ is generally oriented N100° and dips steeply to the north-northeast.

7.3.3.3 Parfouru Fault (PF)

The PF (Figure 7.3 and 7.4) is an ESE-WNW shear zone that dips steeply (75°) to the north or northeast (Daigneault, 1996). The shear zone can reach 300 m wide and has been traced for at least 20 kilometres.

7.3.4 Marbenite Fault (MF)

The MF (Figure 7.3 and 7.4) is an ESE-WNW to SE-NW shear zone that dips steeply to the north or northeast (Trudel and Sauvé, 1992; Sauvé et al., 1993; and Beaucamp, 2010). It is parallel to the Norbenite Fault.

7.3.5 Norbenite Fault (NF)

The NF (Figure 7.3 and 7.4) is a strong second-order shear zone that strikes WNW, subparallel to stratigraphy, and dips 40-60° to the northeast (Trudel and Sauvé, 1992; Sauvé et al., 1993). The NF is 15 to 110 m wide and has been traced for 8 km. It mainly affects the komatiitic units of the JF and occasionally the basaltic units as well. This shear zone splits into two or three branches in some places.

7.3.5.1 K Shear Zone (KSZ)

The KSZ (Figure 7.3 and 7.4) is a shear zone between 300 and 600 m wide that has been traced for 1.7 km. It strikes ESE-WNW and dips 80° toward the northeast, and is composed of talc and chlorite schists, actinolite schists and minor sericite schists, and bodies of pure talc and massive actinolite (Olivo and Williams-Jones, 2002; Olivo et al., 2007).

7.3.5.2 Rivière Héva Fault (RHF)

The RHF (Figure 7.3) is an ESE-WNW shear zone that dips steeply (80°) to the north or northeast (Daigneault, 1996). The shear zone can reach 300 m wide and has been traced for at least 30 kilometres.

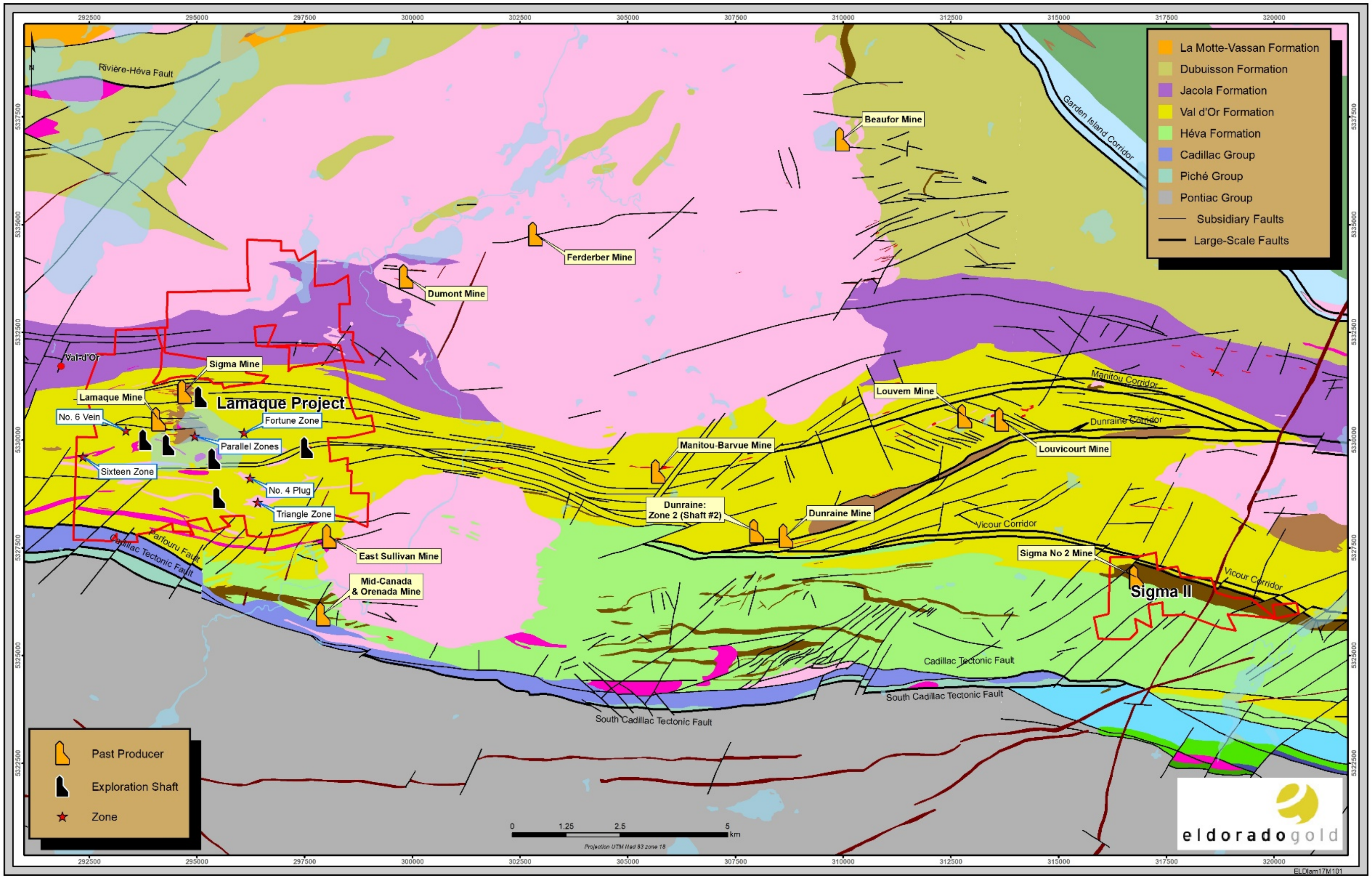


Figure 7.3 – Regional geology of the Lac De Montigny area showing the main faults and gold zones (Adapted and modified from Pilote 2013, 2015a, 2015b).

7.4 LOCAL GEOLOGIC SETTING AND MINERALIZATION

This section describes the geology of the Lamaque Project and was modified from the report by Poirier et al. (2015a) and Girard et al., (2017).

7.4.1 Geology

The Lamaque Project area is principally underlain by volcaniclastic rocks and minor volcanic flows of the Val-d'Or Formation (VDF; 2705–2703 Ma), which have been intruded by numerous intrusions, dykes and sills of compositions varying from mafic to intermediate (Figures 7.3, 7.4 and 7.5). The lithologies young to the south, with the VDF overlying the Jacola Formation in the northern portion of the property and overlain by mafic volcanic rocks of the Héva Formation are found within the southwestern portion of the property.

The VDF is characterized by interstratified massive to pillowed lavas and volcanoclastic rocks of andesitic-basalt to rhyolitic composition. Reports have also recognized one horizon as an exhalative chert horizon. According to Scott (2002) and based on yttrium-zirconium ratios, the volcanic rocks of the lower VDF are tholeiitic to transitional, while the upper Val-d'Or units to the south are tholeiitic to calc-alkalic. Despite the lithlogic variation, units within the VDF exhibit similar response on detailed magnetic surveys, and such surveys have limited use in defining internal stratigraphic and structural relationships.

Because of their intimate spatial association with most of the known gold deposits, the intermediate to mafic plugs and associated porphyritic dykes and sills that intrude the VDF are of particular importance. For example, most of the historical gold production at the Lamaque was contained within the Main Plug. The Main Plug is a vertically plunging chimney-like intrusive body, which measures roughly 245 m by 115 m in diameter. It displays concentric compositional zonation, with an outer rim, which is in contact with the host volcanic rocks, dominantly of dioritic compostion. The diorite rim grades inwards into a porphyritic quartz-diorite phase and the core of the plug is granodiorite. Significant production also came from the East Plug, located due east of the Main Mine and the West Plug, southwest of the Main Plug. Burrows and Spooner (1989) reported that the Main, East and West plugs are unusually sodic and enriched in barium and strontium, and suggest these compositions are magmatic. Risto et al. (2004) suggest that this sodium-, barium- and strontium-enriched composition is more likely a reflection of gold-related hydrothermal alteration. As pointed out by Patton (1988), these plugs are foliated and have a consistent northwest orientation with a steep (70°) northeastern plunge. U-Pb zircon geochronology dates the Main Plug at 2682 to 2685 Ma (Jemielita et al, 1989; Corfu, 1993), making them post-volcanic but early tectonic.

Other similar pipe-like porphyritic intrusions on the property include the Triangle Plug, Plug No. 4 and Plug No. 5.

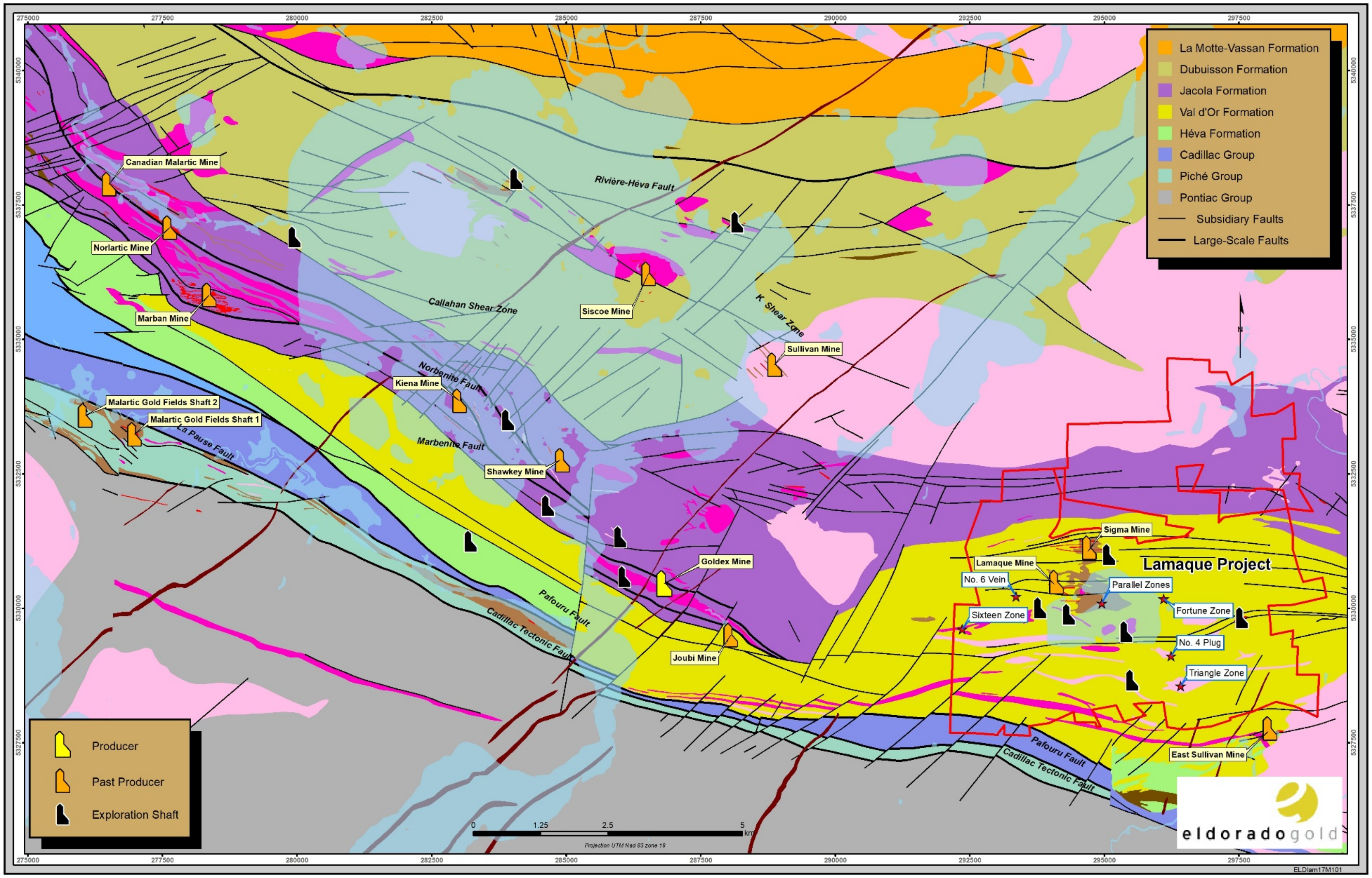


Figure 7.4 – Simplified geology map of the Sigma and Lamaque mine areas. Modified from Sauvé et al. (1993)

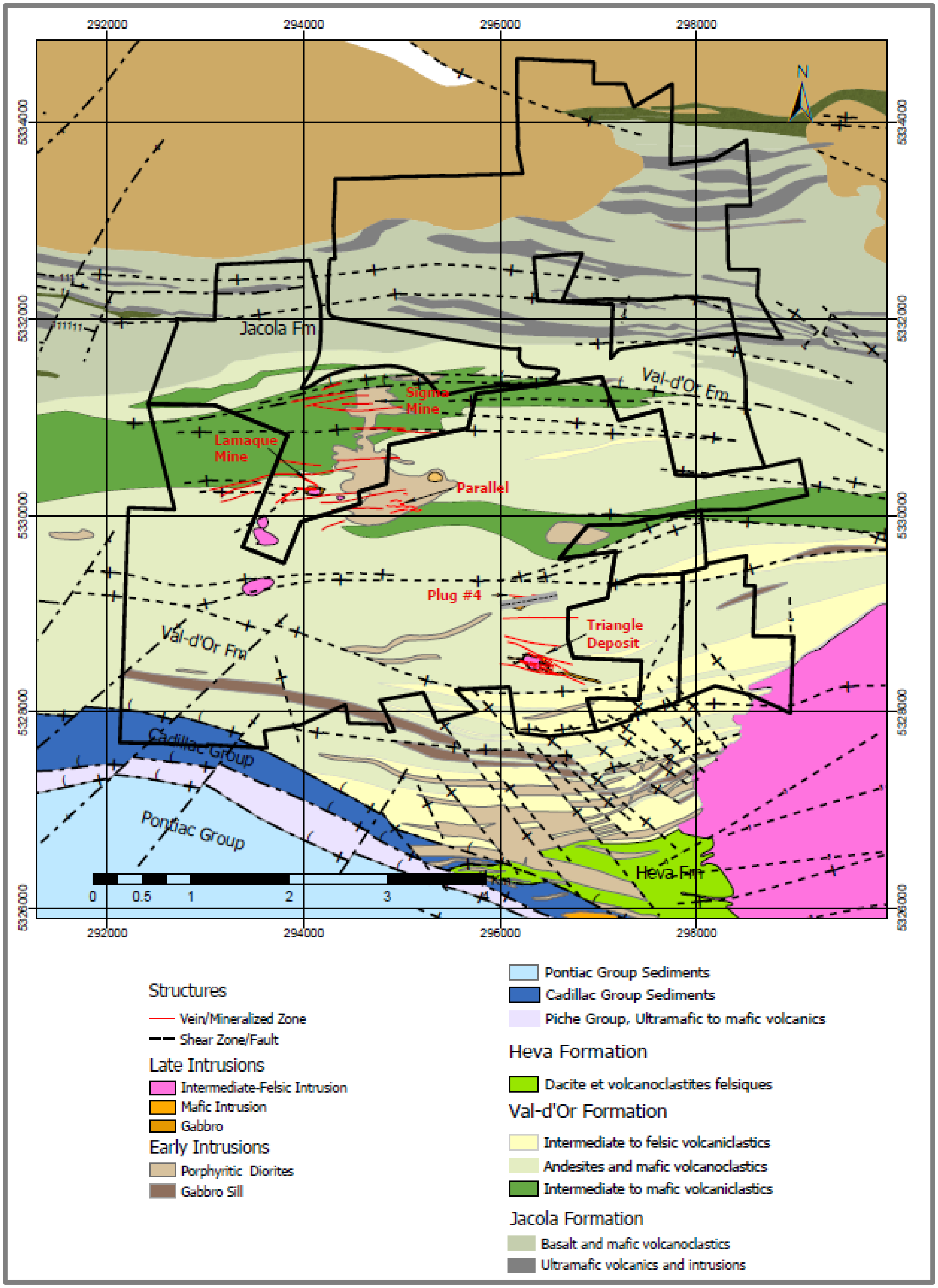


Figure 7.5 – Geology of the Lamaque South and Sigma-Lamaque Properties

Plug No. 4 measures about 100-110 m in diameter at surface and extends to a depth of at least 1300 m. It consists of fine to medium-grained gabbro with a strong magnetic signature. The plug has been cut on its western side by a fine-grained felsic porphyritic diorite dyke, which appear to be post-volcanic like the Lamaque plugs. Plug No. 5 is thought to be very similar to Plug No. 4.

The Triangle Plug, which host the Triangle Gold Deposit, is very similar to the Lamaque Main Plug in terms of its composition, shape, relationship with the surrounding lithologies and style of mineralization. The intrusion is sub circular and consist of two types of porphyritic diorite. Both are medium grained with 3-5 mm feldspar phenocrysts; one is a mafic diorite rich in amphibole (35-40%) in its matrix and high in K; the other is a more felsic diorite with less than 20% mafic minerals, and a matrix consisting of more biotite than amphibole.

Numerous other porphyritic dykes and sills are common in drill holes. These vary from concordant or subparallel to stratigraphy to cross cutting. They are of felsic to intermediate composition and are commonly associated with gold mineralization. Because of the unpredictable geometry and pervasive hydrothermal alteration of some of these intrusions, it is possible that some may represent flows or synvolcanic intrusions.

All lithologies on the property have been metamorphosed to lower greenschist facies (quartz + sericite + albite + chlorite + carbonate + actinolite + chloritoid). Scott et al. (2002) report that all flow units of the VDF have undergone silicification, sericitization and carbonatization to some degree. Various types and extents of alteration have been observed and described in historical and current drill logs. In most cases, the alteration described is found peripheral to, or as enveloping the quartz-tourmaline veins in either volcanic rocks or intermediate porphyritic rocks.

On a regional scale, the volcanic rocks strike E-W to NE-SW and dip steeply to the north or south. In lithological units south of the Bourlamaque Batholith, bedding is overturned with younging directions mainly to the south (Pilote et al., 2000). At the regional scale and on the property, an E-W, steeply-dipping tectonic fabric correlated with regional S1 foliations is superimposed on and obscures primary volcanic textures.

Patton (1988) suggested that the volcanic rocks may be tightly folded, with the intrusive plugs located at the fold noses. Tight folding would be compatible with the regional interpretation. Patton also suggested that the plugs are in fact folded, with thickened sills or dykes occurring in the fold noses. Burrows and Spooner (1989) also suggested that the plugs represented the coalescence of parallel dykes.

7.4.2 Gold Mineralization – Lamaque South

The following description of gold mineralization in the Lamaque Project area is summarized from Beauregard et al. (2011) and references therein, Girard et al., 2017, as well as from information supplied by the issuer.

There are 12 known gold occurrences on the property. All are associated with quartz-tourmaline-carbonate veins, which vary from simple shear hosted and/or extensional vein systems to complex stockworks zones. This section focuses on the Triangle, Plug #4 and Parallel deposits. However, characteristics of the historical Sigma and Lamaque Mines which together produced in excess of 9.5

Moz of Au, are summarized in this report due to their proximity, economic importance and geological similarity to the gold deposits found on the property.

7.4.2.1 Triangle Gold Deposit

The Triangle Gold Deposit was discovered in 2011 by drilling targetting an isolated circular magnetic high anomaly, which corresponds to the massive mafic volcaniclastic rocks surrounding a non-magnetic porphyritic diorite intrusion. The anomalous magnetic zone extends over a sub circular area of roughly 800 m in diameter.

The volcaniclastic rocks in the area of the Triangle Deposit consist of feldspar phenocryst rich (fragments and matrix) lapilli-block tuffs of andesitic to basalt composition. The size of the blocks can reach 70 cm. The texture of the coarse-grained matrix is very massive and competent, however grading can be observed locally. Fine grained beds are rare and turbidite facies have not been observed. Rare thin concordant lava flows, as well as complex and irregular subvolcanic intrusions, are intercalated within the volcaniclastic sequence. The tuffs lack penetrative schistosity,but contain a stretching lineation and a weak flattening and alignment of fragments. The strong competency of the rocks surrounding the Triangle Intrusion is possibly the results of a pervasive albite-quartz-epidote (±chlorite-magnetite-pyrite) alteration.

The main lithological feature at Triangle is the chimney-shaped feldspar porphyritic diorite. This intrusion, called the Triangle Plug, is very similar to the Main Plug at Lamaque. The Triangle Plug is composed of two different facies of the porphyritic diorite: mafic facies composed of 25-40% hornblende with minor biotite in the matrix; and felsic facies, composed of less than 25% mafic minerals in the matrix. For both facies, the 10-30% of the rock consists of zoned fine to medium-grained feldspar phenocrysts.

The Triangle Plug is a cylinder-shaped intrusion which plunges at roughly 70° towards 030°. At a depth of around 700m below surface, the Triangle Plug merges with a large dyke called Triangle North, which extends E-W for a distance of over 4km and dips vertically. Triangle North is also a feldspar porphyritic diorite that shares similarities to both facies of the Triangle Plug. The dyke extends to a depth of at least 1800m below surface.

Gold mineralization at Triangle occurs within quartz-tourmaline-carbonate and pyritic gold veins in the Triangle Plug and in the adjacent massive mafic lapilli-blocks tuffs. The veins are localized in a series of shear zones and fractures that cut both units. The thickest and most continuous veins are localized with E-W striking ductile-brittle reverse shear zones dipping 50-70° south. Veins also occur as extensional shear vein splays dipping 20-45° south and as subhorizontal extension veins (Figure 7.6). The latter generally are not volumetrically significant and are not included in the resource estimation. Gold also occurs in sericite-carbonate-pyrite alteration selvages along the veins, which are commonly foliated to schistose within fault zones.

7.4.2.2 Plug No. 4 Deposit

The Plug No. 4 Deposit (Figure 7.5) is located 550m north of the Triangle Deposit and 1.0km southwest of the historical No. 3 Mine shaft at Lamaque, to which it is connected by drifts on the 450’ and 700’ levels.

No. 4 Plug is a fine- to medium-grained magnetic gabbro intrusion measuring roughly 100 to 150 m in diameter (Figure 7.6). It is enveloped by a younger synvolcanic diorite and diorite breccia similar to intrusive rocks that host the gold bearing veins at the Sigma Mine. These intrusions are cut to the west by a fine-grained porphyritic felsic diorite dyke which extends northwest towards the No. 3 Mine area. Gold mineralization at Plug #4 is restricted to the gabbro intrusion.

A series of E-W striking reverse shear zones dipping between 45° and 75° to the south cuts all units at Plug #4. The shear zones are spaced roughly 25 to 50 m apart and have been identified to depths of more than 1000m from surface in the gabbro. Gold mineralization at the Plug No. 4 Deposit occurs in quartz-tourmaline-carbonate-pyrite veins. These veins have both laminated and breccias textures, and most are associated with and controlled by the reverse shear zones. Low angle extensional shear veins (dipping 35-45° south) are associated with these but appear to have limited extent.

At Plug No. 4, sub-horizontal extensional veins are much more abundant than at Triangle and occur in vein arrays or clusters in the gabbro intrusion that measure with dimensions measuring up to tens of metres. The thickness of individual veins can vary from 1mm to 1.25m, but most are around between 5-10cm. These vein clusters can carry significant gold concentration, but grades are generally very erratic. Where vein concentrations are greatest, average grades can be around 3-4 g/t Au over intervals of 20-30m.

The flat extension vein arrays at Plug No. 4 are spatially associated with the reverse shear zones, occurring most abundantly in zones that extend up to 15m into the hangingwall and footwall of the shears. Commonly, the spacing between veins increases away from the shear zones, while vein thickness, wall rock alteration, tourmaline and pyrite abundance and gold content diminish.

7.4.2.3 Parallel Deposit

The Parallel Deposit is located 650 m northwest of the No. 3 Mine (Figure 7.5) and 900 m east-southeast of Lamaque Mine main shaft.

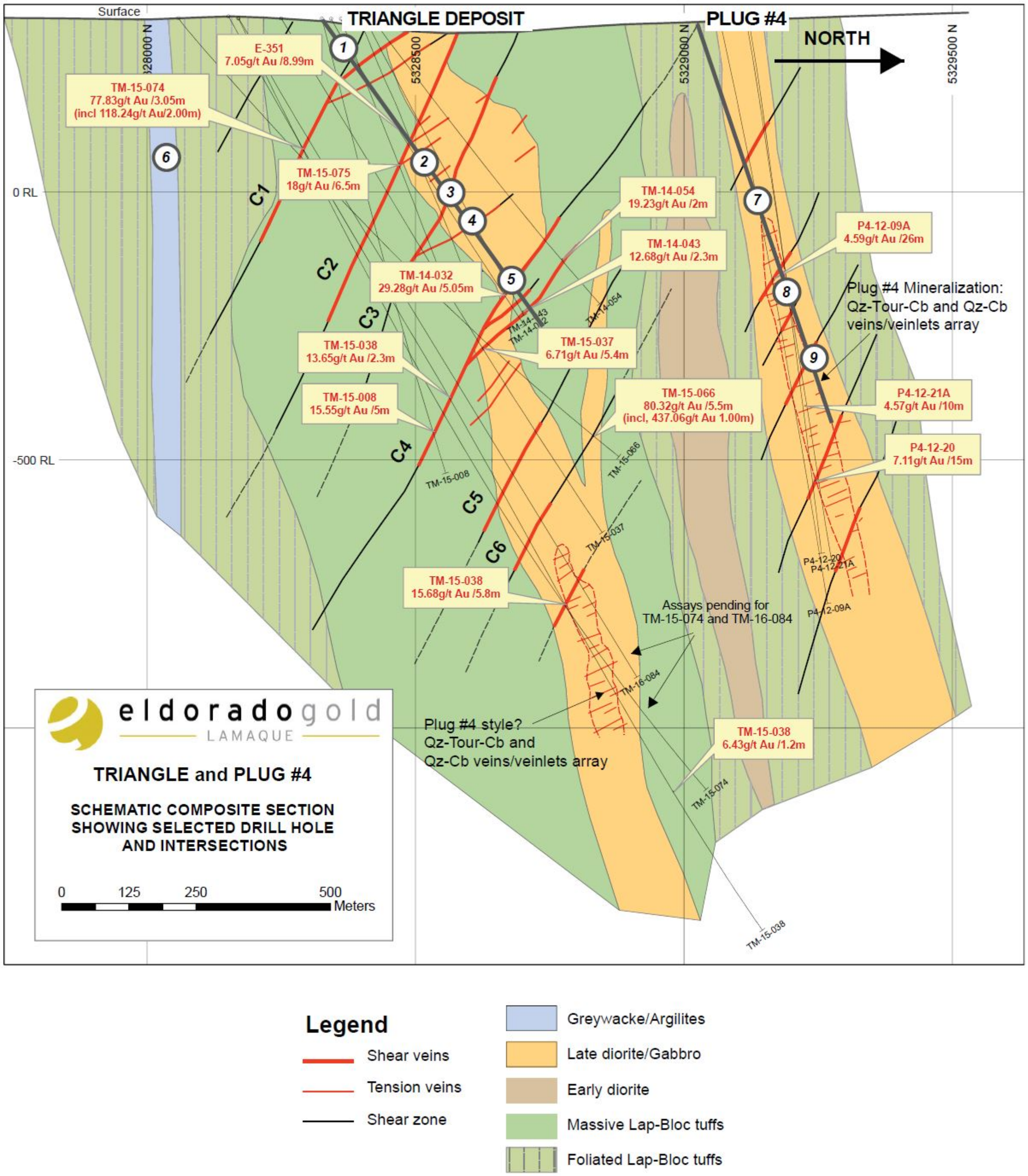


Figure 7.6 – Schematic composite section showing selected drill hole and intersections

Gold mineralization at Parallel is hosted within fine- to medium-grained porphyritic diorite, similar to the host rock at the Sigma Mine. The diorite is medium greenish-gray and contain generally 1 to 5% disseminated pyrite commonly associated with silicification and chloritisation that seem to be part of an early (synvolcanic) pervasive alteration of the diorite.

The ore zones at Parallel occur as sub-horizontal extension veins at shallow depths (70-200 m) and as E-W striking shear veins dipping approximately 30° south at deeper levels.

The mineralized veins consist of quartz and carbonate with lesser amounts of tourmaline, chlorite and sericite. Fine pyrite commonly amounts to 1-3% and rarely up to 5 %. Traces of chalcopyrite occur locally. In wider veins, pyrite and gold are typically confined to vein margins and/or vein contacts, especially in veins composed mainly of quartz and carbonate.

The sub-horizontal veins are laterally extensive (up to 300 m), occur in en-echelon sets and exhibit pinch and swell characteristics. Adjacent wallrocks contain carbonate-albite-sericite and pyrite alteration. In general, they occur in stacked sets 10-25 m thick containing up to 7 or 8 individual veins.

Shear veins also occur in clusters. Typically, up to four en-echelon south dipping veins occur within a 75 m wide vertical corridor that cuts across the porphyritic diorite. The shear veins most commonly range in width from 15 cm and 1.5 m, but can be up to 2.6 m thick locally.

7.4.3 Sigma and Lamaque Mine

Descriptions of the auriferous vein systems of the Sigma and Lamaque Mines have been widely published over decades. Among the many scientific papers and other publications devoted to those two exceptional ore deposits, the most relevant include: Robert (1983), Robert and Brown (1986a, 1986b), Robert et al. (1995), Garofalo (2000), Gaboury et al. (2001) and Olivo et al. (2006) for the Sigma Mine, and Daigneault et al. (1983), Perrault et al.,(1984) and Karvinen, (1985) for the Lamaque Mine.

7.4.3.1 Sigma Mine geology

Three main lithologic units are identified at the Sigma mine (Figure 7.5): undifferentiated volcanic rock, porphyritic diorite and feldspar porphyry dykes (Figure 7.4). Volcanic rock is a general term that includes various tuffaceous rocks and associated pillowed and massive lava flows of andesitic composition. These rocks strike E-W and dip steeply to the north (Figure 7.7). The porphyritic diorite, known locally as the C porphyry, corresponds to a massive plagioclase-phyric diorite of subvolcanic origin that intrudes the lavas. The diorite forms an irregular body located in the central part of the deposit, as well as other irregular and smaller isolated bodies. The feldspar porphyry dykes (the “G dykes”, according to the mine nomenclature), intrude the previous units. These dykes strike approximately E-W and dip steeply to the south. The thickness of individual dykes ranges from a few centimeters to about 10 m and averages 3 m.

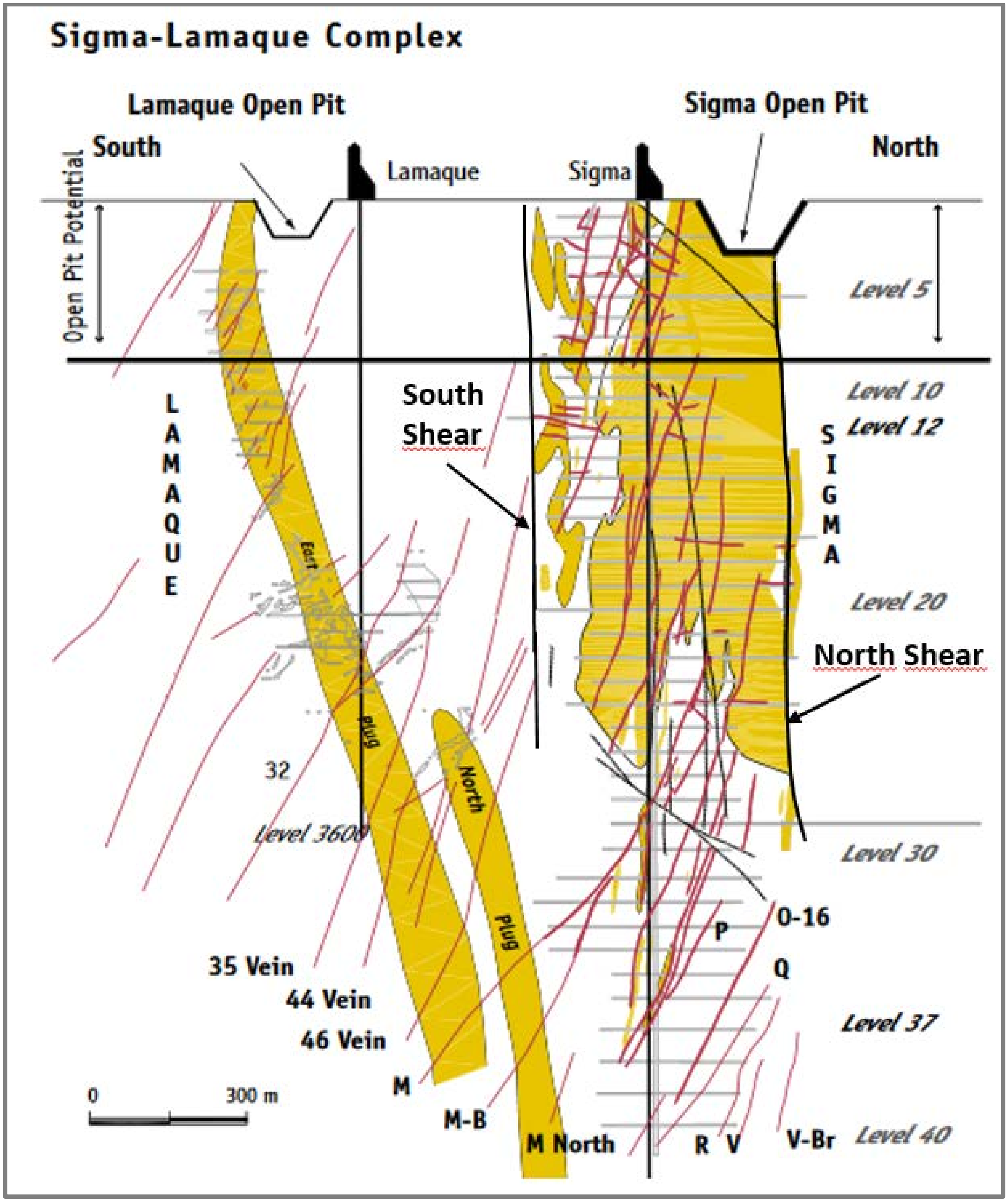


Figure 7.7 – Typical north-south cross section through the Sigma and Lamaque mines at the deposit scale, showing the en-echelon distribution of fault-fill veins between both bordering shear zones, and the branching of moderately and steeply dipping fault-fill veins that isolate lozenge-shaped blocks. Dashed horizontal lines are mine levels. Figure modified from the 1999 Annual Report of McWatters Mining Inc.

The numerous shear zones are the dominant structural features of the deposit. The shear zones are up to 6 m wide, strike E-W (N070° to N110°) and dip moderately to steeply to the south (50°-90°). Most, if not all, of these shear zones have recorded a dominant reverse movement and they overprint the S2 regional schistosity. The weakly penetrative, E-W-striking and subvertical S2 fabric overprints all rock types locally, although it is better developed within less competent rocks such as the lavas.

Auriferous veins at the Sigma mine consist of quartz and tourmaline with lesser carbonates, sericite, pyrite, scheelite, chlorite and chalcopyrite (Robert and Brown, 1986a, b). Four types of veins can be distinguished on the basis of their host rock associations and geometries: (1) steeply to moderately dipping fault-fill veins within shear zones; (2) subhorizontal extensional veins; (3) arrays of subhorizontal extensional veins hosted within the feldspar porphyry dykes, referred to as dyke stringers; and (4) moderately north-dipping extensional-shear veins, referred to as the North Dipper veins.

7.4.3.2 Lamaque Mine Geology

This section on the Lamaque Mine (Figure 7.5) was modified from Poirier et al. (2015a) and Girard et al., (2017).

The volcanic sequence at Lamaque strikes E-W and dips steeply south, with tops facing south. It consists of andesitic basalt lapilli tuffs mixed with lesser andesite flows and flow breccia and mafic lavas. The oldest intrusive rocks at Lamaque are porphyritic diorite dykes and stocks considered equivalent to the C porphyry at the Sigma Mine. Those intrusions are part of a larger intrusive cluster that extends continuously between the two mines, and are volumetrically more significant at Lamaque.

Numerous chimney or plug-shaped intrusions varying from mafic to felsic compositions occur at Lamaque (Figure 7.5), with the Main Plug being the most productive host rock. The Main Plug is roughly elliptical (250 m E-W by 100 m N-S), plunges northeast at 70° and has been traced to a depth of 1,800 m. The core of the Main Plug is medium to fine grained, dark grey and homogeneous tonalite-diorite, which grades outward to quartz diorite and finally diorite. The West and East plugs have the same composition as the Main Plug, but the West plug is coarser grained.

Many types of porphyry dykes have been identified, and are arranged into an alphabetical grouping from A to G, based on characteristics such as mineral composition and grain size. Most are lithologically similar to the feldspar porphyry "G" dykes at the Sigma Mine, with the exception of quartz diorite porphyry (QDP) that is unique to the Main Plug area. All of those dykes are older than the intermediate plug-type intrusions (e.g. Main Plug).

Gold mineralization at Lamaque consists of pyritic quartz-tourmaline-carbonate (QTC) veins, stringers and stockworks. These gold-bearing veins occur within all rock types, but are mostly abundant in intrusions. Roughly 85% of the gold mined historically at Lamaque was from veins hosted by the Main Plug. The QTC veins, as at Sigma, are associated with brittle-ductile reverse shear zones, and occur in multiple orientations and styles. Historical nomenclature for the different veins sets include i) "major veins" (strong subvertical shear vein), ii) "lesser veins" (extensional shears or Riedel shears), iii) "stockworks and stringers" (horsetail structures) and iv) three types of "flat veins", each distinguished by their host rocks or their spatial and temporal relationships with the shear zones.

SECTION • 8 DEPOSIT TYPE

8.1 ARCHEAN GREENSTONE-HOSTED OROGENIC LODE GOLD DEPOSITS

The following description of Archean greenstone-hosted orogenic lode gold deposits is summarized from Simard et al. (2013) and references therein.

Archean greenstone-hosted orogenic lode gold deposits are typically distributed along first-order compressional to transpressional crustal-scale fault zones that mark the convergent margins between major lithological boundaries and are characterized by multiple strain increments (e.g., Larder Lake–Cadillac Fault Zone) Although these major or first-order faults are interpreted as primary hydrothermal pathways to higher crustal levels (Eisenlohr et al., 1989; Colvine, 1989; McCuaig and Kerrich, 1998; Kerrich et al., 2000; Neumayr and Hagemann, 2002; Kolb et al., 2004; Dubé and Gosselin, 2007), only a few significant gold deposits are hosted within the major fault themselves. Examples in the Abitibi Subprovince include the McWatters mine, Lapa mine and the Orenada deposit (Morin et al., 1993; Robert, 1989; Neumayr et al., 2000; 2007; Simard et al., 2013).

Significant mineralized quartz veins and most important gold deposits are commonly hosted in second- and third-order shear zones (Eisenlohr et al., 1989). Structurally, these shear zones vary from brittle–ductile to ductile, depending on their depth of formation (Hodgson, 1993; Robert and Poulsen, 2001). At depths greater than 10 km, quartz veins are less common within shear zones, and gold mineralization is mostly associated with disseminated sulphides (Witt and Vanderhor, 1998).

A widely accepted model for orogenic gold deposit is the continuum model (e.g., Colvine, 1989; Groves, 1993; Gebre-Mariam et al., 1995; Groves et al., 1998, 2003), which involves the migration of hydrothermal fluids from a deep-seated reservoir to mid-crustal level along crustal-scale faults. This model allows for gold deposits to form over a range of crustal depths of more than 15 km, under a variety of P-T conditions ranging from 180 °C at <1 kbar to 700 °C at 5 kbar (Groves 1993).

The timing of gold mineralization relative to metamorphism in higher metamorphic grade rocks has been contentious. A broadly syn-peak metamorphic timing for mineralization has recently been proposed to explain a number of deposits in amphibolite and granulite facies terrains of the Yilgarn Craton (Barnicoat et al., 1991; Witt, 1993, Knight et al., 1993; Neumayr et al., 1993; Smith, 1996; Ridley et al., 2000). Others have interpreted gold deposition as pre- to syn-peak metamorphism at Hemlo, Ontario (Powell and Pattinson, 1997; Powell et al., 1999; Muir, 2002), Campbell–Red Lake, Ontario (Penczak and Mason, 1999; Thompson, 2003), and at Big Bell, Australia (Chown et al., 1984; Phillips and De Nooy, 1988; Phillips and Powell, 2009). The metamorphic devolatization model suggests that gold mineralization forms prior to the peak of metamorphism. In such cases, retrograde metamorphism is likely to have caused redistribution of gold, yielding textures that suggest gold is late (Phillips and Powell, 2009). This timing relationship implies overprinting of early gold mineralization by metamorphism and remobilization of that early gold by subsequent metamorphic events (Tomkins et al. 2004; Tomkins and Mavrogenes, 2001; Phillips and Powell, 2009). In the past two decades, complex gold depositional sequences have been documented in several gold deposits that support the concept that gold deposits form by accumulation during several hydrothermal episodes; examples include Chalice (Bucci et al., 2002, 2004), Kalgoorlie (Kent and McDougall, 1996), Big Bell (Mueller et al., 1996b), Hutti (Kolb et al., 2005) and Lapa (Simard et al., 2013).

8.2 GOLD MINERALIZATION IN THE VAL-D'OR DISTRICT

The following description of Archean greenstone-hosted orogenic lode gold deposits in the Val-d'Or district is compiled from Couture et al., (1994), Olivo and Williams-Jones (2002) and Olivo et al. (2007) and references therein, as well as from syntheses on the structure, mineralogy and alteration of Val-d'Or gold deposits presented in Robert (1990a, 1990b, 1994) and Sauvé et al. (1993).

Archean greenstone-hosted orogenic lode gold deposits occur in all rock types present in the Val-d'Or district, except for late-tectonic Archean granitic batholiths and Proterozoic diabase dykes. The most important feature in terms of deformation is the relationship with shear zones, which host or are spatially associated with the gold deposits (Robert, 1990a, 1990b, 1994). Although the gold deposits are spatially associated with a major first-order shear zone (i.e., the Cadillac Tectonic Zone), most of them are not hosted in this structure; rather, they are hosted by second- and third-order shear zones. The timing of the shear zones in the district is controversial, but there is general consensus that a significant component of the vertical elongation and thrusting along these fault zones occurred during the Kenoran orogeny (Robert, 1990b).

At least two major auriferous mineralizing events have been recognized in the Val-d'Or district on the basis of morphological and structural features, ore and alteration mineral assemblages and crosscutting relationships with intrusive rocks (Robert, 1990a, 1990b, 1994; Sauvé et al., 1993; Couture et al., 1994). The older mineralizing event is manifested by veins and breccias (e.g., the Norlartic, Marban, Kiena mines, and the Main ore zone at the Siscoe mine) that are mainly associated with second-order shear zones and commonly folded or boudinaged by D1 deformation. These veins and breccias are cut by diorite and tonalite dykes, which have U-Pb zircon ages of 2692 ± 2 (Pilote et al., 1993) and 2686 ± 2 Ma (Morasse et al., 1995). The younger auriferous event, which produced the Sigma, Lamaque, Perron-Beaufor, Shawkey, Wesdome and Camflo deposits, as well as the C quartz-tourmaline vein at the Siscoe mine, is represented by veins commonly associated with third-order shear zones. These veins clearly crosscut plutonic rocks intruded between 2694 ± 2 Ma (Wong et al., 1991) and 2680 ± 6 Ma (Jemielita et al., 1990), and may have formed durin

SECTION • 9 EXPLORATION

9.1 PROPERTY SCALE EXPLORATION

Exploration in the Val d'Or area dates back to the original discovery of gold on the property in 1923. Documented historical production of 9.5 million ounces of gold, mainly from the Sigma and Lamaque Mines, has motivated numerous periods of exploration activity conducted by several companies. The most recent phase of exploration began in 2015, shortly after Integra Gold Corp. purchased the Sigma- Lamaque Property, which includes the Sigma Mill complex. During this period, in addition to extensive drilling at Triangle, exploration drilling programs have been conducted at the Plug #4 and Parallel deposits, as well as the Aumaque, South Gabbro, Lamaque Deep, Sigma East Extension, and other targets. Development of the exploration decline at the Triangle deposit has provided underground platforms for delineation and exploration drilling programs beginning in 2016.

Due to the low levels of bedrock exposure over most of the project area, exploration targeting relies heavily on geophysical surveying combined with analysis of historical mining and exploration data. Between February 18th and March 22nd, 2017 a high resolution AeroVision (UAV-MAG) survey was completed on the Lamaque project by contractor Abibiti Geophysics, covering most of the claim blocks. Only the portion covered by the town was not surveyed. A total of 650 line-km was surveyed, on 50m spaced lines oriented north-south, with tie lines every 1000m and with a clearance height of roughly 50m. The survey permited to identify several magnetic anomalies of moderate to strong amplitudes. A series of nine exploration targets were recommended by the contractor based on the interpretation of the survey data.

In 2016, a property-scale targeting program was undertaken with the help of consultants SGS of Montreal and InnovExplo of Val-d'Or, Québec. The targeting program used all of the historical and recent exploration data on the property to generate a model for the property in order to help identify high-quality exploration targets. This compilation, along with the help of the knowledge of the local geologists, identified and prioritized a number of additional targets, including the Sigma East Extension, the South-West Target(located due south of the Lamaque West Plug), and the extension to the east of the Triangle Deposit.

9.2 SOUTH-WEST TARGET AND GABBRO SOUTH

The South West Target is located due south of the Lamaque Mine West Plug. It was identify mainly by the interpretation of a relatively small and isolated magnetic anomaly that shows similar characteristics to the Triangle Deposit. An initial drill program was successful in intersecting mineralized shear zones hosting quartz-tourmaline veins, within an intrusion of similar composition as the Triangle Plug. Follow-up drilling is planned for 2018.

Located south of the South-West Target, the Gabbro South Target is interpreted as a large E-W trending gabbro sill, near the contact between the Val-d'Or Formation and the Cadillac Group. In May of 2017, an Orevision time domain resistivity/induced polarization survey was completed on the south-western area of the property. The survey was conducted by contractor Abibiti Geophysics. A total of 27.1 line-km were surveyed on 100m spaced lines. A total of 25 chargeable sources were identified with the survey. The anomalies are believed to be related with disseminated sulphide mineralization associated with potential E-W faults and shear zones.

9.3 SIGMA EAST EXTENSION

A small isolated magnetic anomaly was identify on the extension to the east of the Sigma Mine trend, roughly 1km east of the open pit. Compilation of historical work showed that the limited drilling performed there had returned significant gold intercepts associated with quartz-tourmailine veins within large vertical shear zones.

In 2016, a small drill program consisting of six drill holes were completed to test this potential mineralization. The drilling intersected a series of sub-vertical shear zones with minor quartz veining. The zone is open and warrants further testing.

9.4 AUMAQUE BLOCK

During July to October 2015, prospecting and outcrop sampling were completed by over the Aumaque Block. Stripping revealed outrops consisting mainly of blocky lapilli tuffs with traces to 1% pyrite-pyrrhotite and well-developed schistosity locally. Several quartz-calcite-chlorite and locally tourmaline veins and veinlets with 1-5% pyrite, traces to 1% chalcopyrite and traces of pyrrhotite were identified.

A total of 285 channel samples of 1 m each were collected. Assay results vary from 0.005 to 51.1 g/t Au.

9.5 TRIANGLE DEPOSIT UNDERGROUND EXPLORATION PROGRAM

After installation of surface infrastructure in the fall of 2015 and spring of 2016, an underground exploration program was initiated in July 2016 for the Triangle Deposit. The objectives of the program were to verify the present geological model, ascertain mineralization geometry, continuity, grade and gold recovery, gather detailed information to assist in validation of future resource estimation, and complete engineering evaluation in regard to rock mass conditions, hydrogeology, and stoping. Underground exploration program is a typical advanced evaluation process for vein type deposit such as Triangle.

SECTION • 10 DRILLING

Diamond-drill holes are the principal source for geologic, grade and metallurgical data on the deposits comprising the Lamaque Project. All diamond drilling was done with wireline core rigs supplied and operated by Orbit-Garant Drilling of Val-d'Or, Québec (since 2009). Drilling was done by wireline method with N-size (NQ, 47.6 mm nominal core diameter) equipment using up to nine (9) drill rigs. Drillers placed core into wooden core boxes with each box holding about 4.5 m of NQ core. Geology and geotechnical data were collected from the core and core was photographed before sampling.

Drilling totals on the Triangle, Parallel and Plug #4 deposits are shown in Tables 10.1, 10.2, and 10.3 respectively, and Figures 10.1, 10.2 and 10.3, respectively as well. From July 2015 to the present, Integra / Eldorado Gold was responsible for planning, core logging, interpretation and supervision and data validation of the various diamond drill campaigns. Drilling during this time totaled accounts for over 70% of the total holes and meters drilled at the Triangle deposit, which hosts 83.3% of the measured and indicated mineral resources and all of the mineral reserves at the Lamaque Project. Also during this period, the critical mineral resource holes for Plug #4 were drilled. Most of the earlier drilling on this deposit was deemed unusable for mineral resource estimation due to significant influence by magnetic lithologies on the downhole surveys. Drill campaigns from earlier years, including those covering the Parallel deposit, are well described in pre-existing Technical Reports (e.g. NI 43-101 Technical Report on the Spring 2017 Mineral Resource Update for the Lamaque Project, March 22, 2017). Compaigns that targeted the Triangle and Plug #4 deposits during 2015 to end of 2017 are the focus of this section and report.

Table 10.1: Summary of Triangle Deposit drilling (surface only)

Period	# of Completed holes	# of Parent holes	# of Extended holes	# of Wedged holes	# of Abandoned holes	Meters
Pre -2009	52	52				11,244
2010-2014	121	121			8	50,308
2015	92	82	5	5	9	61,295
2016	185	157	7	21	53	106,203
2017	141	86	9	46	26	68,496
TOTAL	591	498	21	72	96	297,546

Table 10.2: Summary of Plug #4 Deposit drilling (resource eligible holes only)

Period	# of Completed holes	# of Parent holes	# of Extended holes	# of Wedged holes	# of Abandoned holes	Meters
Pre -2009	8	8				6,410
2010-2014	15	15				15,292
2015	12	12			1	8,942
2016	4	4				3,118
2017	30	19		11	10	22,921
TOTAL	69	58		11	11	56,683

Table 10.3: Summary of Parallel Deposit drilling

Period	# of Completed holes	# of Parent holes	# of Extended holes	# of Wedged holes	# of Abandoned holes	Meters
Pre -2009	104	104				25,580
2010-2014	105	105			2	29,570
2015	30	30			1	6,655
2016	0	0				0
2017	0	0				0
TOTAL	239	239			3	61,805

At the Triangle deposit the surface diamond-drill holes were augmented by 16,505 m of underground drilling in 193 drill holes. The purpose of the underground program was to define an area within the C2 Zone at an in-fill spacing of approximately 10m by 10m. The definition drilling program was carried out by contractor Forage Orbit-Garant using an hydraulic mobile drill rig. The Triangle diamond-drill holes generally ranged in length from 200 to 850 m, averaging 358 m. The longest hole drilled at Triangle totalled 2,198 m. Plug #4 drill holes ranged in length from 198 m to 1275 m, averaging 761 m.

The drill holes were drilled at an inclination of between -50deg and -75deg, and drilled mostly along 350deg to 10deg UTM N azimuths. During this period, the use of wedges and directional drilling became more and important to help control deviations to intersect the targeted zones at Triangle and Plug #4. Down-hole surveys were taken every 3 to 6m intervals using a Reflex EZ-Trac multi-shot instrument. Drill collars from surface drill holes were surveyed by a local contractor Geoposition Arpenteurs Géomètres of Val-d'Or. Underground drill holes were surveyed by the mine survey team. Also, all underground diamond drill holes were grouted from 1.5m to 6.5m using cement, in accordance to Quebec mine safety regulations.

Standard logging and sampling conventions were used to capture information from the drill core. The core was logged in detail directly into the Master Database using the Geotic software. The core was photographed before being sampled.

Eldorado reviewed the core logging procedures at site, and the drillcore was found to be well handled and maintained. Core boxes were stored into metal racks in an organized "core farm" for easy access. Data collection was competently done. Core recovery in the mineralized units was excellent, averaging over 95%. Overall the Triangle and Plug #4 drill programs and data capture were performed in a competent manner.

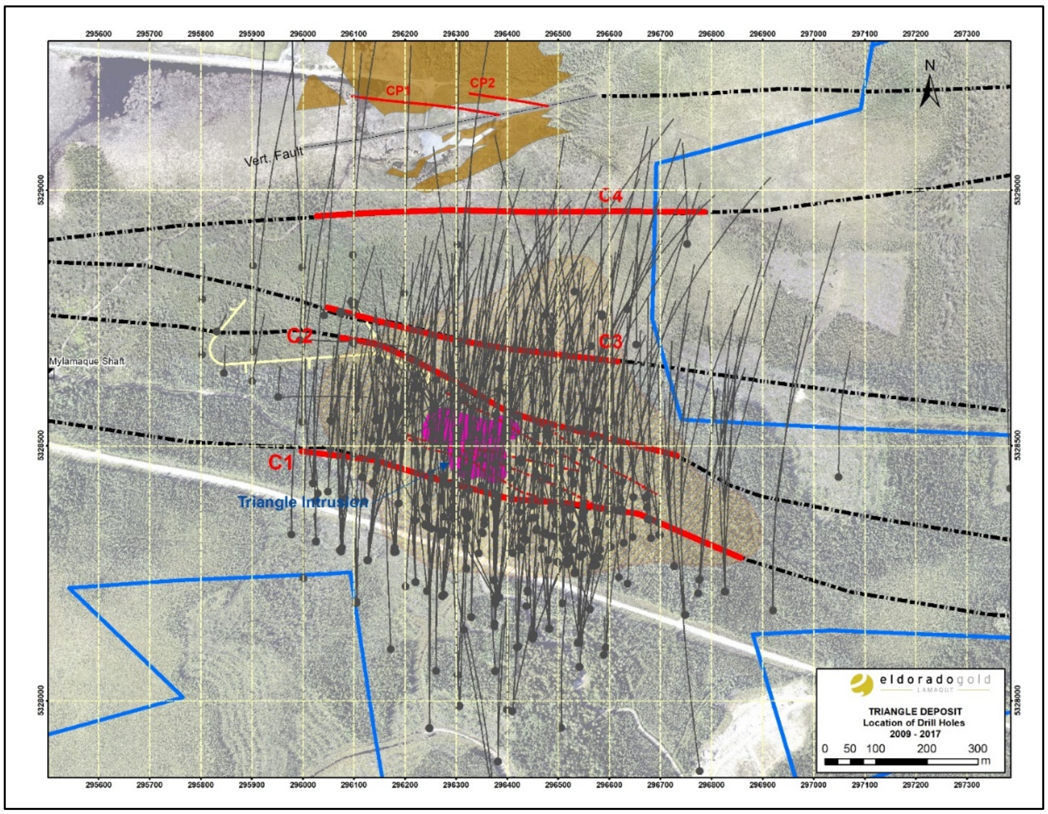


Figure 10.1: Triangle Deposit Drillhole Location Map

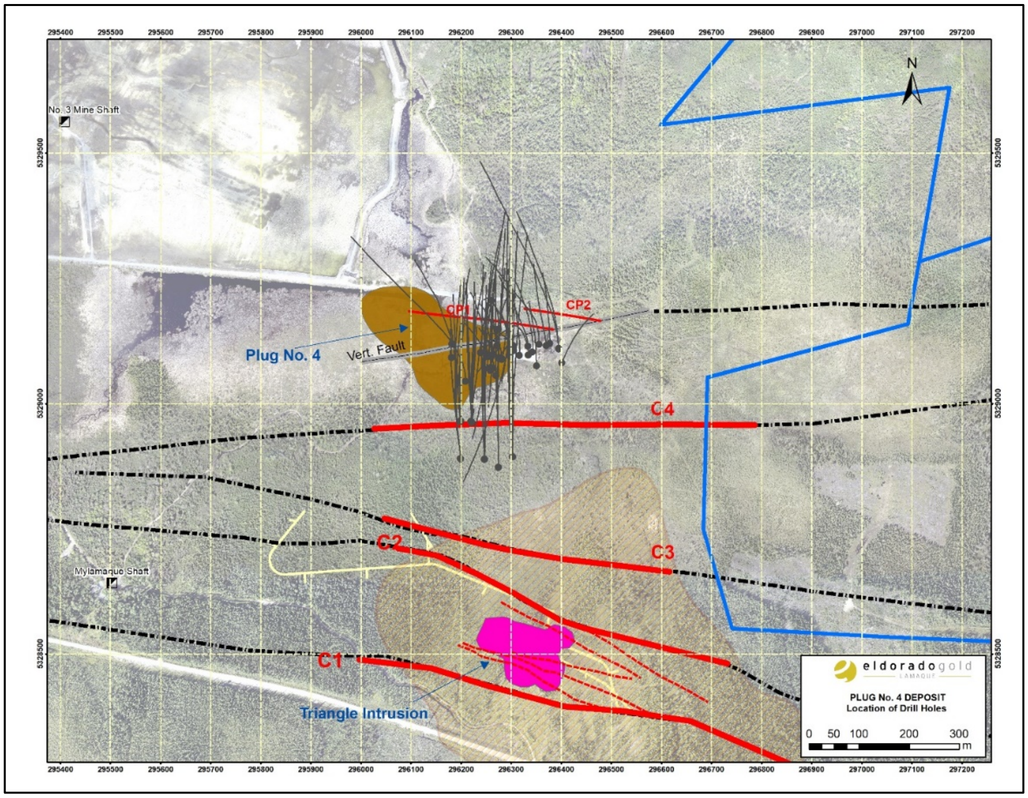


Figure 10.2: Plug #4 Deposit Drillhole Location Map

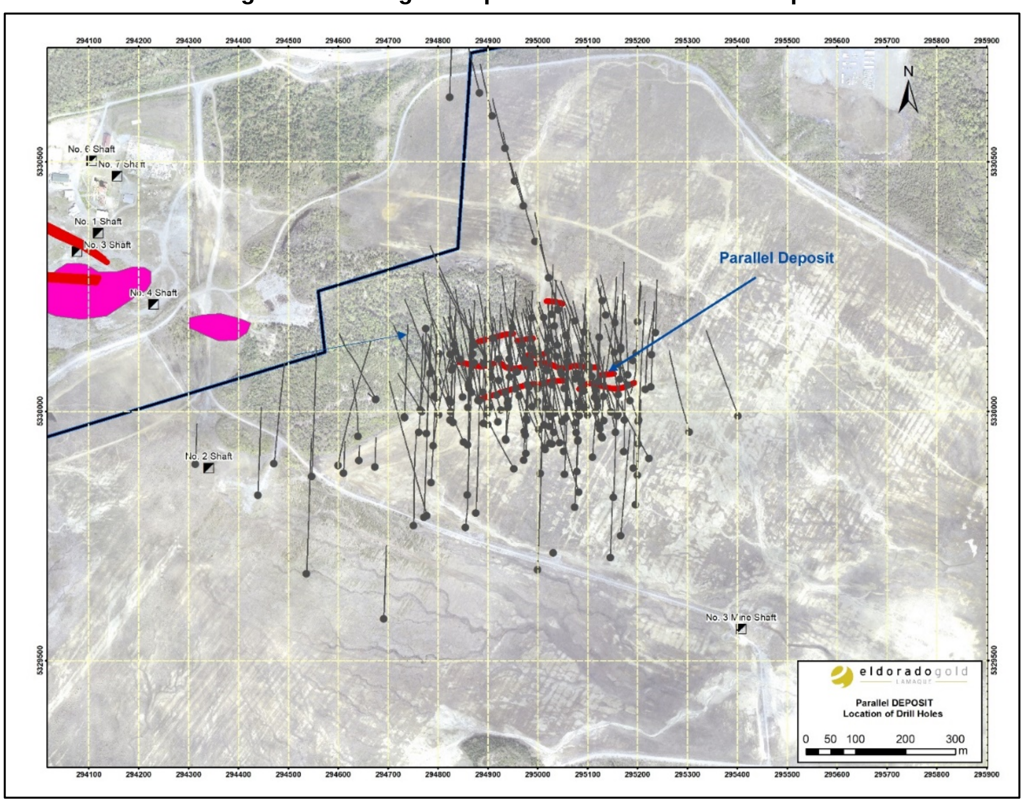


Figure 10.3: Parallel Deposit Drillhole Location Map

SECTION • 11 SAMPLE PREPARATION, ASSAYING AND SECURITY

The sample preparation and QA/QC protocols initially established in 2009 were followed for all subsequent drilling programs at Lamaque. As discussed in Section 10, this section will focus on the sampling and analyses, and quality assurance and control (QA/QC) results from samples derived from drill campaigns completed during 2015 to 2017.

Sample intervals were marked up on the drillcores by the logging geologist. All vein and shear zone occurrences were sampled with additional “bracket sampling” into unmineralizated host rock on both sides of the veins or shear zones. Typically about 40% of a hole was sampled. The core was cut at the Company’s core shack facility in Val-d’Or, Québec. For security and quality control, diamond drill core samples were catalogued on sample shipment memos, which were completed at the time the samples were being packed for shipment. Standards, duplicates and blanks were inserted into the sample stream by Eldorado staff.

The cut core samples were sent for preparation and analyses to Bourlamaque Assay Laboratories Ltd of Val-d’Or (the primary laboratory). At times a secondary laboratory, , ALS Chemex in Val-d’Or, was used.

The remaining core is stored at the Company’s core handling and storage facility in Val-d’Or, Québec. Drill core from the 2015 to 2017 campaigns are stored in metal racks which permits easy access for any additional work.

11.1 SAMPLE PREPARATION AND ASSAYING

Sample preparation procedures for routine fire assaying are to initially crush the entire sample to 10 mesh. A 250 g subsample is split by a riffle unit and pulverized to 85% minus 200 mesh. This subsample is sent for assay where a 30 g subsample is taken and fire-assayed with an atomic absorption (AA) spectrometry finish. Any values greater than or equal to 5 ppm Au were reassayed by fire assay using a gravimetric finish.

The sample batches contained QA/QC samples comprising standard reference materials (SRMs), duplicates and blanks. These were inserted at a general rate of 1 in 20, 1 in 50 and 1 in 20, respectively. The SRMs were purchased from commercial facilities specializing in their manufacture (Rock Labs and OREAS SRMs purchased from Analytical Solution Ltd.). All material used for blank samples consisted of locally-sourced barren limestone. Laboratories also inserted their own quality control samples.

11.2 QUALITY ASSURANCE / QUALITY CONTROL (QA/QC)

The QA/QC procedures assured that the assay results from a sample batch met certain rules in order to be considered “passed” and allowed to be included into the database. The pass-fail criteria were:

- Automatic batch failure if a standard result is greater than the round-robin limit of three standard deviations

- Automatic batch failure if the field blank result is over 0.02 g/t Au.

If the batch fails, it is re-assayed until it passes.

11.2.1 Blank Sample Performance

The field blank sample was taken from a gold-barren sample of crushed white marble and inserted at every 20th sample. The quality control protocol stipulated that if any result returns a value above 20 ppb Au, the batch of samples containing the blank should be re-assayed. Exception was given to results returned from unmineralized intervals. Assay performance of field blanks is shown on Figure 11.1

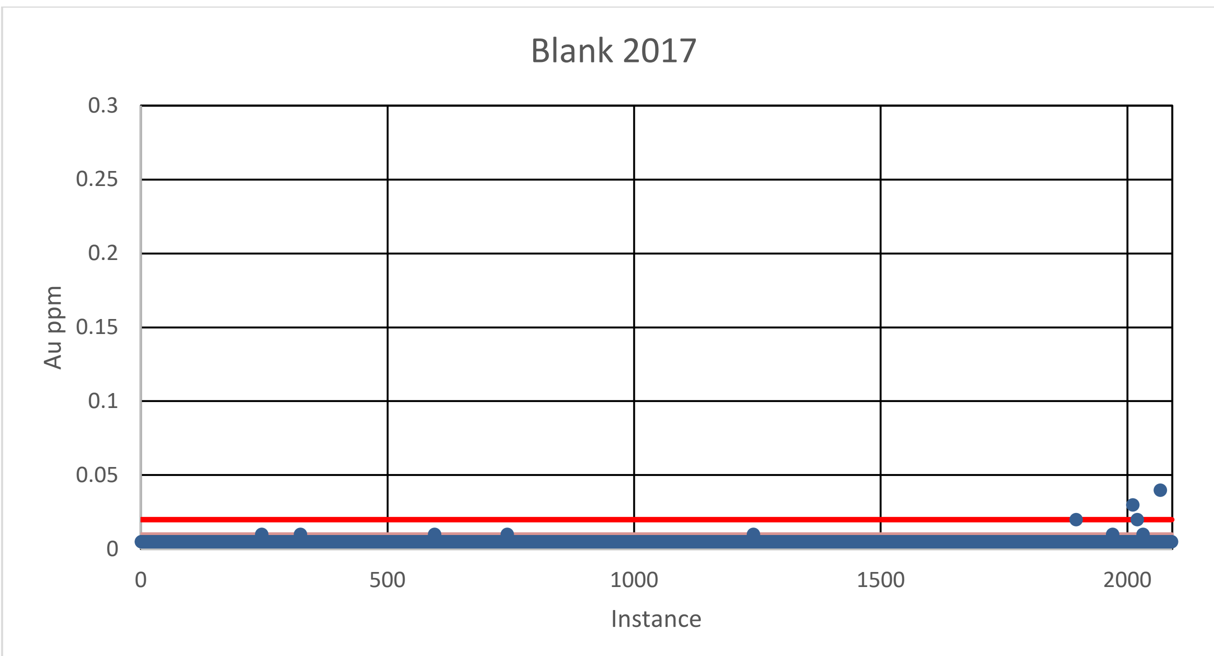


Figure 11.1 – Lamaque Blank Data – 2015 to 2017.

11.2.2 Standard Reference Material (SRM) Performance

Eldorado strictly monitors the performance of the SRM samples as the assay results arrive on site. Assaying during 2015-2017 drill campaigns used 16 different SRMs that covered a grade range from 0.34 g/t Au to 18.17 g/t Au (Table 11.1). These were inserted into the sample stream at every 20th sample. Charts of the individual SRMs are shown on Figure 11.2 and Figure 11.3. At the data cutoff date of December 31, 2017, all samples had passed acceptance criteria. Some failures represent SRMs that upon investigation were found to have been inserted amongst unmineralized samples. These were ignored and not used in any trend analysis of that SRM sample.

Table 11.1 - Standard Reference Material (SRM) samples used at Lamaque Project, 2015 to 2017.

Name	Source	CODE	Element	Au g/t Mean	Standard Deviation	Period used From	To
SN74	Rocklab	STD12	Au g/t	8.981	0.222	Mar-14	Sep-15
SL77	Rocklab	STD13	Au g/t	5.181	0.156	Feb-14	May-15
SE68	Rocklab	STD15	Au g/t	0.599	0.013	Mar-14	Mar-16
SK78	Rocklab	STD16	Au g/t	4.134	0.138	Apr-14	Apr-16
SF67	Rocklab	STD17	Au g/t	0.835	0.021	Jul-14	Aug-15
SJ80	Rocklab	STD18	Au g/t	2.656	0.057	Aug-14	Feb-16
SN75	Rocklab	STD19	Au g/t	8.671	0.199	Feb-15	Apr-16
SL76	Rocklab	STD20	Au g/t	5.960	0.192	Jan-15	May-16
SF85	Rocklab	STD21	Au g/t	0.848	0.018	Oct-15	Apr-16
SP73	Rocklab	STD22	Au g/t	18.170	0.420	Oct-15	Apr-16
OREAS 200	Oreas	STD23	Au g/t	0.340	0.012	Apr-16	present
OREAS 205	Oreas	STD24	Au g/t	1.244	0.053	Apr-16	Dec-16
OREAS 215	Oreas	STD25	Au g/t	3.540	0.097	Apr-16	present
OREAS 216	Oreas	STD26	Au g/t	6.660	0.155	Apr-16	present
OREAS 209	Oreas	STD27	Au g/t	1.580	0.044	Feb-17	present
OREAS 217	Oreas	STD28	Au g/t	0.338	0.010	Oct-17	present

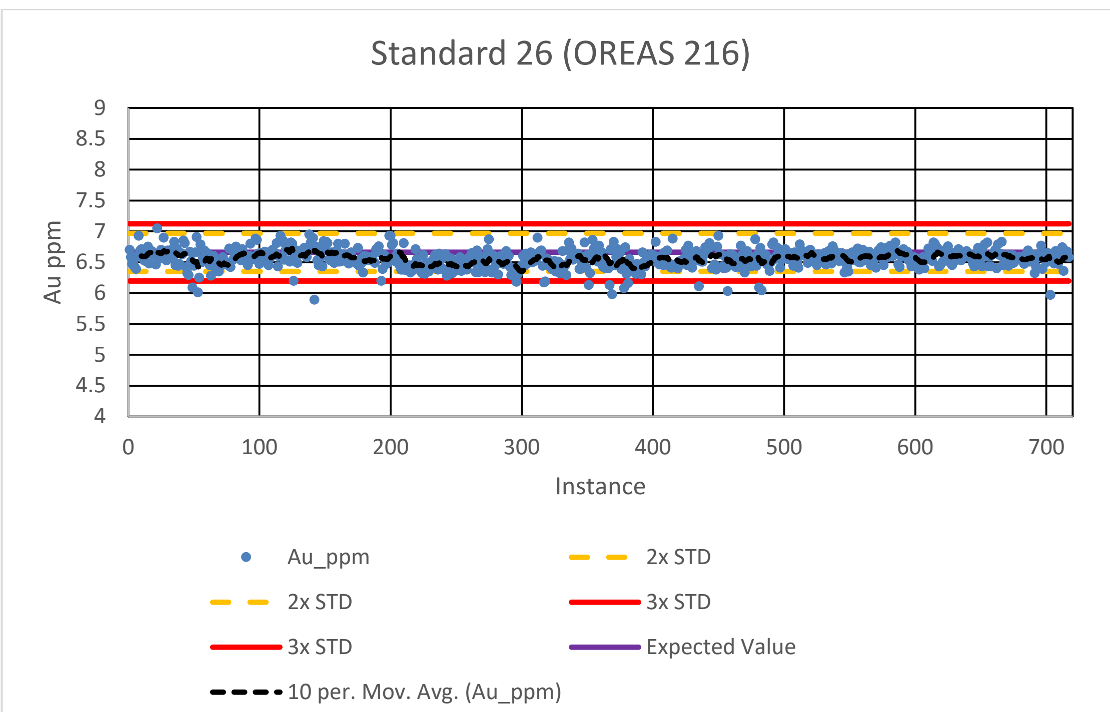


Figure 11.2 – Standard Reference Material Chart for Standard 26 (Oreas 216), 2016 to 2017.

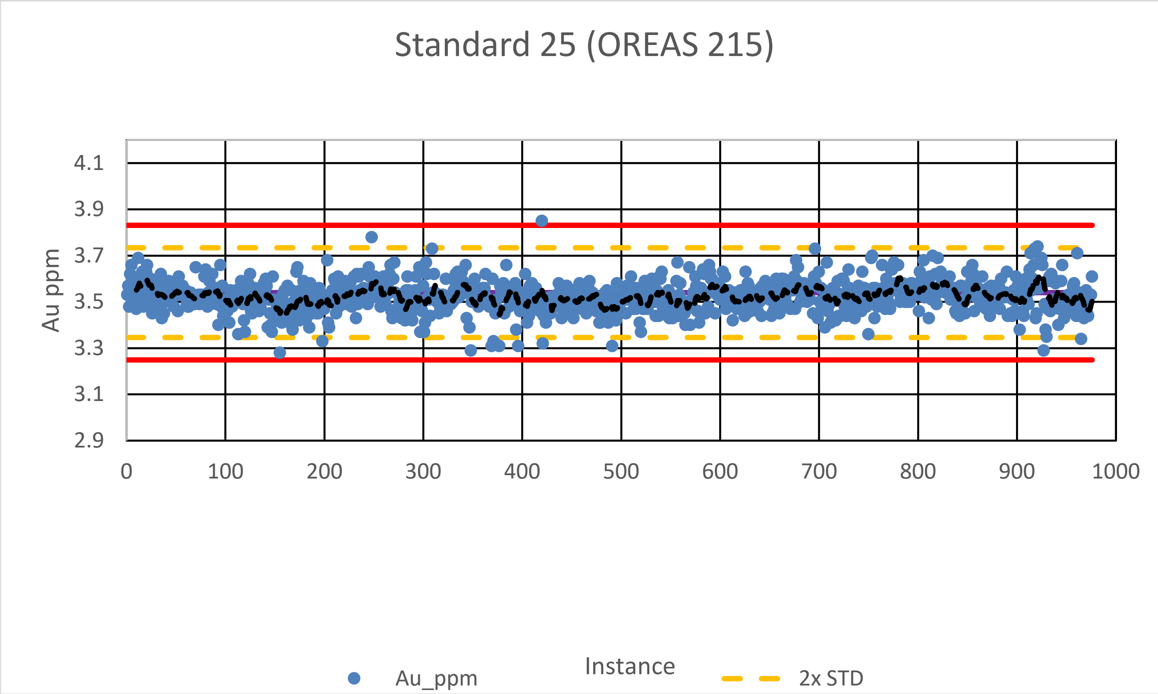


Figure 11.3 – Standard Reference Material Chart for Standard 25 (Oreas 215), 2016 to 2017.

11.2.3 Duplicate Performance

Eldorado implemented a program which monitored data from regularly submitted coarse reject duplicates. The duplicate is prepared by taking half of the crushed material derived from the original sample. These were inserted at every 50th sample. Additionally, every mineralized interval also contained at least one duplicate sample. The duplicate data are shown in a relative difference chart on Figure 11.4. In general, a maximum difference of 20% is recommended for the coarse reject duplicates. However, in gold mineralized systems that typically display effects due to extreme grades combined with the propensity for readily liberated gold during comminution (i.e. sample preparation), a higher scatter of between 30 to 40% may occur. This is what is observed at Lamaque.

To confirm that the extreme grades and liberation issues associated with the gold mineralization at Lamaque are random, effects due to potential bias were investigated in a recent re-submittal of duplicate samples for assay. The results are displayed on a Quantile – Quantile (Q-Q) plot in Figure 11.5. If the distribution lies on or oscillates tightly about the 1:1 line, then the sample population is unbiased. This is the pattern observed.

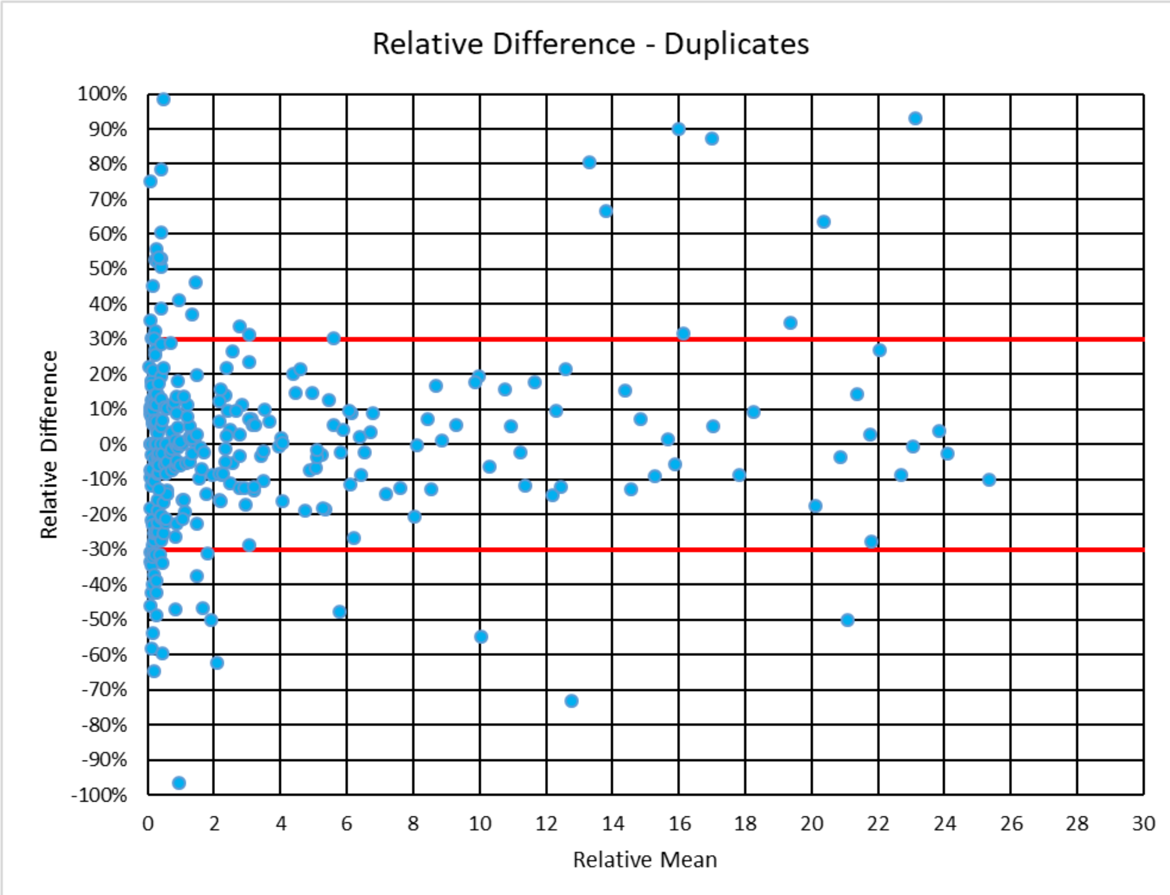


Figure 11-4 – Relative difference plot of gold duplicate data, Lamaque Project, 2016-2017.

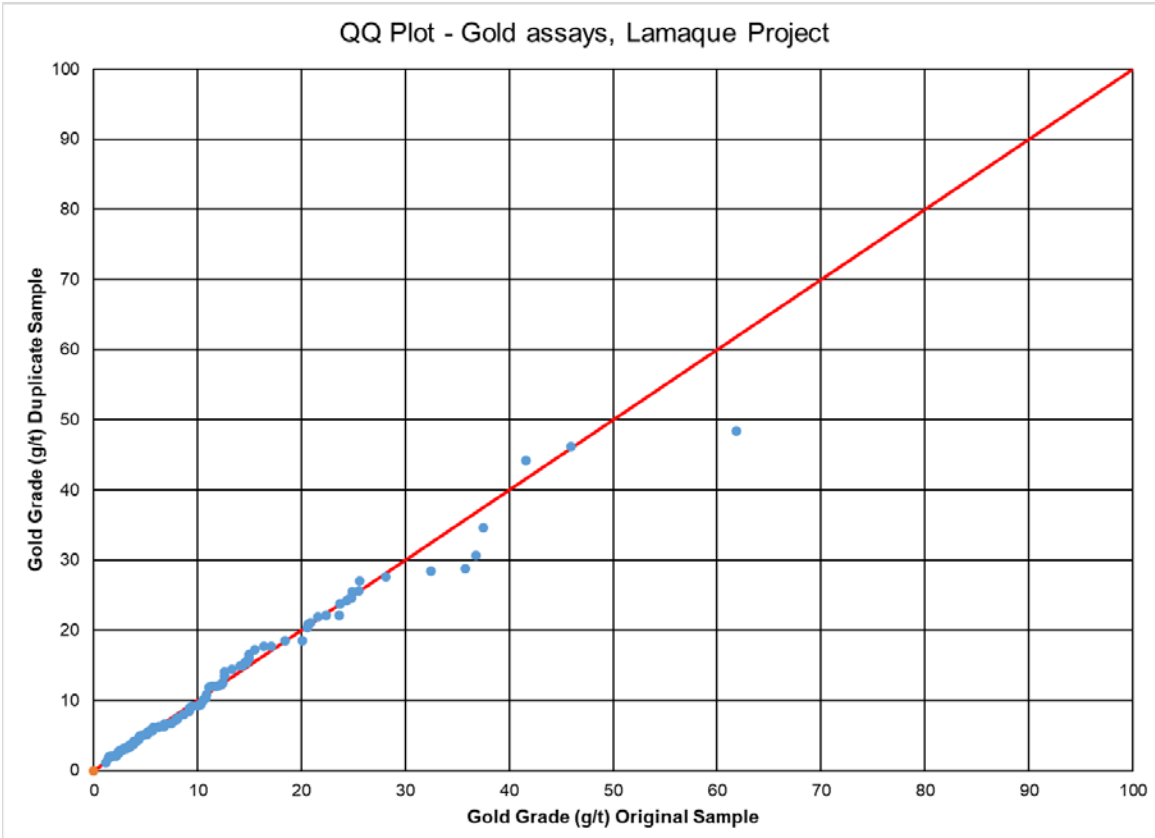


Figure 11.5 – Q-Q Plot for Gold duplicate data, Lamaque Project, 2016-2017.

11.3 CONCLUDING STATEMENT

In Eldorado's opinion, the QA/QC results demonstrate that the Lamaque Project database for assays obtained from 2015 to 2017 is sufficiently accurate and precise for resource estimation.

SECTION • 12 DATA VERIFICATION

Checks to the entire drillhole database were undertaken. Comparisons of the digital database were made to original assay certificates and survey data. Any discrepancies found were corrected and incorporated into the current resource database. Eldorado concluded that the data supporting the Lamaque Project resource work is sufficiently free of error to be adequate for estimation.

SECTION • 13 MINERAL PROCESSING AND METALLURGICAL TESTING

Initial testwork consisted of three series of laboratory testwork programs carried out with ore samples from various zones of the Lamaque deposit in 2012 and 2013 by ALS Metallurgy in Kamloops, British Columbia (Roulston D., Johnston H., Shouldice T., Metallurgical Testwork on the Lamaque Deposit, April 3, 2013, May 29, 2013 and January 16, 2014). The ore samples used for these testwork programs were prepared by Integra Gold and WSP cannot determine if they were representative of the deposit.

The first series of metallurgical testwork that concentrated on the Triangle zone was performed by SGS Minerals Services (SGS) in Lakefield, Ontario in spring 2016 to characterize the ore hardness and the achievable gold recovery using the Sigma Mill flowsheet (Verret F., Imeson D., An Investigation into the Metallurgical Response of Samples from the Triangle Zone of the Lamaque Sud Deposit, October 26, 2016).

A more extensive test program was conducted for the Triangle zone in 2017-2018 by Bureau Veritas Commodities Canada in Richmond, British Columbia.

A summary of the results obtained is presented in this section.

13.1 INITIAL TESTWORK

13.1.1 Cluster 1 and 2 testwork

In winter 2012 and spring 2013, ALS Metallurgy Kamloops carried out two testwork programs on ore samples from the Lamaque Project. The deposit consists of four independent zones named: Plug 4, Triangle, Parallel and Fortune. Table 13.1 presents the origin of the ore samples and proportion of each zone.

Both reports, KM3569 of April 3rd, 2013 and KM3876 of May 29th, 2013, present results of this initial testwork, which included:

- Ball mill work index (Bond method):
  - Performed on the Master Composite and two (2) Cluster Composites.
- Chemical assays and bulk mineral analysis via QEMSCAN:
  - Performed on each of the six (6) composite samples.
- Trace mineral search via QEMSCAN:
  - Performed on the two (2) high grade composite samples.

Some exploratory work was performed to assess the metallurgical response of Lamaque samples to:

- Gravity concentration (Knelson centrifugal concentrator and hand panning);
- Cyanide leach (standard bottle roll procedure);
- Rougher flotation and regrind.

13.1.1.1 Sample Description

The samples arrived at the ALS Metallurgy facilities on November 5, 2012. The samples weighed approximately 247 kilograms and had been crushed to < 6 mesh. They were split into six composites predetermined by Integra Gold, based on sample origin and gold grade.

They were identified as follows:

Table 13.1 – Composite Samples Origin and Weight

Location	Sample Name	Sample Origin	Weight (kg)
South	Cluster 1 – High Grade	90% Plug 4 + 10% Triangle	41.4
	Cluster 1 – Average	86% Plug 4 + 14% Triangle	44.4
	Cluster 1 – Cutoff Grade	70% Plug 4 + 30% Triangle	47.6
North	Cluster 2 – High Grade	65% Parallel + 35% Fortune	37.1
	Cluster 2 – Average	63% Parallel + 37% Fortune	42.1
	Cluster 2 – Cutoff Grade	53% Parallel + 45% Fortune	33.7

13.1.1.2 Sample Preparation

The composite samples were homogenized and rotary split into two (2) kilogram charges. The laboratory prepared a Master Composite consisting of eight (8) kilograms of each of the six (6) composite samples for a total of 48 kilograms. In the second testwork program, Cluster 1 and Cluster 2 Composites were also prepared, each with an equal weight of the Cutoff, Average and High grade samples. All the composites were stored under nitrogen in a sealed container until they were needed in the test program.

13.1.1.3 Sample Characterization

COMMINUTION TESTWORK

A Bond ball mill work index (BWI) test, with a closing screen size of 106 µm, was conducted on the Master Composite, the Cluster 1 Composite and the Cluster 2 Composite. The BWI can be used with Bond's Third Theory of comminution to calculate the net power requirement for grinding. The ball mill work index ranged from 13.8 to 14.9 kWh / tonne. This is considered as average in hardness.

Table 13.2 – Ball Mill Work Index

Report	Sample Description	kWh/t
KM3569	Master Composite	13.8
KM3876	Cluster 1 Composite	14.8
KM3876	Cluster 2 Composite	14.9

The two Cluster Composites generated very similar BWI indices. However, further tests will have to be performed with material of the hardest zones of the mineral deposit in order to clearly establish the overall grinding performance.

CHEMICAL ASSAY AND MINERALOGY

Duplicate chemical assays were conducted on the six (6) composite samples, on the Master Composite and on the Cluster 1 Composite. The gold assay of the Master Composite sample gave about 8 g/t. The individual composite samples ranged from 3 g/t to 15 g/t Au. Bulk Mineral Analyses via the QEMSCAN were also conducted on the six (6) composite samples and on the Cluster 1 Composite to determine the mineral content of each sample.

Table 13.3 – Head Assays Summary

Sample Description	Au	Ag	Fe	STOTAL	CTOTAL	As	TOC*
	g/t	g/t	%	%	%	%	%
Master Composite	8.15	4	5.1	1.47	-	0.007	-
Cluster 1 – High	15.3	4	6.4	2.08	1.56	0.003	0.02
Cluster 1 – Average	6.28	1	6.4	1.72	1.76	0.002	0.02
Cluster 1 – Cutoff	3.16	1	6.3	1.49	1.59	0.003	0.02
Cluster 2 – High	14.6	3	4.1	1.57	1.04	<0.002	0.05
Cluster 2 – Average	6.14	2	4.5	1.20	1.02	<0.002	0.02
Cluster 2 – Cutoff	3.21	<1	4.0	0.99	0.79	<0.002	0.02
Cluster 1 Composite	8.66	-	6.3	1.97	-	-	-

*TOC = Total Organic Carbon

The total sulphur content of the ore samples ranged from 1 to 2%. Sulphur was present primarily as pyrite with traces of chalcopyrite.

Iron ranged from 4 to 6%. The Cluster 1 samples contained more pyrite and amphibole which explains the greater presence of iron (6%).

The assays showed a 1 to 2% total carbon content but only a very small portion of it present in the organic (elemental) form. Organic carbon can potentially have a preg-robbing effect and consume cyanide, and is thus often detrimental to cyanidation.

The gangue minerals were primarily quartz and feldspars. The quartz content varied between 22% and 36% and the feldspars between 15% and 24%.

A "Trace Mineral Search" was conducted via QEMSCAN on the high grade samples to determine liberation levels, size and association of the gold. The study was conducted at 80% passing 84 µm particle size for Cluster 1 and 80% passing 80 µm particle size for Cluster 2. Three slides were scanned for each of the two composites. A total of 40 gold particles were observed for Cluster 1 and 31 particles for Cluster 2.

It showed that 69 and 77% of the gold particles in Cluster 1 and Cluster 2 High grade samples, respectively, were liberated or in binary particles with sulphide minerals.

Table 13.4 – High Grade Gold Trace Mineral Search1

Sample Description	Primary Grind Size 80% passing	Liberated Gold	Au Binary – Sulphide Minerals		Au Binary – Gangue Minerals		Au Multiphase	
	µm	%	% adhesion	% inclusion	% adhesion	% inclusion	% adhesion	% inclusion
Cluster 1 – High	84	47.5	10.0	12.5	5.0	15.0	2.5	7.5
Cluster 2 – High	80	41.9	32.2	3.2	9.7	9.7	-	3.2

1 Percentages shown relate to number of particles without taking into account particle size.

The portion of the gold particles present in inclusion was greater in Cluster 1 than in Cluster 2. Gold in inclusion is more difficult to leach as the exposed surface is low or nonexistent. In this case, finer grinding may minimize this impact.

13.1.1.4 Metallurgical Test Program

The purpose of the test program was to establish a preliminary flowsheet. The program consisted of:

- Gravity concentration followed by cyanide leach of the gravity tailings;
- Rougher flotation test of the whole ore samples.

GRAVITY CONCENTRATION AND CYANIDE LEACH

Four gravity concentration tests were performed on the Master Composite at different grind sizes (80% passing 130, 105, 79 and 56 µm). The tests consisted of processing a 4-kg pre-ground sample in a laboratory 3” Knelson centrifugal concentrator. The concentrate thus obtained was then hand panned for upgrade. Both Knelson gravity concentration tailings and pan tailings were leached together in a 1,000 ppm sodium cyanide concentration and pH 11 for a 48-hour retention time.

The highest overall gold recovery obtained was 89% at the 80% passing 79 µm grind size. About 23% of this overall recovery was via the upgraded gravity concentrate.

The same procedure was applied to the six (6) zone composite samples at an 80% passing 75 µm grind size. These samples were subjected to three optimization tests as follows:

- Increase of the retention time from 48 to 96 hours;
- Reduction of the grind size of the generated Knelson concentrate to 80% passing 50 µm and increase of the retention time to 96 hours;
- Increase of the cyanide concentration to 5,000 ppm NaCN for the Cluster 1 samples.

A summary of the tests conditions and the recovery results is displayed in Table 13.5.

Table 13.5 – Gravity Concentration and Cyanide Leach - Conditions and Recoveries

Test conditions								
Grind Size, 80% Passing, µm	~ 75		~ 75		~ 75		~ 50	
Sodium Cyanide Concentration, mg/L	1,000		5,000		1,000		1,000	
Retention Time, h	48		48		96		96	
pH	11		11		11		11	
Gold Recovery (%)								
	Gravity	Total	Gravity	Total	Gravity	Total	Gravity	Total
Master Composite	23	89.0						
Cluster 1								
Low Grade	12.3	81.0	18.5	86.8	18.4	88.7	20.3	91.6
Average	22.8	84.3	24.4	90.0	25.1	88.9	26.2	92.2
High Grade	14.8	79.3	17.3	86.4	16.4	87.5	20.9	91.1
Cluster 2								
Low Grade	27.7	92.6			37.8	97.4	38.5	97.8
Average	31.6	94.3			29.9	96.9	44.3	98.2
High Grade	36.1	93.0			30.2	97.1	44.3	98.3

There was a notable difference in the performance of Clusters 1 and 2 samples. Gravity and leach recoveries were better for Cluster 2. Depending on the test conditions, the total gold recovery ranged from 79 to 92% for Cluster 1 and from 93 to 98% for Cluster 2. In both cases, recovery increased with both the fineness of grind and retention time.

Increase of the leach retention time to ninety six (96) hours with the same cyanide concentration, increased the overall recovery.

Increase of the sodium cyanide concentration to 5,000 ppm, with a forty eight (48) hours leaching time, increased the overall recovery.

Diagnostic leaching of the tailings showed that part of the gold present in the Cluster 1 samples was not in a leachable form (gold was locked in the sulphide matrix and gangue minerals).

ROUGHER FLOTATION

Rougher flotation tests were done on the Master Composite and the three (3) Cluster 1 samples. The objective was to evaluate the viability of incorporating flotation into the overall process.

A two (2) kg sample of the Master Composite was ground to 80% passing 130 µm while Cluster 1 samples were ground to 80% passing 108-127 µm. Flotation was conducted in 4.4 liter lab cells with Potassium Amyl Xanthate (PAX) as the collector and Methyl Isobutyl Carbonyl (MIBC) as the frother.

The Master Composite produced concentrate containing seventy one grams per tonne (71 g/tonne) of gold with a recovery of 89%. The concentrate produced from the Master Composite sample represented approximately 11% of the feed mass and 96% of the feed sulphur.

Table 13.6 – Summary of Flotation Gold Recoveries Obtained

Sample	% Gold Recovery	% Mass Recovery
Master Composite	89	11
Cluster 1 - High	88	14
Cluster 1 - Average	90	14
Cluster 1 - Cutoff	82	13

FLOTATION, REGRIND AND CYANIDATION

Testing consisted in a rougher sulfide flotation using Potassium Amyl Xanthate (PAX) as the collector and Methyl Isobutyl Carbonyl (MIBC) as the frother, at natural pH. Then, the flotation concentrate was finely ground to 80% passing 7 µm. The concentrate was then intensively leached at 5,000 ppm of sodium cyanide for 48 hours, and the tailings leached at 1,000 ppm of sodium cyanide also for 48 hours.

Cluster 1 Composite was submitted to rougher flotation at three grinding sizes of 80% passing 107, 133 and 206 µm, respectively. The three composite samples (high, average, cutoff) were submitted to rougher flotation at a single grind size only.

Table 13.7 – Flotation, Regrind and Cyanide Leach of Concentrate and Tailings – Gold Recoveries

Description	Total Gold Recovery (%)			
	107 µm	133 µm	206 µm	200 µm*
Cluster 1 Composite	96.0	95.5	93.6	
Cluster 1 – High				94.7
Cluster 1 – Average				95.1
Cluster 1 – Cutoff				89.7

*236 µm for Cluster 1 Cutoff.

Decreasing the primary grind size from 206 to 107 µm, increased the overall recovery.

More than 12% gold in the feed ended up in the flotation tailings. Therefore, leaching these tailings increased the overall gold recovery. Without that, gold recovery would average at 73, 87 and 83% respectively for the cutoff, average and high grade samples.

13.1.2 Potential Processing Facility

A third series of metallurgical testwork was undertaken in October 2013 (report KM4025, January 16, 2014). The objective was to compare the potential metallurgical results of four (4) different flowsheets. These flowsheets were based on six (6) existing milling facilities that could potentially process the Lamaque Sud mineralized material on a custom milling basis. These flowheets were as follows:

- Flowsheet 1: Gravity and carbon-in-leach (CIL);
- Flowsheet 2 : Whole ore cyanidation;
- Flowsheet 3: Whole ore cyanidation and carbon-in-leach (CIL);
- Flowsheet 4: Flotation followed by cyanidation of concentrate and flotation tailings.

13.1.2.1 Sample Description and Preparation

Two drums containing 183 samples were received at the ALS Metallurgy Kamloops laboratory in October 2013. The material, already crushed to minus 6 mesh, was identified according to the mineralized zone it was sampled from: Plug 4, Triangle, Parallel and Fortune, and grouped according to the gold grade (cutoff, average and high grade). The laboratory prepared a composite sample for each zone of the mineral deposit.

For this part of the testwork, the grades were blended. The composites were homogenized, split into two (2) kilogram lots and stored in a sealed container under nitrogen in a freezer until they were ready to be used for testing.

Chemical assays were performed on duplicate head cuts from each of the four composites and then the selected flowsheets were tested. All of them were tested with a primary grind size of 80% passing 75 µm.

13.1.2.2 Properties of the Four (4) Zone Composites

Table 13.8 shows a summary of the head sample assays. The gold grade varies from 4.5 g/t to 9.13 g/t, depending on the ore zone.

Table 13.8 – Summary of Chemical Assays

Composite	Assays					
	Cu	Zn	Ag	STOTAL	S2-	Au1
	%	%	g/t	%	%	g/t
Fortune	0.024	0.03	4	1.11	1.08	6.3
Parallel	0.029	0.02	3	1.49	1.46	9.1
Triangle	0.009	0.01	5	1.58	1.54	8.8
Plug 4	0.011	0.01	2	1.78	1.74	4.5

1Gold assays were completed using a screened metallics assay method.

13.1.2.3 Metallurgical Performance

FLOWSHEET 1: GRAVITY AND CARBON-IN-LEACH (CIL)

Ten (10) kilograms of each composite ore sample were fed into a batch Knelson gravity concentrator with hand panning of the gravity concentrate for upgrade. Tailings from gravity concentration and upgrade were submitted to a carbon-in-leach test under the following conditions: 30 g/L of activated carbon, 1,000 ppm of sodium cyanide, pH 11 and 96-hour retention time. Table 13.9 presents the obtained results.

Table 13.9 –Flowsheet 1- Recoveries Obtained

Gold Recovery (%)		
Composite	Gravity Concentration	Total
Plug 4	13.7	87.6
Triangle	17.6	93.0
Parallel	47.6	97.8
Fortune	26.8	96.6

FLOWSHEETS 2 AND 3: WHOLE SAMPLE CYANIDATION AND CARBON-IN-LEACH (CIL)

For the second and third flowsheets, the same conditions were applied to each flowsheet with 1,000 ppm sodium cyanide concentration, pH 11 and 96-hour retention time. However, for the third flowsheet, 30 g/L of activated carbon were added to the leach slurry. Tables 13.10 and 13.11 present the results obtained.

Table 13.10 – Flowsheet 2 - Recoveries Obtained

Gold Recovery (%)	
Composite	Total
Plug 4	83.2
Triangle	92.9
Parallel	97.1
Fortune	95.6

Table 13.11 – Flowsheet 3 - Recoveries Obtained

Gold Recovery (%)	
Composite	Total
Plug 4	85.1

Triangle	93.4
Parallel	96.6
Fortune	97.1

FLOWSHEET 4: FLOTATION WITH CYANIDATION OF CONCENTRATE

Flotation (rougher and cleaner) was carried out at a natural pH with Potassium Amyl Xanthate (PAX) as the collector and with Methyl Isobutyl Carbonyl (MIBC) as the frother. The final concentrate recovered, which was about 3 to 4% of the feed mass, was leached under 2,000 ppm of sodium cyanide and pH 11 for a period of 96 hours. The flotation tailings were also leached with a 1,000 ppm sodium cyanide concentration, the same pH and retention time.

Table 13.12 – Flowsheet 4 - Recoveries Obtained

Gold Recovery (%)			
Composite	Concentrate Leach	Tails Leach	Total
Plug 4	58.4	24.0	82.4
Triangle	71.1	20.8	91.9
Parallel	85.8	8.9	94.7
Fortune	82.0	13.1	95.1

Overall gold recoveries from the first three flowsheets were comparable. With these flowsheets, gold recoveries from the Parallel, Triangle and Fortune composites ranged from 93 to 98% and recovery for the Plug 4 composite varied from 83 to 88%.

The Trace Mineral Search performed during the first series of tests (April 3, 2013 report) shows that the Cluster 1 sample, the majority of which is composed of the Plug 4 zone, presents gold particles in inclusion and is therefore more difficult to leach. But tests done on the Cluster 1 sample with a finer grind demonstrated a better gold recovery.

Gravity concentrates graded between 257 and 1,218 g/tonne of gold.

The sodium cyanide consumption doubled for a flowsheet using the CIL process. On average, it varied from 0.95 to 1.98 kg NaCN/tonne of feed.

Flowsheet 4, using flotation, showed lower recoveries compared to the other three. Leaching of the concentrate recovered from 58 to 86% of the feed gold. Leaching of the flotation tailings recovered 9 to 24% of the feed gold.

13.2 SGS TRIANGLE ZONE TESTWORK

In spring 2016, SGS conducted a series of metallurgical tests on samples of the Triangle zone to characterize the hardness and the achievable gold recovery using the Sigma Mill process flowsheet.

Two composite samples were produced: one for the Superieur section of the zone and the other for the Inferieur section. An effort was made to represent the various sub-zones within the composite samples and to match the expected average mined grade (including dilution).

13.2.1 Sample Description and Preparation

Approximately 200 kg of drill core samples of the Superieur and Inferieur sections of the Triangle zone of the Lamaque Sud deposit were received at the SGS Lakefield site in March 2016.

Samples from the Superieur and Inferieur sections of the Triangle zone were first prepared separately for grindability tests, head characterization and cyanidation tests at 80% passing 75 µm grind size. Once it was confirmed that the two composites had similar compositions and metallurgical responses, a Triangle Blend composed of ⅓ Superieur and ⅔ Inferieur was prepared.

13.2.2 Chemical Analyses

Screened metallic tests conducted at 150 mesh showed that the Inferieur composite contained more gold and silver than the Superieur composite. The results of the screened metallic tests and the chemical analyses are summarized in the Table 13.13.

Table 13.13 – Triangle Zone Composites Head Assays Summary

Sample Description	Head Grade		+ 150 mesh			- 150 mesh					
	Au	Ag	Weight	Au	Ag	Weight	Au	Ag	Cu	S2-	TOC
	g/t	g/t	g	g/t	g/t	g	g/t	g/t	%	%	%
Superieur	5.94	< 2.0	26.79	7.45	< 5.0	982.91	5.90	2.1	0.007	0.91	0.07
Inferieur	9.41	3.1	26.33	21.34	5.7	960.97	9.09	3.0	0.009	1.46	0.09

13.2.3 Mineralogical Analyses

Pyrite was the main sulphide mineral and ranged from 1.91% in the Superieur composite to 2.44% in the Inferieur composite. The pyrite particles were found to be 87.9% and 97.0% free and liberated for the Superieur and Inferieur composites respectively. Locked pyrite particles were mostly found in complex associations.

13.2.4 Acid Generation Potential

The Superieur and Inferieur composites were found to be non-acid generating and analysis of the leach liquor did not find any acid generating elements to be present in significant concentration.

13.2.5 Grindability Testing

Grindability tests characterized the Superieur and Inferieur composites as hard in terms of Bond Rod mill Work Index, moderately hard in terms of Bond Ball mill Work Index, and moderately abrasive. The overall test results are summarized in Table 13.14.

Table 13.14 – Overall Grindability Test Summary

Sample Name	RWI kW·h/t	BWI kW·h/t	AI g
Superieur[1]	17.3	16.41	0.262
Inferieur[2]	16.7	15.61	0.247
Triangle Comp.	-	15.52	-

1 P80 = 81 µm.
2 P80 = 46 µm.

13.2.6 Metallurgical Test Program

Gravity separation and cyanidation tests were performed on the three composites in order to verify their metallurgical performance using the Sigma processing plant.

13.2.6.1 Gravity Separation Tests

Gravity tests were performed on the three composites at a target grind size of 80% passing 75 µm, and 50 µm for the Triangle Blend composite. The tests consisted of processing a 10 kg test charge in a Knelson MD-3 concentrator in a single pass. The concentrate obtained was recovered and upgraded on a Mozley Laboratory Separator (MLS). The Mozley concentrate was assayed to extinction for gold. The Knelson and Mozley tailings were combined for cyanidation testwork.

A summary of the results is presented in Table 13.15. For the Triangle Blend composite, gravity recovery was 30.8% at a P80 of 71 µm and around 35% at a P80 of 49 µm.

Table 13.15 – Gravity Separation Test Results Summary

Test #	Composite	Primary Grind Size	Mozley Concentrate		
		P80 (µm)	Grade Au (g/t)	Distribution (%)	
				Weight	Au
G-1	Inferieur	74	4 568	0.048	24.3
G-2	Superieur	72	7 112	0.028	33.6
G-3	Triangle	49	4 821	0.056	35.1
G-4	Triangle	49	4 484	0.062	34.6
G-5	Triangle	71	1 223	0.198	30.8

13.2.6.2 Cyanidation Tests

Bottle roll cyanidation tests on gravity tailings were conducted at a cyanide concentration of 1 g/L NaCN with and without pre-aeration and the use of lead nitrate. Two grind sizes were tested: 80% passing 75 and 49 µm, which was the lowest considered achievable with the three mills available in the Sigma plant.

The gold extraction ranged from 87.6% to 90.2% at a P80 around 75 µm without pre-aeration. Under the same conditions, cyanide and lime consumptions varied from 0.83 to 1.06 kg/t NaCN and 0.47 to 0.56 kg/t CaO. Results are shown in Table 13.16.

Grinding to 80% passing 49 µm increased recovery to between 91.7 and 92.6%. Pre-aeration significantly reduced cyanide consumption, from 1.42 ~ 1.55 to 0.20 ~ 0.25 kg/t NaCN at P80 of 49 µm and from 0.89 to 0.19 kg/t NaCN at P80 of 71 µm. Pre-aeration also considerably improved cyanidation kinetics.

Table 13.16 – Cyanidation Test Results Summary

Test #	Gravity Test	P80	Lead Nitrate	Pre-aeration	Reagent Cons. (kg/t CN Feed)		Gold Extraction (%)						Assays (g/t)	
	Tailings	(µm)			NaCN	CaO	24h	36h	48h	60h	72h	84h	Residue *	Calc. Head
CN 1	G-1	74	✓	-	0.83	0.53	87	84	88	89	90	90.0	0.69	6.87
CN2	G-2	72	✓	-	1.06	0.56	83	83	85	84	86	87.6	0.49	3.90
CN3	G-3	49	✓	-	1.55	0.61	80	80	83	83	85	91.7	0.42	5.06
CN4	G-3	49	-	✓	0.20	0.94	87	89	90	91	92	92.0	0.41	5.08
CN5	G-3	49	✓	✓	0.25	0.91	87	88	91	91	92	92.3	0.39	5.05
CN6	G-3	49	-	-	1.42	0.61	82	84	85	86	87	92.4	0.38	4.97
CN7	G-4	49	✓	✓	0.30	1.05	-	-	-	-	-	92.6	0.39	5.29
CN8	G-5	71	-	✓	0.19	0.87	84	85	87	88	89	90.4	0.52	5.40
CN9	G-5	71	-	-	0.89	0.47	82	85	87	88	89	90.2	0.54	5.47

*Average of duplicate assays

13.2.6.3 Overall Gold Recovery

The overall gold recovery results for the gravity and cyanidation process are presented in Table 13.17. For the Triangle composite, overall gold recovery averaged 93.3% at a P80 of 71 µm and increased to an average of 94.9% at a P80 of 49 µm.

Table 13.17 – Overall Gold Recovery Summary

Composite	P80	Gold Extraction (%)			Gold Head Grade (g/t)		Gold Grade CN Residue
Test ID	(µm)	Gravity	CN Leach	Overall	Calc.	Direct	(g/t)
Inferieur G-1 + CN1	72	24.3	90.0	92.4	9.07	9.41	0.69
Superieur G-2 + CN2	74	33.6	87.6	91.7	5.87	5.94	0.49
Triangle G-3 + CN3	49	34.9	91.7	94.6	7.76	-	0.42

Composite	P80	Gold Extraction (%)			Gold Head Grade (g/t)		Gold Grade CN Residue
Test ID	(µm)	Gravity	CN Leach	Overall	Calc.	Direct	(g/t)
Triangle G-3 + CN4			92.0	94.8			0.41
Triangle G-3 + CN5			92.3	95.0			0.39
Triangle G-3 + CN6			92.4	95.1			0.38
Triangle G-4 + CN7	49	34.6	92.6	95.2	8.09	-	0.39
Triangle G-5 + CN8	71	30.8	90.4	93.4	7.84	-	0.52
Triangle G-5 + CN9			90.2	93.2			0.54

13.2.7 Gold Deportment Study on Cyanidation Residue

A microscopic gold deportment study was performed on the combined residue from cyanidation tests CN3, CN4 and CN5. The study showed that the main gold mineral is calaverite ($AuTe_2$), with moderate amounts of petzite (Ag_3AuTe_2) and native gold, accounting for 65%, 19% and 15%, respectively. Based on the study, 39% of the microscopic gold particles (>0.5 µm) were liberated or exposed while 61% were present as locked gold particles. The major host minerals for exposed and locked gold were found to be pyrite (69%) and apatite (30%).

13.2.8 Cyanidation Destruction

Cyanide destruction testwork using the SO_2/air process, with sodium metabisulphite as the SO_2 source, was conducted on the residue from test CN7. All continuous tests used a retention time of around 84 minutes. The addition of 6.92 g SO_2/g CN_{WAD}, 3.23 g lime/g CN_{WAD} and 0.08 g Cu/g CN_{WAD}, were necessary to achieve a final concentration of < 1.0 mg/L CN_{WAD}.

13.3 BV TRIANGLE ZONE TESTWORK

In November 2017, Bureau Veritas (BV) started a new metallurgical test campaign on samples from the Triangle zone and rod mill feed samples collected during the first bulk sample campaign at Camflo mill.

13.3.1 Sample Preparation and Head Assays

Eight different Triangle zone samples made up of coarse assay rejects were collected, one for C1, three for C2, three for C4 and one for C5. In addition, two core samples, one for C2 and one for C4, were also collected in order to be able to measure the sample rod mill work indexes. From the coarse assay rejects, three composite samples were assembled: the C2, C4 and Master composites. The individual samples from which these composites were assembled are listed in Table 13.18

Table 13.18 – Assembly of Composite Samples

Composite	Assembly
C2 composite (30 kg)	10 kg C2 Upper
	10 kg C2 Mid
	10 kg C2 Lower
C4 composite (30 kg)	10 kg C4 Upper
	10 kg C4 Mid
	10 kg C4 Lower
Master composite (80 kg)	10 kg C1
	10 kg C2 Upper
	10 kg C2 Mid
	10 kg C2 Lower
	10 kg C4 Upper
	10 kg C4 Mid
	10 kg C4 Lower
	10 kg C5

Sample and composite specific gravities and head assays are presented in Table 13.19. Head assays were measured both by direct fire assay and the screened metallics procedure. As can be seen, in some cases results from the two methods differ significantly, most likely due to the presence of free gold.

Table 13.19 – BV Test Campaign Sample Head Assays

	Unit	Master Comp.	C2 Comp.	C4 Comp.	C1	C2 Upper	C2 Mid	C2 Lower	C4 Upper	C4 Mid	C4 Lower	C5	C2 Core	C4 Core
SG	g/cm3	2.84	2.84	2.82	2.79	2.85	2.84	2.81	2.84	2.77	2.83	2.84	2.82	2.89
Au (FA)	g/mt	7.13	5.07	8.06	5.93	3.91	3.59	6.25	8.94	6.48	6.07	8.14	4.37	7.37
Au (SM)	g/mt	5.73	6.11	7.73	7.67	5.63	5.72	6.73	9.07	7.56	5.80	8.22	6.25	8.18
Ave. Au	g/mt	6.43	5.59	7.90	6.80	4.77	4.65	6.49	9.01	7.02	5.94	8.18	5.31	7.78
Ag (MA/ICP)	g/mt	3.1	2.8	3.8	3.0	3.7	1.9	2.9	4.7	3.0	3.7	3.6	2.9	2.8
Hg	g/mt	0.25	0.74	0.05	0.08	0.1	1.08	0.29	0.04	0.09	0.02	0.01	0.27	0.14
STOTAL	%	1.79	1.8	2.02	0.65	1.69	1.59	2.3	2.05	1.75	2.3	2.21	1.83	2.32
S2-	%	1.42	1.39	1.63	0.33	1.3	1.18	1.75	1.64	1.37	1.81	1.69	1.4	1.86
CTOTAL	%	1.78	2.01	1.48	1.72	2.08	1.65	2.36	1.7	1.17	1.66	1.85	1.84	1.4
Inorg. C	%	1.47	1.69	1.21	1.47	1.73	1.3	2.02	1.43	0.94	1.37	1.52	1.52	1.16
Cu	g/mt	96	108.7	120.1	169.1	167	55	136.7	101.1	50.8	172.5	113.3	108.6	85.4
Pb	g/mt	10.1	9.9	13.2	11.4	12.9	9.4	13.2	13.7	14.5	9.5	8.8	13.7	21
Zn	g/mt	83	79	77	135	112	80	92	85	56	105	90	73	81
Mn	g/mt	1014	958	1114	799	997	674	1017	1287	656	1556	1413	830	1038
Fe	%	4.5	4.49	4.91	3.81	4.72	3.96	4.71	4.53	3.48	6.15	4.79	3.84	4.97
As	g/mt	5	<5	6	<5	<5	<5	<5	7	<5	6	6	<5	9

FA = Fire assay, SM = Screened metallics procedure.

From the first Camflo bulk sample campaign, seven composite samples were also collected, each representing 3 days of production. The Camflo composite sample head assay and assay ranges for individual samples are shown in Table 13.20. The Camflo bulk sample was mined from the C2 zone.

Table 13.20 – Camflo Sample Head Assays

	Unit	Camflo#1 Composite	Camflo samples (range)
Au (FA)	g/mt	5.51	5.05 – 9.84
Au (SM)	g/mt	5.77	4.56 – 6.97
Ave. Au	g/mt	5.64	4.96 – 8.13
Ag (MA/ICP)	g/mt	2.8	2.9 – 4.4
Hg	g/mt	0.04	< 0.01 – 0.07
STOTAL	%	1.32	1.25 – 2.13
S2-	%	0.91	0.88 – 1.85
CTOTAL	%	1.9	1.9 – 1.98
Inorg. C	%	1.6	1.6 – 1.75
Cu	g/mt	129	112 - 134
Pb	g/mt	95	37 - 237
Zn	g/mt	213	204 - 362
Mn	g/mt	1043	991 - 1075
Fe	%	5.1	5.0 – 5.3
As	g/mt	23	<5 - 19

FA = Fire assay, SM = Screened metallics procedure.

13.3.2 Comminution Testwork

Bond Rod and Ball Mill Work Indexes as well as Abrasion Indexes were measured for the C2 and C4 core samples as well as for the Camflo rod mill feed composite. Results are presented in Table 13.21. Further Bond Ball Mill Work Indexes were measured using the coarse assay reject samples. Values measured for these samples are shown in Table 13.22.

The Rod Mill Work Indexes varied between 15.8 and 18.1 kWh/t. Values that had been measured in the SGS tests also fall within this range.

Ball Mill Work Indexes were measured with a closing screen size of 105 µm, resulting in P80 values of 70 to 77 µm. Values measured ranged between 11.2 and 13.5 kWh/t. These are all lower than values previously measured by SGS at P80 of 81 or 46 µm.

Abrasion work indexes ranged from 0.123 to 0.206 g, also lower than values that had been measured by SGS.

Table 13.21 – Rod Mill and Ball Mill Work Indexes

Sample	Rod Mill Work Index (kW·h/t)	Ball Mill Work Index (kW·h/t)	Abrasion Index g
C2 core	15.8	12.7	0.206
C4 core	17.8	13.1	0.170
Camflo composite	18.1	12.7	0.123

Table 13.22 – Ball Mill Work Index Variability Samples

Sample	Ball Mill Work Index (kW·h/t)
Master Composite	12.9
C2 Composite	12.4
C4 Composite	13.1
C1	13.4
C2 Upper	11.5
C2 Mid	13.0
C2 Lower	11.2
C4 Upper	12.8
C4 Mid	13.3
C4 Lower	13.5
C5	13.0

13.3.3 Gravity Recovery Testwork

E-GRG tests were conducted on 20 kg samples of the Master, C2 and C4 composites as well as the Camflo composite. Results are summarized in Table 13.23.

As can be seen, cumulative gold recoveries after 3 passes range between 37 and 56%, with the C2 composite presenting the lowest gold recovery. The fact that the Camflo composite, which also comes from the C2 zone, produced the highest gold recovery, shows that gravity gold content can be quite variable.

Table 13.23 – E-GRG Test Results

Sample	Parameter	Unit	1st pass	2nd pass	3rd pass
Master Composite	Feed P80	µm	512	234	70
	Cumulative weight recovery	% of initial feed	0.54	1.03	1.54
	Cumulative Gold Grade	g/t	271	238	222
	Cumulative Gold Recovery	% of initial feed	20	33	46

Sample	Parameter	Unit	1st pass	2nd pass	3rd pass
C2 Composite	Feed P80	µm	992	216	72
	Cumulative weight recovery	% of initial feed	0.46	0.95	1.44
	Cumulative Gold Grade	g/t	149	134	129
	Cumulative Gold Recovery	% of initial feed	14	25	37
C4 Composite	Feed P80	µm	813	220	71
	Cumulative weight recovery	% of initial feed	0.46	0.90	1.40
	Cumulative Gold Grade	g/t	291	262	245
	Cumulative Gold Recovery	% of initial feed	20	35	51
Camflo Composite	Feed P80	µm	903	235	63
	Cumulative weight recovery	% of initial feed	0.51	1.00	1.47
	Cumulative Gold Grade	g/t	305	260	233
	Cumulative Gold Recovery	% of initial feed	26	43	56

13.3.4 Whole Ore Cyanidation

Numerous whole ore cyanidation tests were conducted on the C2, C4 and Master composites to determine the impact on recovery of various parameters such as grind size, pH, lead nitrate, cyanide concentration and aeration using oxygen or air. The main trends are reported here. Some tests were also conducted on the Camflo composite to compare with results obtained at the Camflo plant.

Based on results obtained in the SGS test campaign, all tests included 4 hours of pre-aeration. All samples were then submitted to a cyanide leach for a given duration (varies depending on test series) followed by CIP with 15 g/L activated carbon for 8-16 hours (depending on tests series).

13.3.4.1 Baseline tests

The first tests were done under the same conditions as the SGS tests, except with CIP added. Leach conditions were as follows:

- P80 of 50 µm
- 1.0 g/L NaCN
- 40% solids pulp density
- pH 10.5 - 11.0

- aeration using oxygen (but DO levels achieved were only between 9.0 – 12.9 mg/L, with most values below 10)
- Leach duration: 56 hours
- CIP duration: 16 hours (total 72 hours)

Tests results are shown in Table 13.24. As can be seen, recovery for the C2 composite is significantly lower than for the C4 composite, with 90.9% compared to 94.2%. In addition, cyanide consumptions are much higher than those measured during the SGS tests with pre-aeration.

Table 13.24 – Baseline Leach Test Results

Sample	Measured Head[1]	Calc. Head	Residue	Leach recovery (%)			CIP rec. (%)	Reagent Consumption (kg/t)	
	g Au/t	g Au/t	g Au/t	24 h	48 h	56 h	72 h	NaCN[2]	CaO[3]
Master composite	6.43	7.26	0.409	89.9	90.6	92.1	94.3	1.31	1.36
C2 composite	5.59	6.46	0.585	84.5	87.3	88.2	90.9	1.41	1.39
C4 composite	7.90	7.57	0.447	92.9	93.2	92.8	94.2	1.38	1.17

1 Average of fire assay and screened metallics procedure.

2 Consumption after 56 hours, prior to carbon addition.

3 Amount added. Residual not subtracted.

13.3.4.2 Effect of Lead Nitrate, pH and Dissolved Oxygen

Recovery obtained in the Camflo mill, where lead nitrate was used and the circuit was operated at high pH (around 11.75) was higher than what was observed in the lab test results. Thus tests were conducted to determine the effect of lead nitrate addition and higher pH.

This series of tests was conducted with a shorter leach duration (32 h + 16 h CIP = 48 hours total), thus not allowing direct comparison with the final recovery from baseline tests. Some tests were conducted with a relatively small oxygen flow, some with air and some with enough oxygen to maintain dissolved oxygen concentrations above 20 mg/L. Grind size and cyanide concentration were kept the same as in the baseline tests. Pulp density was increased to 45% solids and kept at this value for all subsequent tests.

Results are presented in Table 13.25. Addition of 1.5 kg/t lead nitrate and increasing the pH both improved recovery individually but even more so when combined. The recovery improvement was higher (by about 1.4 to 2.0% for the C2 composite and 0.5% for the C4 composite) when the pH was maintained between 11.7 and 12.5 at all times. At this pH level, the impact of lead nitrate on recovery became insignificant for the C4 composite (when comparing residue grades). It however still increased the recovery for the C2 composite. The highest pH level also resulted in the lowest cyanide consumptions, regardless if lead nitrate was added or not.

The impact of high pH and lead nitrate on recovery can possibly be explained by the presence of gold tellurides. Both have been reported in the literature to improve gold telluride leaching rates (J.O. Marsden and C. I House, 2006). Gold tellurides were identified as the main gold species in the cyanidation tailings in the SGS gold deportment study on the cyanidation residue (refer to section **Error! Reference source not found.**). In addition, other gold mines processing gold telluride minerals in the Val d'Or area operate or have been operated at high pH to improve gold telluride dissolution kinetics and perhaps gold recovery.

The impact of higher dissolved oxygen concentrations seems insignificant when comparing the residue grades of the tests with air and those with DO > 20 mg/L, rather than comparing the recovery values which vary with calculated head grades. These are often considerably higher than the measured head grade, resulting in variations in recovery due to this factor. This is particularly true for test C62 with a calculated head grade of 15.81 g/t vs a measured head grade of 7.90 g/t, likely due to the presence of coarse gold. The recovery value from this test cannot be considered for this reason.

Table 13.25 – Effect of Lead Nitrate, pH and Dissolved Oxygen

Sample	Test no.	Lead nitrate	pH	DO	Measured Head[1]	Calc. Head	Residue	Leach recovery (%)		CIP rec. (%)[2]	Reagent Consumption (kg/t)	
		kg/t		mg/L	g Au/t	g Au/t	g Au/t	24 h	32 h	48 h	NaCN[3]	CaO[4]
Master composite	C32	1.5	10.4-11.2	9-15	6.43	7.34	0.485	90.5	91.2	93.5	0.80	0.91
	C33	-	10.7-12.2	12-16		7.37	0.515	88.5	90.2	93.0	0.41	1.73
	C18	1.5	10.8-12.3	10-14		8.85	0.473	90.9	92.9	94.6	0.54	1.73
	C46 (air)	1.5	11.9-12.8	9-10		8.08	0.349	93.4	95.4	95.6	0.27	5.30
C2 composite	C37	1.5	10.3-11.2	11-12	5.59	6.20	0.634	85.5	87.3	89.8	1.00	0.82
	C38	-	10.6-12.2	10-13		6.63	0.656	85.0	87.2	90.0	1.35	1.79
	C22	1.5	10.6-12.2	12-15		6.44	0.462	87.7	89.7	92.7	0.50	1.63
	C50 (air)	1.5	11.8-12.7	9-10		5.95	0.353	92.4	-	94.1	0.26	5.30
	C56	1.5	11.7-12.5	21-23		6.36	0.336	93.3	93.7	94.7	0.24	5.15
	C57	0.5	11.9-12.4	20-24		6.28	0.499	90.5	90.7	91.9	0.23	5.15

Sample	Test no.	Lead nitrate	pH	DO	Measured Head[1]	Calc. Head	Residue	Leach recovery (%)		CIP rec. (%)[2]	Reagent Consumption (kg/t)	
		kg/t		mg/L	g Au/t	g Au/t	g Au/t	24 h	32 h	48 h	NaCN[3]	CaO[4]
	C58	-	11.9-12.4	21-23		6.10	0.482	89.2	90.2	92.0	0.23	5.15
C4 composite	C41	1.5	10.5-11.3	11-14	7.90	7.90	0.438	92.2	93.6	94.4	0.85	0.73
	C42	-	10.9-12.3	14-15		8.48	0.507	92.3	92.9	94.0	0.60	1.82
	C26	1.5	10.6-12.3	10-13		8.40	0.415	93.1	93.8	95.1	0.49	1.76
	C54 (air)	1.5	11.4-12.7	9-10		7.93	0.375	93.5	-	95.3	0.16	5.30
	C62	1.5	11.7-12.5	20-27		15.81	0.383	97.4	97.2	97.4	0.32	5.15
	C63	0.5	11.7-12.5	20-28		8.71	0.380	95.1	94.9	95.6	0.37	5.15
	C64	-	11.8-12.5	20-23		8.93	0.417	95.6	94.7	95.4	0.20	5.15

1 Average of fire assay and screened metallics procedure.

2 Except for tests with air, for which 56 h recovery is reported.

3 Consumption prior to carbon addition.

4 Amount added. Residual not subtracted.

13.3.4.3 Effect of grind size

Effect of grind size was investigated at the highest pH (saturated with lime) with addition of 1.5 kg/t lead nitrate. Again, some tests were conducted with a relatively small oxygen flow, some with air and some with enough oxygen to maintain dissolved oxygen concentrations above 20 mg/L. Cyanide concentration and pulp density were kept same as in the baseline tests.

Results are shown in Table 13.26. As can be seen, a finer grind results in higher recovery, with about a 2% increase when going from 80% passing 75 to 50 µm at the tested retention times. Once again, the impact of the dissolved oxygen level is not conclusive. As mentioned before, the recovery value from test C62 cannot be considered due to the extremely high calculated head grade.

Table 13.26 – Effect of Grind Size and Dissolved Oxygen

Sample	Test no.	Grind size P80	pH	DO	Measured Head1	Calc. Head	Residue	Leach recovery (%)		CIP recovery (%)	Reagent Consumption (kg/t)	
		µm		mg/L	g Au/t	g Au/t	g Au/t	24 h	48 h	56 h	NaCN[2]	CaO4
Master composite	C44	74	12.1-12.6	12-15	6.43	8.59	0.473	88.1	92.7	94.1	0.24	5.30
	C47 (air)	74	12.0-12.8	9-10		8.12	0.505	91.4	93.2	93.6	0.14	5.30
	C45	65	12.1-12.7	12-13		8.95	0.539	90.2	92.9	93.4	0.20	5.45
	C46 (air)	48	11.9-12.8	9-10		8.08	0.349	93.4	95.4	95.6	0.27	5.30
C2 composite	C48	72	12.2-12.7	13	5.59	6.84	0.538	88.3	91.0	91.9	0.20	4.99
	C51 (air)	71	11.5-12.7	9-10		6.41	0.468	89.3	92.0	92.8	0.25	5.30
	C61	74	11.8-12.4	20-25		5.23	0.540	81.4	-	91.5	0.12	4.69
	C49	62	12.2-12.7	13-15		7.02	0.518	91.2	92.5	92.6	0.17	5.45
	C50 (air)	48	11.8-12.7	9-10		5.95	0.353	92.4	94.0	94.1	0.26	5.30
	C56	47	11.7-12.5	21-23		6.36	0.336	93.3	-	94.7	0.24	5.15
C4 composite	C52	78	12.0-12.7	11-17	7.90	7.04	0.489	89.0	92.5	93.0	0.15	5.30
	C55 (air)	78	11.4-12.8	9-10		7.69	0.546	90.2	92.4	92.9	0.16	5.30
	C67	77	11.8-12.5	21-26		7.25	0.563	89.5	-	92.4	0.27	4.84
	C53	63	11.9-12.7	12-14		7.39	0.467	91.3	93.3	93.8	0.12	5.30
	C54 (air)	51	11.4-12.7	9-10		7.93	0.375	93.5	94.8	95.3	0.16	5.30
	C62	51	11.7-12.5	20-27		15.81	0.383	97.4		97.4	0.32	5.15

1 Average of fire assay and screened metallics procedure.

2 Consumption prior to carbon addition.

3 CIP recovery after 48 hours.

4 Amount added. Residual not subtracted.

13.3.4.4 Effect of Cyanide Concentration

Tests to determine the impact of cyanide concentration were conducted on the Master composite. Tests were conducted at 80% passing 47-48 µm grind size, at high pH (11.0 – 12.3), with 1.5 kg/t lead nitrate addition and at high DO levels (> 20 mg/L). Pulp density remained at 45% solids as in the baseline tests.

Results are presented in Table 13.27. As can be seen, pH was not maintained quite as high as in the previous tests, which could have affected the results. Final recovery at 1.0 g/L NaCN was about

1.0% lower than in the previous test conducted with air (C46) at a higher pH. Residues are all above 0.4 g/t while at the very high pH levels, values below 0.4 g/t had been achieved.

Lower cyanide concentrations resulted in slower kinetics and reduced cyanide consumption. The effect on final recovery is however not clear from the tests.

Table 13.27 – Effect of Cyanide Concentration

Sample	Test no.	NaCN concentration	pH	Measured Head[1]	Calc. Head	Residue	Leach recovery (%)		CIP recovery (%)	Reagent Consumption (kg/t)	
		g/L		g Au/t	g Au/t	g Au/t	24 h	32 h	48 h	NaCN[2]	CaO[3]
Master composite	C68	1.00	11.3-12.2	6.43	8.48	0.457	94.3	94.1	94.6	0.25	3.86
	C69	0.75	11.0-12.2		7.18	0.435	92.6	93.7	93.8	0.22	4.68
	C70	0.50	11.2-12.2		8.03	0.425	92.1	93.8	94.5	0.11	3.94
	C71	0.25	11.2-12.2		7.28	0.446	91.4	93.5	93.7	0.06	4.92

1 Average of fire assay and screened metallics procedure.

2 Consumption after 32 hours, prior to carbon addition.

3 Amount added. Residual not subtracted.

13.3.4.5 Variability Tests

Two series of variability tests were conducted, one with oxygen keeping the dissolved oxygen concentration above 20 mg/L and the other with air. Both series of tests were conducted under the following conditions:

- 80% passing 50 µm grind size
- 1.0 g/L NaCN concentration
- 45% solids pulp density
- 1.5 kg/t lead nitrate
- 4 hours pre-aeration retention time
- 32 hours leach retention time
- 16 hours CIP retention time (total 48 hours)
- 15 g/L activated carbon concentration in CIP.

Although the intent was to conduct the tests at a pH ≥ 11.7, lime consumption resulted in pH dropping to lower values during the tests. In the test series with oxygen which was done first, pH values for the C1, C2, C4 and C5 ore samples dropped to around 10.6-10.8 after 24 hours. In the test series with air, pH was originally adjusted to higher values in an attempt to maintain the pH ≥ 11.7, but nonetheless dropped to around 11.2-11.4 after 24 hours.

Test results for C1, C2, C4 and C5 ore samples are presented in Table 13.28. Test results for the 3-day Camflo rod mill feed composites are presented in Table 13.29. As can be seen from the results for the C1, C2, C4 and C5 ore samples, the higher pH in the test series with air resulted in systematic higher recoveries and lower cyanide consumptions, showing that the pH has a much higher impact than the dissolved oxygen concentration. Based on comparison with previous test results with pH ≥ 11.7, it is believed that even higher recoveries could have been achieved if the pH had been maintained above this value throughout the leach.

In general, however no particular variability sample stood out with results considerably different from the composite ore samples. C2 Upper performed a little better than C2 Mid and C2 Lower. C1 showed very good recoveries but this is probably at least partially due to the high calculated head grades. Nonetheless, C1 residue grades were lower than for all other samples. C5 recoveries were similar to those of the C2 zone.

Table 13.28 – C1, C2, C4 and C5 Ore Samples Variability Test Results

Sample	O2/air	pH	Measured Head[1]	Calc. Head	Residue	Recovery	Reagent Consumption (kg/t)	
			g Au/t	g Au/t	g Au/t	%	NaCN[2]	CaO[3]
C1	O2	10.6-12.3	6.80	9.94	0.207	97.9	0.43	2.91
	Air	11.2-12.9		10.90	0.209	98.1	0.29	3.66
C2 Upper	O2	10.6-12.2	4.77	6.55	0.437	93.1	0.36	2.91
	Air	11.3-12.7		6.44	0.370	94.0	0.28	3.51
C2 Mid	O2	10.6-12.3	4.65	5.92	0.563	90.4	0.41	2.63
	Air	11.2-12.8		6.71	0.420	93.3	0.28	3.24
C2 Lower	O2	10.6-12.2	6.49	6.56	0.595	90.5	0.49	2.77
	Air	11.3-12.8		7.32	0.506	92.7	0.27	3.38
C4 Upper	O2	10.6-12.3	9.01	9.30	0.609	93.4	0.39	2.92
	Air	11.2-12.8		9.78	0.513	94.8	0.23	3.68
C4 Mid	O2	10.6-12.3	7.02	7.68	0.332	95.6	0.39	2.88
	Air	11.4-12.8		9.89	0.458	95.0	0.24	3.48
C4 Lower	O2	10.8-12.3	5.94	7.36	0.433	93.9	0.36	2.97
	Air	11.4-12.7		7.15	0.377	94.6	0.22	3.57
C5	O2	10.7-12.3	8.18	9.17	0.729	91.9	0.39	3.03
	Air	11.3-12.8		9.46	0.633	93.1	0.22	3.48
C2 core	O2	10.6-12.3	5.31	6.07	0.527	91.2	0.56	2.75
	Air	11.0-12.8		6.03	0.356	93.9	0.22	3.36
C4 core	O2	10.7-12.3	7.78	7.28	0.375	94.8	0.47	2.91
	Air	11.4-12.9		8.61	0.301	96.5	0.16	3.27

1 Average of fire assay and screened metallics procedure.

2 Consumption after 32 hours, prior to carbon addition.

3 Amount added. Residual not subtracted.

When comparing test results for the 3-day Camflo rod mill feed composites to those for the C2 ore samples, it can be seen that for the Camflo composites:

- Except for the last 3-day composite, pH dropped less despite initial adjustments to the same values;

- Residue grades were generally lower, with many composite residue grades below 0.3 g/t;
- Recoveries were generally higher, especially for the 2nd to the 5th 3-day composites;
- Cyanide consumptions were lower;
- High dissolved oxygen concentration seems to have had a beneficial effect for the last two 3-day composites.

It thus seems that the ore processed during the first Camflo bulk sample campaign is not representative of the C2 zone as a whole.

Table 13.29 – 3-Day Camflo Composite Samples Variability Test Results

Sample	O_2/air	pH	Measured Head[1]	Calc. Head	Residue	Recovery	Reagent Consumption (kg/t)	
			g Au/t	g Au/t	g Au/t	%	NaCN[2]	CaO[3]
Sept. 27/28/29	O_2	11.3-12.2	5.58	6.22	0.455	92.8	0.23	4.45
	Air	12.0-12.9		5.72	0.224	96.0	0.17	4.45
Sept. 30/ Oct. 1/2	O_2	11.1-12.2	6.73	7.75	0.296	96.2	0.26	4.30
	Air	11.9-12.9		7.94	0.219	97.2	0.14	4.39
Oct. 3/4/5	O_2	11.0-12.2	5.31	6.15	0.223	96.5	0.25	4.09
	Air	11.9-12.9		6.41	0.216	96.6	0.17	4.18
Oct. 6/7/8	O_2	11.0-12.2	5.20	6.73	0.305	95.5	0.23	3.57
	Air	11.4-12.9		6.50	0.216	96.6	0.16	3.69
Oct. 8/10/11	O_2	11.0-12.1	6.03	6.44	0.265	95.8	0.27	4.09
	Air	11.9-12.9		6.30	0.192	97.1	0.18	4.09
Oct. 12/113/14	O_2	10.8-12.2	8.13	7.07	0.315	95.6	0.34	4.09
	Air	11.9-12.9		7.16	0.406	94.3	0.18	4.09
Oct. 15/16/17	O_2	10.6-12.2	4.96	6.73	0.312	95.3	0.51	2.66
	Air	11.1-12.9		7.07	0.363	94.7	0.15	3.27

1 Average of fire assay and screened metallics procedure.

2 Consumption after 32 hours, prior to carbon addition.

3 Amount added. Residual not subtracted.

13.3.5 Gold Loading Isotherm

A gold loading isotherm was established using a pregnant leach solution (PLS) sample from a bulk leach of the Master Composite. The initial PLS solution was at pH 11.1, contained 0.95 g/L NaCN, 5.75 mg/L gold, 2.54 mg/L silver and 5.2 mg/L copper. Pre-attritioned and washed Norit GCN 612G activated carbon was used. The samples were allowed to come to equilibrium for 72 hours. The resulting equilibrium curve is presented in Figure 13.1.

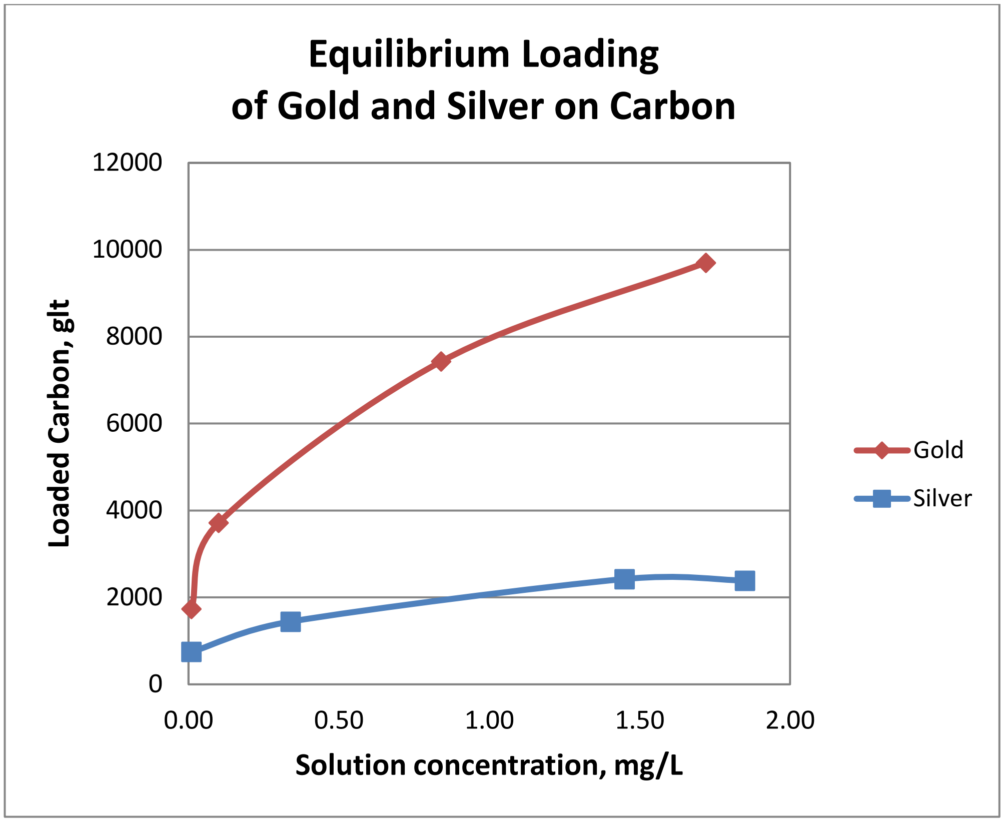


Figure 13.1 Equilibrium Loading of Gold and Silver on Carbon

13.3.6 Cyanide Destruction Testwork

Continuous cyanide destruction tests were conducted in two 1.5 L reactors in series each with 60 minutes retention time. The tests were operated for 6 hours. Air feed to each reactor was 1.5 L/min. Based on the amount of copper already in the solution, various amounts of copper sulphate was added to reach the predetermined test copper concentration. Sodium metabisulfite ($Na_2S_2O_5$) was used as SO_2 source.

Test results are presented in Table 13.30. At a SO_2/CN_{WAD} ratio of 4.5 and a pulp density of 45% solids, residual CN_{WAD} concentrations below 1.0 mg/L were achieved in the first reactor at both pH 8.0 and 9.0. At a pulp density of 55% solids, the concentration out of the first reactor slightly exceeded 1.0 mg/L CN_{WAD}.

Table 13.30 – Cyanide Destruction Test Results

Sample	Test no.	Feed Slurry				Cyanide Destruction Conditions			CNWAD in Product Slurry[1]		Reagent Consumption		
		Grind size P80	Pulp density	pH	CNWAD	SO_2/CN_{WAD}	Cu	pH	R1	R2	$Na_2S_2O_5$	CaO	Cu
		µm	% solid	-	mg/L	g/g	mg/L	-	mg/L	mg/L	g/g CNWAD		kg/t
Master composite	MC-CD-1	50	45	11.4	288	3.00	20	8	3.93	3.25	4.5	0.38	0.018
	MC-CD-2	50	45	11.4	288	4.50	40	8	0.29	0.28	6.7	1.44	0.043
	MC-CD-3	50	45	11.4	288	3.75	30	8	1.35	0.79	5.6	1.14	0.031
C2 composite	C2-CD-1	50	45	10.3	244	4.50	40	8	0.70	0.54	6.7	1.51	0.040
C4 composite	C4-CD-1	50	45	10.3	243	4.50	40	8	0.74	0.54	6.7	1.44	0.041
Mixed: Master, C2, C4 and Camflo composites	BT-CD-1	Mixed	45	10.6	154	4.50	40	8	0.57	0.41	6.7	0.45	0.036
	BT-CD-2	Mixed	55	10.6	140	4.50	40	8	1.06	0.89	6.7	0.61	0.027
	BT-CD-3	Mixed	45	10.6	244	3.75	40	8	1.35	0.89	5.6	0.76	0.036
	BT-CD-4	Mixed	45	10.6	244	4.50	40	9	0.58	0.48	6.7	2.50	0.036

1 Average of last three hours of operation (R1 = reactor 1, R2 = reactor 2).

13.3.7 Thickening, Rheology and Filtration Testwork

Pocock Industrial, which is based in Salt Lake City, Utah, conducted solid-liquid separation tests at Bureau Veritas' laboratory on 3 samples from the Master composite: Leach feed, CIP tailings and cyanide destruction tailings, all at a P80 grind size of 50 µm. Samples characteristics are shown in Table 3.31. Settling, rheology and filtration tests were completed.

Table 13.31 – SLS Testing Sample Characteristics

Sample	Liquor S.G.	Solids S.G.	pH
50 µm Leach feed	1.00	2.83	11.7
50 µm CIP tailings	1.00	2.83	11.8
50 µm Detoxed tailings	1.00	2.83	9.3

13.3.7.1 Thickening Tests

Flocculant screening tests were first conducted. Based on these, flocculant SNF AN910SH, a medium to high molecular weight, 15% charge density anionic polyacrylamide, was selected. Based on the screening tests, flocculant dosage was also determined. A minimum dosage of 10-15 g/t was determined for the leach feed and CIP tailings and a minimum dosage of 15-20 g/t for the detoxed tailings.

Static thickening tests were conducted in two liter graduated cylinders equipped with slow turning picket rake mechanisms. Tests were done on the samples at various flocculant dosages and feed solids concentrations.

Dynamic high rate thickening tests used a continuous bench scale unit equipped with a motorized rake mechanism and pumps for continuous slurry and flocculant feed and withdrawal of underflow solids as required to maintain a constant pulp level.

For both static and dynamic tests, samples were diluted to a selected feed solids concentration using decant solution or simulated process solution. Recommended thickening design parameters for both conventional and high rate thickeners (from static and dynamic tests) are summarized in Table 13.32.

Table 13.32 – Recommended Thickening Design Parameters

Sample	Floc. Dosage	Floc. Conc.	Max Feed Solids Conc.	Minimum Unit Area for Conventional Thickener Sizing[1]	Hydraulic Rate for High Rate Thickener Sizing	Estimated Underflow Density
	g/t	g/L	%	m^2/mtpd	$m^3/m^2\cdot h$	%
Leach feed	15 - 20	0.1 – 0.2	Conv. type: 15 - 25 High rate: 16 - 21	0.174	3.48	Conv. type: 61 - 65 High rate: 63 - 67
CIP tailings	20 - 25	0.1 – 0.2	Conv. type: 15 - 25 High rate: 16 - 21	0.172	3.68	Conv. type: 61 - 65 High rate: 63 - 67
Detoxed tailings	20 - 25	0.1 – 0.2	Conv. type: 15 - 25 High rate: 16 - 21	0.175	3.22	Conv. type: 61 - 65 High rate: 63 - 67

1 Includes a 1.25 scale up factor.

13.3.7.2 Rheology Tests

Pulp rheology of the non-Newtonian thickener underflow pulps was measured using a Fann Model 35A true coaxial cylindrical rotational viscometer. Underflow slurries were pre-sheared (i.e. the flocculant structure destroyed) using a laboratory mixer prior to testing.

A summary of the apparent viscosities at reference shear rates and varying percent solids is presented in Table 13.33. The thickener underflow slurries exhibit a shear thinning behavior with apparent viscosity decreasing as shear increases.

Yield stresses required to initiate flow are also indicated in the table. Underflow slurries with yield stress values in excess of 30 N/m^2 (Pa) are normally beyond the capabilities of conventional thickening and pumping systems. In addition, the shape of the yield stress versus solids concentration curve must also be considered. Design density should be selected such as to avoid the exponential region of the curve as the material could quickly become solidified beyond pumping capability with only a slight increase in solids concentration.

13.4 EXPECTED RECOVERY

The expected recovery was estimated separately for zones C2 and C4, due to the different recoveries obtained during the metallurgical testwork for each of these two zones. In both cases, the average of the four best leach tests was used. Values from the variability tests were not used as it is expected that higher recoveries would have been obtained if the pH had been better controlled to maintain a value above 11.7 throughout the tests.

All the tests were for an 80% passing 50 µm grind, with 48-56 hours leach + CIP retention time, even though actual plant retention time is longer. All recoveries except one were obtained at pH values above 11.7 throughout the tests. Individual test recoveries and the average used as basis for the financial model are indicated in Tables 13.33 and 13.34 for zones C2 and C4 respectively.

Table 13.33 – C2 Zone Expected Recovery

Test no.	Lead nitrate[1]	NaCN concentration	pH	DO	Retention time	Recovery
	kg/t	g/L		mg/L	h	%
C56	1.5	1.0	11.7-12.5	21-23	48	94.7
C50 (air)		1.0	11.8-12.7	9-10	56	94.1
C60		0.5	11.9-12.5	21-27	48	93.4
C591		1.0	11.9-12.4	21-23	48	92.9
AVERAGE						93.8

1 Lead nitrate added after pre-aeration rather than in grinding mill.

Table 13.34 – C4 Zone Expected Recovery

Test no.	Lead nitrate	NaCN concentration	pH	DO	Retention time	Recovery
	kg/t	g/L		mg/L	h	%
C63	0.5	1.0	11.7-12.5	20-28	48	95.6
C64	-	1.0	11.8-12.5	20-23	48	95.4
C54 (air)	1.5	1.0	12.0-12.7	9-10	56	95.3
C26	1.5	1.0	10.6-12.2	10-13	48	95.1
AVERAGE						95.3

As mentioned, the ore mined in 2018 prior to the Sigma mill start-up will be processed via toll milling agreements at the Camflo and Westwood mills. This ore will come from the C2 zone.

Both mills operate at high pH and are capable of grinding the ore to an 80% passing 50 µm grind. The Camflo mill uses lead nitrate, however this reagent would not be added at the Westwood mill. Based on this, recovery for the Camflo mill is considered to be equivalent to the one achievable at the Sigma mill (93.8%) however recovery for processing at the Westwood mill was estimated at 92% based on test C58 at high pH without lead nitrate.

Table 13.33 – Thickener Underflow Apparent Viscosities and Yield Stress

Sample	Solids Conc.	Coefficient of Rigidity	Yield Stress	Apparent Viscosity (Pa·s)								
				5	25	50	100	200	400	600	800	1000
	%	Pa·s	N/m2	s-1	s-1	s-1	s-1	s-1	s-1	s-1	s-1	s-1
Leach Feed	69.3	0.178	55.1	7.334	2.685	1.742	1.130	0.733	0.475	0.369	0.308	0.268
	67.6	0.139	44.0	5.759	2.157	1.413	0.925	0.606	0.397	0.310	0.260	0.227
	65.4	0.085	31.8	2.935	1.364	0.981	0.705	0.507	0.365	0.301	0.262	0.236
	62.5	0.045	19.7	1.883	0.832	0.586	0.412	0.290	0.204	0.166	0.144	0.128
	58.0	0.024	9.4	0.954	0.422	0.297	0.209	0.147	0.103	0.084	0.073	0.065
	49.1	0.018	4.1	0.516	0.232	0.165	0.117	0.083	0.059	0.048	0.042	0.037
CIP Tailings	68.8	0.184	56.6	7.389	2.715	1.764	1.146	0.745	0.484	0.376	0.314	0.274
	67.9	0.152	48.0	5.997	2.295	1.518	1.004	0.664	0.439	0.345	0.290	0.254
	66.0	0.086	34.9	4.330	1.566	1.011	0.652	0.421	0.272	0.210	0.175	0.152
	63.1	0.074	19.5	2.960	1.037	0.660	0.420	0.268	0.170	0.131	0.108	0.094
	57.0	0.019	6.8	0.875	0.321	0.209	0.136	0.088	0.057	0.044	0.037	0.032
	49.1	0.006	2.1	0.287	0.100	0.063	0.040	0.025	0.016	0.012	0.010	0.009
Detoxed Tailings	68.7	0.209	57.4	6.062	2.743	1.949	1.385	0.984	0.699	0.573	0.497	0.445
	67.0	0.129	42.5	4.459	1.976	1.392	0.981	0.691	0.487	0.396	0.343	0.306
	65.4	0.092	30.4	3.255	1.443	1.017	0.716	0.504	0.355	0.290	0.250	0.224
	62.5	0.048	15.5	1.911	0.751	0.502	0.336	0.224	0.150	0.119	0.100	0.088
	58.7	0.030	7.1	1.132	0.409	0.263	0.170	0.110	0.071	0.055	0.046	0.040
	49.9	0.012	2.4	0.414	0.149	0.096	0.062	0.040	0.026	0.020	0.016	0.014

13.4.1.1 Vacuum Filtration Tests

Vacuum filtration tests were conducted using a filter leaf supported vertically on a vacuum flask. The filter cloth used was a National Filter Media (NFM) 8-10 cfm/ft^2 multifilament polypropylene cloth. All tests were conducted at an applied vacuum of 67.7 kPa. No flocculant was added as filtration aid.

Two samples were tested: unthickened detoxed tailings and thickened detoxed tailings. Vacuum filtration tests were performed to examine the effect of cake thickness and air-dry duration on production rate and filter cake moisture. Results are summarized in Table 13.34.

Table 13.34 – Summary of Vacuum Filtration Test Results

Sample	Feed Solids Conc.	Cake Thick.	Filter Cake Moisture	Bulk Cake Density	Production Rate[1]
	%	mm	%	dry kg/m^3	dry kg/m^2·h
Unthickened detoxed tailings	42.2	10	19.6%	1412	570
		15	20.4%		495
Thickened detoxed tailings	64.3	10	17.6%	1523	1248
		15	18.5%		1582

1 Production rate includes a 0.8 scale up factor.

13.4.1.2 Pressure Filtration Tests

Pressure filtration tests were conducted in a lab scale pressure filtration device consisting of a 250 mm section of a 50 mm pipe with drainage grid supporting the filter media in the lower flange. National Filter Media (NFM) 8-10 cfm/ft^2 multifilament polypropylene cloth was used. Pressure applied during the tests was 550 kPa (80 PSI).

Tests were conducted on the two same samples as for vacuum filtration. Tests were performed to examine the effect of cake thickness, air blow duration and air blow on sizing requirements and filter cake moisture. Results are summarized in Table 13.35.

Table 13.35 – Summary of Pressure Filtration Test Results

Sample	Feed Solids Conc.	Half Cake Thick.	Filter Cake Moisture	Bulk Cake Density	Bulk Cake Density	Sizing Basis[1]	Air Blow Time	Total Filter Cycle Time
	%	mm	%	dry kg/m^3	wet kg/m^3	m^3/dry tonne	min	min
Unthickened detoxed tailings	42.3	15	14.5%	1231	1440	1.016	3.0	16.0
Thickened detoxed tailings	64.3	15	9.8%	1608	1783	0.777	3.0	16.0

1 Includes a 1.25 scale up factor.

SECTION • 14 MINERAL RESOURCE ESTIMATE

14.1 INTRODUCTION

The Mineral Resource estimate for the Lamaque deposits (Triangle, Parallel and Plug #4) used data predominantly from surface diamond drillholes. The resource estimates were made from 3D block models created by utilizing commercial geological modelling and mine planning software. The block model cell size is 5 m east by 5 m north by 5 m high.

14.2 MINERALIZATION DOMAINS

Gold mineralization occurs within the moderately to steeply dipping main shear zones and associated more moderately dipping splay zones. The interpretation of all mineralization zones is underpinned by a geological review of structure, alteration and veining carried out by site geologists. The geological elements defining mineralization were captured in a separate composite field which defined and labeled each mineralized zone. The 3D mineralized domains were created using the vein modeling module in Seequent’s Leapfrog Geo software from an interval selection largely based on the composite field. The selection was locally changed to ensure spatial coherence and continuity in 3D. Two sets of solids were produced: 1) mineralization solids, which connect all similar intervals defined by the composite field, irrespective of grade; these solids track the geological elements supporting the mineralization, and 2) resource solids, which are based on those created in step #1 but restrict/clip zones laterally by removing material below a resource cut-off grade of ~2.5 g/t Au.

At Triangle, six main shear zones were modelled: C1, C2, C3, C4, C5 and C7 (Figure 14.1). Of these, C2 and C4 contain the bulk of the gold mineralization. Except for C7, each of the main shear zones has associated mineralized splay zones (Figure 14.2) with the splays related to C1, C3 and C4 being the most significant with respect to gold mineralization.

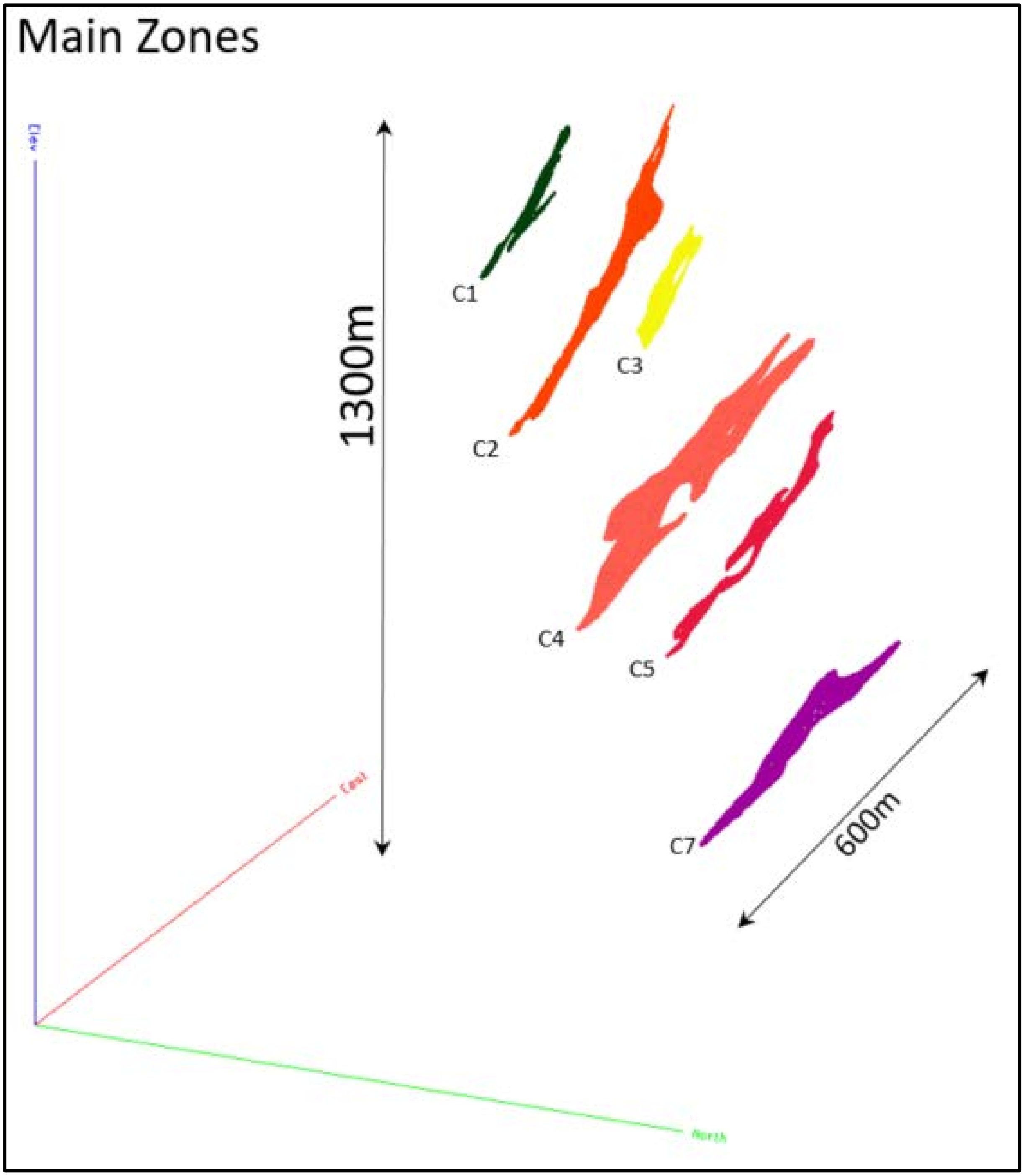


Figure 14.1 – 3D view of the modelled resource solids associated with the main shear zones at Triangle

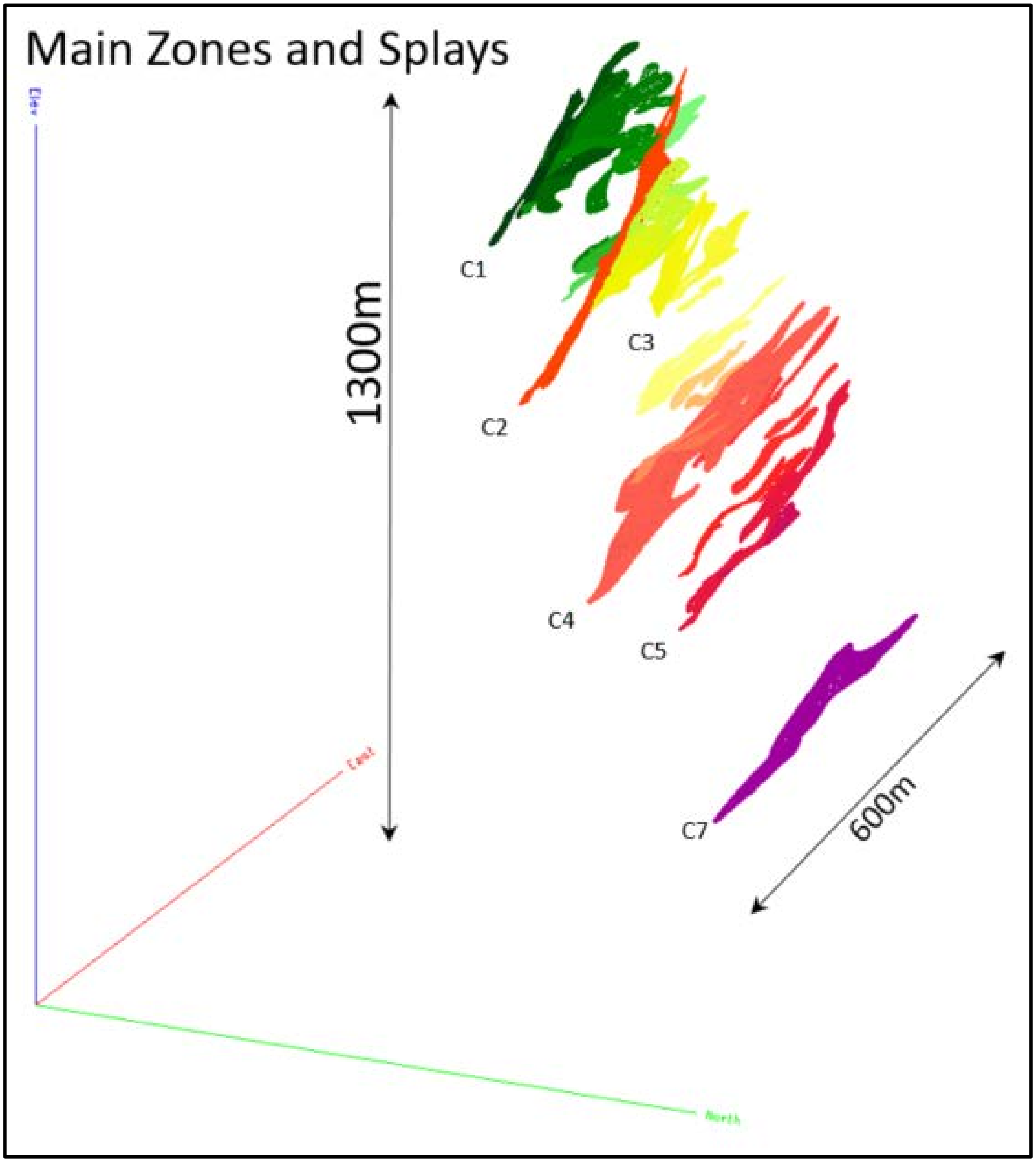


Figure 14.2 – 3D view of the modelled Splay zones associated with the Main shear zones at Triangle

The most significant mineralization at Parallel is found in moderately-dipping, hybrid shear/extensional zones to the footwall of a non-mineralized higher-order steep structure. Minor mineralization associated with horizontal extensional veining is present in the hangingwall of said steep structure (Figure 14.3).

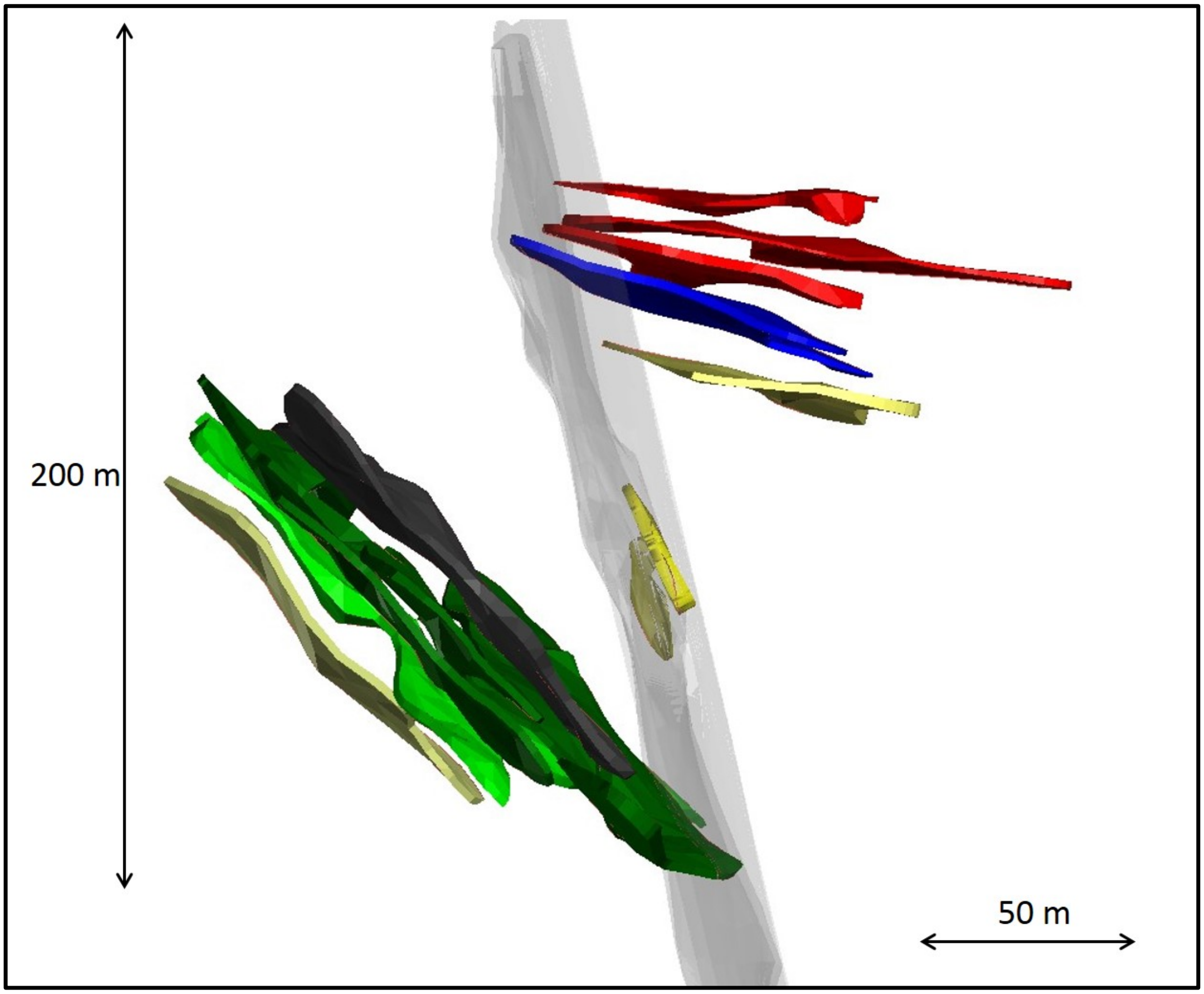


Figure 14.3 – 3D view of the modelled zones at Parallel, looking east.

At Plug 4, there is a distinct set of stacked steep shear zones having similar orientations to those at Triangle, but that are restricted to the core of the Plug 4 intrusive. A feature that is unique to Plug 4 is the occurrence of sub-horizontal vein arrays associated with these steep mineralized shears. The vein arrays are stronger in the footwall side of the main steep zones or between steep zones occurring close together. The flat zones are captured in a single set of solids representing spatially coherent zones above 0.5 g/t Au (Figure 14.4).

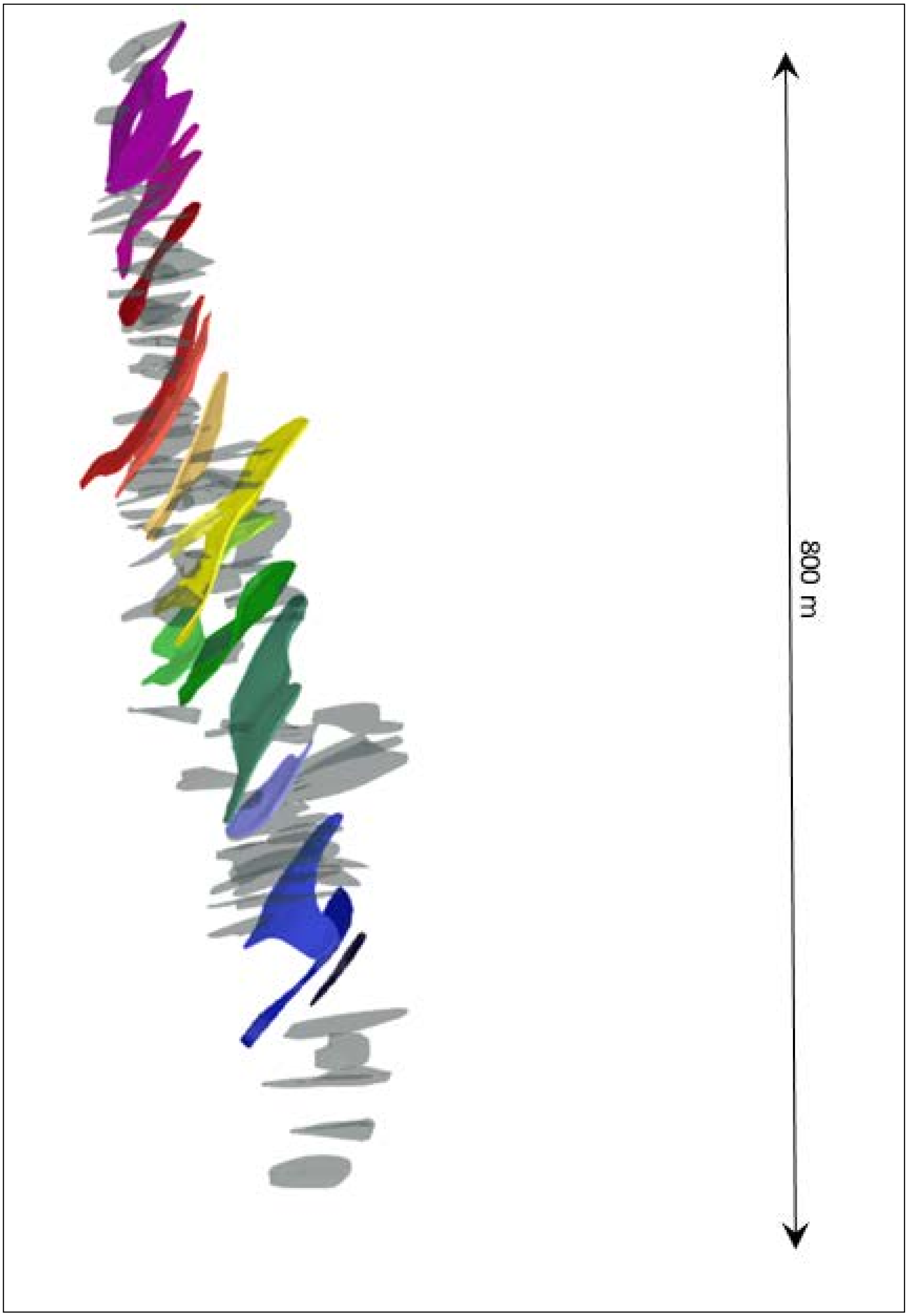


Figure 14.4 – 3D view of the modelled zones at Plug #4. The coloured shapes represent the steep mineralized shapes whereas the gray coloured shapes denote the modelled sub-horizontal vein arrays.

14.3 DATA ANALYSIS

The mineralized domains were reviewed to determine appropriate estimation or grade interpolation parameters. Several different procedures were applied to the data. Descriptive statistics, histograms and cumulative probability plots and box plots have been completed for composite data. The results were used to guide the construction of the block model and the development of estimation plans including treatment of extreme grades. These analyses were conducted on 1-metre composites of the assay data. The statistical properties from this analysis are summarized for both uncapped and capped data in Tables 14.1 and 14.2 for the main Triangle deposit. Tables 14.3 and 14.4 display the same analysis for data from the Parallel and Plug #4 deposits.

Gold grades in the Triangle deposit are highest in C4 and C5 shear zones, followed by C2 and the C4 and C5 Splay veins. Coefficients of variation (CVs) for uncapped data are highest in the upper Triangle shears of C1 through C3. Variation due to extreme grades are notably less in veins and shears associated with C4 and deeper zones at Triangle.

Parallel is a small high grade deposit akin to the deeper Triangle shears in gold distribution. Plug #4 is a distinctly lower grade deposit displaying a highly variable gold distribution.

Table 14.1 – Triangle Deposit composite statistics for 1 m Uncapped Composite Au (g/t) Data

Domain	Number	Min	Max	Mean	Q25	Q50	Q75	SD	CV
C1	300	0.00	390	7.18	0.22	1.28	4.99	27.92	3.89
C1 Splays	904	0.01	87	4.50	0.52	2.13	5.47	8.18	1.82
C2	1612	0.01	1089	9.77	1.42	3.98	9.81	33.90	3.47
C2 Splays	402	0.00	229	4.73	0.29	1.16	3.42	16.77	3.55
C3	131	0.00	97	4.00	0.03	0.29	2.50	12.52	3.13
C3 Splays	389	0.01	142	6.11	0.61	2.04	6.03	13.58	2.22
C4	725	0.01	204	11.72	2.48	5.75	11.98	20.86	1.78
C4 Splays	408	0.01	248	8.20	1.29	3.81	8.10	19.00	2.32
C5	181	0.01	328	13.44	2.41	6.64	11.87	31.14	2.32
C5 Splays	20	0.02	45	7.74	1.35	3.47	11.89	11.28	1.46
C7	45	0.05	80	6.59	1.12	3.37	8.00	12.76	1.94

Note: Min = minimum value; Max = maximum value; Mean = average value; Q25 = value at the 25th frequency percentile of the data; Q50 = value at the 50th frequency percentile of the data, i.e. the median; Q75 = value at the 75th frequency percentile of the data; SD = standard deviation of the data; CV = Coefficient of Variation of the data and equals SD / Mean.

Table 14.2 – Triangle Deposit composite statistics for 1 m Capped Composite Au (g/t) Data

Domain	Number	Min	Max	Mean	Q25	Q50	Q75	SD	CV
C1	300	0.00	80	5.09	0.21	1.28	4.99	10.66	2.10
C1 Splays	904	0.01	80	4.49	0.52	2.13	5.47	8.10	1.80
C2	1612	0.01	80	7.96	1.41	3.98	9.78	11.52	1.45
C2 Splays	402	0.00	80	4.01	0.29	1.16	3.42	9.45	2.36
C3	131	0.00	67	3.48	0.03	0.29	2.51	9.85	2.83
C3 Splays	389	0.01	80	5.48	0.60	2.04	6.04	9.93	1.81
C4	725	0.01	80	10.32	2.48	5.75	11.98	13.60	1.32
C4 Splays	408	0.01	80	7.02	1.28	3.80	8.11	10.71	1.53
C5	181	0.01	80	10.46	2.40	6.64	11.87	13.56	1.30
C5 Splays	20	0.02	45	7.74	1.35	3.47	11.90	11.28	1.46
C7	45	0.05	69	6.35	1.12	3.37	8.01	11.37	1.79

Note: Min = minimum value; Max = maximum value; Mean = average value; Q25 = value at the 25th frequency percentile of the data; Q50 = value at the 50th frequency percentile of the data, i.e. the median; Q75 = value at the 75th frequency percentile of the data; SD = standard deviation of the data; CV = Coefficient of Variation of the data and equals SD / Mean.

Table 14.3: Parallel Deposit composite statistics for 1 m Uncapped and Capped Composite Au (g/t) Data

Zone	Number	Min	Max	Mean	Q25	Q50	Q75	SD	CV
Uncapped Data									
Main	321	0.01	152	10.97	1.18	4.43	11.17	20.20	1.84
Splays	129	0.01	226	10.72	0.93	4.00	11.95	23.92	2.23
Capped Data									
Main	321	0.01	60	9.06	1.18	4.43	11.17	12.44	1.37
Splays	129	0.01	60	8.25	0.93	4.00	11.36	10.68	1.29

Table 14.4 – Plug #4 Deposit composite statistics for 1 m Uncapped and Capped Composite Au (g/t) Data

Zone	Number	Min	Max	Mean	Q25	Q50	Q75	SD	CV
Uncapped Data									
Main	1052	0.00	613	6.37	0.71	2.21	5.46	23.19	3.64
Splays	1813	0.00	314	3.78	0.27	1.05	2.98	14.75	3.91
Capped Data									
Main	1052	0.00	60	5.03	0.70	2.21	5.47	8.33	1.66
Splays	1814	0.00	60	3.07	0.26	1.05	2.98	6.09	1.99

14.4 EVALUATION OF EXTREME GRADES

The shear zones and associated splays at Triangle, Parallel and Plug #4 display effects due to extreme gold grades. As such, the data shows high Coefficient of Variation (CV) values, especially in the upper Triangle zones and in Plug #4. A strategy of capping extreme assay values to limit the risk associated with extreme grades was pursued. For purposes of capping, the probability of achieving or exceeding the predicted annual grade in the production schedule as a measure of risk was used. An 80 percent level of risk was chosen as acceptable. The 80 percent figure is the probability of achieving or exceeding the predicted annual contribution from the high grades. In other words, the actual contribution from the high grade should meet or exceed the prediction in 4 out of 5 years.

The procedure adopted establishes a capping grade through Monte Carlo simulation. It is assumed that mining will encounter the high grades in a more or less random or independent way. Therefore, the total number of high-grade samples likely to be encountered during the mining in any year is dependent on the mining rate and how frequently higher grades occur. The sample grades are then subdivided into low- and high-grade populations at an arbitrary value.

The 20th percentile of the distribution is selected as the risk-adjusted high-grade metal contribution. The analysis for the Triangle mineralized zones showed that a capping strategy should aim to remove about 12 percent of the gold metal in the estimate. This was achieved by implementing an 80 g/t Au cap to the assay data prior to compositing. For the Parallel data, a similar reduction was met by applying a 60 g/t Au cap grade. Plug #4 extreme grade analysis indicated that 20 percent of the gold metal needed to be removed which was achieved by using a 60 g/t Au cap grade on the assay data.

The number of capped Triangle samples was 88, 66 in the main shear zones and 22 in the splay zones. C2 and C4 had the most assays capped at 28 and 18, respectively. Parallel saw 20 samples capped whereas Plug #4 had 49 samples capped. The resultant distributions, shown in the capped composite data above, display notably reduced CV values (e.g. 1.45 and 1.32 for the larger zones of C2 and C4, respectively).

14.5 VARIOGRAPHY

Variography, a continuation of data analysis, is the study of the spatial variability of an attribute. Eldorado prefers to use a correlogram, rather than the traditional variogram, because it is less sensitive to outliers and is normalized to the variance of data used for a given lag. Correlograms were calculated for gold inside the more largely populated zones of C2, C3, C4 and C5 at Triangle. Correlogram model parameters are shown in Table 14.5.

Gold inside the shears display high nugget small ranged structures, typical of deposits of this type. The smaller nugget in C2 may be the effect of more closely spaced data from the underground infill drilling (~10 m by 10 m spacing) that was started in this zone in 2017.

Table 14.5 – Correlograms parameters for Triangle Main Zones

Domain	C_0	C_1	C_2	Range 1 (Z)	Range 2 (X)	Range 3 (Y)	Rot1 (Z)	Rot2 (X)	Rot3 (Y)
C2	.35	.61	.04	14	3	8	16	12	74
				128	20	28	122	-28	-4
C3	.65	.03	.32	15	160	28	9	-10	-57
				91	2	29	-127	-15	-44
C4	.75	.19	.07	3	19	37	-58	89	103
				304	20	129	-72	34	-28
C5	.80	.08	.12	9	16	22	25	-27	-51
				80	6	500	-90	0	16

Models are spherical. The first rotation is about Z, left hand rule is positive; the second rotation is about X', right hand rule is positive; the third rotation is about Y'', left hand rule is positive.

14.6 BULK DENSITY

A constant bulk density of 2.8 t/m^3 was used for the whole deposit. This is based on measurements from earlier work and extensive experience in the Val d'Or camp with similar deposits.

14.7 MODEL SET-UP

Eldorado carried out the grade estimation using MineSight mining software. The block size for the Lamaque models was in part selected based on mining selectivity considerations (underground mining). The assays were composited into 1 m fixed length downhole composites, honoring the individual modelled main vein or splay vein 3-D shape. Intervals of less than 0.5 m lengths were merged into the preceding composite. A second set of composites, composited over the full thickness of the zone if less than 5 m or limited to 5 m fixed intervals in wider areas, was created for model validation purposes.

All blocks were coded on a whole block basis, by zone type and percent of 3-D shape within a model block (termed ore percent). Main vein zone coding took precedent over splay zones for the few occurrences where both zones were within 5 m of one another. Mined openings as of December 31, 2017 were also tagged as a mined out percentage. The tagged ore percent values were adjusted by subtracting any mined out percentage. This only affected areas of C2 and C1.

14.8 ESTIMATION

Grade modelling consisted of interpolation by ordinary kriging (OK) and Inverse Distance Weighting (ID) to the second power (a few domains in Parallel and Plug #4 used third power ID). ID interpolation was use for zones that had limited data (individual splays at Triangle, C1 and C7 at Triangle, all Parallel and Plug #4 domains). Nearest-neighbour (NN) grades were also interpolated for validation purposes but where interpolated using the longer length composite data set. Blocks and composites were matched on mineralized zone or domain.

The search ellipsoids were oriented preferentially to the orientation of the respective domain as defined by the attitude of the modelled 3-D shape. The sizes were somewhat guided by the results of the spatial analysis. Searches had mostly 60 to 80 m long ranges along the main axes of the domains, 15 to 50m across the zone. Splay zones utilized a more equant search (70 m in all directions) as composites from different splay shapes belonging to a main shear (e.g. C3 Splays) were allowed to estimate grades irrespective of which shape they came from (that is, a soft boundary approach). Block discretization was 4 m x 4 m x 1 m.

A two-pass approach was instituted for interpolation. The first pass required a grade estimate to include composites from a minimum of two holes from the same estimation domain, whereas the second pass allowed a single hole to place a grade estimate in any uninterpolated block from the first pass. This approach was used to enable most blocks to receive a grade estimate within the domains, including the splay domains. The blocks estimated in the second pass have a lower confidence level, which is reflected in their Mineral Resource classification. For the Triangle model first pass or two-hole interpolation, blocks received a minimum of 5 and maximum of 4 composites from a single drill hole. Maximum composite limit ranged from 16 to 28. The second pass for the Triangle model simply used a minimum of 2 and maximum of 10 to 16 composites to interpolate a gold grade. Parallel and Plug #4 interpolation utilized a minimum of 3 to 4 with a maximum of 2 to 3 composites from a single drill hole for their first pass interpolation. Maximum composite limit ranged from 12 to 18. The second pass protocol mimicked those used for Triangle.

In all domains, an outlier restriction was used to control the effects of high-grade composites in local areas of less dense drilling. The threshold grades were generally set through inspection of the cumulative probability plots for the mineralized units. The outlier value was 40 to 50 g/t Au for Triangle domains, 20 to 40 g/t Au for Parallel and Plug #4. The maximum distance imposed on these outliers ranged from 20 to 40 m.

14.9 VALIDATION

14.9.1 Visual Inspection

Eldorado completed a detailed visual validation of the Triangle, Parallel and Plug #4 resource models. They were checked for proper coding of drillhole intervals and block model cells, in both cross-section and plan views. Coding was found to be accurate. Grade interpolation was examined relative to drill hole composite values by inspecting cross-sections and plans. The checks showed good agreement between drill hole composite values and model cell values. The hard boundaries appear to have constrained grades to their respective estimation domains. The addition of the outlier restriction values succeeded in minimizing grade smearing in regions of sparse data. Examples of representative sections containing block model grades, drill hole composite values, and domain outlines are shown in Figure 14.5 to Figure 14.7.

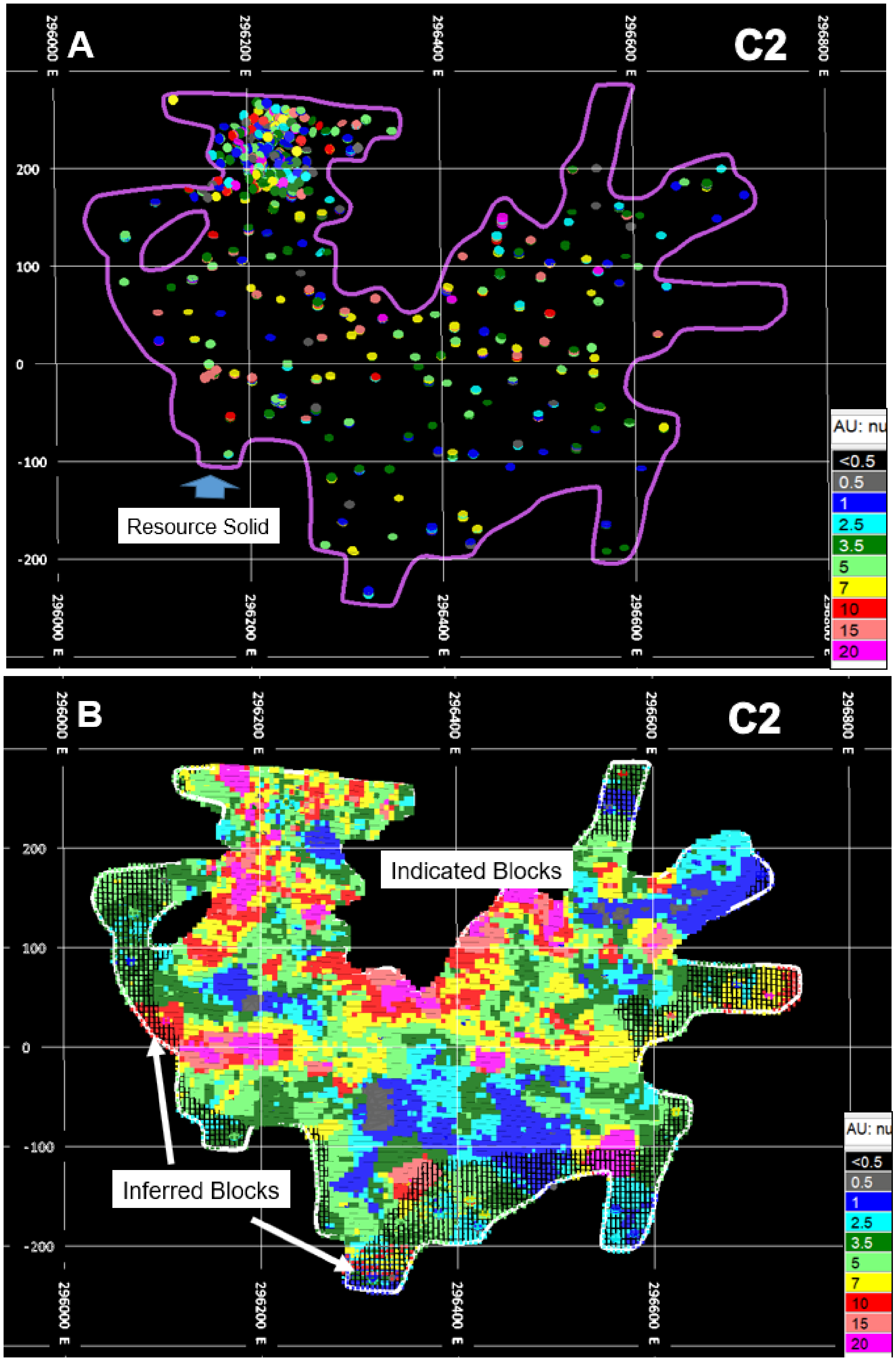


Figure 14.5 – C2 Main shear zone showing resource solid outline and A) gold composite data and B) gold block model. Note that in B) the smaller blocks denote Inferred mineral resources.

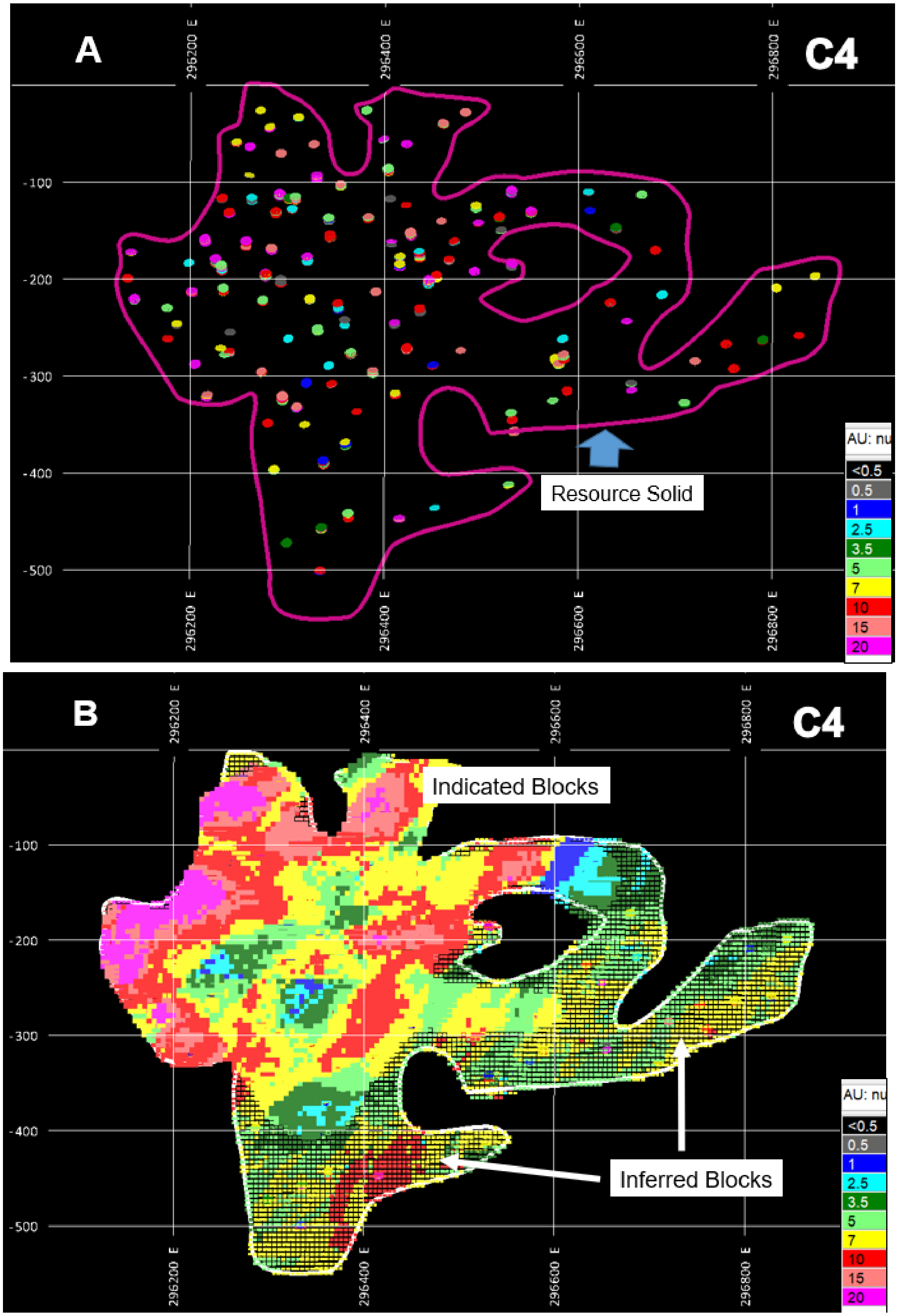


Figure 14.6: C4 Main shear zone showing resource solid outline and A) gold composite data and B) gold block model. Note that in B) the smaller blocks denote Inferred mineral resources.

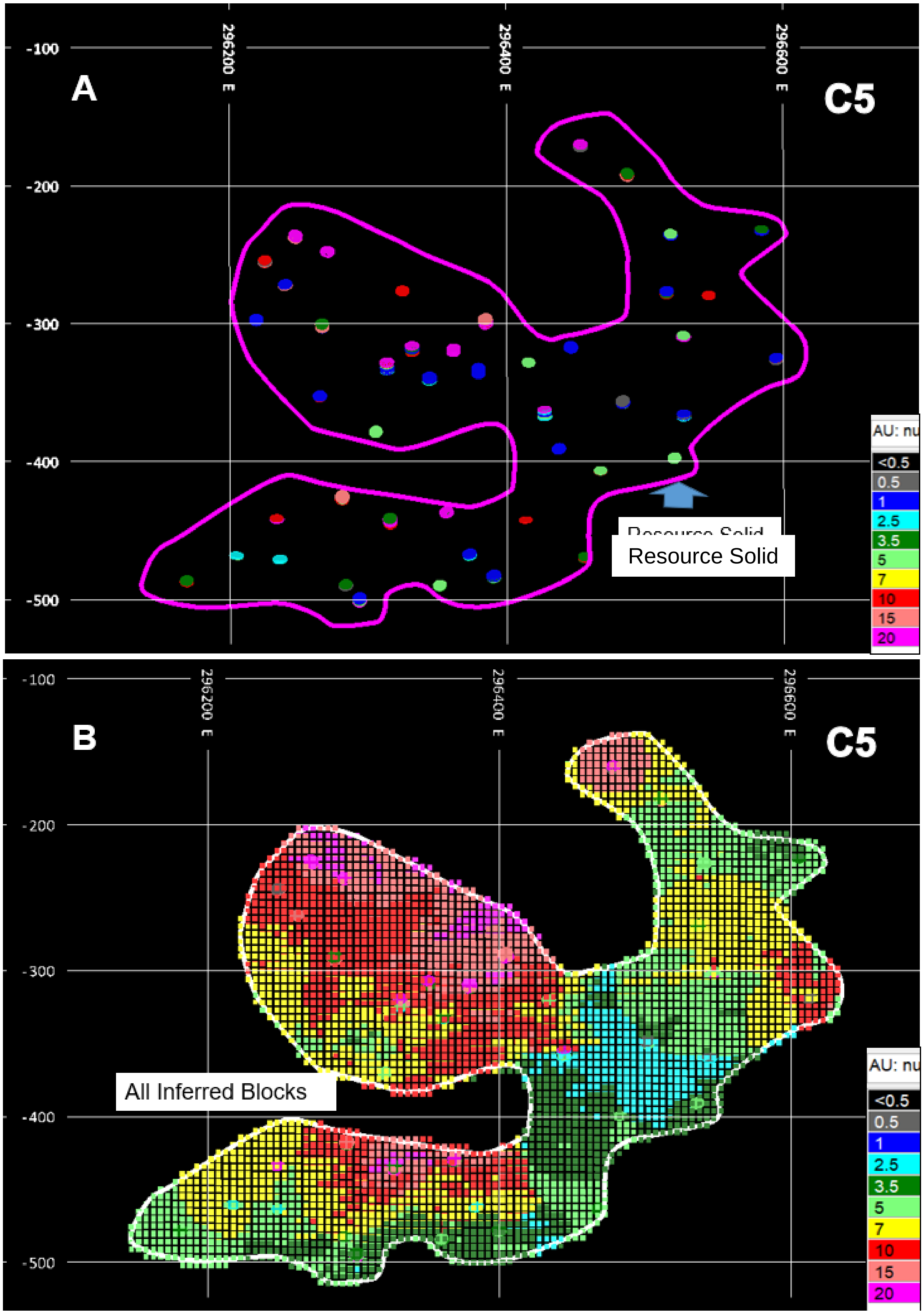


Figure 14.7: C5 Main shear zone showing resource solid and A) gold composite data and B) gold block model (in the case of C5, all classified as Inferred mineral resources).

14.9.2 Model Check for Change-of-Support

An independent check on the smoothing in the grade estimates for the major mineralized domains - C2 and C4 Main zones of the Triangle deposit - were made using the Discrete Gaussian or Hermitian polynomial change-of-support method. This method uses the “declustered” distribution of composite grades from a NN or polygonal model to predict the distribution of grades in blocks. The histogram for the blocks is derived from two calculations:

- The block-to-block or between-block variance
  - The frequency distribution for the composite grades transformed by means of Hermite polynomials (Herco) into a less skewed distribution with the same mean as the declustered grade distribution and with the block-to-block variance of the grades.

The distribution of hypothetical block grades derived by the Herco method is then compared to the estimated grade distribution to be validated by means of grade-tonnage curves.

The distribution of calculated 5 m x 5 m x 5 m block grades for gold in the mineralized domain is shown with dashed lines on the grade-tonnage curves in Figure 14.8. This is the distribution of grades obtained from the change-of-support models. The continuous lines in the figures show the grade-tonnage distribution obtained from the block estimates. The grade-tonnage predictions produced for the model show that grade and tonnage estimates are validated by the change-of-support calculations over the range of mining grade cutoff values (2.5 g/t to 3.5 g/t Au).

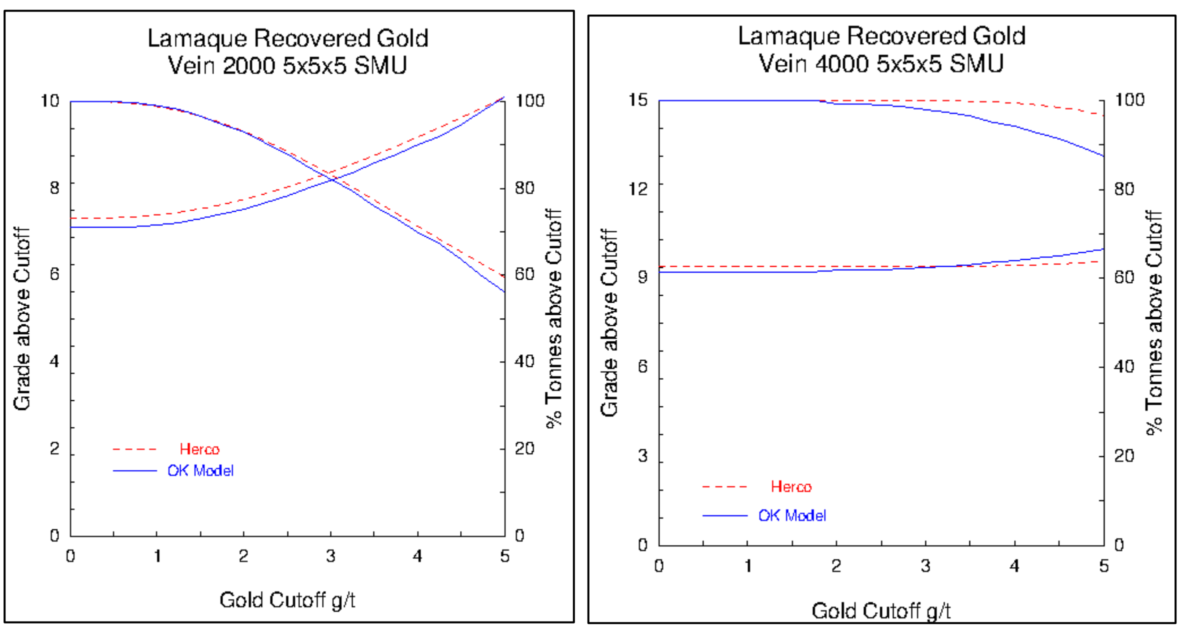


Figure 14.8 – Herco Plots for C2 (Vein 2000) and C4 (Vein 4000), Triangle Deposit

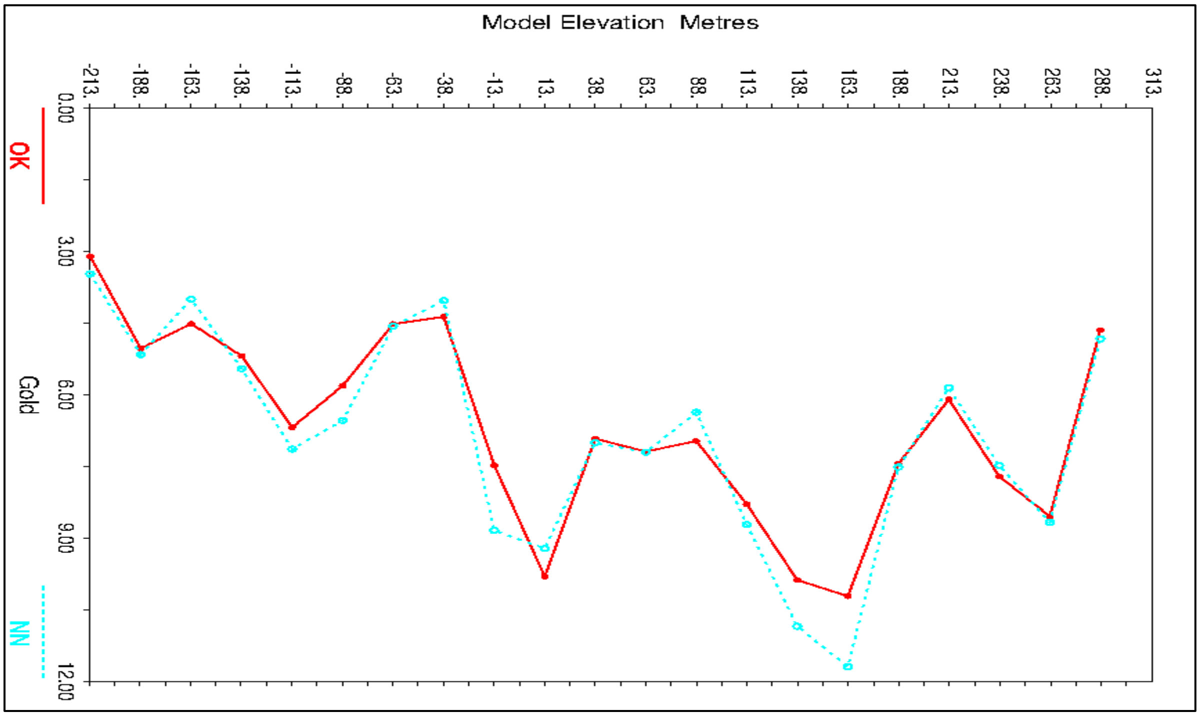
14.9.3 Model Checks for Bias

The block model estimates were checked for global bias by comparing the average metal grades (with no cutoff) from the model with means from NN estimates. The NN estimator declusters the data and produces a theoretically unbiased estimate of the average value when no cutoff grade is imposed and is a good basis for checking the performance of different estimation methods. Results, summarized in Table 14.6, show no global bias in the estimates.

Table 14.6 – Global Model Mean Gold Values by Mineralized Domain

Domain	OK / ID Estimate	NN Estimate	Difference (%)
C1	5.27	5.38	2.1
C2	7.32	7.58	3.4
C3	3.78	3.69	-1.9
C4	9.27	9.55	2.9
C5	9.66	10.03	3.7
C7	5.05	5.28	4.2
Triangle Splays	5.52	5.71	3.3
Parallel Main	8.98	9.17	2.2
Parallel Flats	7.69	7.90	2.7
Plug #4 Main	4.88	4.73	-3.2
Plug #4 Flats	3.16	3.02	4.5

C2 and C4 Main zones of the Triangle model were also checked for local trends in the grade estimates by grade slice or swath checks. This was done by plotting the mean values from the NN estimate versus the kriged results for benches and eastings (both in 25 m swaths). The kriged estimate should be smoother than the NN estimate, thus the NN estimate should fluctuate around the kriged estimate on the plots. The observed trends, displayed in Figures 14.9 and 14.10 behave as predicted and show no significant trends of gold in the estimates.



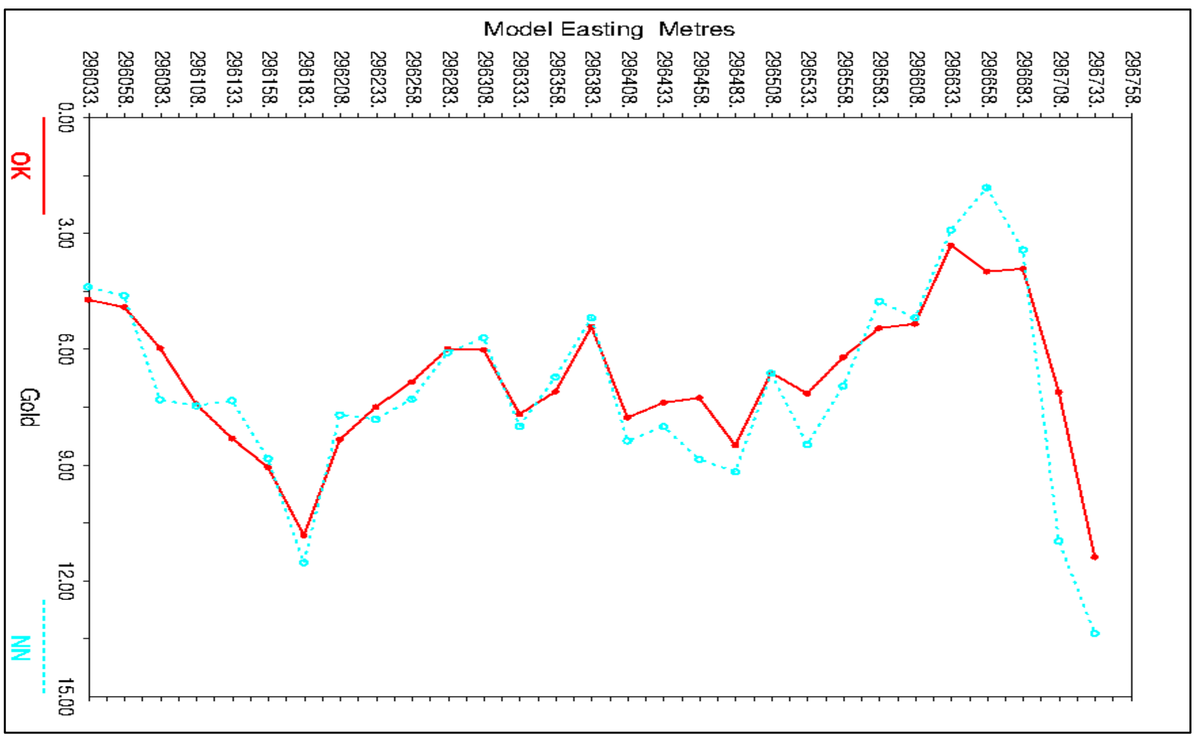


Figure 14.9 – Model trend plots showing 25 m binned averages along eastings and elevations for kriged (OK) and nearest neighbour (NN) gold grade estimates, C2 Main Zone, Triangle deposit

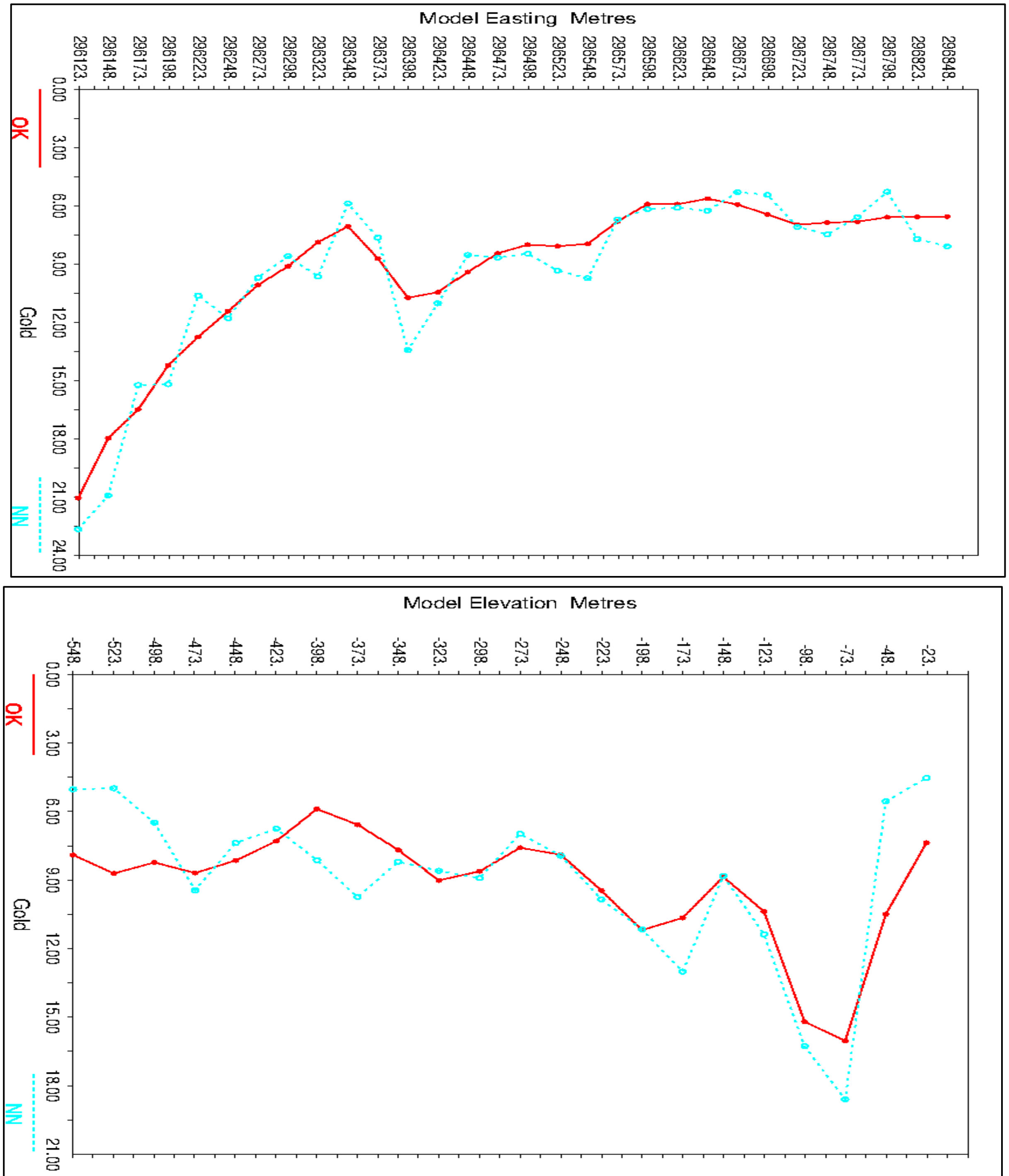


Figure 14.10 – Model trend plots showing 25 m binned averages along eastings and elevations for kriged (OK) and nearest neighbour (NN) gold grade estimates, C4 Main Zone, Triangle deposit

14.10 MINERAL RESOURCE CLASSIFICATION

The mineral resources of the Triangle, Parallel and Plug #4 deposits were classified using logic consistent with the CIM Definition Standards for Mineral Resources and Mineral Reserves referred to in National Instrument 43-101. The mineralization of the project satisfies sufficient criteria to be classified into measured, indicated, and inferred mineral resource categories.

Inspection of the Triangle model and drillhole data on plans and cross-sections, combined with spatial statistical work and investigation of confidence limits in predicting planned annual and quarterly production, contributed to the setup of various distance to nearest composite protocols to help guide the assignment of blocks into measured or indicated mineral resource categories. Reasonable grade and geologic continuity is demonstrated over most of the C2 and C4 zones in the Triangle deposit, where the average distance of the samples to a block center interpolated by the first pass, i.e. samples from at least two drill holes, is up to 30 m. Blocks that met these criteria were classified as indicated mineral resources. Indicated resource blocks within the area covered by underground development and infill diamond drilling in C2 were re-classified as measured mineral resources. All remaining model blocks containing a gold grade estimate were assigned as inferred mineral resources.

For the other two deposits, similar protocols were used. At Parallel, Indicated and Inferred mineral resource classification followed the logic used at Triangle. Plug #4 model used an average distance of 25 m to a block center interpolated by samples from at least two drill holes to be classified as Indicated mineral resources. No measured mineral resource were declared at Parallel or Plug #4.

14.11 MINERAL RESOURCE SUMMARY

The Mineral Resources for the Lamaque Project, as of 31 December 2017, are shown in Table 14.7. The mineral resources are reported at a 2.5 g/t gold cut-off grade.

Table 14.7 – Lamaque Mineral Resources, as of 31 December 2017

Deposit Name	Categories	Tonnes (x 1,000)	Grade Au (g/t)	Contained Au (oz x 1,000)
Triangle	Measured	132	10.4	44
	Indicated	3,582	8.84	1,018
	Measured + Indicated	3,714	8.9	1,062
	Inferred	4,648	7.42	1,109
Parallel	Measured	-	-	-
	Indicated	221	9.92	70
	Measured + Indicated	221	9.92	70
	Inferred	206	8.7	57
Plug #4	Measured	-	-	-
	Indicated	762	5.84	143
	Measured + Indicated	762	5.84	143
	Inferred	514	5.56	92
Total Resources	Measured	132	10.4	44
	Indicated	4,565	8.39	1,231
	Measured + Indicated	4,697	8.45	1,275
	Inferred	5,368	7.29	1,258

SECTION • 15 MINERAL RESERVE ESTIMATES

The Mineral Reserve estimate is based on Measured and Indicated Mineral Resources for the Triangle Deposit upon which the mining plan and economical study have demonstrated economical extraction. Mineral reserves are reported using a gold price of US$ 1,200/oz and an exchange rate of 1,300 CAN$/US$. An estimated cut-off grade of 3.5g/t Au was applied at stope scale for the discrimination of material to be retained in reserves and all stopes falling below the cut-off grade were taken out of the mine plan. Isolated blocks with grade slightly above cut-off were taken out of the reserves if their extraction could not support the cost of development. From a marginal cut-off grade perspective that considers sunk costs, mandatory developments in the mineralized zone were included in reserves if they graded at least 1.0 g/t.

Although the operational cost projection early in the PFS suggested that 3.5 g/t Au would be an appropriate cut-off grade, the final detailed estimation of costs was compared back to the chosen cut-off grade to ensure its validity. The results are presented in Table 15.1. The final breakeven cut-off grade was determined to be 3.41 g/t Au and the application of 3.5 g/t for the estimation of reserves remains valid.

The costs supporting the analysis of the breakeven cut-off grade were compiled from 2020 onwards, reflecting full steady-state production such that the high unit costs associated with the ramp up in production did not impact the reserve base.

Table 15.1 – Cut-off Grade Definition

Description	Total cost, CA$M *	Unit cost (CA$) * ($/t)
Mining	248.96	78.22
Milling (including ore transport and refining)	72.70	39.42
G&A	125.46	22.84
Sustaining development	53.48	16.80
Royalties	11.35	3.56
Total	511.94	160.85
Gold recovery **	%	94.51%
Gold price	$US/on	1,200
Exchange rate	CND/USD	1.30
Gold price	$CAN/oz	1,560
Breakeven grade	Gr Au/t	3.39
Cut-off grade applied to resources	Gr Au/t	3.50

* Note: Cost for years 2020 and later

** Note: Gold recovery at Sigma mill

15.1 FACTOR THAT MAY AFFECT THE MINERAL RESERVES

Areas of uncertainty that may materially impact the Mineral Reserve estimates include and is not restricted to:

- Gold market price and exchange rate.
- Costs assumptions, in particular cost escalation.
- Geological complexity and continuity.
- Dilution and recovery factors.
- Geotechnical assumptions concerning rock mass stability.

15.2 UNDERGROUND ESTIMATES

The Mineral Resources model provided by Eldorado Gold Corporation in Vancouver served as the basis for estimating mineable tonnage and metal content in the mine plan. Orebody wireframes were produced on LeapFrog Geo software and an interpolated block model was produced by MineSight Software. Using Autocad and Promine, stopes shapes were cut from the LeapFrog wireframes which represent undiluted volumes. The resource model was interrogated for the tonnes and grades in each shape and the information gathered in MS Excel files. The following constraints and modifying factors were then applied to the stope shape physical quantities to determine the final reserve tonnes and grades:

- Only those stopes in the Measured and Indicated resource categories were retained for inclusion in Mineral Reserves

- Mining method
  - Long-hole extraction where the orebody dips steeper than 42°
  - Room-and-pillar extraction where the orebody dips less than 42°

- External dilution
  - 0.75 m over-break estimated for long-hole mining except for the zones C4, C4-10 and C4-30, where 1.0m of over-break was applied to reflect the less favorable ground conditions at lower elevations. Over-break is assumed to be barren (0 g/t Au)
  - Additional 5% dilution by mass was applied to long-hole stopes to reflect mucking of backfill from the floor which becomes intermingled with the ore

- Development
  - The development minimal excavation dimensions were determined in consideration of the mining method and the orebody dip, thus incorporating internal, planned dilution
  - 100% mining recovery and no over-break were applied to development.
  - Inclusion of development grading 1.0 g/t to 3.50 g/t if the development is mandatory

- Mining recovery
  - 95% was applied to long-hole stopes
  - 85% was applied to room and pillar areas

- Milling Au was set to 95%.
- Global cut-off grade of 3.5 g/t.

15.3 MINERAL RESERVES STATEMENT

Mineral Reserves for the Triangle deposit were prepared by Eldorado Gold Lamaque Technical Services staff. The Mineral Reserve estimate is summarized in Table 15-2 and has an effective date of December 31st, 2017. All Mineral Reserves are classified as Proven or Probable in accordance with the 2014 CIM Definition Standards.

Table 15.2 – Lamaque Project Mineral Reserves as of December 31, 2017

Category	Tonnes (x 1,000)	Grade (g/t Au)	Contained Au (oz x 1,000)
Proven	111	8.78	31
Probable	3,698	7.25	862
Proven + Probable	3,809	7.30	893

15.4 QUALIFIED PERSON COMMENTS ON RESERVES ESTIMATE

As of the effective date of this report, the QP is not aware of any risks, legal, political or environmental factors that would materially affect the potential development of the mineral reserves

SECTION • 16 MINING METHODS

16.1 INTRODUCTION

The present mine plan is based entirely on the proven and probable reserves of 3.8 million fully diluted and recovered tonnes presented in Section 15. The Triangle Zone will be the only area of extraction of this PFS representing the most advanced part of the property. This mining plan herein excludes the No. 4 Plug and Parallel zones which will be examined over the course of 2018.

In July 2016, Integra contracted Promec Mining to develop a ramp and access drifts to reach the Triangle Zone for the purpose of extracting a 60,000-tonne bulk sample. As of February 28th 2018, more than 5,810 metres of lateral development, in waste and ore, and more than 340 meters of raises have been excavated following the mine plan detailed herein.

From the early phases of the project it was recognized that the reserves in the Triangle zone would be optimally recovered through a haulage ramp system to surface, as opposed to a vertical shaft which could not be economically justified at this time. The ramps will be used to haul ore and waste and provide access for personnel, equipment, materials and services, and form part of the exhaust air circuit. Exploration at depth below the lowest elevation of current reserves is ongoing, and other material handling options may need to be considered in the future (discussed further in Section 24).

The Triangle Zone long-hole stopes are designed within four sub-parallel major zones (the C-ones) dipping 50-70° to south. Approximately 95% of all reserves are located within these zones, whereas a minor portion (5%) of the reserves are located in low-dipping (20-45°) splays off the main shears (C-splays). Each of the C-zones (C2 West, C1-C2 East and C4) will be serviced by a dedicated haulage ramp. A total of 36 levels are required to extract the reserves over the vertical extent of the orebody. The total lateral development requirements are estimated at 54,000 meters over the LOM. Sector C2 West and C1-C2 East are connected via level 184 and 148. Sector C2 West and C4 are tied together at level 328 (Figure 16.1).

The dimensions of the main ramp were determined according to required clearances for the selected mobile equipment, considering ventilation requirements during development and production. It was determined that a 5.1 m wide by 5.5 m high profile would be suitable for a 45 t haul truck, although it would be possible to use 60t trucks upon removal of the ventilation ducts. In general, all the ramps are designed to be driven at a 15% gradient while minimizing intersections with major structures and optimized to reduce development requirements to access each production area. The main ramp is fully serviced for ventilation, power (compressed air and electricity), communications, industrial water and dewatering.

Remuck bays are laid along the main ramp typically at 125 m intervals (to a maximum of 150 m) and will eventually be converted to store equipment and materials, sumps, refuge stations or explosives magazines. Level access crosscuts and haulage drifts are planned to be developed off the ramp at a 5.0 m by 5.0 m profile. All infrastructure development, which would not be used for access, i.e., remucks, ventilation drifts and sumps, were designed with a 5.0 m by 5.0 m profile, (Table 16.1).

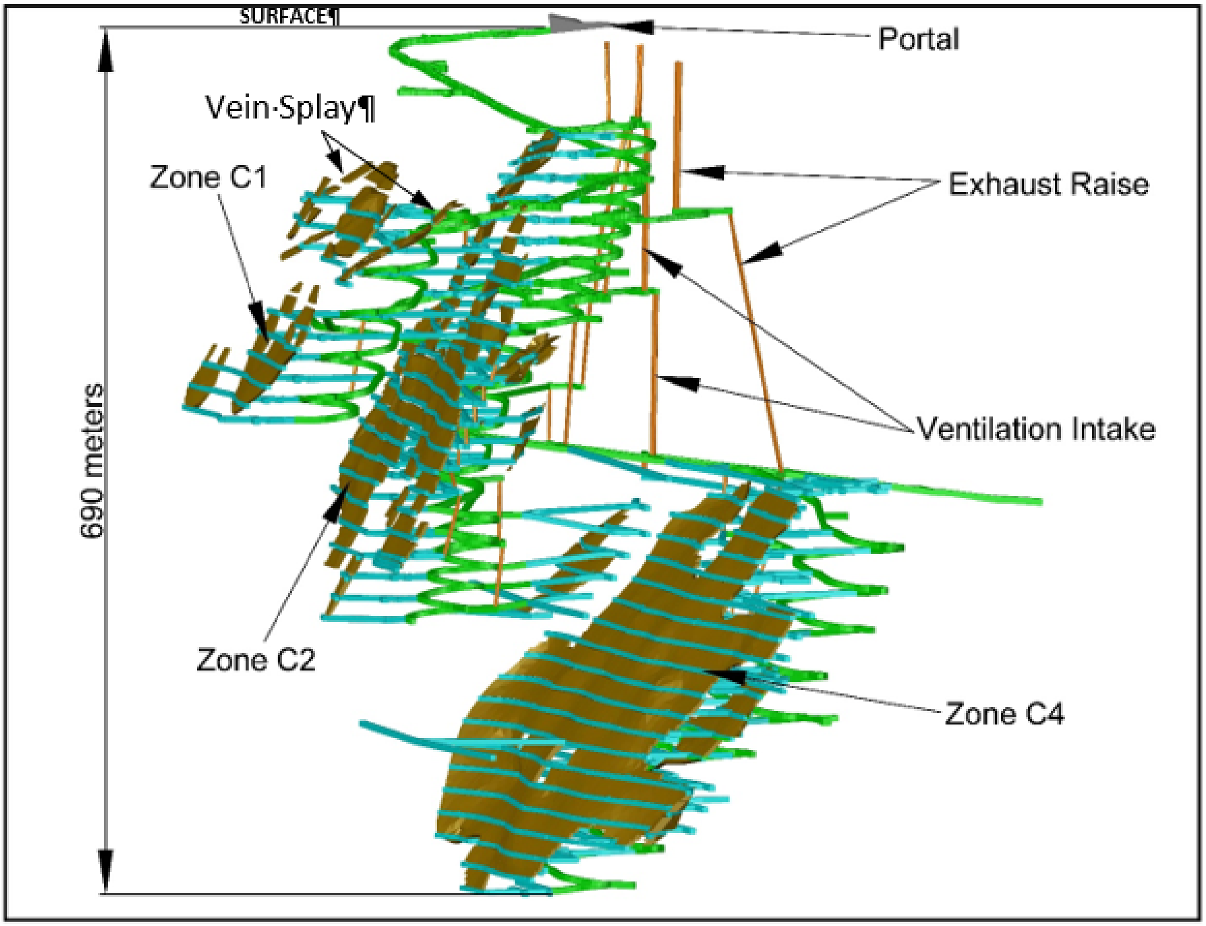


Figure 16.1 – Isometric view of the Triangle deposit and related development

A network of ventilation raises is planned to bring fresh air from surface to key points in the mine where it will be distributed to every level. This network will be mainly composed of two series of raises of which one will be equipped with ladders to provide a means of emergency egress. Stale air will travel through specific exhaust raises and the main ramp system. Raises in the upper part of the mine are designed to 5.0 m diameter whereas raises at the lower elevations of the mine will be developed to 3.4 m diameter (Table 16.1).

Table 16.1 – Underground design parameters

Parameter	Design Criteria
Minimum mining width	2.0 m
Vertical distance between sublevels	18.0 m
Service raise (egress) dimensions	3.4 m diam
Vent raise dimensions	5.5 m, 5.0 m and 3.4 m diam.
Main ramp and haulage ramp	5.1 m wide x 5.5 m high (max 15% grade)
Main haulage drive and x-cuts	5.0 m wide x 5.0 m high (max 3% grade)
Ore drive	4.2 m wide x 4.2 m high (max 3% grade)

A longitudinal retreat long-hole mining method was selected for the Triangle zone based on the orebody characteristics, including grade, dip, continuity, thickness, etc. Although some part of the C-zones have hanging walls dipping shallowly (as low as 43 degrees), the approach in this study has been to promote the application of long-hole mining where possible.

The project will use some of the existing surface infrastructure including the mill and tailings facility of the Sigma site. At the Triangle site, the temporary surface buildings for mining operations and management are in place, and at the time of writing, the expansion of these installations into permanent facilities is under way.

The ore extracted from the bulk sample program and the ore produced in Q1 of 2018 to Q4 of 2018 will be transported and processed under custom milling contracts at the Camflo and Westwood mill facilities. During this time the Sigma mill will be refurbished and upgraded to process Triangle ore production as of Q1 2019. Mill refurbishment work and the tailings pond expansion requirements have been evaluated by AMEC.

The overall life of mine based on Proven and Probable reserves is estimated at approximately 8 years, including one (1) year of pre-production followed by 7 years of commercial production.

16.2 MINING METHOD

The Triangle mine plan will primarily use mechanized longitudinal long-hole retreat method on most of the C-zones. The stopes that have been developed, and in some instances mined for the bulk sample will serve to trial and optimize the cemented rock-fill batch plant. A minor portion of the reserves in shallow dipping areas such as the upper part of C1 will be mined using the room-and-pillar mining method, and backfill will not be required.

Ore will be hauled by 45 tonnes underground mining trucks via the main access ramp to the surface ore pad. The broken ore will then be transported by a local contractor via an off-road route from the Triangle ore pad to the Sigma mill site.

16.3 MINING METHOD SELECTION

The proposed mining methods has been used successfully in neighboring operations that share similar ore body geometry. Local manpower and mining professionals alike are familiar with these methods and possess an extensive underground mining background in highly mechanized and narrow vein mining operations. In consideration of practicalities, the minimum mining width for long-hole stoping has been set to 2.0 m whereas the minimum mining height in room-and-pillar mining has been set at 1.8 m.

The various zones will be further subdivided into local production centers in order to increase the number of working areas that sustain the target production level. From the main ramp, a main level haulage drift will be driven on each level followed by the ore drives which will be developed east and west from the approximate midpoint of each production center.

Near the junction of the ramp with any given level, infrastructure will be constructed to provide and distribute electrical power, ventilation, water and compressed air through cables and steel piping.

Each level will host an electrical sub-station which will be fed by either the 13.8kV, 4.16 kV or 600 V underground feeders.

16.3.1 Long-hole Method

Most of the Lamaque reserves are amenable to using the long-hole approach with only few isolated exceptions. Sublevels will be developed along the vein strike with an 18.0 m vertical separation. Based on geotechnical considerations, planned stoping will be implemented according to the parameters set out in Table 16.2.

The stope preparation sequence involves marking all the reference lines on the walls for drilling, followed by raise boring an initial opening raise using a V-30 boring head[1]. Production drilling is then carried out following a pattern defined by the engineering group. The blasting sequence will generally involve two separate firings; the first blast will increase the initial void according to the available volume for broken material whereas the second will encompass the remaining stope volume. Production blast holes will be typically 64-mm (2.5”) or 76-mm (3.0”) in diameter following a pattern parallel to the walls. Mucking will be performed longitudinally from the ore drift using remote controlled loades. Each stope will be filled with a combination of cemented rock-fill and free running waste material as described in Section 16.4.7. Figure 16.2 illustrates a typical production center mined by the long-hole retreat method.

Table 16.2 – Stope dimensions and introduction hole diameters for long-hole mining

Stope Dimension			Drilling Diameter
Length	Height	Thickness	
25.0 m – in the C2 Zone 22.0 m – in the C4 Zone	18.0 m (vertically)	2.0 m min	64 mm or 2-1/2 “
		2.0 m to 3.5 m	64 mm or 2-1/2 “
		>3.5 m	76 mm or 3-0 “

[1] Machine Roger International

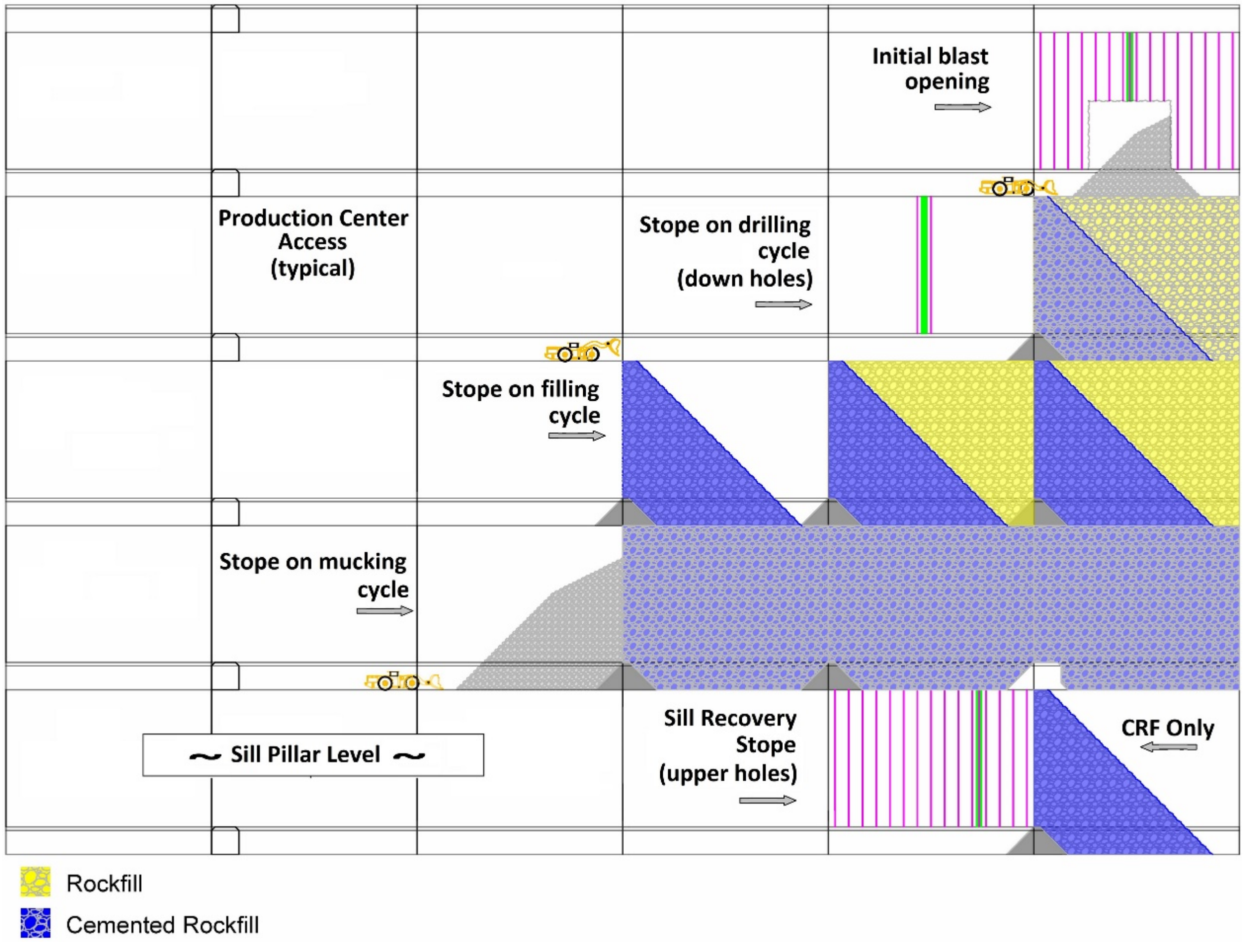


Figure 16.2 – Typical longitudinal stopes using rockfill

16.3.2 Room-and-pillar method

The proposed room-and-pillar stope configuration is based on typical industry practices for currently operating mines with similar vein geometry. The typical mining height will vary from 1.8 m to 3.0 m. The room-and-pillar mining method involve the excavation of a series of “rooms” following the vein, leaving “pillars” (columns) of rock in place to support the mine roof. In conventional room and pillar mining, drilling is achieved using hand-held drill equipment and holes are loaded with explosives. After blasting, bolts are then installed in the mine roof to ensure the roof is properly supported. The broken rock is moved to either a raise or a drawpoint using an electric slusher with a scraper. The broken material is the recovered by an LHD to be hauled to surface by truck.

A room-and-pillar design is proposed for the upper part of the C1 Zone using 6 m wide rooms with 3 m x 6 m pillars. Figure 16.3 presents a schematic of this typical room-and-pillar design. Mechanized sublevel development from which the blasted material is loaded by LHDs will be completed at 60 m intervals along the vein.

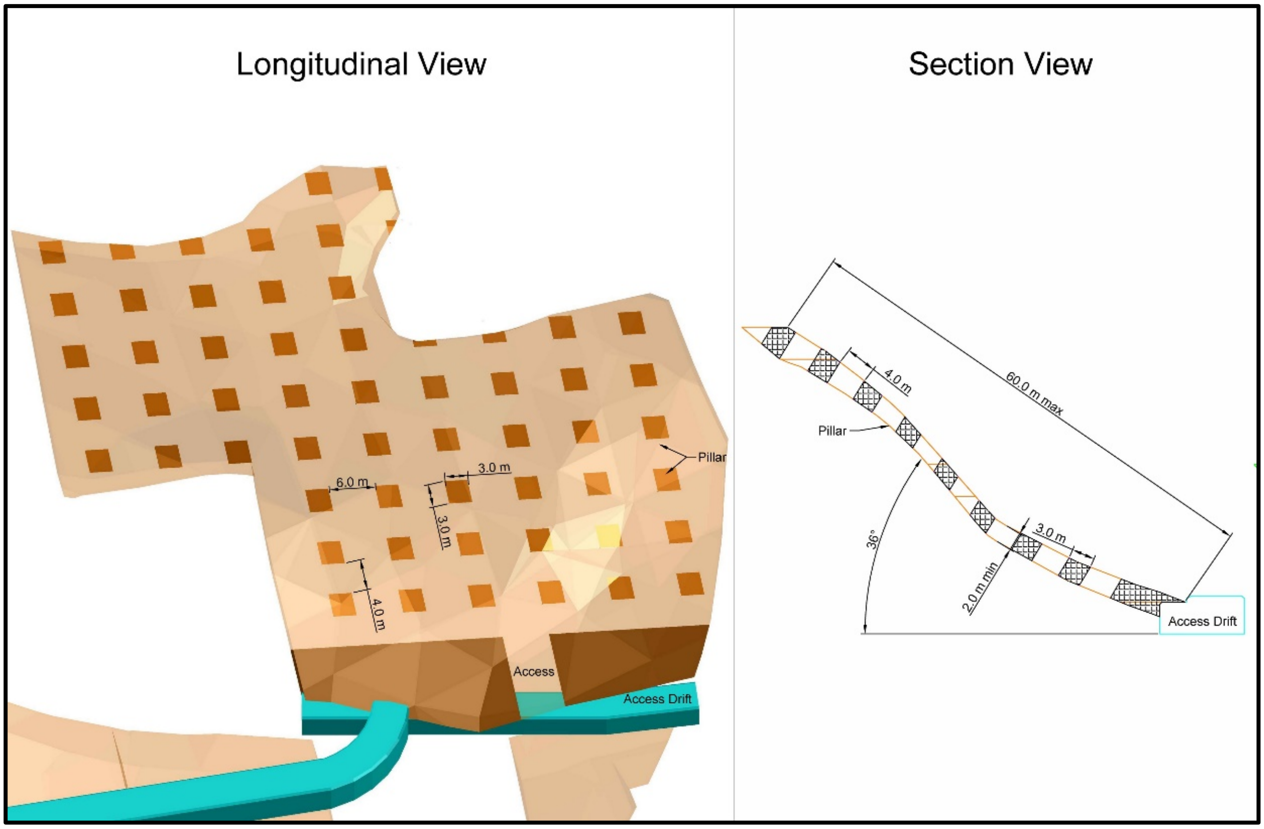


Figure 16.3 – Typical room and pillar design

16.4 GEOTECHNICAL ASSESSMENT

A detailed geotechnical evaluation of the Triangle Zone has been completed by Golder Associates Ltd. (Golder, 2014 & 2017). This evaluation includes a rock mass classification and geomechanical mine design recommendations for the C2 and C4 zones. It is considered that these design parameters can be used for the other mineralized zones as the geological context is similar. The recommendations are based on data coming from seven NQ-3 geotechnical drill holes with core orientation, hydraulic conductivity testing, structural data from eight televiewer surveys, fifteen laboratory testing samples, and the exploration drill hole database with 953 boreholes where RQD, fracture count and structure types are available. In addition, this evaluation also includes the structural and rock mass data collected from wall mapping on traverses located at the 76, 94 and 130 levels along the exploration ramp and ore drifts, as well as field observations of the exposed rock masses.

16.4.1 Rock mass characterization

16.4.1.1 Intact rock strength testing

The intact rock strength was estimated through field assessment of the strength index (ISRM, 1981) and laboratory rock tests; resulting in the main rock types being classified as very strong (uniaxial compressive strength, UCS in the range of 100 to 250 MPa). The intact rock strength parameters based on laboratory testing and interpreted Hoek-Brown strength parameters are presented in Table 16.3.

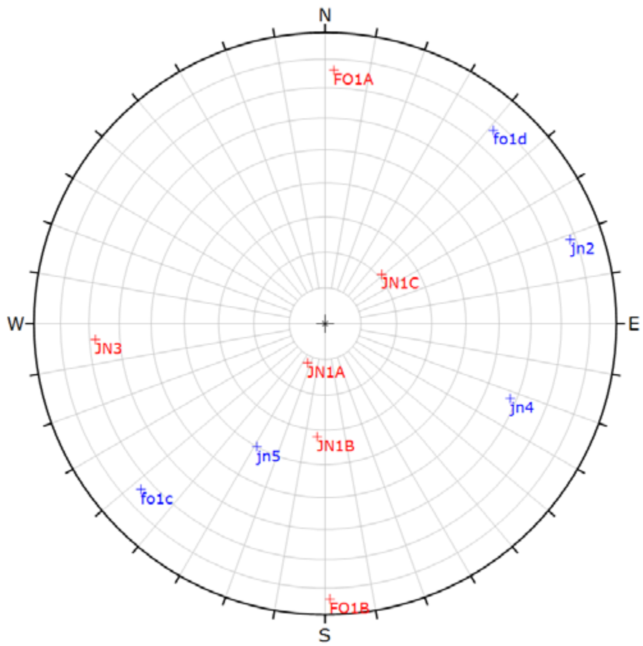
Table 16.3 – Intact rock strength parameters – Triangle Zone (from Golder 2017)

	Parameters	Representative Range	Engineering Average Value
Laboratory Intact Rock Strength Results	Unconfined Compressive Strength (UCS)	100 to 200 MPa	145 MPa
	Young's modulus (E)	30 to 40 GPa	35 GPa
	Poisson's ratio (ν)	0.10 to 0.25	0.15
	Density	2.7 to 3.0	2.8
Hoek-Brown Intact Rock Strength Parameters	Intact Compressive Strength (σci) (35% Percentile)	166 to 189 MPa	175 MPa
	Tensile Strength (σt)	-8 to -15 MPa	-13 MPa
	Hoek-Brown material coefficient (mi) (35% Percentile)	12.6 to 16.0	14.0

16.4.1.2 Structural analysis

Major and minor structural sets have been identified using the oriented core drill hole data, structural televiewer data, and structural wall mapping along the exploration ramp and ore drifts. The stereonet presented on Figure 16.4 indicates that there are, in general, three major discontinuity sets: i) the foliation set (Fo1A and Fo1B), ii) the sub-horizontal set (JN1A to JN1B) and iii) inclined set JN3. In addition, there are other minor discontinuity sets, which may occur locally. The foliation set is persistent and dominant throughout the mining area, with increasing intensity closer to the mineralized zones.

Field observations along the ramp, down to a depth of 130 m, indicate that: a) the rock mass is sparsely jointed, b) the sub-horizontal set is persistent and dominant, with variable spacing from 0.2 to 3 m and c) near the main mineralized zones, there are quartz veinlets, with orientations similar to the minor joint sets. These quartz veinlets tend to be healed and do not create separation planes. However, when occurring with calcite and tourmaline, they become weakness planes.



	Color	Dip	Dip Direction	Label
User Planes				
1		76	182	FO1A
2		84	359	FO1B
3		74	48	fo1c
4		77	221	fo1d
5		12	24	JN1A
6		32	4	JN1B
7		21	229	JN1C
8		78	251	jn2
9		68	86	JN3
10		58	292	jn4
11		40	29	jn5

Figure 16.4 – Stereonet plot of major (in red) and minor (in blue) structural sets (from Golder 2017)

16.4.1.3 Rock Mass Classification

Two geotechnical domains have been initially defined in the Golder (2016) report prepared for the Preliminary Economic Assessment (PEA) study; they are Diorite (intrusive) and Tuff. These domains are lithology-driven and based on the available data. Further evaluation of the geotechnical data, including the new exploration holes and wall mapping, suggest that the differences between the rock mass quality that might exist between the Diorite and Tuff are found to not be substantially different enough to justify the representation of separate parameters for the rock mass classification. Instead, they can be treated together, as summarized in Table 16.4 for the Triangle Zone. On average, the rock masses are classified as Good (average) quality.

Table 16.4 – Rock mass classification (from Golder 2017)

	Diorite	Tuff
RQD	*Very good* (75-100%) (assumed RQD = 90% - Average and 75% - Lower Bound for rock mass classification) *Very Poor* and *Poor* (0-50%) in less than 5% of data Areas within the planned hanging walls with low RQD values (<50%) have been identified, mainly in the HW of the C4 zone. RQD = 90% (Average, rating 18) and 75% (Lower Bound, rating 15)	
Fracture Spacing	Mostly 0.3-1 m spacing, with some 0.05-0.3 m values. Ratings of 18 (Average) and 12 (Lower Bound).	
Intact Rock Strength	Mostly *Very Strong* (R5, 100-250 MPa) to *Strong* (R4, 50-100 MPa) Locally, may show weaker strength (R4, some R3) when examining 0-400 m interval RMR rating of 12 (Average) and 7 (Lower Bound)	
Joint Conditions	Clean to slightly altered, predominantly planar, rough to smooth (Jr = 1.5 – Average and Lower Bound, Ja = 1 – Average to Ja = 2 Lower Bound) and (Jcon = 20 – Average to 12 – Lower Bound)	
Number of Joint Sets (Jn)	Two major joint sets plus a random set (Jn = 6) to three joint sets (Jn = 9)	
Groundwater Conditions	Dry, rating 10 (note if significant seepage water is observed during the development of the upper drilling drift, then consider reducing the stope strike length)	
RMR_{1976} (Average / Lower Bound)	**78** (Good) / **56** (Fair)	
Q' (Average / Lower Bound)	**23** (Good) / **6** (Fair)	
Q' (fault/shear zones)	**1** (RQD = 25%, Jn = 6, Jr = 1 and Ja = 4)	

Intervals with lower RQD values (or high number of fractures per metre) were identified within some of the planned hanging walls (HW) of stopes for the C4 zone. These lower values are related to fault and shear zones and are represented by poor rock mass quality, as shown in Table 16.4. These fault zones tend to be narrow (thickness < 3 m) and occur at distances varying from a few meters to more than 10 m from the planned stope HW; thus, minimizing their potential impact on the stope dilution.

16.4.2 Stope Sizing Assessment

Numerical and empirical design methods have been considered to find the optimal stope dimensions. Using Mathew's Stope Stability Chart and Map3D numerical modelling, it was proposed to limit stope strike length to 25 m for the planned sublevel vertical intervals of 18 m where there are typical (or average) rock mass conditions for for both the C2 and C4 Zones (Golder, 2017). Where there are anticipated poor rock mass conditions due to the proximity of fault/shear zones in the HW of a stope, then it was estimated, for planning purposes, that the stope strike length would be reduced by half (i.e., strike length of 12.5 m) in about 15% of the planned stopes within the C4 zone.

Dilution was estimated by considering that up to 0.75 m of rock could fall from the stope HW. This would represent a percentage of dilution varying from 37.5 % to 15% for a stope width of 2 m and 5 m, respectively. To reduce the potential dilution, cable bolts will be installed from the sub-levels into the hanging wall.

16.4.3 Sill Pillar Assessment

Sill pillars will be temporarily left in place and will serve to increase the number of production areas in operation at the same time. These pillars are designed to 18 m vertical height, similar to the regular long-hole stope height, and will be placed every 90 m vertical interval, the equivalent of 5 stopes vertical mining progression. The sill pillars will be recovered as the mining progresses upwards to the sill level. 3D numerical modelling results indicate that 18 m vertical height sill pillars are considered adequate for both the C2 and C4 zones.

Since the mining sequence will follow a longitudinal retreat toward the centre of the mineralized zone, the sill pillar geometry will be reduced, subsequently causing an increase of the induced stresses at the central portion of the sill pillar. For the planned C2 and C4 extractions evaluated in this PFS study, the potential overstress of the sill pillar is not considered to substantially impact its recovery. However, the numerical results suggest that the mining sequence to recover the sill pillar located at 4650 m (estimated depth of 670 m) at the lower portion of the C4 zone or, in the future, at greater depths will likely require modification. For example, one alternative for mining at these lower depths is creating two accesses at each lower sublevel to allow a chevron (or triangular) type of mining sequence progressing from the center towards the outside abutments, reducing the induced stresses in the central portion of the sill pillar.

16.4.4 Sill Pillar Recovery

The sill pillars will have a similar height to regular L-H stopes. However, they will be mined by drilling upper blastholes (including the V-30 slot raise) instead of downhole and will be primarily backfilled with cemented rockfill (CRF), as shown schematically on Figure 16.5. The stopes located at the extremities of sill pillar will only be partially backfilled. Ore mucking will be done similarly to regular L-H stoping (i.e. driving the LHD remotely along the lower ore drive) but a series of short cross-cuts will be driven between the haulage drive and the ore drive to allow CRF to be dumped into the stope once it is mucked out

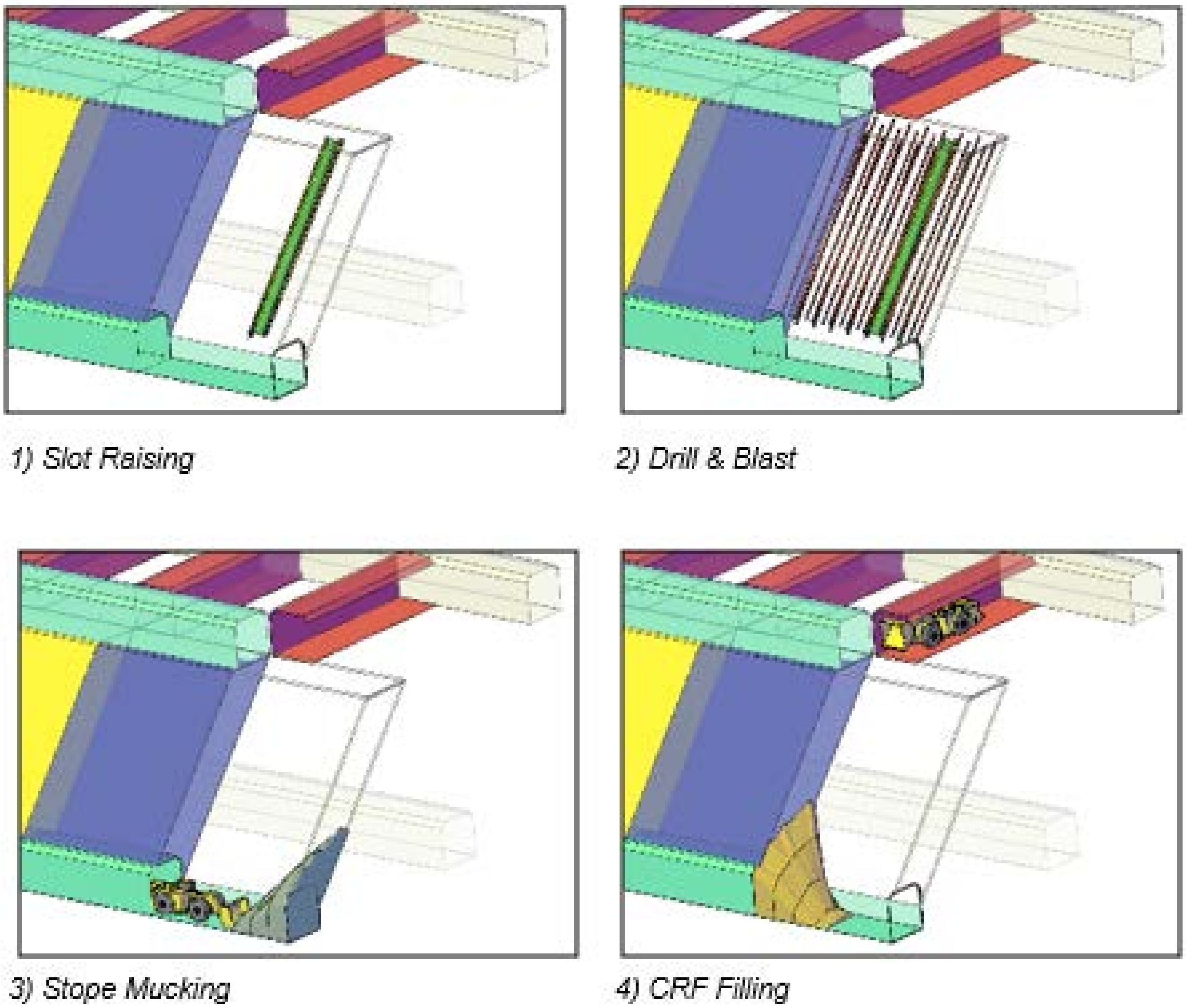


Figure 16.5 – Sill Pillar Stope extraction phases

It was assumed that about 95% of the sill pillars will be recovered.

16.4.5 Crown Pillar Assessment

A crown pillar stability assessment was carried out in 2014 with Golder recommending that the back of the nearest stope back to the ground surface be located at a minimum depth of 40 m and 46 m for a planned stope span of 3 m and 5 m, respectively. The minimum depths include the average overburden thickness of 25 m plus the crown pillar (bedrock) thickness.

For the PFS, it is assumed that mining will not take place above El. 58 m, which corresponds to a depth of approximately 58 m.

16.4.6 Typical Ground Support Patterns

Ground support patterns for both temporary and permanent excavations have been validated by Golder Associates Ltd. (Golder) for the PFS study. The ground support consists of using 19 mm (3/4" or # 6) diameter resin grouted rebars in the back (or roof) of the ramp and access drifts and Swellex Pm12 bolts in the back of the ore drifts. Split sets (SS33) will be installed in the walls. The ramp and drifts will be excavated with flat backs and rounded shoulders. Table 16.5 describes the general support for different excavation types.

Table 16.5 – Typical ground support (from Golder 2017)

Openings	Dimensions	Planned Ground Support by Integra	Comments
Ramp (permanent)	6 m (W) x 5.5 m (H) Flat back	Roof: 1.8 m fully resin grouted rebar (1.2 m x 1.2 m), #6 welded mesh Wall: 1.5 m galvanized split set (1.2 m x 1.2 m), #6 welded mesh, screen support to within 1.8 m of floor	Increase the bolt length to 2.1 m long rebar if span is more than 6 m. In the presence of fault zones, fiber reinforced shotcrete (min. 50 mm) is recommended. 0.9 m (3') Split Sets to pin the mesh
Ramp Intersection (permanent)	Intersection between ramp and access drift. Span < 7.5 m	Roof: 3.0 m (10') fully resin rebars (1.2 m x 1.2 m) and #6 welded mesh Wall: 1.5 m (1.5') galvanized split set (1.2 m x 1.2 m), #6 welded mesh, screen support to within 1.8 m of floor	Care should be taken to maintain the corners of intersections in order to avoid making the effective span from becoming larger than necessary.
	Intersection - Span 7.5 m to ≤ 10 m	Roof: 3.0 m (10') resin rebar (1.2 m x 1.2 m) and #6 welded galvanized mesh Secondary, 6 m long, 25 tonnes, 7 strand, 15 mm (5/8"), bulged cable bolts on a 1.5 m x 1.5 m Wall: 1.5 m split set (1.2 m x 1.2 m), #6 welded galvanized mesh, screen support to within 1.8 m of floor	These bolts should also be used for the 8 m to 10 m wide garage. Extend to a minimum of 2 bolting lines (or rings) before and beyond all intersection sides.
Access Drift (permanent)	4.5 m (W) x 4.5 m (H) to 5.5 m (W) x 5 m (H) Flat back	Roof: 1.8 m resin rebar (1.2 m x 1.2 m), #6 welded galvanized mesh Wall: 1.5 m split set (1.2 m x 1.2 m), #6 welded galvanized mesh, screen support to within 1.8 m of floor	Requires galvanized split-sets. In the presence of wide fault zones, fiber reinforced shotcrete (min. 50 mm) is recommended.

Openings	Dimensions	Planned Ground Support by Integra	Comments
Ore Drift (temporary)	3.5 m (H) x < 6.5 m (W) Flat back (not at intersection)	Roof: 2.1 m (7') Swellex Pm12 (1.2 m x 1.2 m), #6 welded mesh Wall: 1.5 m (5') split set (1.2 m x 1.2 m), #6 welded mesh, screen to provide support on the shoulders down to 1.5 m from the back.	Split set 0.9 m (3') to hold the mesh.
	3.5 m (H) x 6.5 m (W) to 10 m (W) (not at intersection)	Roof: 3.0 m (10') Swellex Pm12 (1.2 m x 1.2 m), #6 welded mesh Wall: 1.5 m (5') split set (1.2 m x 1.2 m), #6 welded mesh, screen to provide support on the shoulders down to 1.5 m from the back.	When crossing wide fault zones, fiber reinforced shotcrete (min. 50 mm) is recommended.
Intersection Ore Drift and Access Drift	Span < 10 m	Roof: 3.0 m (10') Swellex Pm12 (1.2 m x 1.2 m), #6 welded mesh Secondary 3.6 m (12') long, coated, Super Swellex Pm24 (1.5 m x 1.5 m) Wall: 1.5 (5') m galvanized split set (1.2 m x 1.2 m), #6 welded mesh, screen support to within 1.8 m of floor	

When drifting through poor ground conditions two coats of shotcrete will be applied. A flash coat of 25 mm of shotcrete will be applied on the back and shoulders before starting mucking activities, and a second coat varying from 50 mm to 75 mm will be applied after the installation of bolts and mesh.

Longer bolts consisting of 25 tonnes cable bolts or connectable Super Swellex Pm24 bolts will be required at intersections and along the backs of top cuts in places with more than 10 m wide long-hole stope spans.

6 m to 9 m long cable bolts will also be installed into the HW rock mass from the sub-levels in a fan patterns at inclinations of 20°, 40° and 70° and a 2 m x 2 m spacing.

16.4.7 Backfill Strength

Rockfill will be used to backfill the stopes. Since a longitudinal retreat mining sequence will be used, only about 1/3 to 1/2 of the volume is planned to be filled with cemented rockfill (CRF) and the rest of the void will be filled with non-cemented (or unconsolidated) rockfill. There will be a minimum of 2 m of CRF from the stope wall, as illustrated on Figure 16.6.

The main requirement of the cemented fill is for it to remain stable (and self-supporting). When a fill wall is exposed by mining an immediately adjacent stope, a cement rockfill (CRF) with 4 wt% binder content based on a water: cement ratio of approximately 0.7:1 will be used to provide a strength greater than approximately 600 kPa. This estimated strength considers the partial backfilling of the upper portion of the stope (Figure 16.6) and the binder content takes into account the normal variability of the cemented rockfill mix during mining operations. With the implementation of a QA/QC program, and through experience, the mine will be in a position to evaluate the opportunity to reduce the binder content to 3.5%.

The rockfill size gradation will be comprised of approximately 25% < 1", 25% between 1" and 4" and 50% > 4". The maximum grain size will be 12 inches.

For planning purposes, it has been assumed that there will be a delay of 7 days after backfilling before mining of an adjacent stope commences.

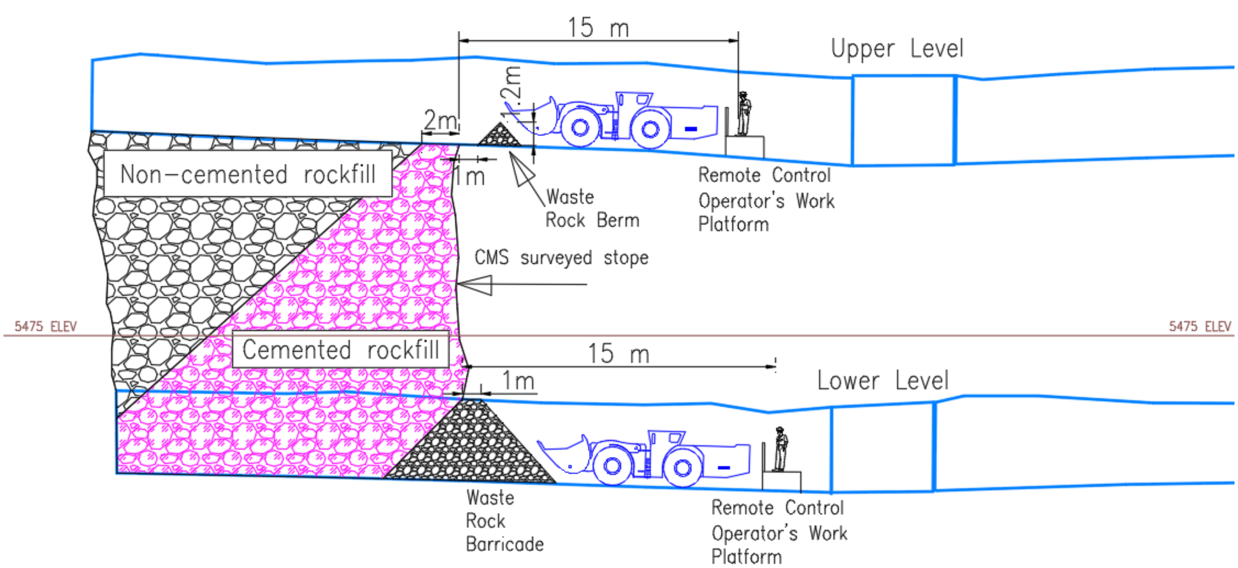


Figure 16.6 – Schematic representation of the proposed partial use of cemented rockfill

Stopes located above the temporary sill pillar will be backfilled with a higher strength cemented rockfill (CRF) to allow future sill pillar recovery. The estimated CRF strengths varies with the stope span (or width) from about 1 MPa to 3 MPa for spans ≤ 3 m to up to 10 m, respectively.

Based on the current laboratory test results, it is recommended that a 6% and 7% binder content be used for excavated stope spans (or widths) of ≤ 3 m and 3 to 10 m, respectively. A minimum delay of 28 days curing time was used for planning purposes.

Any future change to the minimum binder content for the stopes located above sill pillars will be based upon; a) laboratory test results with specimens taken directly from the CRF mix at the backfill bay, b) actual performance of CRF and with minimum to no unravelling after the sill pillar is recovered and c) mine experience gained during the recovery of the upper sill pillars.

16.5 UNDERGROUND MINE DESIGN

The various zones will be accessed through the main ramp from the portal next to the Triangle laydown area on surface. From the ramp, sublevels will be developed to access mineralized material.

Excavation in mineralized material is generally 4.2 m high by 4.2 m wide as a minimum, but the width varies in relation to vein thickness to provide the optimal geometry for production drilling. Ore and waste will be hauled by LHDs from the production area to either a remuck bay or a loading point close to the ramp and then loaded into trucks to be hauled directly to surface.

The compilation of mine development quantities considered in the mine plan is presented in Table 16.6.

Table 16.6 – Mine development quantities for the C- zones group

Development	Pre-prod. years (tonnes)	Production years (tonnes)							Total
	2018	2019	2020	2021	2022	2023	2024	2025	
Ramp (5.1 m x 5.5 m)	3 550	2 564	2106	1 886	276	0	125	0	10 507
Acces drifts (4.5 m x 4.5 m) (5.0m x 5.0m)	2 845	5 172	6 183	5 352	3 896	1 812	836	0	26 096
Ore Drives (4.2m x 4.2m)	2 136	2 910	3 172	4 298	3 208	1 428	247	0	17 398
Alimak raise 5m diam. and 3.4m diam.	201	693	566	84	80	0	0	0	1 947

16.5.1 Mining sequence

In the first quarter of 2018, the first year of pre-production, three development crews will be working 11-hours shifts day and night. Two of these teams will concentrate on ramp development. The third team will be focussed on developing level accesses and ore drifts. A fourth development crew will be added in the second quarter of 2018 and will also be dedicated to the development of accesses and ore drifts.

An Alimak (contractor) team will develop the ventilation raise network.

Production from long-hole stopes will commence in the first quarter of 2018. The number of active production centers will increase until it reaches six centers in Q4 of 2019. Room and pillar stopes will start in the third quarter of 2019. At this point, 1 crew comprising two miners will work each shift. In the first quarter of 2022, a second crew will be required. The room and pillar method represent 6% of the total ore tonnes in contrast to the long-hole method which represents 67% of the total ore tonnes.

Figures 16.7 and 16.8 below are longitudinal views of C2 and C4 Zones respectively, showing the outline of each production center within each zone.

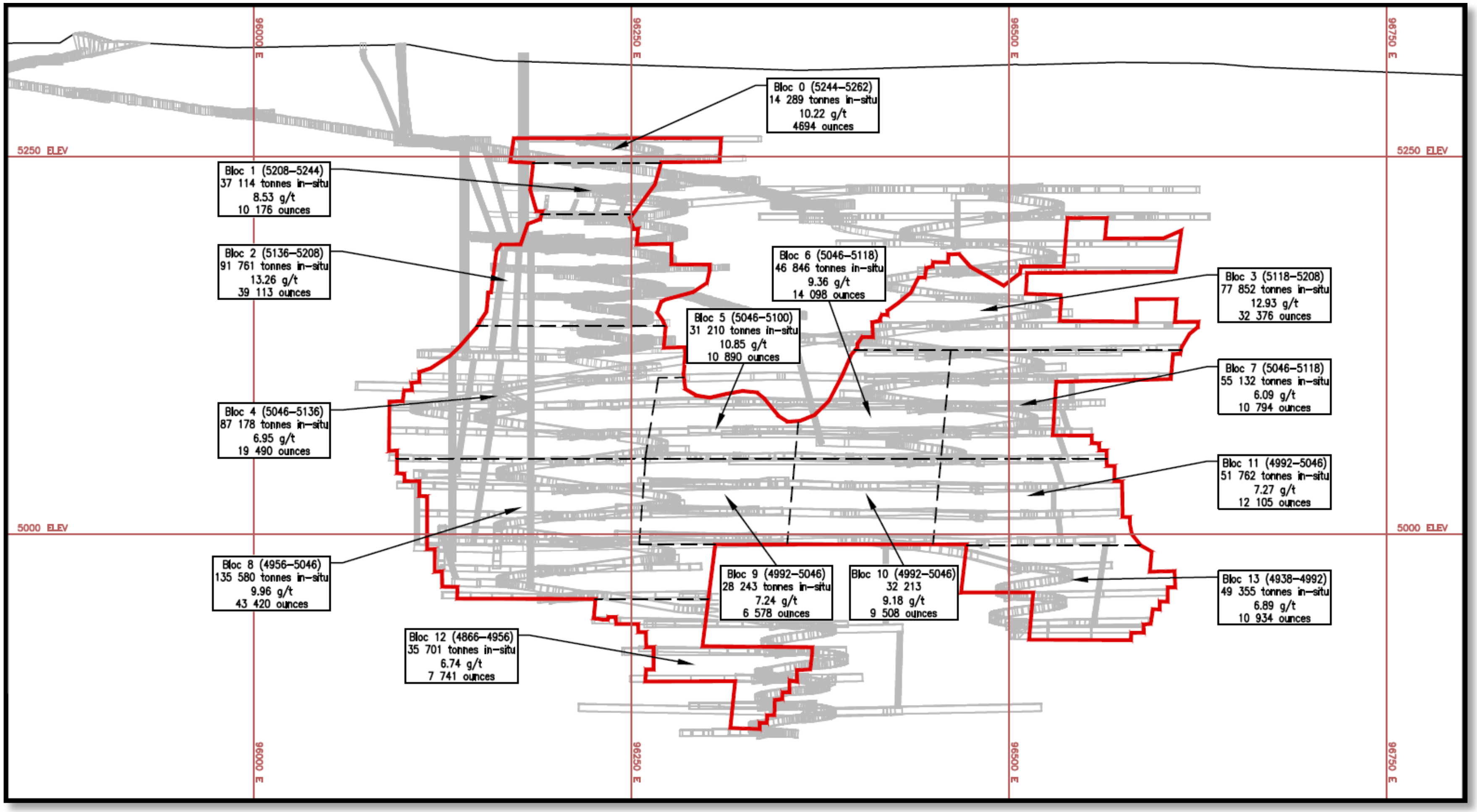


Figure 16.7 – Longitudinal view of the C2 Zone with production centers

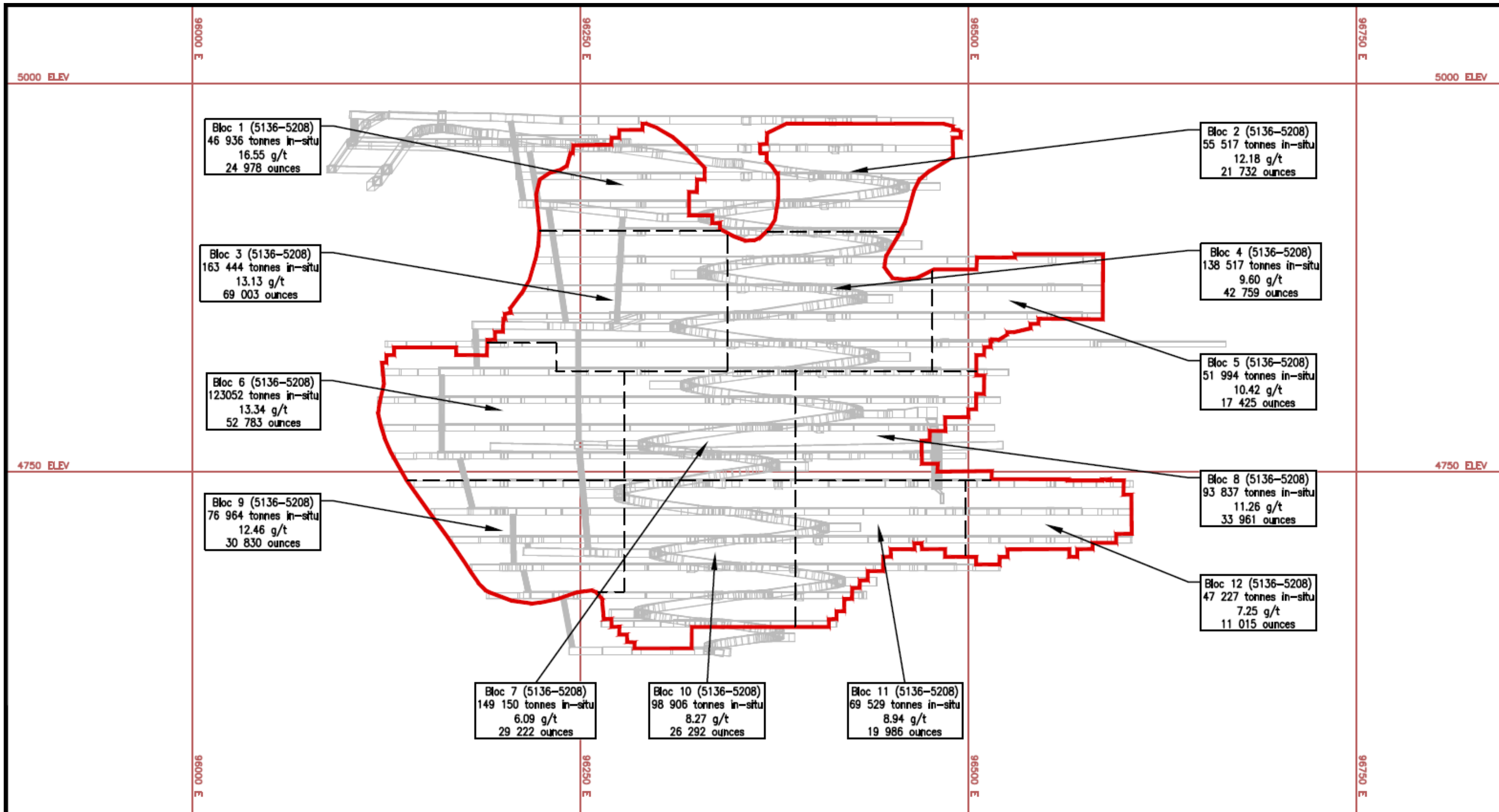


Figure 16.8 – Longitudinal view of the C4 Zone with production centers

16.6 MINING RATE

The production rate will ramp-up in pre-production from 165 tpd of mineralised material in 2018 to 1,200 tpd in 2019. The mine plan has a duration of approximately 8 years including the pre-production period. The average production rate from the Triangle Zone is 1,481 tpd over the commercial period, calculated on a 360 operating days per year.

Limited production will occur during the pre-production stage, accounting for approximately 54,225 ounces over the course of year 1 pre-production period.

16.7 MINE DEVELOPMENT AND PRODUCTION ASSUMPTIONS

The development and production assumptions have been prepared on the basis of demonstrated performance, at Lamaque and elsewhere. Stope cycle times were estimated from a first principles build of activities and associated productivities:

- Lateral development: (as set out in Table16.7);
- Alimak raises: 120 m/month;
- Conventional raises: 60 m/month;
- Long-Hole stoping: from 145 t/day to 310 t/day depending on the stope width;
- Room and pillar stoping: 18 t/per man-shift, 11 hours per shift (day-night).

Table 16.7 – Lateral development assumptions

Type of heading	Projected Development Rate			
	First Month	Second Month	Third Month	Forth Month and more
Efficiency factor (learning curve)	40%	60%	80%	100%
Single face	70 m/ mth	105 m/ mth	140 m/ mth	175 m/mth
Double face	82 m/ mth	123 m/ mth	164 m/ mth	205 m/ mth
Triple Face	94 m/ mth	141 m/ mth	188 m/ mth	235 m/ mth
Multiple face (4 +)	108 m/ mth	162 m/ mth	216 m/ mth	270 m/ mth

16.8 MOBILE EQUIPMENT

The selection of underground mining equipment is based on the mining methods, drift and stope dimensions, production rate, operating costs and capital costs. Given the overall life of mine is 8 years, it was assumed that only new equipment would be purchased under financing agreements. The list of equipment considered for the Project is presented in Table 16.8.

Table 16.8 – Mining equipment for the Lamaque Project

Mining Equipment Triangle	Pre-prod (year)	Production (year)						
	2018	2019	2020	2021	2022	2023	2024	2025
Backhoe Loader	1	3	4	4	4	3	3	2
Bolter	7	9	9	9	7	4	3	0
Boomtruck	1	2	3	3	3	2	2	1
Cement Mixer	1	1	1	1	1	1	1	0
Emulsion Truck	2	2	2	2	2	0	0	0
Grader	1	1	1	1	1	1	1	1
Jumbo	6	6	6	6	4	2	1	0
LHD	8	12	12	12	9	9	5	3
Men Carrier	3	4	4	4	4	3	3	2
Scissor Lift	6	6	6	6	6	4	4	2
Service Tractor	14	16	16	16	17	16	13	12
Service Truck	2	4	5	5	5	4	3	2
Truck	4	6	7	7	7	7	7	4

16.9 SHIFT SCHEDULE

The mine will operate seven (7) days a week, night and day. This annual schedule is equivalent to 360 days per year of operation.

- Technical services and administration will be on a schedule of 5 days work – 2 days rest. Some workers from technical services will be on a 7 days work – 7 days rest to support development crews.
- Development and production crews will be on a schedule of 7 days work – 7 days rest – 7 days work (night shift) – 7 days rest, for 11 hours per shift.
- Room & pillar crews will be on a schedule of 7 days work – 7 days rest – 7 days work (night shift) – 7 days rest, for 11 hours per shift.

16.10 UNDERGROUND AND SERVICE MANPOWER

A local mining contractor was originally hired to undertake all the lateral development, the Alimak raises and for specific surface work during the pre-production period. In Q1-Q2 2018, Eldorado will gradually hire mine workers, to be largely selected from the contractors team, as the mine development progresses until it reaches the full employee complement. With the exception of Alimak raising, production drilling and definition drilling, Eldorado will employ its own mine crews to fill all the positions related to mine development, production, construction and mine services by the end of pre-production.

The manpower resources on each working shift used to prepare the following mine schedule includes:

- 2 crews of 2 men, working in cross-shift day and night, will be assigned to the room and pillar stopes;
- 6 long-hole crews (contractor)
- 7 truck drivers
- 12 LHD operators
- 4 development crews, for a total of 6 jumbos
- Each development crew consists of 1 jumbo operator and 2 workers for ground support and services; the truck operator is not included as part of the crew count.

Details of the manpower requirements for the duration of the LOM is listed in Tables 16.9 and 16.10.

16.11 TECHNICAL STAFF - ENGINEERING, GEOLOGY AND SURVEYING

The engineering and geology departments will provide the technical support to the mine operations. A senior mine engineer will be in charge of both technical departments. A team composed of senior and junior engineers, geologists, mine and geological technicians will be hired, along with engineering students for work terms. Surveying will be done by a dedicated surveying group who will split duties on a rotation. Geologists will maintain grade control and outline the valuable mineralization to keep dilution to a minimum.

Table 16.9 – Manpower requirements: Administration and surface services

	2018	2019	2020	2021	2022	2023	2024	2025
Direction and departement superintendant	14	14	14	14	14	14	14	10
Administration, purchase and IT	17	17	17	17	17	16	13	10
Human resources and training	11	11	11	11	10	10	8	7
Geology	8	8	9	9	9	7	5	3
Engineering	14	16	16	16	16	13	9	4
Environment	3	3	3	3	3	3	3	2
Surface	6	6	6	6	6	6	4	3
Maintenance	38	47	47	47	47	44	39	14
Mine	150	181	185	195	147	123	82	43
Mill	37	42	42	42	42	42	42	42
TOTAL	**298**	**345**	**350**	**360**	**311**	**278**	**219**	**138**

16.11.1 Ore production schedule

The underground mine plan provides for recovering 3,808,737 tonnes of ore grading 7.30 g/t Au.

Table 16.10 – Mine plan ore production distribution

Mining	C1, C2 & C3 Zones (Tonnes)	C4 Zone (Tonnes)	Total (Tonnes)	Percentage
Development	461,109	552,718	1,013,827	27%
Long Hole	1,098,062	1,474,031	2,572,093	67%
Room & pillar	222,818	- -	222,818	6%
Total	1,781,989	2,026,749	3,808,737	100%

The annual development quantities and the ore production by mining method are summarized in Table 16.11

Table 16.11 –Mining plan, yearly tonnage distribution

	Pre production				Production										
	Year 1				Year 2				Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	TOTAL
	Q1	Q2	Q3	Q4	Q1 Preprod	Q2	Q3	Q4							
Production	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Triangle															
Development (t)	-	27 303	47 491	38 415	37 473	42 100	45 548	39 243	185 260	247 790	198 231	91 154	13 819	-	1 013 827
Grade (g/t)	-	8,61	7,30	5,56	6,08	7,57	6,33	4,30	6,20	7,29	6,17	5,80	5,86	-	6,50
Long Hole (t)	14 850	14 850	31 010	36 568	51 038	51 038	65 824	94 901	351 735	287 802	335 149	448 992	508 141	280 194	2 572 093
Grade (g/t)	7,21	7,21	9,44	10,52	9,05	9,05	8,89	8,09	6,42	7,73	7,88	7,90	7,76	7,20	7,70
Room and Pillar (t)	-	-	-	-	-	-	6 480	6 480	25 920	25 920	51 840	51 840	51 840	2 498	222 818
Grade (g/t)	-	-	-	-	-	-	8,07	8,07	7,72	7,34	5,68	5,74	5,84	5,77	6,30
Total (tonne milled)	6 100	20 000	65 000	65 000	110 000	110 000	110 000	140 000	560 000	560 000	580 000	600 000	585 000	297 638	3 808 738
Grade (g/t)	7,21	8,12	8,14	7,98	7,83	8,31	7,86	7,03	6,43	7,52	7,10	7,38	7,54	7,20	7,30
Ore stockpiled (t)	8 750	30 903	44 404	54 387	32 898	16 036	23 889	24 512	93 638	108 033	137 709	105 588	89 503	27 108	-
Grade (g/t)	7,21	7,86	7,95	7,95	7,95	7,95	7,92	7,90	7,50	7,40	7,42	7,41	7,42	7,42	-
Mill Recovery (%)	93,8%	93,8%	91,2%	91,2%	93,8%	93,8%	93,8%	93,8%	94,1%	94,5%	94,8%	95,0%	95,1%	95,2%	94,5%
Gold Produced (oz)	1 326	4 895	15 515	15 200	25 963	27 583	26 066	29 677	108 880	127 987	125 646	135 335	134 858	65 577	844 507

16.12 MINING SERVICES

16.12.1 Ventilation

16.12.2 Fresh air demand

The fresh air demand for the Lamaque mine has been determined so as to satisfy the Quebec Regulation respecting occupational health and safety in mines (RROHS (en) / RSSTM (fr).

The fresh air required to dilute emissions from each unit of mobile machinery listed in Table 16.12, is specified by its (CANMET) engine certificate. In estimating the aggregate rate of fresh air flow for the entre mine, an utilisation rate has been applied to account for time when machines may be mechanically unavailable, or simply not in use. The utilisation rates are, respectively, 80% for production equipment, 60-75% for most service equipment, and 50% for machinery that functions primarily with electricity.

A contingency of 15% has been applied on the total estimated fresh air requirements to prevent for additional equipment that could be added during the life of mine. It also allows a quantity of air for potential leaks in the system.

Table 16.12 – Mobile equipment list with technical specifications.

Equipment		Model	Engine	Power		Quantity
				(HP)	(kW)	
Scoop 10yd	Cat	R2900G	CAT C15	409	305	3
Scoop 8yd	Cat	R1700	CAT C11	353	263	2
Scoop 6yd	Cat	R1600	CAT C11	279	208	2
Scoop 4yd	Cat	R1300	CAT 3306B	165	123	1
Scoop ELEC	Sandvik	LH514E	Volvo TAD1340VE	177	132	4
Truck 30T	Cat	AD30	CAT C15	409	305	2
Truck 45T	Cat	AD45	CAT C18	589	439	5
Jumbo 2 mats	Atlas Copco	M2C	Deutz TCD2013 L04	161	120	2
Jumbo 2 mats	Atlas Copco	S2C	Deutz TCD 2012 L04 2V, Tier 3	121	90	2
Jumbo 1 mat	Atlas Copco	S1D H	Deutz D914 L04	78	58	0
Bolter	Maclean	928	Mercedes OM 904=DD9043 MU32	147	110	0
Bolter	Maclean	SSB	Mercedes OM 904=DD9043 MU32	147	110	2
Bolter	Maclean	975	Mercedes OM 904=DD9043 MU32	147	110	2
Bolter	Cementation	SB-9	Deutz BF4M1013C	150	112	0
Boomtruck	Maclean	BT3	Mercedes OM 906	201	150	3
Cisor lift	Maclean	SL3	Mercedes OM 904=DD9043 MU32	147	110	4
Cisor lift	Maclean	SL 2.5	Deutz 2012-4	138	103	2
Cisor lift	Ciseau	CS3	Mercedes OM 904=DD9043 MU32	147	110	4
Blockholer	Maclean	BH3	Mercedes OM 904=DD9043 MU32	147	110	1
Cement truck	Maclean	TM3	Mercedes OM 906	201	150	1
Mining rescue truck	Maclean	BT-3	Mercedes OM 904=DD9043 MU32	147	110	1
Backhoe	CAT	420Fit	C4,4 Acert	124	92	3
Jeep	John Deer	5425	Toreon	50	37	1
Emulsion tractor	MineCat		Iveco N45	99	74	2
Tractor	Kubota	M5-111	V3800CR-TIEF4	106	79	7
Tractor	Kubota	L 6060	V2403	62	46	10
Bus (20 passengers)	Trottier	NA	Mercedes OM 904=DD9043 MU32	147	110	1
Bus (14 passengers)	Abiquip	NA	NA	147	110	1
Grader	Cat	MC100	135H Motorgrader	156	116	1

The mine fresh air demand was evaluated according to the equipment on hand annually and is presented in Table 16.13. The maximum demand of 606 kcfm (286 m^3/s) is expected to be reached in year 2022.

Table 16.13 – Airflow requirements for the C1 to C4 Zone

	2018		2019		2020		2021		2022		2023		2024		2025	
	(cfm)	(m3/s)	(cfm)	(m3/s)	(cfm)	(m3/s)	(cfm)	(m3/s)	(cfm)	(m3/s)	(cfm)	(m3/s)	(cfm)	(m3/s)	(cfm)	(m3/s)
TOTAL	337,190	159	452,550	214	518,290	245	524,540	248	526,865	249	447,155	211	444,830	210	284,025	134
Contingency 15%	50,579	24	67,883	32	77,744	37	78,681	37	79,030	37	67,073	32	66,725	32	42,604	20
TOTAL + 15% Contingency	387,769	183	520,433	246	596,034	282	603,221	285	605,895	286	514,228	243	511,555	242	326,629	154

16.12.3 Ventilation Infrastructure

16.12.3.1 Fresh air supply

The main fresh air intake consists of three parallel raises from surface to 346mL. The raise diameters are respectively; 5 m (except portion from 0 to 70mL which is 5.5 m), 3.43 m, and 3.43 m with escapeway. At 346mL a transfer drift will permit the fresh air to enter an internal network of raises upward into Zone C1.

From 346mL two raises (5 m, and 3.43 m with escapeway) bring the remaining fresh air down to 472mL. From 472mL two raises (3.43 m, and 3.43 m with escapeway) bring the remaining fresh air down to 688mL. The raise fitted with an escapeway connects every level.

16.12.3.2 Main Fans

The main fans were selected with the assistance of a Ventsim™ ventilation simulation model. Using the largest life-of-mine airflow value (606kcfm) as a constraint, the fan total pressure is estimated at 4.1 kPa (16.5 in. W.G). The most appropriate fan arrangement is considered to be two forcing fans in parallel, each with a nominal power of 1500 hp.

16.12.3.3 Heating system

The heating system consists of direct flame burners and has been sized to heat a total flow of 286 m^3 / s (606 kcfm) and a temperature differential of 40 ° Celsius which corresponds to the average maximum of the city from Val d'Or at a set point of 4° Celcius.

Mine air will be heated by natural gas with two burners, each rated at 29,646 MBTU/H, located upstream of the main fans.

16.12.3.4 Mine exhaust

The ramp (5.1m X 5.5m) will serve as the primary exhaust from the bottom of the mine to 472mL. Above 472mL, an exhaust air will split from the ramp via the addition of a 3.43 m diameter raise to surface (Figure 16.9).

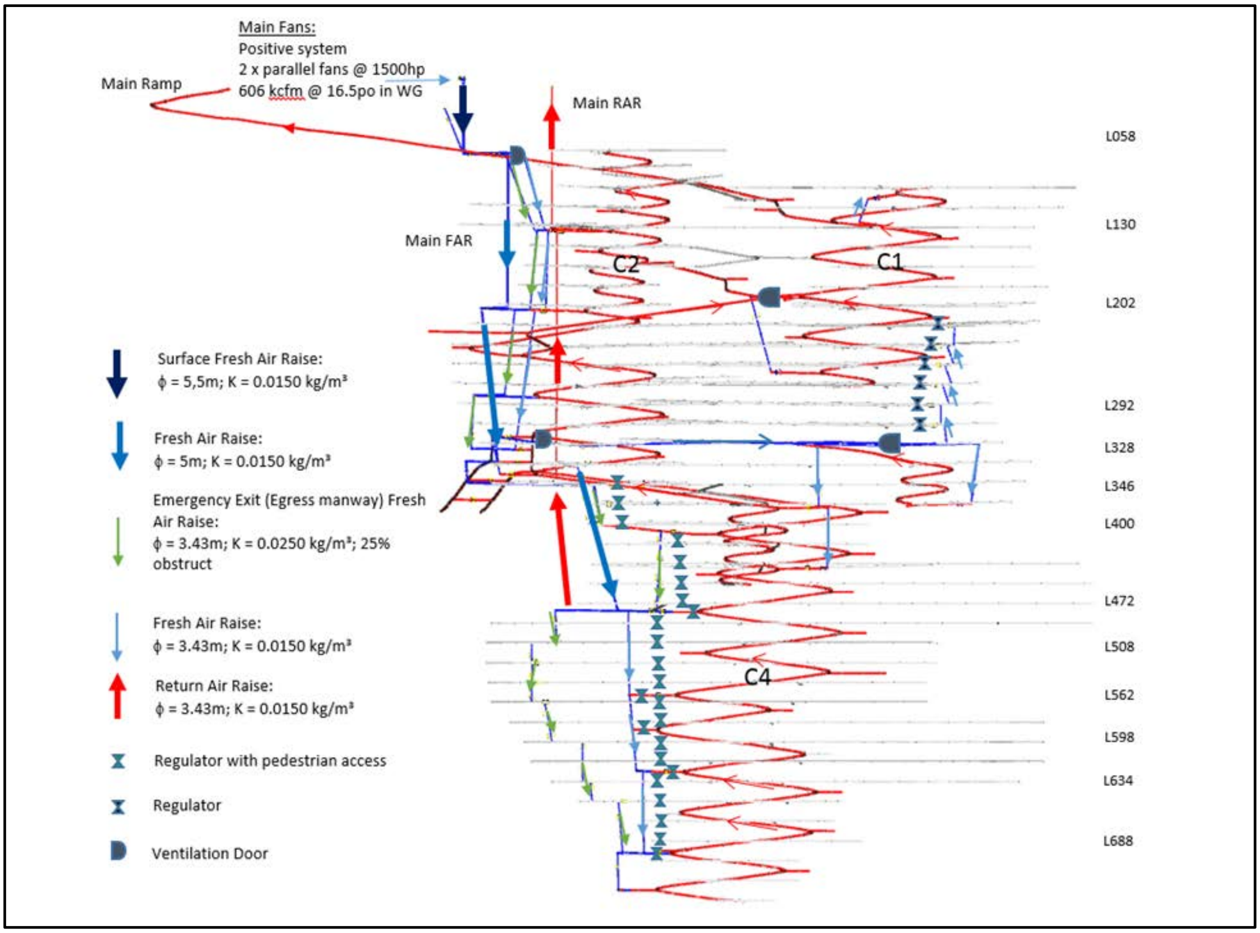


Figure 16.9 – Ventilation network overview (looking east)

16.12.4 Ventilation Network

The Lamaque deposit is currently distinguished by the following zones; C1, C2 West, C2 East, C3 and C4.

16.12.4.1 Zone C1

Zone C1 is ventilated by 150 hp auxiliary fans cascading on and off each sublevel from the ramp. Each fan will supply approximately 75kcfm (11 in W.G.) to each level. See figure 16.10 for typical level plan.

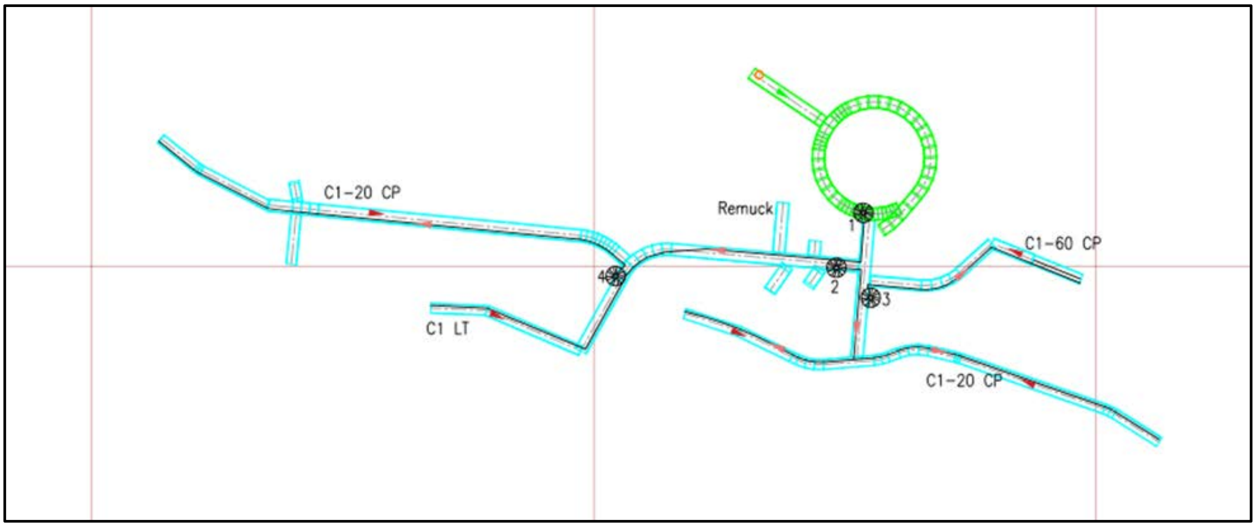


Figure 16.10 – Level plan of Zone 1 typical ventilation (forced ventilation)

16.12.4.2 Zones C2 and C3

The C3 and C2 (lower) Zones will be ventilated throughout the ramp as the main fresh air intake. A 350hp fan installed at the bottom of a 3.43m diameter raise will draw the air from the ramp. The Figure 16.11 presents the C3 and C2 ventilation system.

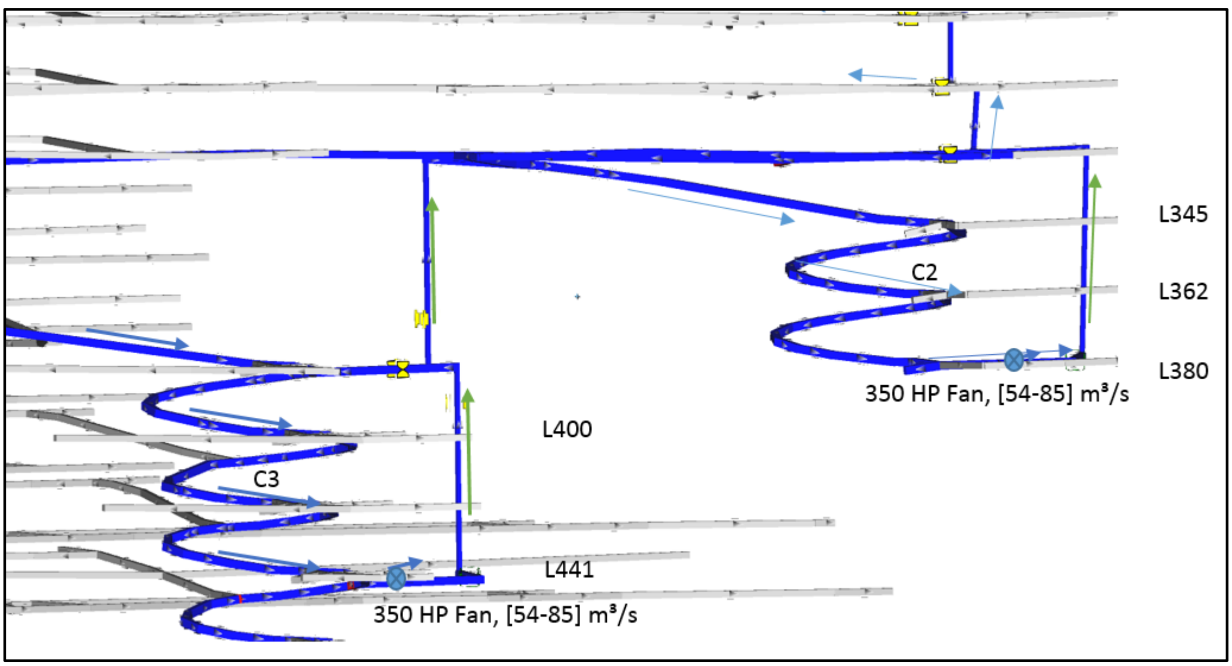


Figure 16.11 – Primary ventilation of zone C3 and part of C2 (lower)

The levels of Zone C2 and C3 are ventilated as zone C1, with forced ventilation system.

Zone C2 East features an internal vent raise (3.43m diameter) with regulators at each level. Regulators will permit to control fresh air entering each level according to actual demand (Figure 16.12).

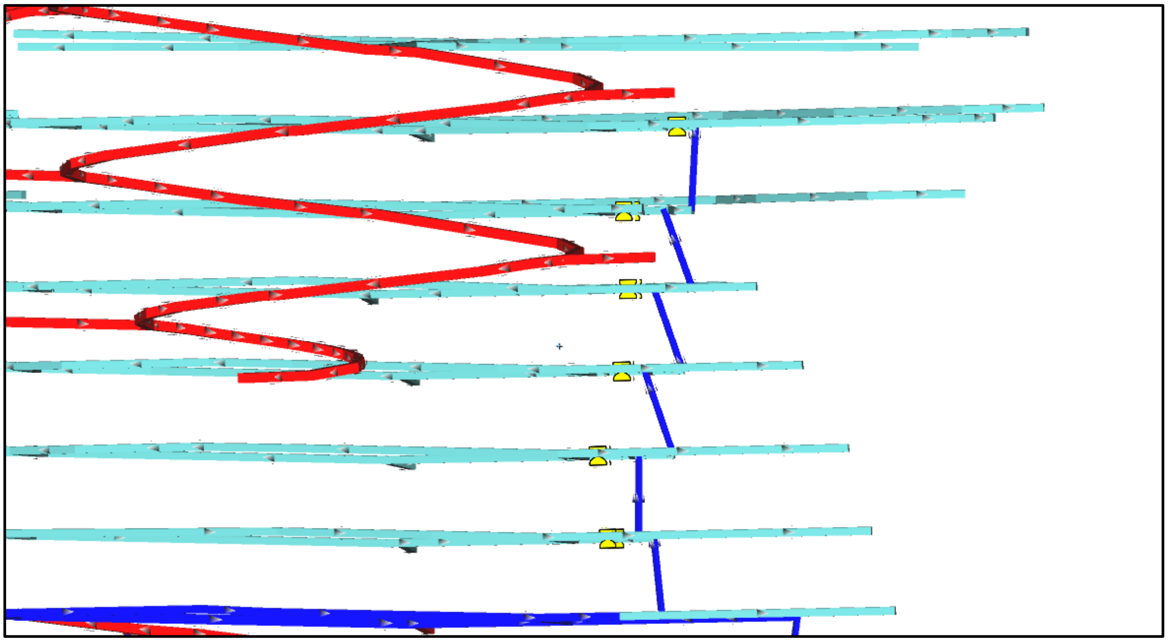


Figure 16.12 - C2 Est, ventilation system

16.12.4.3 Zone C4

Zone C4 is primarily ventilated by a raise located on the West side. Like zone C2 East, regulators will permit air flow control according to demand on each level. This raise has an escape way which is accessible on every level in the zone.

The Figure 16.13 shows the configuration of a typical ventilation arrangement during the production phase of the mine. The fresh air raise provides the fresh air on the level. A regulator located in the fresh air access controls the airflow according to the demand on the level.

Production area, stopes and West part of the level are ventilated by forced ventilation. Haulage drift will be equipped with permanent 48 inches rigid ducting and orebody zone will be equipped with a 48'' flexible duct.

150 hp fans are installed when an airflow of 76 kcfm is required and when the entire zone is necessary to be ventilated (for instance zone Est C4). 75 hp fans will be installed when two zones will be producing at the same time. A 25 hp fan will be installed when only one heading, or one equipment will be working simultaneously.

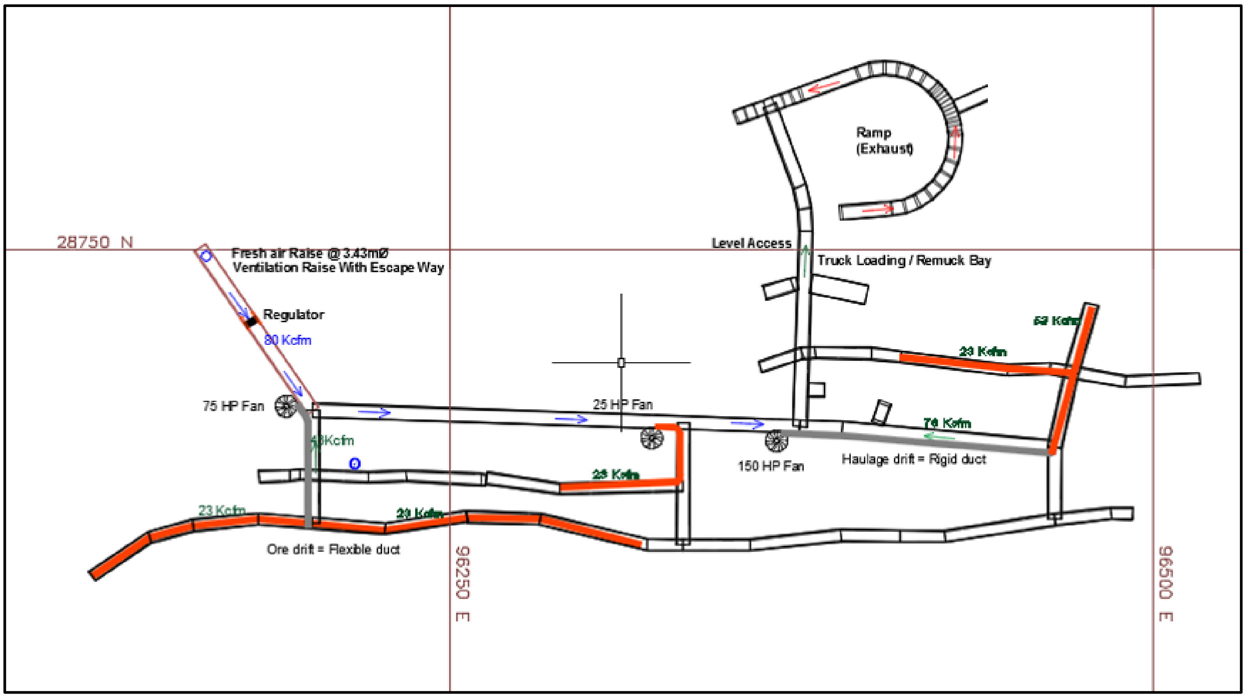


Figure 16.13 - Typical arrangement of the zone C4 during the production phase

16.12.5 Mine Dewatering

The design capacity for the dewatering system was estimated based on the water usage of the planned machinery and dust control when the production will be at his highest level. The seepage water estimation is based on the hydro geological study titled ''Integra Gold Corp. – propriété Lamaque, Étude hydrogéologique, Projet de mise en valeur souterraine de la zone Triangle'' (March 2015) (Table 16.14).

Table 16.14. Dewatering system - Main design criteria

Properties	AV.	Max.	Unit
Production	20	73	m3/h
Seepage water	63		m3/h
Total dewatering Capacity	83	136	m3/h
Solid content	0,3%	15%	
Total solid in the slurry	170		Dry t/ month

16.12.5.1 System description

The dewatering system will be deployed in two stages, the development phase during which the installation will be temporary the final system that will be in function for the mine life.

The following table shows the development identification, and the final identification of the pumping system equipment. The development skids will be integrated in the final system.

16.12.5.2 Development

For excavations, submersible pumps will be used at each level to pump into a skid-mounted tank. The skid will have one tank and a set of pumps in parallel. Each pump will be designed to pump the nominal estimated flow and the other will be a backup. During peak flow, both pumps can be simultaneously operated. The skid will follow the development faces, and the pumps, speed will be adjusted according to the static head. In some zone there could be several pumping skids in series.

16.12.5.3 Final system

The final system will use development skids in the deepest zones of the mines, which will pump water to the treatment stations. In the C4 zone three pumping steps will be required to pump from the 688 level up to the 418 level.

There will be two main water treatment stations: one the level 180, in the center ramp, and one at level 418 at the top of the C4 zone. These stations will remove solids from the water. The clarified water will be pumped by sets of multistage pumps (2X250 hp). The level 418 will pump to the clear water station of at level 160. The station at level 180 will also pump to the level 160, and the combined flows will be pumped to the mine Sigma, via an already installed 6’’ three km long HDPE pipe.

16.12.5.4 Water treatment stations

The water treatment stations consist of a charged water skid, who will receive untreated water from the 6’’ black steel pipe in the ramp. From this station the water will be pumped through flocculation. The function of this system is to facilitate the decantation and dehydration of solids by mixing between 1 to 5 gr of polymers per cubic meter of treated water. After, this step the water will be sent to one of the three dehydration bays, where the solid will settle. The clear water will flow through a filter wall, and it will end up by gravity in the clear water sump. During normal operations, the clear water will be pumped into a tank and from there to the multistage pumps set (2X250 hp units).

At nominal flow rate, it is expected that the settling bay will operate in a six (6) weeks three-phase cycle. For two weeks the settling bay will be in fill mode, and the polymerized solid charged water will be pumped in the bay. Once the bay will be filled, it will be at rest for two weeks in dehydration mode. Finally, there will be two weeks available for the clean-up procedure.

This buffer has been allowed for operational flexibility and also because of the uncertainty on the sludge volume that will be managed by the system.

The sludge removal from the bay will be done with a production scoop. During the clean-up, to avoid contamination, the clear water sump will be pumped to the untreated tank, the bay in filling mode will then be isolated with temporary barricades and pump so the clear water can be pumped directly at the clear water pumping station.

16.12.6 Compressed air

Compressed air supply is provided by two (2) 1,476 CFM electric compressors that had been installed temporarily when the underground development work was initiated in July 2016. There are plans to add 2 additional units that will be installed in the newly built compressor room. The two temporary units will then be relocated and installed alongside the others in the compressor room.

The compressed air piping network will be installed along the ramp, the main drifts and the escapeways throughout the mine. Compressed air shall provided power to pump for dewatering development work, handheld drills (for specific and limited use in planned development and, room and pillar stope) as well as provide an emergency supply of air to the refuge station.

16.12.7 Industrial Water

The industrial water supply for Triangle is temporarily provided by the MyLamaque shaft pumping station. The available capacity at Mylamaque is limited to 220m3/day.

At this time a clean water sump is in construction at Level 135 and will be available at Q1 2018. The sump will collect seepage water from upper part of the Triangle development (Level 70m) close to ventilation raise. The sump has a capacity of over 200m3 and water will be pumped to industrial water network to serve mining operation. The Mylamaque shaft will continue to be used for surface water protection and to serve the upper portion of Triangle.

16.12.8 Underground Explosive Storage

There will be two (2) different locations where explosives will be stored underground. One powder magazine is already in service on Level 50 where blasting agents and accessories are stored in two (2) separate rooms (Figure 16.14). This storage facility will be dedicated to serve the needs between Level 76 and Level 350. The capacity of these storage are 90,000 kg for explosives material and 80,000 detonators (caps).

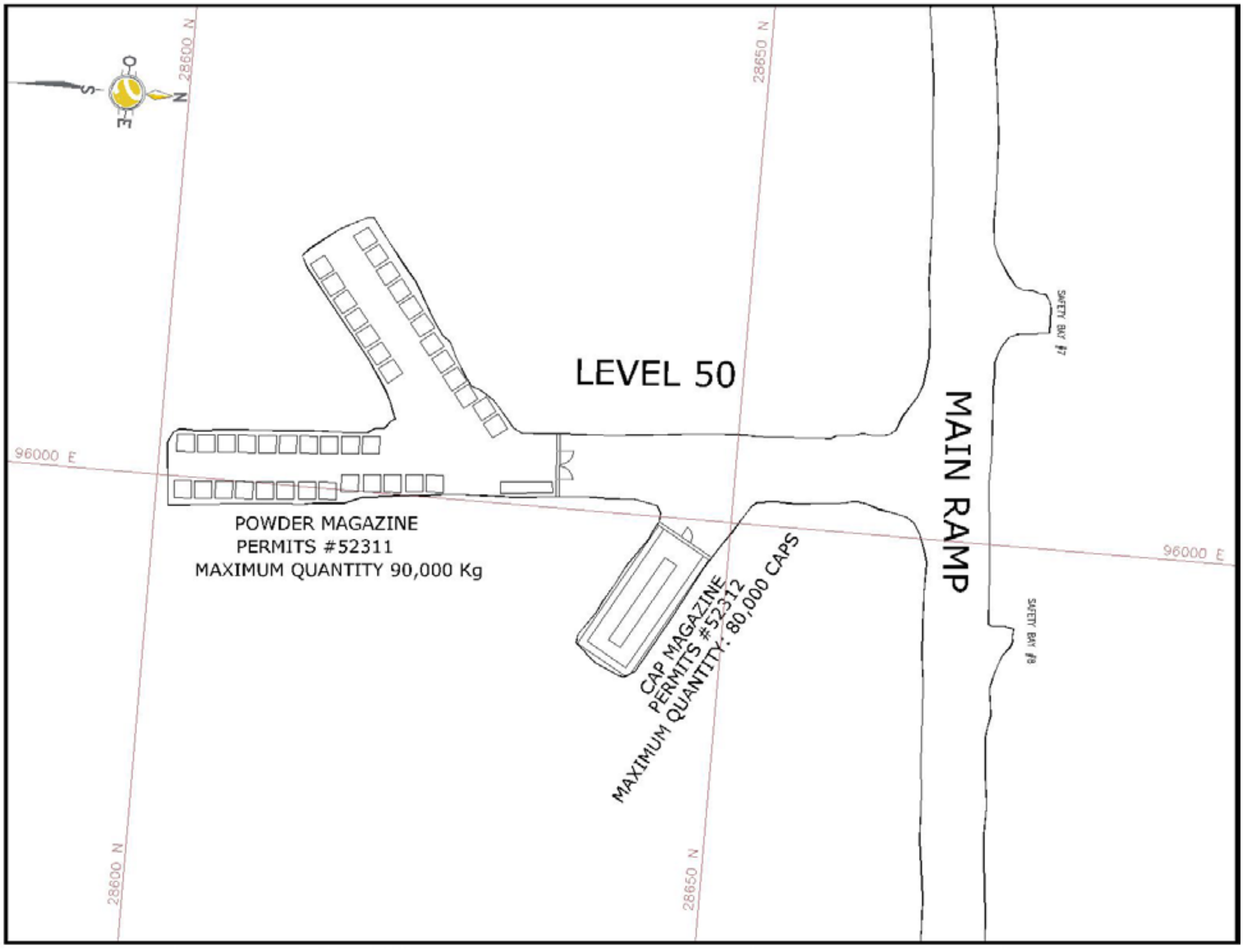


Figure 16.14 – Level 50 Underground Explosive Storage

16.12.9 Underground power distribution

One 4 160 V feeder (riser teck cable), through the portal, will be used to supply the first level underground substation (and the future substation to be located on the 400m level). One 13.8 kV feeder, through dedicated services hole from surface, will supply the lower levels of the mine. Junction boxes will be installed at each level to feed underground the main substations. The 13.8 kV and 4.16 kV feeder will ensure satisfactory voltage regulation throughout the ramp and raise, while minimising the size of conductors.

Underground main substations will be installed, in conformity with the Mine Code and Canadian Electrical Code (CEC), at each main level. Substations will provide 4.16 kV and 600 V power to underground loads such as pumps, ventilation, garages, 4.16 kV mobile substations, etc. All substation equipment will be located inside dedicated electrical room excavations along the main ramp.

A common modular diesel generator station will be installed near surface electrical substation sites to accommodate underground emergency loads (compressors, and pumps).

The total emergency load is estimated (under full mine operation) as indicated in the following Table 16.16.

Table 16.16 – Estimated Emergency Load

Description	KW
Compressors	300
Pumping	200
Ventilation	500
Total	1000

The generators will be installed in individual waterproof shelters and the 4,16 kV switchgear will be installed in a separate building along with control, protection, metering and batteries.

Generator grounding will be connected to the isolated grounding grid in order to keep a constant ground fault level.

One (1) above- ground fuel tank will be required, having a capacity of 10,000 litres. At full load, there will be enough fuel to supply all the generators for a period of 24 hours.

16.12.10 Underground Communication

16.12.10.1 System overview

The mine will be provided with a fiber optic and "leaky feeder" network to support all required communications to operate the mine and undertake data handling inside the mine and to the surface. The communication infrastructure will have all the capabilities to support voice communications, PLC monitoring and control, video, operation data and to control and monitor the electrical network. The communication network will be based on a communication infrastructure composed of two major technologies: a fiber optic cable backbone and a radiating cable technology. This communication network will meet all data transmission requirements and ensure redundancy and improved reliability for most of the Ethernet/IP applications. The network equipment will be specified to enable reconfiguration of the network without communication services outage as the exploration and operation shafts are being developed.

16.12.10.2 Design Standards

Unless otherwise specified, the design standards take into consideration the fact that communication equipment will be installed from the very first stages of the mine construction. The installed equipment must therefore be operational from the construction stage to the mine production stage. A provision has been made for installing temporary communications infrastructure.

Whenever possible, all systems must respect the following criteria:

- Reliability (redundancy implementation);

- Flexibility;
- Scalability.

16.12.10.3 Fiber optic backbone

The fiber optic cable backbone will be installed in the vent raise in a ring configuration wherever possible. Fiber termination point will be provided for each main electrical room in a communication cabinet.

RADIATING CABLE TECHNOLOGY (“LEAKY FEEDER”)

The radiating cable is primarily used for the radio mobile communication services but can also transmit all critical data communications in case of either failure of maintenance work of the fiber network. This technology also makes it very easy to move mobile, temporary and long-run temporary equipment, without modifying the configuration and programming of existing systems. This system will have the largest extent in the mine and will extend up to wireless sensors at the very end of each working areas.

COMMUNICATION SERVICES

Both the fiber network and the radiating cable technology are expected to facilitate the integration of the services listed in Table 16.17.

Table 16.17 – Communication Services and Cable Technology

Communication Services	Fiber Optic	Radiating Cable
Radio communication system		X
Interface to the surface communication system	X	
Surveillance and process camera network (High-Bandwidth)	X	X
Clock synchronization of the electrical protection relays	X	
Monitoring and control of the electrical system	X	
Control network for stationary and mobile equipment	X	X
Telemetry system	X	
Specific emergency frequency as well as dedicated phone to be used exclusively for mine rescue	X	X
Bridging to the surface radio link	X	
3-gas detection system (CO, NO2, O2) and LEL	X	X
Blasting system		X
Auxiliary fan control	X	X
Control circulation lighting	X	X
Geolocation of personnel and vehicles	X	X

EMERGENCY POWER FOR COMMUNICATION SYSTEM

It is essential that communication facilities be supplied by a reliable power source. Generally, communication services are heavily solicited during power outages, major events or emergencies.

Having access to different power sources to supply the control and communication systems improves reliability during critical operation. Among others, proper power autonomy is quite critical as the mine relies on communications to supervise and control electrical elements of the system.

The primary source is based on independent and isolated battery supply such as a double-conversion uninterruptible power supply (UPS). This equipment will additionally protect the communication systems from electrical grid problems and temporary power failures. Sizing of this power source will be determined by the equipment installed and the charge required to ensure at least one (1) hour of autonomy at full load with no assistance from another source.

A secondary power source should also be considered to improve reliability in case of a localized electrical issue that could affect the communication equipment in the event of extended power outage.

16.12.11 Mine Safety

16.12.11.1 Fire Prevention

Fire extinguishers would be provided and maintained in accordance with regulations and best practices at the underground refuge stations, electrical substations, pump stations, fueling stations, explosives magazines, and other strategic areas. Every vehicle would carry at least one fire extinguisher; the correct size and type would depend on the type of vehicle. Underground heavy equipment is equipped with automatic fire suppression systems (Ansul system).

16.12.11.2 Mine Rescue

A fully trained and equipped Mine Rescue Team is essential to the safe operation of any mine and is already in place at Triangle with its dedicated local and equipment. The mine rescue teams will be trained for surface and underground emergencies. A dedicated vehicle for mine rescue is planned in 2018.

16.12.11.3 Refuge Station

Self-contained portable refuge stations will be provided in the main underground work areas. The refuge stations are designed to be equipped with compressed air, potable water, and first aid equipment. They will also be supplied with a fixed telephone line and emergency lighting. The refuge chambers will be sealable to prevent the entry of gases. The portable refuge stations are planned to be moved to new locations as the work areas advance. Permanent refuge station have also been incorporated in the mine plan.

16.12.11.4 Emergency Egress

The main ramp is planned to provide primary egress from the underground workings. The fresh air raise (FAR) system with a dedicated manway would provide the secondary egress in case of emergency. The manway would be equipped with steel ladders and platforms.

16.12.11.5 Emergency Stench System

A stench gas system is installed on each fresh air intake and may be triggered to alert underground personnel in the event of an emergency. A new system will be installed on the main permanent ventilation system in 2018.

SECTION • 17 RECOVERY METHODS

17.1 INTRODUCTION

For the PFS, an average treatment rate of around 575,000 tonnes per year is anticipated, starting in 2020. It is planned that a portion of the ore mined in 2018 will be processed via toll milling agreements at the Camflo and Westwood mills, with processing at the Sigma mill starting in Q4 of 2018. For 2019, the plant will operate at a lower capacity, processing around 440,000 tonnes.

This section describes the process equipment available at the Sigma Mill as well as modifications that are planned based on the metallurgical testwork results. Modifications to the plant will be completed in two phases: modifications required for plant start-up in 2018 and additional changes to the flowsheet in 2019. The Sigma plant tailings will be sent to the tailings pond until Q3 of 2019 at which time the paste plant will become operational and from then on the tailings will be sent to the paste plant.

This section also discusses the estimated gold recovery considering the metallurgical testwork results obtained so far. A summary of the process design criteria, mass and water balance are then presented. A list of the major plant equipment is provided. Required plant personnel, energy, consumables and water are listed. Finally the plant layout is presented.

17.1.1 Sigma Mill Process Description and Flowsheet

The Sigma gold plant is situated at the East entrance of the city of Val-d'Or. This plant started operation in 1937. The plant capacity and the flowsheet were modified a number of times along the years. The flowsheet uses gravity concentration, cyanide leaching and carbon-in-pulp to recover the gold. The Sigma Mill simplified flowsheet, including modifications planned as part of this PFS, is provided in Figure 17.1. Preliminary detailed flowsheet are presented in Appendix X.

The simplified flowsheet shown in Figure 17.1 reflect the plant as it is planned to be after 2019, once all modifications have been incorporated.

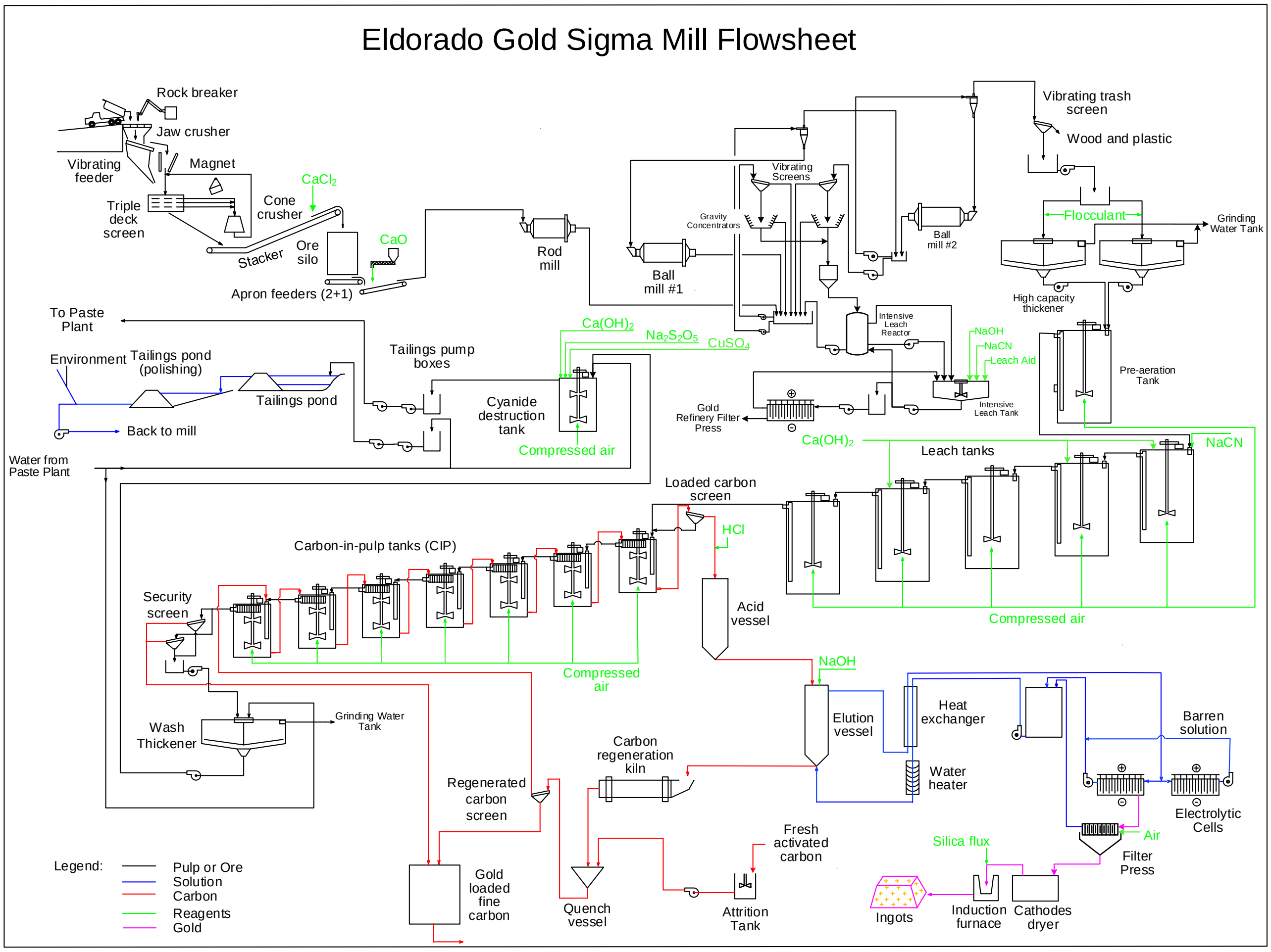


Figure 17.1 – Sigma Mill Simplified Flowsheet

17.1.2 Crushing Circuit

The ore from the mine will be trucked to the Sigma mill site 24 hours per day. The crushing circuit will however only be operating during a 12 hour day shift. The crushing circuit will be fed by both ore coming directly from the mine and ore from a stockpile at the Sigma mill site.

The plant feed is first dumped on a heavy duty grizzly screen, either directly by the truck coming from the mine or by a loader transferring ore from the Sigma mill site stockpile. An impact hammer breaks the oversize rock directly on the grizzly screen.

The crushing circuit is composed of a vibrating grizzly feeder followed by a Metso C110 jaw crusher. A HP400 cone crusher is used to obtain the final crush size and is installed in closed circuit with a triple deck screen, which will be replaced by a new one. The final screen product is transported by a belt conveyor to the fine ore bin for storage. In the winter, calcium chloride will be added onto the conveyor belt to prevent ore freezing.

17.1.3 Ore Storage

The mill, which at one time operated using a stockpile, is equipped with three apron feeders. In order for the fine ore bin to be able to feed two of the three existing apron feeders, rather than a single one as is currently the case, the bin will be moved slightly. The third apron feeder will nonetheless also be rehabilitated to be able to use it to feed the rod mill in case ore freezes in the silo, despite the use of calcium chloride. In such a case, ore from the crushing circuit will be diverted to an outdoor stockpile to feed the third apron feeder. The fine ore bin has an approximate live capacity of 2,300 tonnes.

A 100 tonne quicklime silo is located near the ore silo, above the ore silo discharge conveyor. It feeds solid quicklime directly onto the ore conveyor via a screw feeder.

A second 40 tonne quicklime silo will be installed near the main plant building. This silo will feed a lime slaker and milk of lime storage tank and distribution pumps. The milk of lime will be used for pH control in the leach circuit and pH control in the cyanide destruction tank.

17.1.4 Grinding Circuit

The grinding circuit is composed of a primary rod mill and two ball mills. The 9 x 12 ft, 400 hp rod mill runs in open circuit. The two ball mills are currently set up to run in parallel with both mill discharges going to the same common cyclone. The circuit will be rearranged to use the 11.5 x 14 ft, 1,000 hp ball mill as the secondary grinding step and the 12 x 14 ft, 1,250 hp ball mill as a tertiary grinding step.

The first ball mill will operate in closed circuit with the existing cyclone cluster which is composed of fourteen cyclones, six units of which would be in operation and eight in standby. For the first year of operation, a portion of the cyclone underflow will be sent to one of the existing two gravity concentrators, via the existing static screen, and the remainder returned to the first ball mill. In 2019, when a new gravity circuit will be installed, two new pumps will be connected to the combined rod mill/ball mill pump box to feed the new gravity recovery equipment. The entire cyclone cluster underflow will then be returned to the ball mill. The cyclone cluster overflow is sent to the secondary ball mill pump box.

For the secondary ball mill circuit, the pump box exists but new cyclone feed pumps must be purchased. In 2019, two additional new pumps will be connected to the secondary ball mill pump box to be able to feed one of the new gravity concentrators. A new secondary cyclone cluster will also be purchased and installed. The underflow feeds the secondary ball mill, and the overflow is sent to a vibrating trash screen.

Based on testwork conducted in 2018 by Bureau Veritas (refer to section 13.0), the addition of lead nitrate enables an increase in recovery especially for the C2 zone. For this reason, a new lead nitrate system composed of bag discharge hopper, mixing tank, transfer pump, storage tank and dosing pumps will be installed. Lead nitrate will be fed directly to the grinding circuit.

17.1.5 Gravity Circuit

The gravity circuit is currently composed of two static screens and two 20-inch Knelson gravity concentrators operating as parallel units to recover free gold. For the first year of operation, the XD20 unit will be rehabilitated and the CD20 unit dismantled. The Knelson concentrate will be treated on the existing shaking table. The table concentrate is then further processed in the refinery.

In 2019, two new 20-inch gravity concentrators will be installed each preceded by a new vibrating screen. One gravity concentrator will be installed in the primary ball mill circuit while the other will be installed such that it can be fed by either the primary or the secondary ball mill circuits. The old XD20 gravity concentrator will be dismantled at that time. As mentioned previously, the new gravity recovery equipment will be fed directly from the ball mill pump boxes as the existing plant height does not allow installation of all equipment underneath the cyclones. For the time being, it is planned to install the new gravity equipment in a building extension to the grinding building.

A new intensive leach reactor skid will also be installed in 2019 to treat the gravity concentrate. This includes a concentrate feed tank, the intensive leach reactor, an agitated solution tank as well as solution and slurry pumps.

17.1.6 Thickening

The trash screen underflow is pumped to the thickener feed box. A sampler and particle size monitor are installed on this line. The plant is equipped with two 30 ft diameter high rate thickeners. It is assumed that both will need to be rehabilitated although based on thickening test results, for a production rate of 575,000 tpa, one might be sufficient. Operation at lower tonnage the first year will enable us to confirm thickener performance. Flocculant is supplied to the thickeners by a flocculant preparation system.

17.1.7 Leach Circuit

After thickening, the slurry will go to a new agitated pre-aeration tank before being sent to the leach circuit where cyanide is used to dissolve the gold. The circuit is composed of five tanks for a total of approximately 5,100 m^3 of active leach volume. The pre-aeration tank will be the same size as the leach tanks. Slurry flows from one tank to the other by gravity. Every tank can be by-passed to allow maintenance on any given tank. Each tank will be equipped with an agitator mechanism and compressed air lines. Currently, one agitator is missing and needs to be replaced. As mentioned before, the plant will be equipped with a new milk of lime system, which will enable fine tuning of the leach circuit pH.

17.1.8 Sodium Cyanide

The plant is currently equipped with a sodium cyanide receiving and storage tank and sodium cyanide dosing pumps. They are still in good condition but are located in the middle of the plant with no separate containment area and fairly close to the diluted HCl tank. In order to prepare the plant for potential compliance to the International Cyanide Management Code, Eldorado are moving the sodium cyanide tank to a new annex of the mill building where it will have its own containment area and access to the area can be restricted. A new truck delivery pad with containment will also be added. The same pad will be used for both sodium cyanide and sodium hydroxide deliveries.

17.1.9 Carbon-in-Pulp Circuit

The discharge of the leach circuit first flows through a sampler. It then goes on to the carbon-in-pulp (CIP) circuit, composed of one larger 280 m^3 tank and six smaller 170 m^3 tanks. Four of the smaller tanks will need to be replaced due to their poor condition. The slurry goes from one tank to the other by gravity. Every tank can be by-passed to allow maintenance on any given tank. Inter-stage screens prevent carbon from being transferred with the slurry. These screens will also be replaced, the existing ones being in poor shape. All tanks are equipped with agitators. As part of the CIP circuit rehabilitation, all tanks will also be re-equipped with compressed air lines and air distribution cones.

Carbon is pumped counter-current to the slurry using existing vertical pumps. Fresh carbon is fed to the last tank via the regenerated carbon vibrating screen which removes carbon fines prior to feeding the carbon to the CIP circuit.

After going through the CIP circuit, the slurry proceeds to two parallel safety vibrating screens to recover any smaller carbon particles that may have passed through the inter-stage screens. The screen underflows feed into a pump box whereas the overflows, which contain fine loaded carbon, are loaded into bags.

The combined volume of the leach circuit and CIP tanks is about 6,400 m^3. At 575,000 tpa, this provides a residence time of about 66 hours.

17.1.10 Elution and Carbon Regeneration

Loaded carbon is pumped from the first CIP tank onto a screen, which returns the underflow slurry to the same tank. Carbon is screened out and then drops into a bin prior to the acid wash column, whose purpose is to eliminate carbonates fouling the carbon. Hydrochloric acid is used for the acid wash.

From the acid wash, the carbon goes into a 3 tonne capacity pressure elution column where a high temperature, high pressure sodium cyanide and caustic solution desorbs gold from the carbon. The elution solution is pre-heated by the hot pregnant solution via a heat exchanger and then further heated using an electric heater.

Carbon from elution is regenerated in an electric rotary furnace, cooled in a quench tank, and returned to the regenerated carbon screen which feeds the last CIP tank. As required, fresh carbon will be fed to a new agitated carbon attrition tank and pumped to the CIP circuit via the regenerated carbon quench circuit. Carbon fines collected from the regenerated carbon screen, the kiln feed dewatering screen and other carbon transfer waters are collected in two fine carbon tanks, one overflowing into the other. A new fine carbon filter press will be added to process the tank underflows. This fine carbon will then be sold.

17.1.11 Hydrochloric Acid and Caustic Soda

The plant is equipped with two hydrochloric acid (HCl) tanks: one for receiving of the concentrated HCl and one for dilute acid at the concentration used in the acid wash. A new truck delivery pad with containment will be added for acid delivery. This pad is on the other end of the plant from the sodium cyanide delivery pad to avoid any possible contact between the two chemicals.

The existing caustic soda tank will remain where it is currently located. As mentioned before, trucks delivering the caustic soda will use the same truck delivery pad as for sodium cyanide.

17.1.12 Refinery

The cooled pregnant solution from elution is sent to the electrolysis cells located in the refinery. There are two electrowinning cells running in parallel. Gold forms on steel wool cathodes. A fan is used to evacuate fumes from the electrowinning cells.

Gold is removed from the cathodes in a wash booth and the resulting gold sludge is fed to a filter press. The filter cake will be dried in a new calcining oven and then melted with flux in the induction furnace. A dust collector is used to treat the off-gas from the induction furnace. Initially, the drying oven and furnace will also be used to treat the shaking table concentrate. The doré ingots are stored in a vault.

In 2019, a third electrolysis cell will be installed in the refinery to process the solution from the intensive leach reactor skid. For this the filter press must be moved. This electrolysis cell uses basketless cathodes that can be washed in-situ. The gold sludge will thus be pumped directly from the electrolysis cell to the filter press.

17.1.13 Cyanide Destruction and Related Reagents

In 2019, tailings from the safety screens underflow pump box will be fed to a new cyanide recovery thickener (wash thickener). In this thickener, CIP tailings will be mixed with part of the water returned from the paste plant. The thickener underflow will be pumped to cyanide destruction. The overflow will go to the grinding water tank. In the cyanide destruction tank, the thickener underflow will be rediluted using water from the paste plant as cyanide destruction is more efficient at lower solids concentrations.

Before the paste plant and cyanide recovery thickener are put into service, CIP tailings from the safety screens underflow pump box will be sent directly to the cyanide destruction tank. There, reagents and air are used to reduce cyanide concentrations to environmentally acceptable levels.

Sodium metabisulphite will be used as the SO_2 source. New mixing and distribution tanks as well as transfer and dosing pumps are required for this reagent. A new plant extension will be built for reagent storage and to house the cyanide destruction reagent preparation equipment. The plant is already equipped with copper sulphate tanks but these will be moved to the new building section. New copper sulphate transfer and dosing pumps are required. As mentioned before, the plant will be equipped with a new milk of lime system which will be used for cyanide destruction pH control.

17.1.14 Tailings

The plant is equipped with two tailings pump boxes each connected to two tailings pumps in series with a third spare set of two pumps in standby. When the plant will initially start up, tailings from cyanide destruction will be sent to the tailings pond via one or the other pump boxes.

Once the paste plant is in operation, one of the pump boxes will be used to send the cyanide destruction tailings to the paste plant while the other will be used to return excess paste plant thickener overflow to the tailings pond.

17.1.15 Water services

The plant is equipped with two water tanks. The grinding water tank collects water from the pre-leach thickeners and eventually the cyanide recovery thickener. Currently the recirculated water tank collects water from the tailings pond recovery basin (locally called Lake Tremblay) and city water is used as make-up.

A Sigma Mine underground water line will be connected to the recirculated water tank to provide fresh water to the plant. This line will become the main water supply to the tank once the paste plant goes into operation and tailings are no longer pumped to the tailings pond. The recirculated water tank provides make-up to the grinding water tank when required.

17.1.16 Air services

The plant was previously equipped with two high pressure air compressors for both plant air and leach requirements, as well as a third spare diesel compressor. These compressors will be replaced as more air is required for pre-aeration and cyanide destruction. To meet the pre-aeration, leach, CIP and cyanide destruction air requirements, it was decided to install two low pressure compressors in operation and one stand-by unit. In addition, one duty and one stand-by high pressure air compressors will be installed to supply plant air to both the concentrator and paste plant.

Instrumentation air is supplied by two separate existing air compressors (one operating, one stand-by) connected to a common air dryer. A second air dryer is located in the crushing section, fed by plant air.

17.2 WATER BALANCE

The preliminary plant water balance is presented in Figure 17.2. Once the paste plant is in operation, all water removed from the tailings in the paste plant thickener and filter, as well as seal water used within the paste plant, will be returned to the Sigma mill. However only a portion of this water can be used by the mill, thus the excess amount will be sent to the tailings pond.

The mill also requires an average of around 27 m^3/h fresh water for reagent mixing, gland water, gravity concentrators, intensive cyanidation, loaded and regenerated carbon screens as well as elution and carbon regeneration. Water from the underground Sigma mine will be used for this purpose. The Sigma mill is also connected to city water which will be used for sanitary purposes and safety showers, and serves as back-up for fresh water.

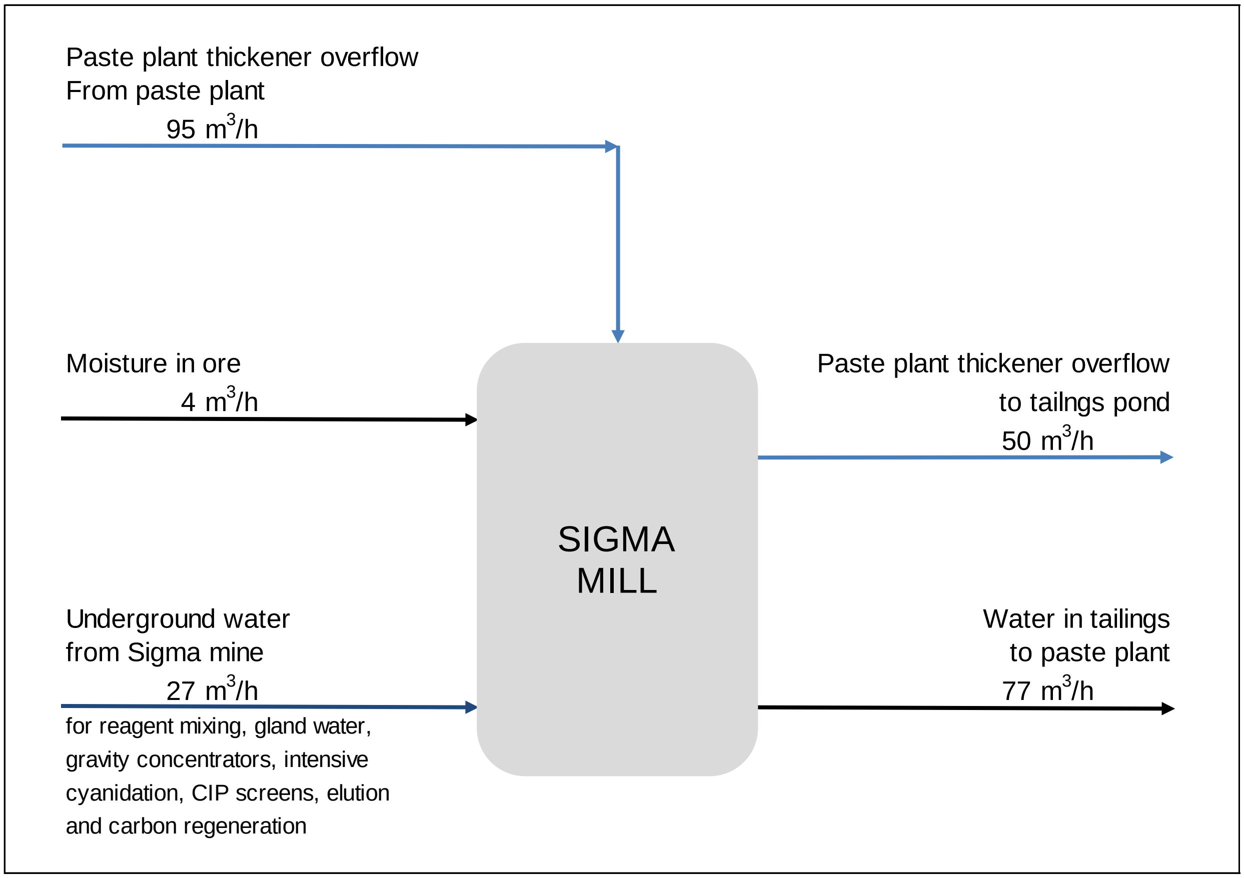


Figure 17.2 Preliminary Sigma Mill Water Balance

17.3 PLANT PERSONNEL

The list of the planned plant personnel is provided in Table 17.1.

Table 17.1 Planned plant personnel

Description	Number of persons
Staff	
Mill superintendent	1
Metallurgist	1
Metallurgical technician	1
Operations supervisor	1
Mechanical supervisor	1
Electrical supervisor	1
Surface Supervisor	1
Mechanical planner	1
Reliability engineer (shared with	0.5

Description	Number of persons
Subtotal staff	8.5
Operations	
Grinding operator	4
Solutions operator	4
Crushing operator	2
Reagents operator	2
Refiner	1
Loader operator	1
Back-up operator	4
Tailings pond operator	2
Subtotal operations	20
Maintenance	
Mechanic	6
Electrician	5
Carpenter	2
Janitor	1
Subtotal maintenance	14
TOTAL	42.5

17.4 POWER, REAGENTS AND CONSUMABLES

17.4.1 Power

Power consumption for the plant has been estimated in three parts: grinding power, existing equipment power and new equipment power. Grinding power has been estimated based on average rod and ball mill work indexes. Historical power consumption has been used for the time being to estimate power consumption of existing equipment. Power consumption of new equipment has been estimated using the load list, taking into account expected utilization factors and subtracting any items that are replacing existing equipment whose loads would thus already be included in the historical power consumption.

Based on this method, average grinding power consumption has been estimate at 22.4 kWh/t and average total power at 56 kWh/t.

17.4.2 Reagents and Consumables

Estimated consumption rates of reagents and consumables are presented in Table 17.2. It should be noted that for the C4 zone, the use of lead nitrate has not been considered since tests results showed similar recoveries with and without the use of lead nitrate.

Table 17.2 Consumption of reagents and consumables

Reagent or Consumable	Unit	Consumption	
		C2 zone	C4 zone
Grinding media (rods)	kg/t	0.40	0.39
Grinding media (1st ball mill)	kg/t	0.88	0.85
Grinding media (2nd ball mill)	kg/t	0.95	0.91
Sodium cyanide (100% NaCN)	kg/t	0.64	0.74
Lime (CaO)	kg/t	4.24	3.62
Lead nitrate ($Pb(NO_3)_2$)	kg/t	1.50	-
Flocculant	kg/t	0.04	0.04
Carbon	kg/t	0.053	0.053
Hydrochloric acid (HCl)	kg/t	0.17	0.17
Caustic soda (NaOH)	kg/t	0.079	0.079
Leach aid (for intensive cyanidation)	kg/t	0.0013	0.0013
Calcium chloride	L/t	0.25	0.25
Scale inhibitor	kg/t	0.04	0.04
Sodium metabisulphite (Na2S2O5)	kg/t	2.13	2.13
Copper sulphate ($CuSO_4.5H_2O$)	kg/t	0.06	0.06

17.5 PLANT LAYOUT

The layout of the main plant buildings, excluding the crushing and ore storage areas, is shown in Figure 17.3. As can be seen on the drawing, a new building extension is currently planned to the north of the grinding building for the new gravity concentration circuit. The lead nitrate mixing and distribution tanks are planned to be installed in the former SAG mill building, next to the grinding area.

Building extensions are planned to the south-west of the main mill building for the cyanide tank and the new milk of lime system. To the south, the current mill compressor building will be extended to house the three new low pressure process air compressors. To the south-east, a building extension is added for reagent storage and for cyanide destruction reagents mixing and distribution tanks (sodium metabisulphite and copper sulphate).

The new cyanide recovery thickener is located to the south-east of the main plant building, near the new cyanide destruction reagents building. The new pre-aeration tank will be installed next to the existing leach tanks.

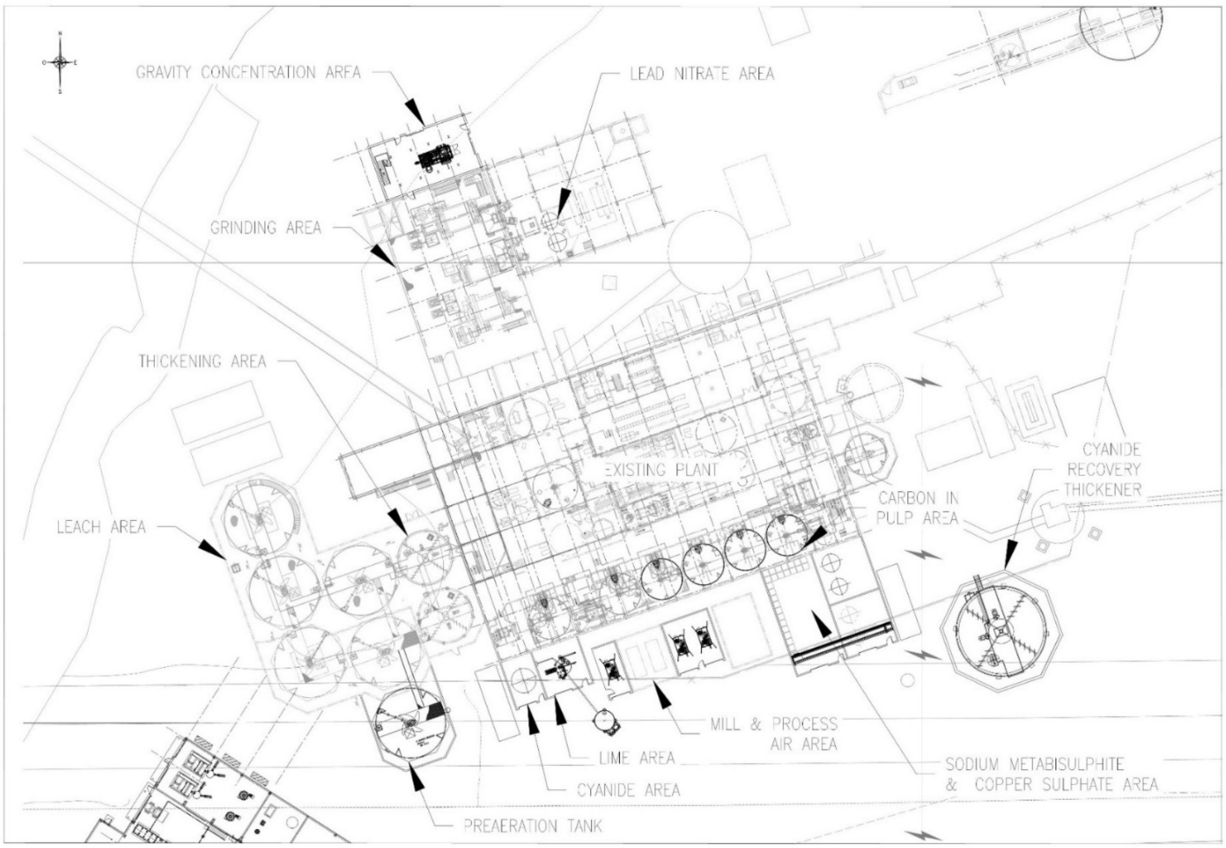


Figure 17.3 Plant Layout

SECTION • 18 PROJECT INFRASTRUCTURE

18.1 OVERVIEW

Eldorado's acquisition of the Sigma-Lamaque Complex in 2014, provided the Company with a permitted 2,200 tpd milling complex and tailings facility adjacent to the Lamaque South Property, as well as permitted underground infrastructure that includes three portals, a mechanical workshop, offices, a dry, equipment, and all mining concessions and mineral claims on the past-producing property. However, it must be noted that none of the underground infrastructure that were mentioned in the 2017 PEA will be considered in this PFS.

In early 2015, Eldorado (formerly operated by Integra Gold Corp) had initiated the construction of the mine at the Triangle deposit (Triangle) to support the bulk sample program proposed in the 2014 PEA. The Triangle site facility is still under construction but is nearly 65% complete. The facilities include road access, underground access (portal), workshops, warehouse, office, mine dry and various service facilities. An overhead power line, fresh and waste water conduits as well as natural gas line supply has been brought to Triangle site.

At Triangle, $7.5M CAD worth of infrastructure was put in place during the summer and fall of 2015 to develop an underground exploration ramp and conduct a bulk sampling program.

Land clearing, road construction and site preparation were carried out in the summer of 2015. A 25 kV power line (Figure 18.1) was erected between the Sigma site and Triangle. A pipeline (two 6" HDPE pipes) was also built to transport water to the Sigma mine for the dewatering of the Triangle underground workings. Another pipeline (two 3" pipes) at the Triangle site was set up for the potable water supply and wastewater disposal. The final connection to the electric grid and the Val-d'Or water network was completed in 2016.

A modular building that will serve as an office and dry was set up in the fall of 2015 on the Triangle site. A garage (17m x30m) was built for equipment maintenance and to serve as a warehouse (Fig. 18.2).

A portal (Figure 18.3) was excavated to accommodate a ramp 5.1 meters wide by 5.5 meters high.

In 2017, Triangle site was expanded to support the future production phases. Modular buildings were added to bring dry capacity from 100 to 200 workers. As of Q1 2018, another extension to the dry facility is currently in progress which will extend capacity to 400 workers by adding extra modular buildings. A modular building to serve as both a technical services and administration office was added in 2017 as well as a number of other buildings to serve for health and safety, training and surface personnel. During Q3 2017, an expansion of the garage-ware-house building was built to add 2 services bay, 1 wash bay, to triple warehouse capacity and to add office space for maintenance personnel. This building is currently in the completion phase .A Megadome building was added in Q1 2018 to serve as garage for underground transport vehicle in order to facilitate transport of personnel at shift change . During 2017, a temporary heating unit building was built over the escape way at surface with the heating system to serve for underground ventilation until installation of the permanent system in 2018.

Figure 18.1 – Triangle Zone access road construction and power line installation in 2015 (credit: Integra)

Figure 18.2 – Expanded Warehouse-Garage at Triangle, February 2018(credit: Integra)

Figure 18.3 – Construction site: Triangle Zone portal and surface infrastructure in 2015 (credit: Integra)

18.2 CAMP

Given the proximity to Val-d'Or, provision for workforce housing accommodation or transportation arrangements has not been considered.

18.3 SITE ACCESS

The Lamaque Project is located approximately 3 km south of Val-d'Or. The Sigma-Lamaque Complex encompasses the Triangle mine site and the Sigma mill and tailings facility all accessible via the national Highway 117. The Triangle site is accessible using the existing Goldex-Manitou access road which was improved in 2017 to allow heavier traffic to drive through.

18.4 MINE SITE ENTRANCE, GATE HOUSE AND PARKING LOT

There are two (2) security entrance gates controlling personnel, supplies and visitors accessing the property. One gate is located close to the Sigma mill facility and the other one is located at the Triangle mine site.

The parking lot at Sigma site can accommodate 85 personal vehicles while the one at Triangle is laid out to accommodate 220 vehicles.

18.5 TRIANGLE INDUSTRIAL SITE AND LAYDOWN AREA

The construction of the Triangle mine site was initiated in June 2015. Preparation of the laydown area required the grubbing of 9.7 Ha where trees were cut down and top soil removed and stored on overburden piles for future use at the end of the mine operation. A network of ditches was excavated around the perimeter and within the laydown area to provide adequate site drainage and water management control. Each section of roads crossing a natural stream or a ditch was outfitted with a culvert of the appropriate size.

A network of internal local roads has been built, from the Triangle site, to access surrounding service facilities. Ore will be transported from Triangle ore pad to the Sigma mill through 9.1 km off-road circuit. Figure 18.5 and 18.6 provides an overview of Triangle existing and final layout.

The plan as outlined in the PFS will have minimal impact on the community as there are no homes, businesses or other infrastructure where the proposed mining will take place.

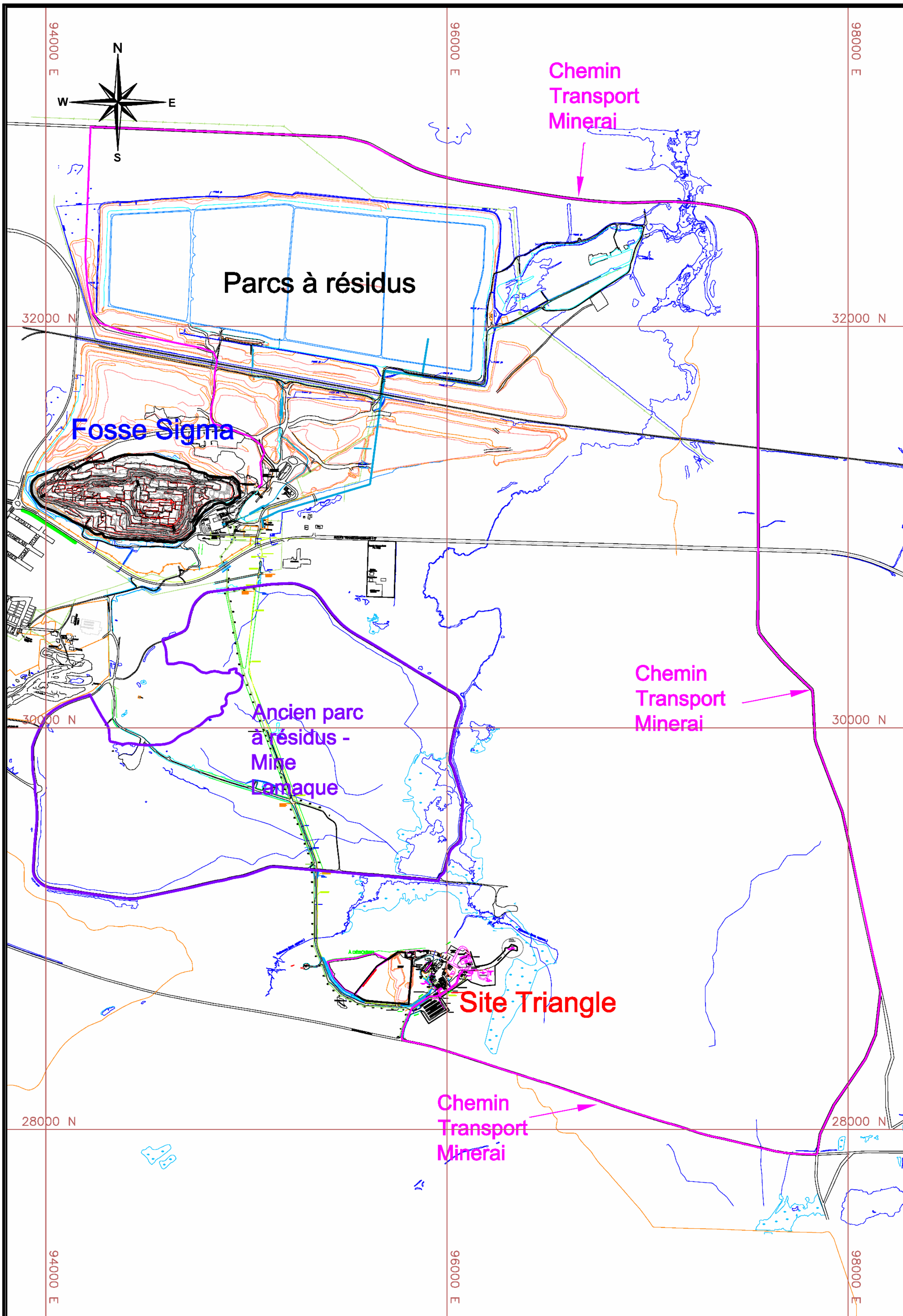


Figure 18.4 – Surface view of the Sigma-Lamaque Complex.

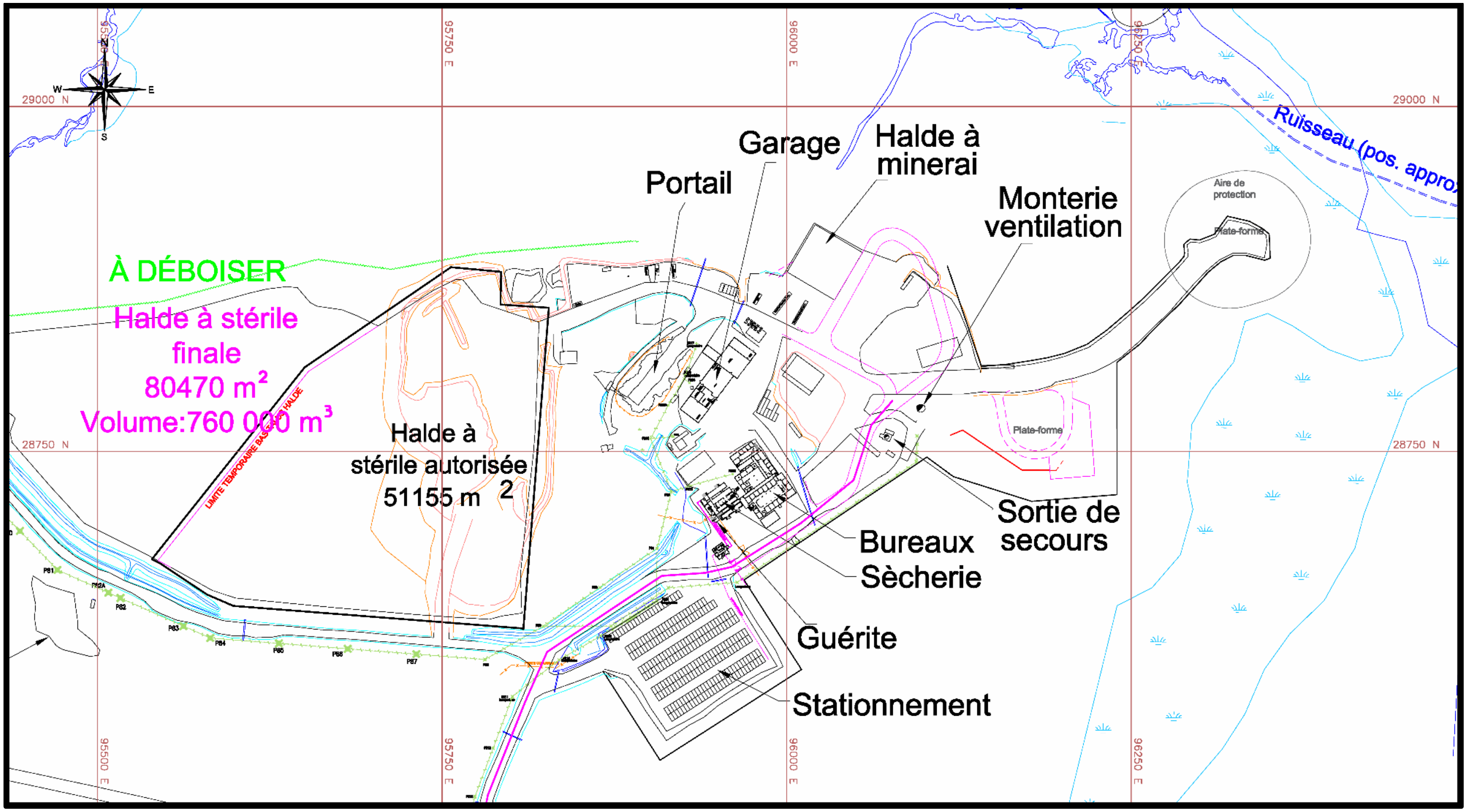


Figure 18.5 – Actual Surface layout of Triangle Mine Site.

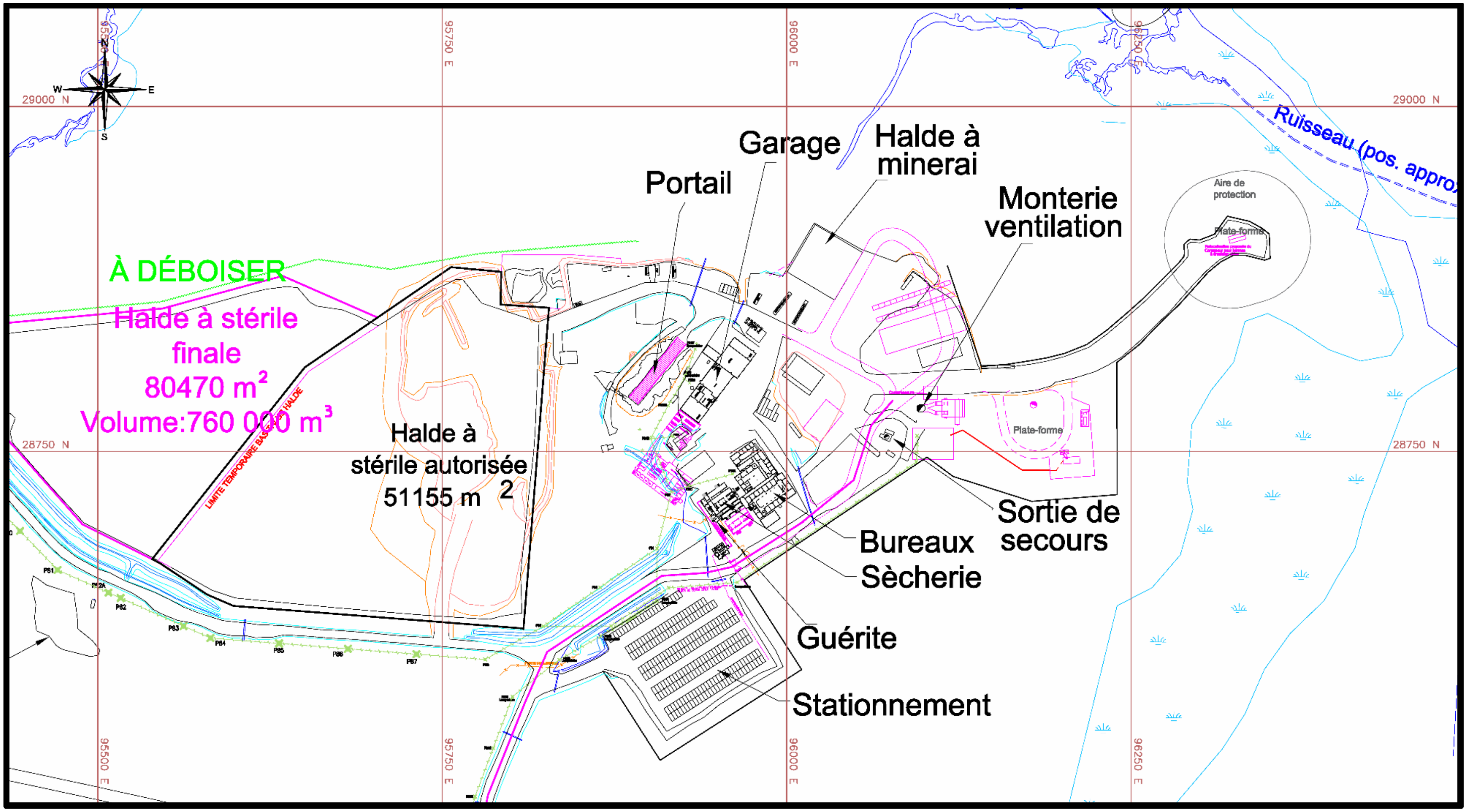


Figure 18.6 – Final Surface layout of Triangle Mine Site (pink to build)

18.6 SIGMA-LAMAQUE PROPERTY BUILDINGS

The Sigma-Lamaque property already has several existing buildings that were erected or renovated to service the future needs of the Triangle project. The Triangle mine site consists of various services buildings that were used to shelter project management crew during the ongoing construction and underground development work. At the time of writing this report, quotations have been received from contractors for key buildings like the administration building, core logging facility and the surface ventilation power house which should conclude the infrastructure construction plan, Once the construction phase is complete, some of the temporary buildings will be removed and some will remain and be part of the final mine infrastructure layout.

18.7 TRIANGLE SITE BUILDINGS

The following building are built at Triangle mine site (Figure 18.6):

- A garage (16. 7 m by 60 m) with 6 working bay, a warehouse, a compressors room and offices to serves maintenance and procurement teams;
- An office for mine department (18 m by 14.6 m) made of prefab modules;
- An office for technical services and admin (18 m by 27 m) made of prefab modules;
- An office for health and safety, training, construction (2 x 18m x 6m) made of prefab modules;
- A mine rescue local (18m x 6m), made of prefab modules;
- A dry-house for 400 men and 25 women built with eight prefab modules of 3.6m by 19.5 m.
- A fabric building (19.5 m by 9.8 m) used has a cold storage
- A fabric building (30 m by 15 m) heated and used as parking garage

The following building shall be built and put into service by the end of 2018:

- A new fabric building will be installed to serve as cold storage.
- The underground service compressors will be relocated in a new building constructed next to the main ventilation raise.
- A new core logging house (12.2m by 12.2 m).

The following building shall be built and put into service by the end of 2020:

- Administration and service building having 1,755 m2 spanning on two floors for Eldorado's staff. This includes a dry facility that can accommodate 400 workers including a separate area for 30 female workers. The actual dry facility will be kept in service and dedicated to cater contractors needs for the remaining LOM

18.8 FUEL STORAGE AND EXPLOSIVES

Diesel and gas stations are already present at the Sigma-Lamaque Complex to service small vehicles. Diesel fuel for the mine equipment and vehicles is stored in 50,000-litre above-ground tanks installed near the portal entrance.

In the future, two other reservoirs (20,000 L and 4,000 L) will be install near the ventilation raise of Triangle for fuel distribution to a 9,600 L reservoir locate underground using service holes drilled off and lined with fuel proof material from the surface down to the underground fuel bay located on Level 360.

Initially, all the explosives and accessories were stored in temporary magazines located on surface at about 800 meters north of the Triangle laydown area. Since November 2017, all the explosives and related supplies and accessories are stored in the newly commissioned explosives storage facilities underground located on Level 50.

A small container (2.5 m by 12.0 m) located 100m East of Triangle laydown area is used to store explosive wrapping and boxes. A permitted burning facility for explosive wrapping and boxes is located about 400m West of Triangle and is used when required.

18.9 CEMENT PLANT FOR BACKFILL

The surface cement plant will be built near the ventilation raise. A 150mm Victaulic steel pipe would run down the vent raise and laid along the ramp and drift up to an underground storage tank. The plant consists of mixers and two containers of 5 m by 12 m joint together. Two silos of 100 Tonnes of cement will feed the plant.

18.10 WATER SYSTEMS

The Sigma-Lamaque Complex is already connected to the Val-d'Or potable water network. The Triangle site is connected to the Val-d'Or potable water network via a 2.5-km private line.

18.11 COMMUNICATION SYSTEM

The Sigma-Lamaque Complex is connected to the public telephone service and internet.

A new IP telephone network was installed at the Sigma-Lamaque Complex, and at Triangle.

The communication between buildings will use single mode optic fiber cable. To allow employees to have wireless access, a network access point (WI-FI Unifi AP Pro) will be installed in each of the buildings to permit cellular and computer connections.

The surface radio system consists primarily of channels with local short-distance coverage or extended coverage. The following channels are planned for the site:

- Security/emergency;
- Surface operations;

- General and maintenance (mechanical/electrical/housekeeping/etc.);
- Underground operations (underground link with surface).

18.12 VENTILATION AND HEATING INSTALLATION

Two main fans equipped with 1,500 Hp motors will be installed at the surface collar of the ventilation raise (Figure 18.3). There will be two 30 MBTU heating systems that will be couple with the installation.
Another raise, located 30 m away, will be used as an egress. On top of this raise, a small building is already installed with a heating system to serve for 2018.

A natural gas line of 200 mm is installed from Energir main line along the 117 road to the ventilation installation at Triangle. In 2017, the private network was extended to serve future permanent ventilation heating system.

18.13 SEWAGE AND INDUSTRIAL WATER

The Sigma-Lamaque Complex is connected to Val-d'Or services. The South Zones will be connected to the Val-d'Or sewage water network via a 2.5-km private line.

Industrial water is supply at the South zone by a 100 mm HDPE pipe from the #3 Mine well passing by the Mylamaque well then to the ramp portal. A pumping station is install at the Mylamaque shaft and a surface line is available for firefighters.

At this time a clean water sump is in construction at Level 135 and will be available at Q1 2018. The sump will collect seepage water from upper part of the Triangle development (Level 70m) close to ventilation raise. The sump has a capacity of over 200m3 and water will be pumped to industrial water network to serve mining operation. The Mylamaque shaft will continue to be used for surface water protection and to serve the upper portion of Triangle.

18.14 ORE STOCKPILE

An ore stockpile of 3 875 m^2 is built near the Triangle ramp portal (Figure 18.6). The proposed stockpile will have a capacity of about 12,000 m^3 allowing low-grade material to be stored and resampled if required. The ore will then be transported on a daily basis to the Sigma mill stockpile.

The ore stockpile is considered a non-permanent infrastructure therefore, it will be removed and the land restored at the end of mine life.

18.15 WASTE STOCKPILES

Waste stockpiles will serve as permanent storage infrastructure for the waste rock extracted from the underground mines. The Triangle waste pile is located close to the portal to limit transport distance.

The waste stockpiles have been built and permitted authorised waste stockpile is already used. This stockpile outline is covering an area of 52 000 m^2 for a total capacity of 400 000 m^3. In 2019, this stockpile will have to be enlarge to 83 000 m^2 for a final 760 000 m^3 capacity (Figure 18.6).

The waste stockpile is also used to provide waste material to generate construction roadbed material for the underground ramp and for various construction purpose at surface. Part of the waste pile is also used as temporary storage area for material.

18.16 OVERBURDEN STOCKPILE

An overburden stockpile has been created with the Triangle site excavation north of the portal. Part of that material was used to cover the HDPE water line to the Sigma site (Figure 18.6). The overburden material will serve for reclamation purpose at the end of the operation.

At the Sigma site, the overburden pile is 50 m south of the mill.

18.17 PROCESS PLANT SIGMA COMPLEX REFURBISHING

18.17.1 Electrical Infrastructure

18.17.1.1 25 kV Outdoor Substation

Currently, a 25 kV overhead line from Hydro-Québec is feeding an outdoor substation providing power to Sigma and Triangle utilities. In accordance with Hydro-Québec standards, the fuses in the existing substation need to be removed. It was also noted that the existing 7.5/10 MVA 25-4.16 kV power transformer in the substation no longer meets the FM Global requirements due to its proximity to the warehouse and the size of its basin. Furthermore, a modification of the general layout of the substation is necessary (removal of fuses, addition of breakers and disconnect switches, relocation of Hydro-Québec metering device, addition of local metering device), which would require a one-week general shutdown. Consequently, to be able to adhere to Hydro-Québec and FM Global recommendations, to reduce the duration of the shutdown to a minimum and to increase the overall electrical supply reliability, a new 25 kV substation will be built to supply Sigma and Triangle utilities.

The new 25 kV outdoor substation consists of two (2) 25-4.16 kV power transformers feeding a 4.16 kV switchgear. Redundancy is required for reliability and maintenance purposes. Two (2) other 25 kV dedicated feeders are used for the distribution lines of Triangle utilities and for the laboratory/crusher. One additional feeder is planned for future installations. Every 25 kV feeder is equipped with its own circuit breaker and disconnect switches.

The existing power transformer, even if manufactured over 30 years ago, seems to be in a good overall condition. No oil leaks have been detected so far and an oil sample was taken in July 2017. While some gas levels were detected at their limit, the levels remained stable over the years when compared with the samples taken over a ten-year period. This indicates that the power transformer has not deteriorated. This transformer needs to be fully tested so that we can determine if it can be energized while fully loaded.

To be able to meet the objective of pouring the first gold bar by the end of the year, the substation will need to be completed in two phases.

The first phase planned in 2018 consists of the erection of a new outdoor substation, except for the installation of the two (2) power transformers, their foundations and basin. All the electrical equipment will also be installed and connected in the new substation, except for the two (2) power transformers. The existing 7.5/10 MVA power transformer at its current location will feed the Triangle mine and the Sigma utilities, but will be fed from the new substation. There will be no parallel redundancy during that period, resulting in a total plant shutdown upon transformer failure. To mitigate this, a new power transformer will be purchased and stored at the plant.

The second phase planned in 2019 consists of the completion of the substation with the civil and electrical works related to the power transformers. At that time, the substation will be fully redundant, which will significantly increase the flexibility and the reliability of the network.

18.17.1.2 4.16 kV Switchgear

The existing 4.16 kV switchgear is located in the main electrical room of the plant, in the building near the outdoor substation. One of the cells of the existing switchgear was damaged after a cable fault where the feeder protection had malfunctioned. This cell is no longer in service but the main bus bar is still energized. This switchgear is also equipped with electromechanical relays. This type of relay is considered outdated and represents a safety issue. Consequently, the general protection of this switchgear needs to be modernized. Furthermore, each 4.16-0.6 kV transformer needs to be fed by its own circuit breaker from the switchgear to reduce the arc flash level and to be flexible for service or maintenance operations. The number of circuit breakers in the current switchgear is insufficient to allow this distribution. Considering that the switchgear is at the heart of the power distribution, that it is in overall bad condition and that it is critical on the process loads, installation of a new switchgear is highly recommended.

The new main distribution center will be a 4.16 kV 2,000 A AIS (air-insulated switchgear) type switchgear located in a new prefabricated electrical room near the outdoor substation. This switchgear would be equipped with two (2) main breakers and one (1) tie breaker for redundancy, as well as ten (10) feeder breakers, including one (1) spare for future use. This configuration will add flexibility and reliability to the network. The switchgear would be rated arc-resistant and would be operated remotely to reduce electrical risks to a minimum.

18.17.1.3 4.16 kV Distribution

At the moment, the electrical rooms fed from the 4.16 kV switchgear consist of a 4.16 kV fuse disconnect close-coupled with a 1 MVA 4.16 kV-600 V dry type transformer. The secondary side of the transformer is close-coupled with a 600 V distribution center.

The fuse-disconnect on the primary side of the transformer is no longer required due to a withdrawable feeder breaker from the new 4.16 kV switchgear that offers proper means of protection and visible disconnection.

However, since the equipment is close-coupled and that a modification of the cable entry could be complicated, the fuse holder would be modified with a bus bar such that only the disconnect switch would remain.

The three (3) main motors of the plant are currently fed at 4.16 kV with a direct starter. Each 4.16 kV motor is planned to be fed with a soft-start, reducing the network perturbations caused by voltage drops on motor start-ups as per Hydro-Québec standards, reducing the overall stress on the motor and extending its lifespan. The primary, secondary and tertiary crushers are now also fed from the same feeder breaker at the 4.16 kV switchgear.

Finally, a new 4.16 kV feeder is required to power-up the paste plant. One 1,200 A spare breaker, including all the loads mentioned above fed by the new 4.16 kV switchgear, will remain for future use.

18.17.1.4 600 V Distribution

The existing 600 V distribution centers fed from the dry-type transformer are generally in good condition, except for one of them that needs to be replaced because of its outdated protection that does not comply to current protection standards. This distribution center is replaced by a 600 V switchboard. However, the modernization of the metering devices of all distribution centers is required, since the technology used is still analogic. In the meantime, knowing that this equipment is about 30 years old but mainly in good condition, the trip unit modernization of the protection is recommended without changing all the equipment.

Each distribution center feeds a 600 V Motor Control Center (MCC). A general inspection of the MCC has led to the conclusion that they require replacement.

The crusher MCCs are fed from two (2) different locations: one being a transformer in the crusher room and the other being from the laboratory. For process and maintenance optimization, MCC No. 1 fed by the laboratory will be fed by MCC No. 2 in the crusher. This modification implies changing the size of the dry type transformer up to 1.5 MVA.

18.17.1.5 Electrical - General

Grounding is missing in many area of the existing processing facility and will need to be reinstalled in the processing facility. . Additionally, depending on their general condition, the 600 V motors and feeders may need to be replaced. Further to CNESST investigations, it was pointed out that every motor needs to be equipped with a start-stop pushbutton to ensure that the motor is de-energized when a lockout procedure is underway. Finally, the general lighting needs to be changed and will be standardized to a LED technology.

18.17.1.6 Emergency Network

In the event of a power failure, emergency power to operate critical equipment will be provided by a single 600 V, 1,000 kW, emergency standby power generator. This generator will be installed in its own building. The provision of an automatic transfer switch and an emergency MCC for critical loads will be installed in an existing electrical room. This will prevent damage on equipment and supply power for general emergency distribution services in case of an outage.

18.17.2 Process Equipment Refurbishing

As part of this study, detailed inspection of the processing facility was completed by a multi-disciplinary team combining engineering and operations points of view. The outcome of the inspection is described herein.

Crushing Area

In the crushing area, the following equipment will require overhaul prior to recommissioning:

- Jaw Crusher;
- Four Conveyors;
- Vibrating Feeder;
- Rock Breaker.
-

18.17.2.1 Cone Crusher

The triple deck screen is too damaged and will require replacement.

Insulated panels will be installed inside the rock breaker building and the size of the openings will be reduced to minimise noise emissions. In addition, an 8.5 m high by 24.4 m long wall will be installed as a sound barrier.

A new building will be constructed for the crushing conveyors transfer point, which is currently unprotected. This involves construction of new foundations adjacent to the existing ones to support a new building, construction of a building envelope covering the existing structure and equipment, installation of building heating and electrical components as well as lifting structures for mechanical maintenance.

A cover will be added to the crushed ore transfer conveyor, as well as a second walkway to be able to access the conveyor from both sides. A staircase and a platform to give access to the walkway will also be constructed as well as the foundation that supports them.

The primary crusher building overhead crane will be extended by an extra 0.3 m to the west. This work also implies modifications to the crane support structure, end stoppers, building cladding, girts and braces as well as the electrical cables and supports providing power to the crane.

18.17.2.2 Crushed Ore Storage Area

The three apron feeders will be overhauled but the major work element in the ore storage area is the displacement of the storage silo by 5 m to position it over two of the apron feeders. Currently, the silo stands over only one. The existing opening in the ore bin wall will also be enlarged to let loaders and excavators go inside.

18.17.2.3 Grinding and Gravity Concentration

In the grinding area, the following elements will require refurbishing:

- Ball Mill #1 & #2;

- Rod Mill;
- Ball Mill Motors;
- Rod Mill Motor;
- Cyclone;
- Mill gear boxes;
- Feed Conveyors;
- Screens;
- Knelson gravity concentrator.

18.17.2.4 Leaching

In the leaching area, the major elements requiring overhaul are the leach tank agitators and agitator gearboxes.

18.17.2.5 Carbon in Pulp Area

In the carbon in pulp area, the following elements need overhaul:

- Vertical Carbon Pumps;
- CIP Tank Agitators;
- CIP Agitator Gearboxes;
- Screens;
- CIP TANKS #001, 002, and 003.

The major work in this area is the full replacement of the existing CIP tanks #004, 005, 006 and 007.

Maintenance work is difficult over the CIP tanks because of missing overhauled equipment. Rearrangement of roof trusses will be considered on newly installed maintenance lifting devices.

Additionally, concrete basins will be built to provide containment for this area as well as the leaching area.

18.17.2.6 Desorption and Regeneration

The following are the main elements to be refurbished in the desorption and regeneration area:

- Elution Exit Screen;
- Regeneration Kiln;
- Acid Wash Screen.

Complementarily, the gold room filter press needs to be moved to allow installation of the an additional electrolytic cell for the intensive leach circuit will be installed in the gold room and the system will be optimized.

18.17.3 Piping Systems

During the PFS, a survey of the status of the piping systems was done to scope the refurbishing requirements. A line-by-line review of the level of damage to each pipe, the presence of required piping supports, the dimensions and quantity of manual valves by types were identified. The overall plan covers the replacement of approximately 25% of the pipes, the installation of missing or replacement of damaged pipe supports, the replacement of manual valves sized below 6 inches, and the refurbishing of manual valves of 6 inches and larger.

18.17.4 Instrumentation

As for the unused power cables, control and communication cables that are no longer in use will be dismantled to provide space in existing cable trays for new cabling. The fire detection system will be completed for the crusher and process plant building.

Existing instruments will be recalibrated if sufficient but replaced if damaged. Additional instruments will be provided for proper process control. Modulating and on/off control valves will refurbished according to the same rule as for manual valves.

18.17.5 Eye-wash and Safety Showers

Various chemical reagents will be added to the process circuit. These reagents will be supplied in solid or liquid form and will need preparation. Eye-wash and safety showers will be installed in the different reagent preparation and distribution areas for safety purposes, in case of operator contact with the reagents. Some existing eye-wash and safety showers will be upgraded to meet health and safety regulation requirements.

18.17.6 Ventilation of Crushing and Process Plant buildings

Currently, there is no ventilation ensuring a minimum number of air changes per hour, as specified by health and safety regulations in the process plant and crushing buildings. As a result, a new ventilation system will be installed in both buildings to meet health and safety regulation requirements.

18.17.7 Dust Collection Systems

The crushing facility is equipped with a dust collection system that has to be optimized and upgraded. The gold room is equipped with a dust collection system that has to be replaced.

18.17.8 Process Plant Buildings Refurbishing

The perimeter wall structure and cladding of the main plant building is in disrepair and also needs improvement acoustically. To this end, all perimeter walls will be rebuilt with new purlins and metallic cladding.

A survey of the main plant building structure (roof trusses and columns) was done identifying structural members that no longer can fulfill their purpose. They will be replaced or repaired and repainted.

The warehouse structure is also in bad condition and will be repaired.

There are currently water infiltrations in the document vault. Its concrete foundation will thus be repaired.

18.18 PASTE PLANT – PIT TAILLINGS DISPOSAL

18.18.1 Laboratory Test Work

The solid-liquid separation of the tailings and the paste mix are important parameters for the paste plant design, equipment selection, and distribution network.

Laboratory tests were performed by SGS Lakefield for solid-liquid separation investigation and by URSTM for paste mix investigation. The tests were performed on samples collected during the third bulk sample campaign conducted at the Camflo Mill on the C2 mineralized zone (see Section 13). The samples were prepared by a Camflo operator and WSP cannot determine if they were representative of the deposit and the operation of the Sigma mill.

However, the Camflo operation presents similarities with future Sigma mill parameters in terms of grind size, which is an important factor for settling, filtering, and paste properties.

Two steel drums containing approximately 135 litres of cyanide leached pulp were send to SGS Lakefield. The cyanide was detoxified to a concentration smaller than 1 ppm WAD cyanide by batch process prior to solid-liquid separation (SLS) test work and paste recipe determination.

18.18.1.1 Solid-Liquid Separation Test Work (SLS)

A representative sample of about 15 kg (dry mass) was collected by SGS for the solid-liquid separation test work. The remaining sample was shipped to URSTM2 laboratory for paste investigation.

Sample characterization was done to determine particle grind size distribution (Figure 18.7), specific gravity, and pH. Characteristics are shown in Table 18.1.

Table 18.1 - SLS Testing Sample Characteristics

	d80 (µm)	< 20 µm (% vol.)	Dry SG	pH
Camflo Tailings	56	51.7	2.85	9.4

[2] URSTM – Unité de Recherche et de Services en Technologie Minérale, Rouyn-Noranda, Québec

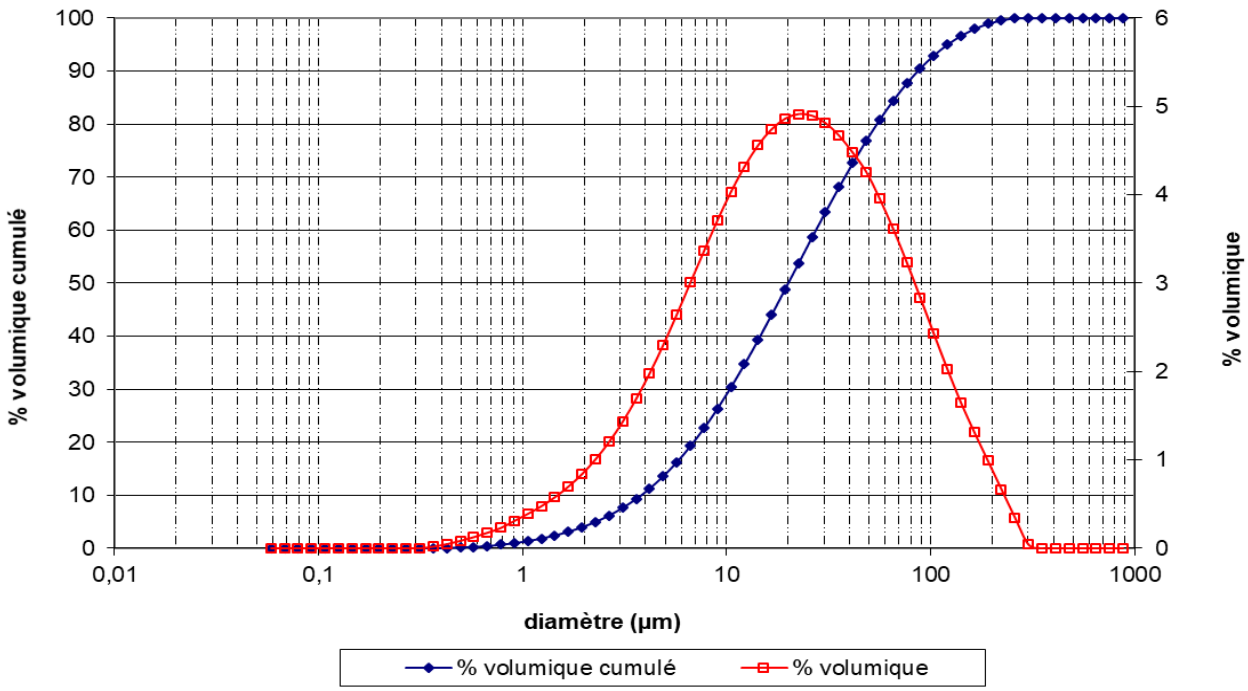


Figure 18.7 – Particle size distribution of tailings (Source: URSTM)

18.18.1.2 Settling Test

Flocculant scoping tests were first conducted. Based on these, flocculant BASF Magnafloc 333, a very high molecular nonionic polyacrilamide, was selected.

Preliminary static settling tests were conducted in two-litre graduated cylinders equipped with rotating picket-style rakes. Test results were used to determine dynamic thickening test conditions. Tests were conducted on a single flocculant dosage and on various feed solids concentrations.

Based on static test results, dynamic thickening tests were conducted. The effect of flocculant dosage was observed at 10% solid diluted feed for a 0.16 m2/(t/d) thickener unit. A dosage of 15 g/t rather than 10 g/t decreased the overflow total suspended solid (TSS) from 78 to 15 mg/l.

The subsequent tests were conducted with a Magnafloc 333 dosage of 15 g/t for various thickening unit areas ranging from 0.08 to 0.16 m2/(t/d).

Dynamic thickening test results are summarized in Table 18.2.

Table 18.2 – Summary of Dynamic Thickening Results by Unit Area

Flocculant dosage g/t	Unit Area $m^2/(t/d)$	Solids Loading[1] $t/m^2/h$	Net Rise Rate $m^3/m^2/d$	Underflow %sol (w/w)	Overflow TSS mg/l	Residence Time h
15	0.16	0.26	54.4	64.2	15	1.63

15	0.13	0.32	66.9	63.4	47	1.32
15	0.11	0.38	79.14	63.0	54	1.12
15	0.09	0.46	96.7	62.4	76	0.92
15	0.08	0.52	108.8	61.3	81	0.82

1 Solid loading I – no scale up factor

18.18.1.3 Vacuum and Pressure Filtration

Both filtration tests were conducted on the thickened sample at 64% w/w solids. Filter cloth scoping was conducted. Testori P4408 TC polypropylene cloth was selected for all tests.

Filtration tests were performed to examine the effect of cake thickness and air-dry duration on the production rate and filter cake moisture.

Vacuum filtration tests were performed on a pour-on leaf filter. All tests were conducted at an applied vacuum of 21 inches mercury.

Pressure filtration tests were conducted in a Bokela "Filtratest" unit. Pressure filtration was conducted at 5.5 bar and 6.9 bar pressure level.

Results are summarized in Table 18.3.

Table 18.3 – Summary of Vacuum and Pressure Filtration Test Results

Vacuum Level in Hg	Pressure Level bar	Feed Solids % w/w	Form Time s	Dry Time s	Cake Thickness mm	Production Rate[1] Dry kg/m^2*h	Cake Moisture %
21	NA	64	34	4	23	3144	24.2
			24	5	18	3322	24.5
			25	15	18	2336	24.5
			25	42	18	1413	24.4
			24	120	18	663	23.8
			10	100	9	409	20.5
			37	150	23	638	23.8
NA	5.5	64	3	51	15	1501	9.7
			5	54	19	1808	9.8
			6	102	25	1206	10.1
			9	119	29	1210	10.6
	6.9		3	45	15	1714	10.2
			4	66	20	1528	9.7
			5	75	24	1652	10.3
			8	112	30	1352	11.5

1 Production rate I – no scale up factor

Technical data sheets based on SGS laboratory results and paste plant design were sent to the supplier for preliminary solid-liquid equipment sizing.

Vacuum disc filtration is considered for this project even though pressure filtration produced lower residual moisture and higher solid throughput. Pocock test work (Section 13) shows better vacuum filtration results for detoxified tailings in similar conditions in term of throughput and cake moisture content. Optimization test work must be done for vacuum disc filter sizing.

18.18.1.4 Paste Mix and Rheology

Physical and geochemical tailings characterization is ongoing at URSTM. Paste mix and rheology testwork are still ongoing. This section presents a summary of the preliminary results available. Figure 18.8 shows the 7 days curing time uniaxial compressive strength (UCS) results with general use (GU) cement and, Figure 18.9 shows the slump behaviour of the paste.

Addition of 1.5% GU cement by weight of dry tailings is estimated as a good average for the disposition of the tailings in the open pit and for the purpose of OPEX estimation. Cement is added to the tailings to prevent for liquefaction, strength targeted is 100 kPa at 7 days (e.g. Belem and Mbonimpa, 2016). This strength will be confirmed during next engineering stage, when longer curing time results will be available as well as with geotechnical stress and consolidation modelled.

As rheological tests are currently not available, an average pipeline pressure loss value of 10 kPa/m was used for the design of the pipeline. This value is assumed to be in a reasonable range based on other projects operated at 71.5% solids and having similar particle size distribution. Cemented paste is anticipated to be discharged with a slump between 178 mm (7 inches) and 254 mm (10 inches).

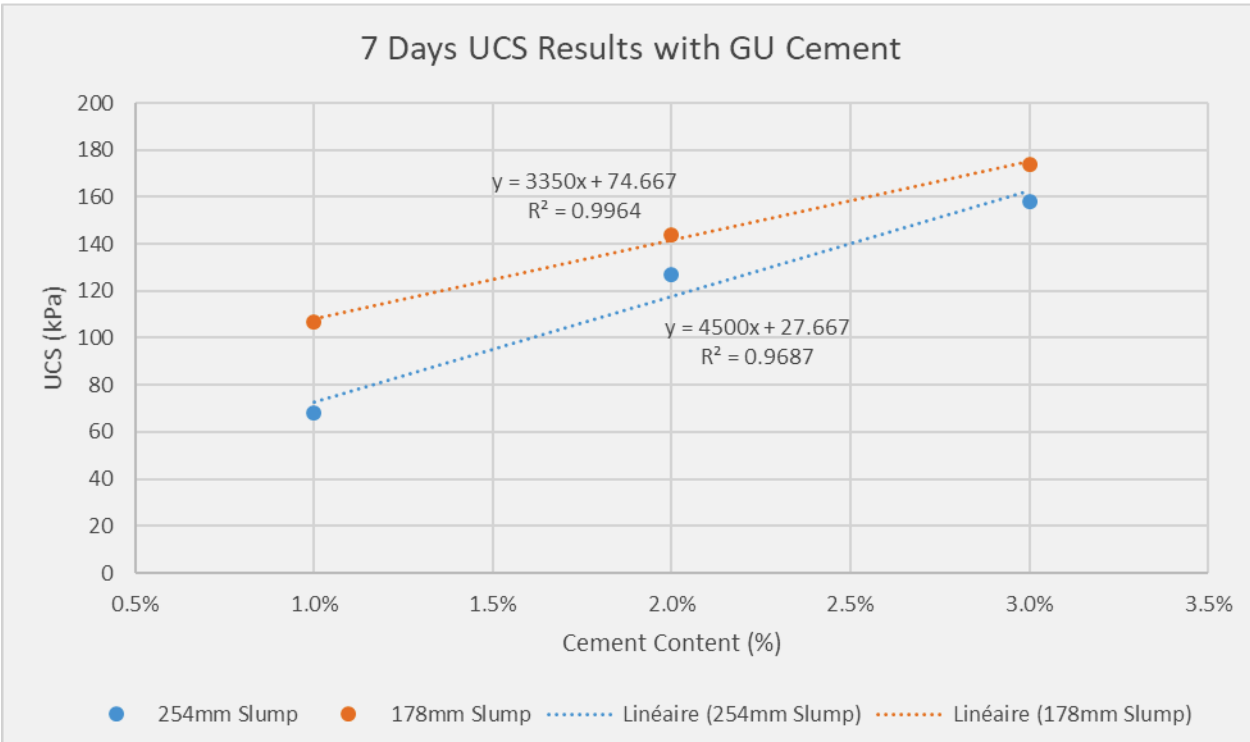


Figure 18.8 – UCS at 7 days (Source: URSTM)

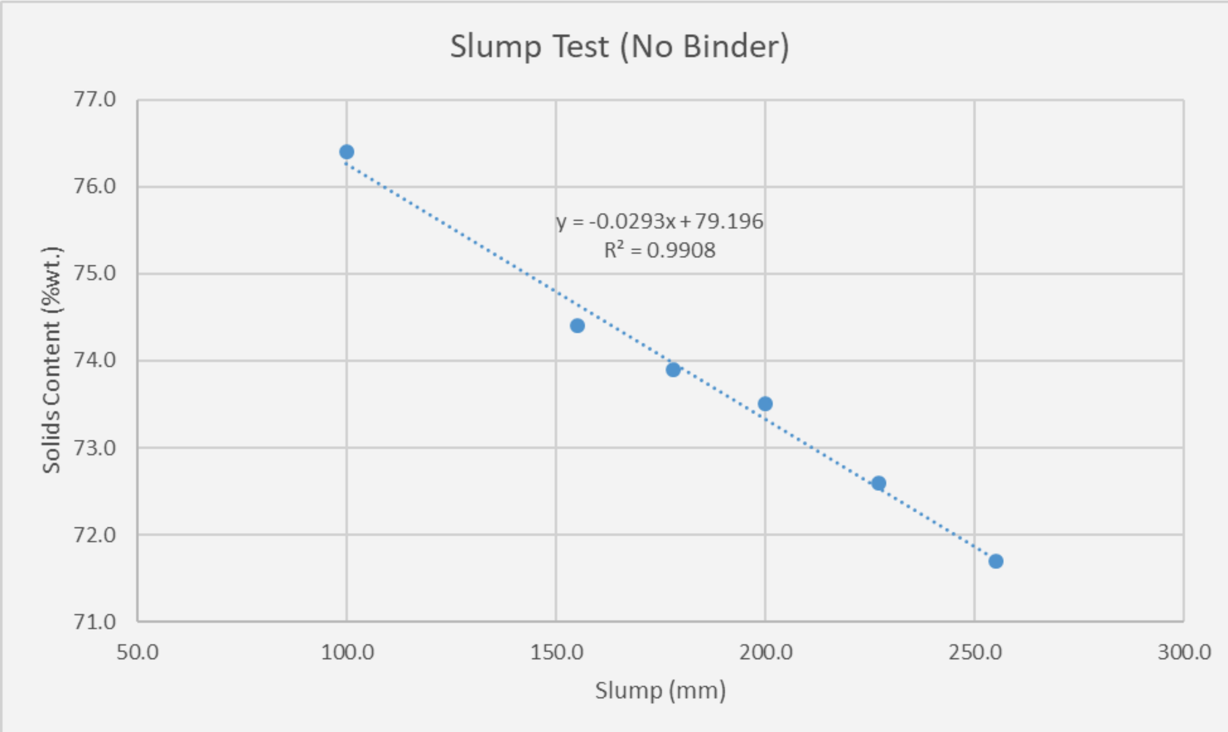


Figure 18.9 – Tailings %Solids vs. Slump (Source: URSTM)

18.18.2 Design Criteria

Table 18.4 presents the main design criteria.

Table 18.4 – Paste Plant Design Criteria Summary

	UNIT	VALUE
Tailings Production	t/d	1 712
Paste Plant Availability	%	100
Paste Solid Content	%	70-73%
Paste Binder Content	%	1.5
Paste Binder		Cement GU (Type 10)
Paste Strenght	kPa	100
Paste Slump	mm	178 to 254
Paste Production	m^3/h	52

18.18.2.1 Process Description

The detoxified tailings are used to produce the paste backfill. Since the design of the paste backfill plant is dependent on the material properties of the tailings streams, preliminary testing was conducted to determine the potential recipe for the paste, which is outlined in Section 18.26.1.5.

The paste backfill tonnage is determined and calculated based on the utilization of all tailings produced by the process plant at an average incoming dry solids rate of 1,712 tonnes per day to a maximum of 2,000 tonnes per day. The plant availability is 100%. This availability will be achieved by installing a second production line as backup to minimize the amount of residues sent to the tailings facility.

The tailings streams feed the paste plant thickener via the detoxification tank. The tailings slurry is characterized by a solids content of 48%, solids specific gravity of 2.85, and a particle size distribution of 80% passing 56 microns and 52% passing 20 microns and is therefore considered to be a fine slurry.

The thickener increases the tails solid content from 48% to 64%. The thickened tailings are pumped to an agitated filter feed tank that provides residence time to manage fluctuating flows and brief stoppages. The required tailings tonnage from the filter feed tank is pumped in one of the two parallel lines in the paste backfill plant.

During the paste backfill plant operation, only one production line is in operation. The second production line is a backup for downtime. Each line consists of one vacuum disc filter, its own discharge conveyor, one paste mixer, and a positive displacement pump. The tailings are pumped from the filter feed tank via a filter feed pump dedicated to each disc filter. The agitation in the tank enables homogenization prior to filtration.

Vacuum filtration increases the slurry density from 64% solids w/w to 80%. It should be noted that further testing will be required to confirm and optimize the use of the vacuum disc filter. The filtration process strategy consists of a continuous process operating with one filter at a time. The filtration parameters will be defined with the chosen supplier.

Cakes from the vacuum disc filter are discharged onto their respective conveyor. The belt conveyor is equipped with a scale and is operated to continuously feed the paste mixer, one per line.

A premix of cement and water are continuously added into the paste mixer based on the filter cake weight. The premix is done in an agitated mixing tank. The cement is stored in a silo adjacent to the paste backfill plant, with a live capacity of 75 tonnes. This allows about 36 hours of retention time.

The silo is equipped with a dust collector as well as a screw feeder conveyor discharging onto a weigh belt conveyor in order to control the binder addition. One additional dust collector is installed close to the weigh belt conveyor and the mixing tank chute for dust control.

The paste mixer is used to combine the various constituents into the final paste product. The filtered tailings cakes are mixed with the premixed cement and water so that the discharge from the mixer is a consistent paste slump at a desired 70-73% solids.

The paste is then discharged from each paste mixer through a paste hopper. Two positive displacement pumps will be installed, one per line, with one common distribution network to ensures the paste distribution around the Sigma open pit (see Section 18.26.2). Figure 18.10 presents a simplified flowsheet.

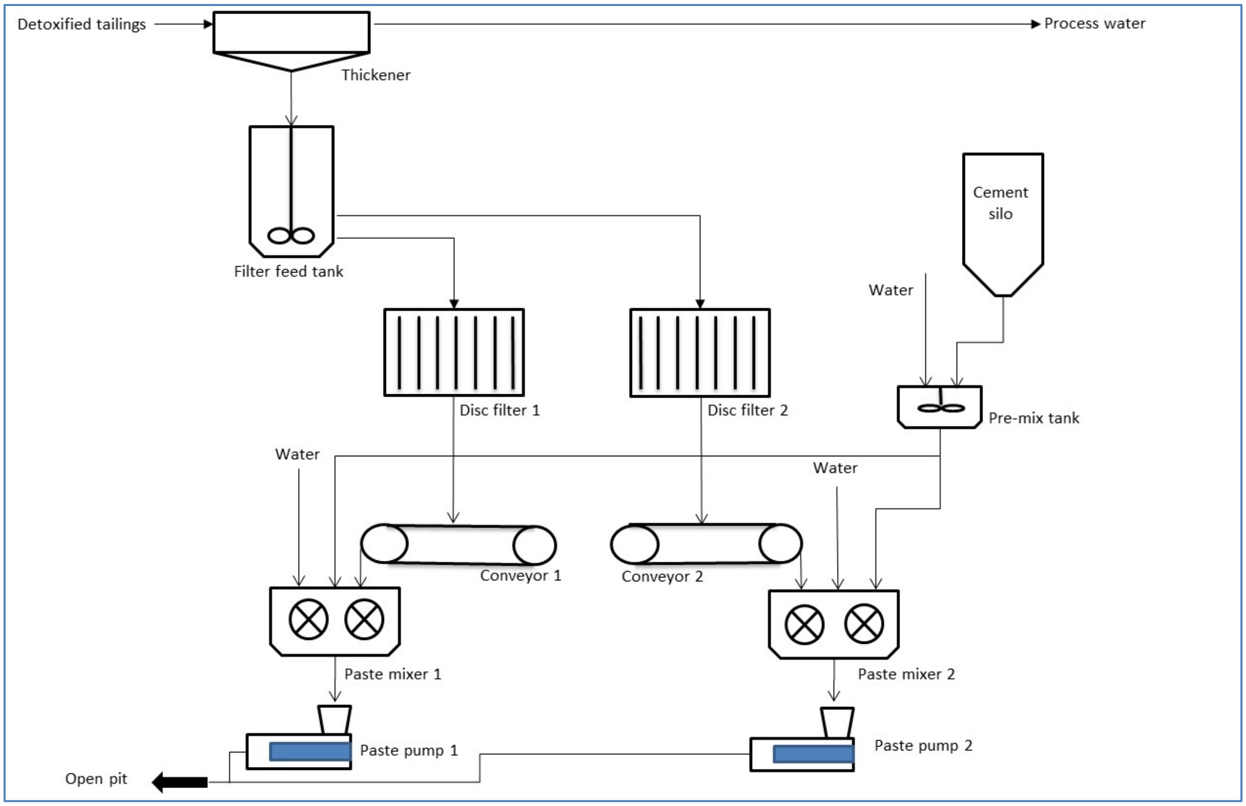


Figure 18.10 – Paste Plant Simplified Flowsheet

The piping network is purged via a borehole wash water pump and by an air blowing process. The existing service air compressor at Sigma mill will be used to provide the service air to a dedicated air receiver at the paste plant.

Paste backfill water requirements (i.e. gland seal water, wash water, fresh water) will be provided by the underground mine dewatering process. The use of paste plant process water was considered but was dismissed for this study. Additional testing is required to increase the level of knowledge on the impacts of process water released in the Sigma pit. A fresh water tank will be installed at the paste backfill plant to supply fresh water to all systems.

The filtrate from vacuum filtration is pumped into the thickener feed. The thickener overflow and the vaccum ring seal water, named process water, are returned to the Sigma mill tailings pump box. Paste plant process water will be diverted from its final destination and used in the grinding circuit, as needed.

It should be noted that in case of major problems with the thickener or the filter feed tank, the thickener underflow can be pumped via the waste water pump box to the Sigma tailings pump box and sent to the pond.

18.18.3 Electrical Distribution and Automation

Electrical power for the paste backfill plant will come from the new medium voltage switchgear located in the concentrator, via a new 5kV teck cable installed in the concentrator, then buried into the services trench between the concentrator and the paste backfill plant. This option was retained because existing 5kV cable to the portal aerial line was installed temporarily and is not compliant. The portal overhead line would now be fed by a buried 5kV teck cable from a fused disconnect switch in the paste backfill plant electrical room, thus not requiring an additional medium voltage breaker in the new switchgear. Interfering poles of this existing overhead line must be dismanteled anyways for paste backfill plant construction.

An insulated prefabricated control room is planned for the paste backfill plant operation. A fibre optics link with the concentrator would be installed in the services trench to allow availability at the paste backfill plant for automation (PLC, HMI), fire alarm, cameras, corporative, and phone networks.

18.18.4 Building and Civil Works

Located on the west side and next to the process plant, the building will be placed on conventional spread footing. Because no geotechnical report was available at the time of this study, it has been considered that rock level would be at four (4) metres from top soil level and that the building foundations would sit on the rock level. No piles are currently considered for the paste plant area. The main building has an 28.8 m by 35.3 m footprint and is 20.950 m high. The building is considered to be of conventional structural steel with a pre-insulated panel covering. Also, the building structure will support a 10MT overhead crane that will run in an east-west direction over its entire length.

The thickener, the filter tanks, and the cement silos will be installed on raft foundations. Additionally, a containment area has been considered for the thickener and the filter feed tank.

In order to mitigate the spill risk of the pulp feed, because of its cyanide concentration, a concrete gutter between the past plant and the process plant was considered.

18.19 SURFACE WATER MANAGEMENT

During past operation, the Sigma Tailings Impoundment managed its own water. Tailings were deposited in one active basin while contaminated water was collected and treated inside two of the three remaining inactive basins; the fourth basin is currently full of tailings. Water was treated for two main contaminants: total suspended solids (TSS) and cyanide. Contaminated water was first diverted/pumped into one of the two inactive basins, where solids could be settled. Once the sedimentation process was completed, water could then be diverted to the second inactive basin for cyanide treatment by exposure to the open-air (evaporation, ultraviolet rays, etc.). This form of treatment was not a continuous process but a “batch” process. Once the treatment process of a “batch” was completed, water could either be directed to the recirculation basin to be returned to the mill or transferred to the polishing pond for final monitoring. Water from seepage and runoff is collected by peripheral ditches around the four basins and is channelled to the polishing pond. Water from the underground dewatering operation (underground mine stope) is transferred to the polishing pond. To increase storage capacity, the impoundment basins were periodically raised using in-place tailings.

The projected tailings production for the Lamaque Gold project is estimated to be 3.8 Mt, for seven years of operation. The current concept consists of using the Sigma open pit to store 3.4 Mt of tailings in form of paste fill. The Sigma Tailings Impoundment will be used as a secondary storage facility only and is expected to hold up to a maximum of 0.4 Mt of tailings.

Currently, the proposed surface water management maintains similar water management cycles as those involved in the past water management strategy: the contaminated water will still be managed and treated inside the tailings basins (B-1, B-2, B-4, and B-9). The Sigma Tailings Impoundment will have a double functionally: as storage for the Directive 019 design flood of the Sigma impoundment area, as well as water treatment, using natural degradation processes including ultraviolet (UV) degradation of cyanide from natural daylight. Once the treatment process completed, water will be finally transferred by pumping to the recirculation basin and from there to the mill, or to the polishing pond for water quality control and monitoring before being released to the environment.

Water from the Triangle zone (infiltration and runoff water), as well as the mine drainage water, is sent to a treatment system call the Mud-Wizard for treatment (mainly for total suspended solids), and then sent to the Sigma-Lamaque underground mine stope. Runoff water from Lamaque Pit is also managed in the underground mine stope. The water from Sigma-Lamaque underground mine stope, similarly to the paste water management strategy, is transferred by pumping to the polishing pond to be discharged to the environment (according to Eldorado Gold and water quality results, water from the underground meets the quality criteria of the current regulation and therefore does not require storage in the Sigma Tailings Impoundment).

The Sigma Pit, which would be used as the main tailings storage area, will have to manage its own water during the design flood in order to avoid overloading the water management works of the Sigma Tailings Impoundment. During normal operations, the contaminated water from the Sigma open pit can be directed inside the Sigma Tailings Impoundment for treatment.

Contaminated water from the paste fill plant or from the mill will also be sent to the Sigma Tailings Impoundment to be managed. A maximum flow rate of 80 m3/hour was considered in the design water balance. A summary of the water management system is presented below in Figure 18.11 in a form of a flow sheet.

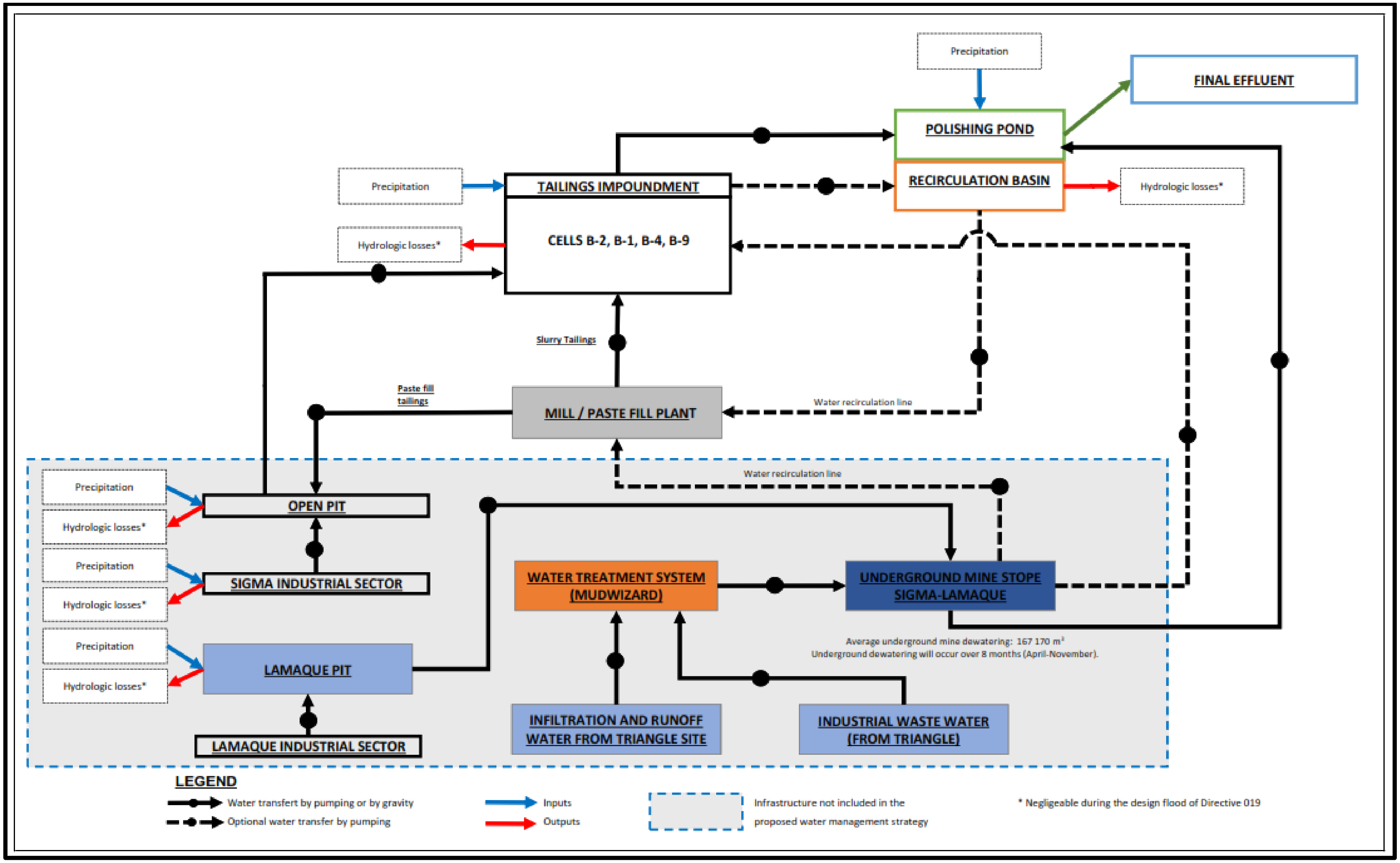


Figure 18.11 – Water Management Schematic of the Lamaque Project.

18.20 WATER TREATMENT AND SETTLING POND

As mentioned previously, during operation, contaminated water from the Sigma Tailings Impoundment, including Sigma Open Pit, paste fill plant or mill will be treated inside the tailings basins (B-1, B-2, B-4, and B-9) for two main contaminants: total suspended solids (TSS) and cyanide.

The Mud-Wizard system is used to treat water from the Triangle zone (infiltration, operation and runoff water), as well as the mine drainage water mainly for total suspended solids. The Sigma-Lamaque underground mine stope, which collected water from the triangle zone and the Lamaque pit, meets the quality criteria of the current regulation and is transferred by pumping to the polishing pond to be discharged to the environment.

In case of an unforeseen situation in which water from the Sigma pit or from the underground must be pumped to the tailings impoundment during the design flood, the water management system of Sigma Tailings Impoundment will be overloaded and contaminated water will be discharged to the environment. Further flexibility for water management is highly recommended and an alternate water treatment system (beside the natural degradation process) should be put in place. A mobile water treatment station can be installed as an added security measure to ensure water quality compliance, before being discharged to the polishing pond and to add more flexibility for the water management of the site.

18.21 TAILINGS STORAGE FACILITIES

18.21.1 Sigma TSF

The current tailings impoundment is located in the northern sector of the Sigma Mine, immediately north of the open pit and railway. The tailings impoundment is composed of four basins: B-1, B-2, B-4, and B-9 (Figure 18.12). The main structures include:

- The West dyke;
- The South dyke;
- The North dyke;
- The East dyke;
- The middle dykes 1, 2 and 3;
- Emergency spillways;
- Peripheral ditches;
- A recirculation pond and a polishing pond.

The dikes of the tailings impoundment were originally built and raised periodically with tailings. At the cessation of mining operations by Century Mining Corporation, tailings deposition was within basins B-1 and B-2, while the B-4 and B-9 basins were used only for storage of excess water. No tailings have been disposed within the tailings impoundment since the suspension of operations in May 2012. Currently, two basins out of four (basins B-4 and B-9) are full.

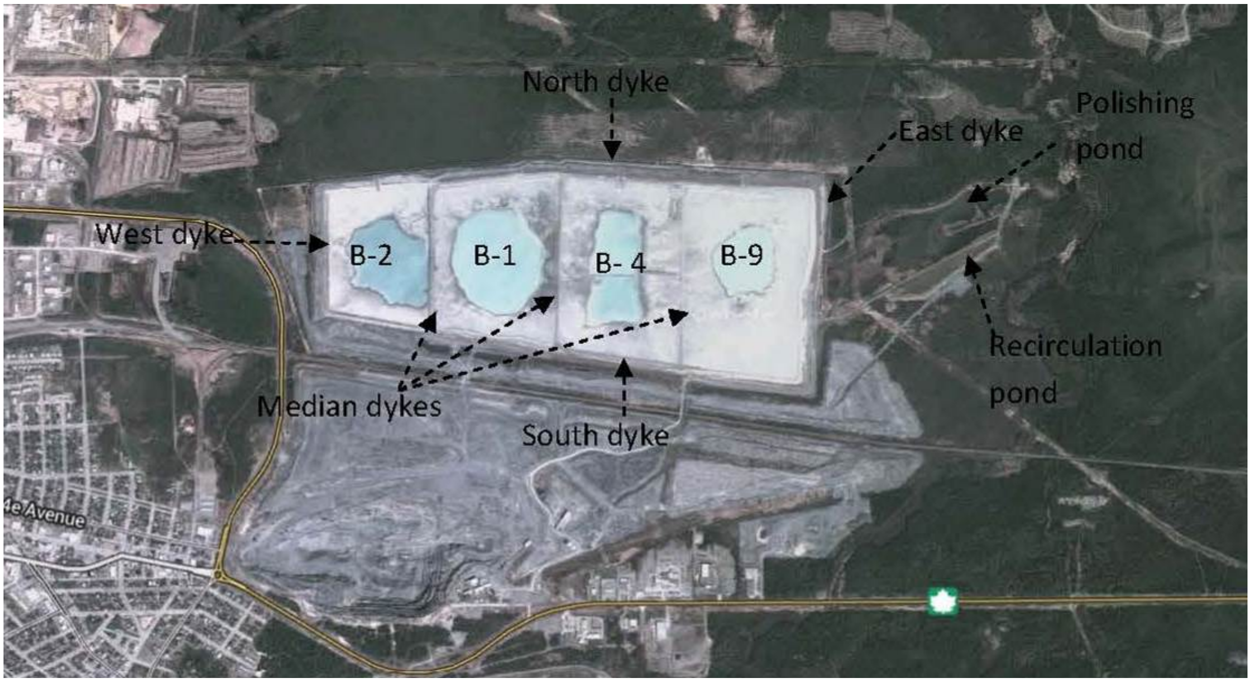


Figure 18.12 – View of the existing tailings impoundments at the Sigma mine.

As mentioned, the Sigma tailings impoundment will be used as a secondary storage facility to store a maximum of 0.4Mt of tailings. In order to store the required tailings volumes, while managing the design flood, basin B-2 must be raised.

Taking into account the proposed water management strategy, for the Sigma tailing impoundment, a high-level tailings fill-plan model was developed. Deposition will progress from the east to the west in basin B-1, sequenced in a way to avoid the obstruction of the water flow from basin B-1 to basin B-2. The following figure (figure 18.13) shows the deposition plan including the proposed location of the discharge points and the tailings pipe road from the mill to the tailings impoundment.

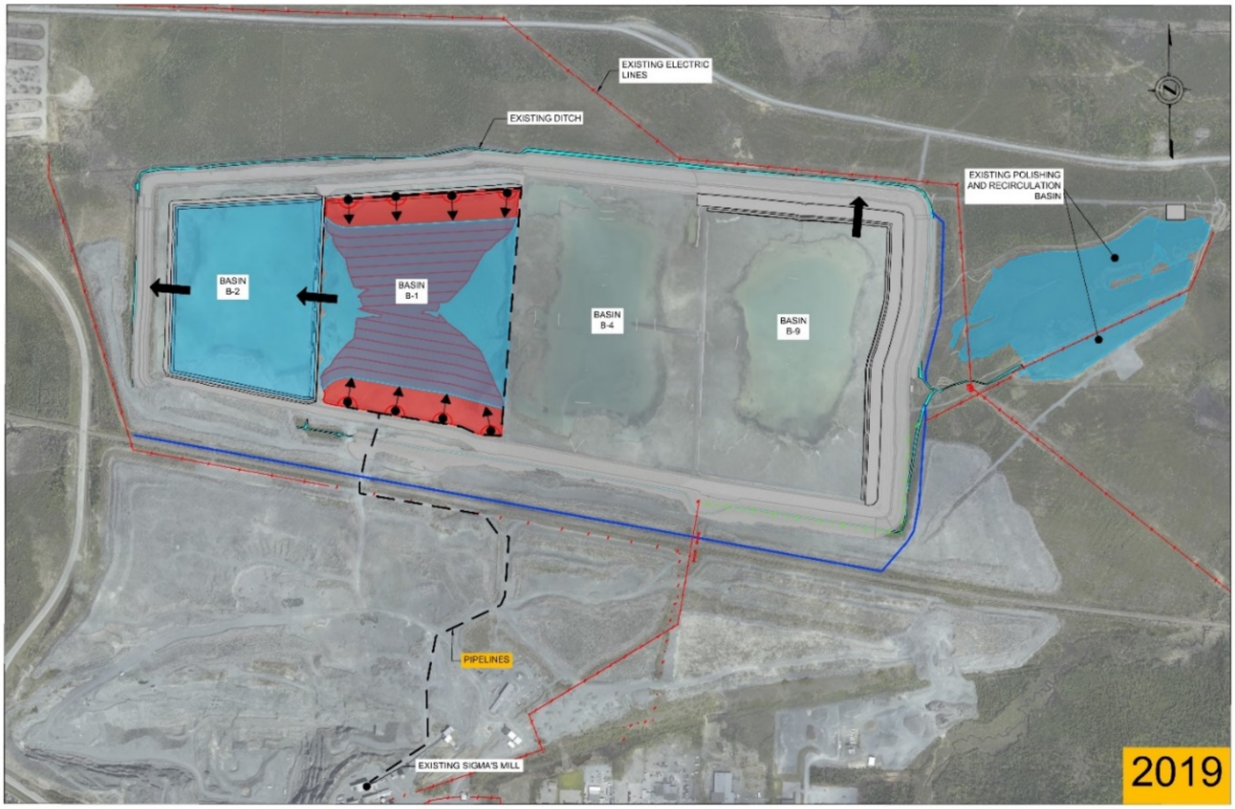


Figure 18.13 – View of the existing tailings impoundments at the Sigma mine.

The Sigma tailings impoundment does not currently meet the minimal factors of safety specified in Directive 019 under static and dynamic conditions. A stability berm and a roc shell will have to be built on the downstream of all the basins of the impoundment before the Sigma tailings impoundment could be reused for tailings and water storage.

The required berm is proposed to be build in 2 phases:

- Phase 1: Stabilizing the Sigma Tailings Impoundment as soon as possible, only for static conditions in 2018;
- Phase 2: Completing the construction of the full width of the required berm for static and seismic loading after 2018.

A specific configuration for the stability berm of Phase 1 at the toe of each basin is proposed. The configurations of the stability berm at Phase 2 remain unchanged at a maximum width of 51 metres. A few sections of the tailings impoundment will need to be lowered from the elevation of 320m to 318.5m (basins B-1 and B-9), so that the berm of Phase 1 would not interfere or obstruct the northern Seepage ditch. A few sections of the seepage ditch at the toe of the tailings impoundment will still be impacted (South and East) at Phase 1 and will need to be rebuilt. A section of the power line, belonging to Eldorado, will also have to be moved. The construction sequence is detailed in chapter 20 of this report.

18.22 TAILINGS STORAGE IN SIGMA PIT

The watershed area of the Sigma open pit corresponds to an area of 958,167 m2 extending from the north of the open pit to beyond Highway 117 to the South (Figure 18.14).

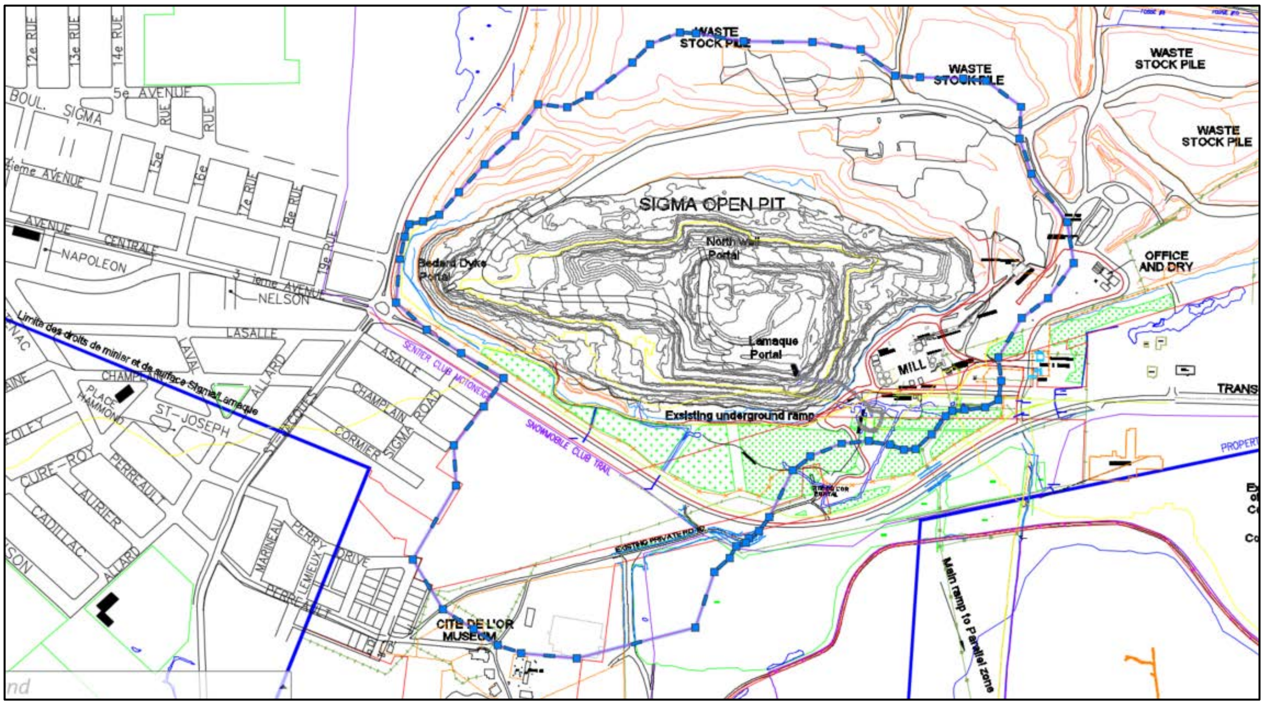


Figure 18.14 – Watershed of the Sigma open pit

The annual water reporting to the open pit was estimated using statistically average year based on data from Environment Canada (mean temperature data from 1951 to 2013 as well as precipitation and snowmelt data from 1955 to 2016). The historical average year is 2016 with 921.0 mm. The corresponding volume of precipitations and surface runoff that would report to the pit is 797,207 m3. The water management facility designs use average year water balance.

Currently, precipitations and run-off reporting to the open pit inflow in the former underground mine under the pit. The phreatic surface is artificially maintained approximately 100 m below the pit through pumping at flowrate of approximately 3,000 m3 per day. The open pit bottom is permeable, as the crown pillar was previously mined out and backfilled with granular material (waste rock). Figure 18.15 shows a projection of anticipated backfilled ore zones on pit floor and walls.

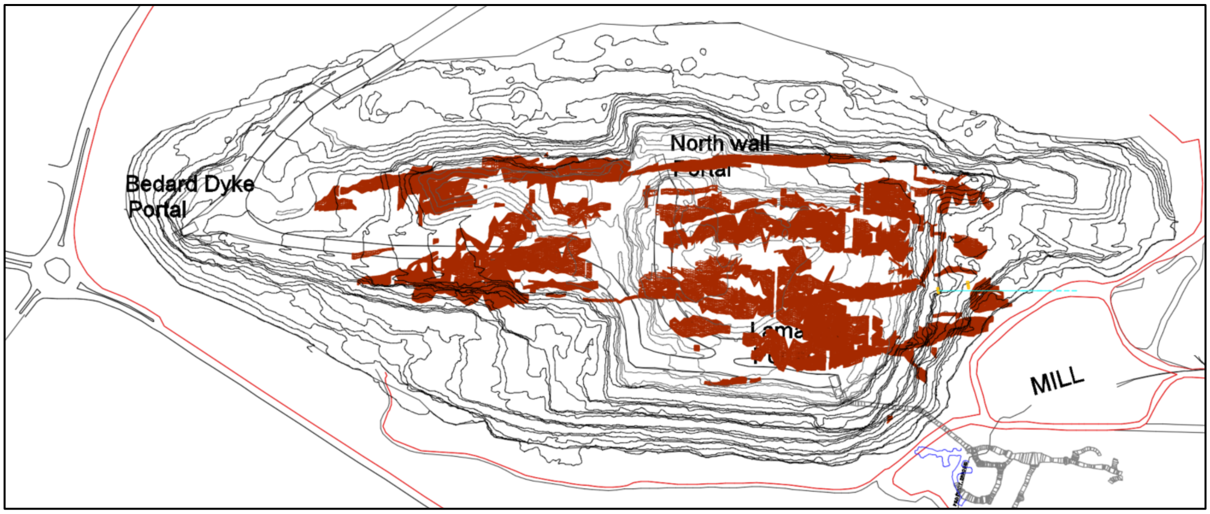


Figure 18.15 – Backfilled ore zones

The tailings management approach selected uses the existing permeability of the open pit to limit the amount of water reporting to the open pit that would turn into contact water and need treatment. A layer of crushed rock will be installed (a pervious surround, see section Tailings Management Strategy) to surround the tailings and act as a drain for run-off. Accordingly, most of the water (88% to 96%) reporting to the open pit over the life of mine will continue to be managed with the existing dewatering system; only the water directly in contact with the tailings surface (direct precipitations and bleed water from tailings) will be managed with a contact water pond and pumped to the mill using a reclaim system. Table 18.5 presents the annual amount of water that would need to be managed in the contact water pond based on an average climate year.

Table 18.5 – Management of water reporting to the open pit (average climate year)

Year	Open pit inflows and precipitations	Runoff draining in pervious surround	Contact water with tailings
	m^3	m^3	m^3
1	797 207	767 640	29 567
2	800 095	741 648	58 447
3	801 310	730 713	70 597
4	802 406	720 850	81 555
5	803 190	713 795	89 395
6	804 154	705 112	99 043

18.22.1.1 Tailings

Tailings generated from ore processing will have been subject to cyanide leaching for gold recovery followed by cyanide destruction. Table 18.6 presents tailings production per year and storage location.

Table 18.6 – Tailings production

Year	Tailings tonnage	Tonnage in TSF	Tonnage in pit	Total volume (m^3)
0	278,190	278,190	0	0
1	457,893	121,810	336,083	257,603
2	621,840	0	621,840	476,631
3	632,760	0	632,760	485,001
4	632,760	0	632,760	485,001
5	625,260	0	625,260	479,253
6	552,919	0	552,919	423,804
Total	3,801,622	400,000	3,401,622	2,607,293

Most of the tailings produced will be discharged in the Sigma open pit under the form of cemented paste backfill. Volume stored in the open pit was estimated based on a slump of 254 mm and a percentage of solids of 71.5%wt. Corresponding wet and dry densities are respectively estimated to 1.85 t/m3 and 1.32 t/m3.

18.22.1.2 Tailings Management Strategy

The following drivers are important for the project-oriented tailings management approach selected to store the tailings in the Sigma pit in the form of cemented paste backfill:

- The existing TSF has capacity for only 400,000 tonnes, as TSF size increase is restricted by Eldorado Gold property limits.
- Sigma pit offers a capacity of approximately 13,5 million m^3 close to the process plant.

Avoid building a new surface facility is expected as a positive asset to the project, for permitting as well as for social acceptability of the project.

Nevertheless, the in-pit storage of tailings comes with constraints. The following present the design basis and management strategy that was developed for technical practicability of the operation as well as the safety of employees and the environment:

- Total cyanides are below 2 mg/L in process water.
- Cement will be added to filtered tailings to form a strengthened pumpable paste; the strength developed will prevent potential liquefaction under seismic loading.
- The cement content and the viscosity of the cemented paste backfill will be adapted, depending on the discharge location in the pit and the vertical position in the stack to

limit the bleed water and the potential infiltration in the former underground mine under the pit.

- Layers of rock, sourced from non acid generating and non metal leachable rock, will be spread on the pit bottom and pit walls to surround the tailings and form a pervious surround system (e.g., MEND, 2015). On the pit bottom, three different sizes of crushed rock will act as a filter to limit infiltration of tailings; on walls, two layers of coarser rock will be used (see Figure 18.16). During operations, the purpose of pervious surround is to divert runoff from becoming contact water. During post-closure, i.e. when the Sigma pit will be flooded, the purpose of pervious surround would be to limit groundwater contact with the tailings mass. The necessity of having a pervious surround will depend on the tailings potential impacts on groundwater quality downstream of the pit. Potential impacts will depend on tailings characteristic, site hydrogeological conditions and compliance point and criteria. Evaluation of potential impacts and closure requirements for tailings in-pit disposal will be carried out at the next stage of the project.
- Under average climate year, direct precipitations on the tailings surface and bleeding water from the tailings will be accumulated in a contact water pond maintained in the tailings. The amount of water in this pond will be maintained as low as is practically possible during operations to avoid interference with the strength development of the cemented paste backfill.
- The size of the contact water pond takes into account the requirements specified in Directive 019 for the mining industry (Gouvernement du Québec, 2012) in terms of water retention capacity for the tailings disposal area; accordingly, the pond size increases yearly as the tailings deposition area increases.

18.22.1.3 Tailings pipelines

From the paste plant, cemented paste backfill will be pumped through a 150 mm pipeline to be discharged at nine to ten end pipe discharge points depending of the year (Figure 18.7). Table 18.x3 shows the pipeline length distribution and estimated pump pressure by year of operation. The total length of the pipeline at year 6 is 1,480 m.

Table 18.7, Paste backfill piping

Year	SCH80 pipe (m)	SCH40 pipe (m)	HDPE pipe (m)	Estimated pump pressure (Bar)
1	600	190	100	46
2	0	270	0	65
3	0	30	0	75
4	0	70	0	77
5	0	0	0	64
6	0	220	0	80
Total	600	780	100	

18.22.2 Reclaim water

The cemented paste backfill deposition strategy plans to maintain a contact water pond at its center, over the tailings beach. A reclaim pumping system is planned from this pond to the process plant.

18.22.3 In-pit old opening management

The pervious surround, to be placed in contact with the pit walls, will prevent paste from flowing through the historic openings obverved at different locations in the open pit. The pervious surround is planned 1 m above the tailings beach for years 1 to 5 and 0.5 m above the tailings beach for year 6.

18.22.4 Tailings deposition plan

The deposition plan was prepared using Muck 3D, a program developed by MineBridge Software. This software simulates tailings deposition from different discharge points simultaneously. The software was used to generate the fillings scheme and capacity curves, as well as support the estimation of construction quantities. The Figure 18.17 shows the yearly development of the tailings in-pit disposal over the life of mine. Yearly volumes of tailings are presented in Table 18.6.

Deposition is expected to use single end pipe discharge point system to build and maintain the contact water pond. For the purpose of planning, an average deposition slope of 2% above and below water was assumed, considering that the viscosity of the cemented paste will be controlled and that the operating objective of minimizing as much as possible the amount of water stored in the internal water pond to allow the cemented paste to strengthen.

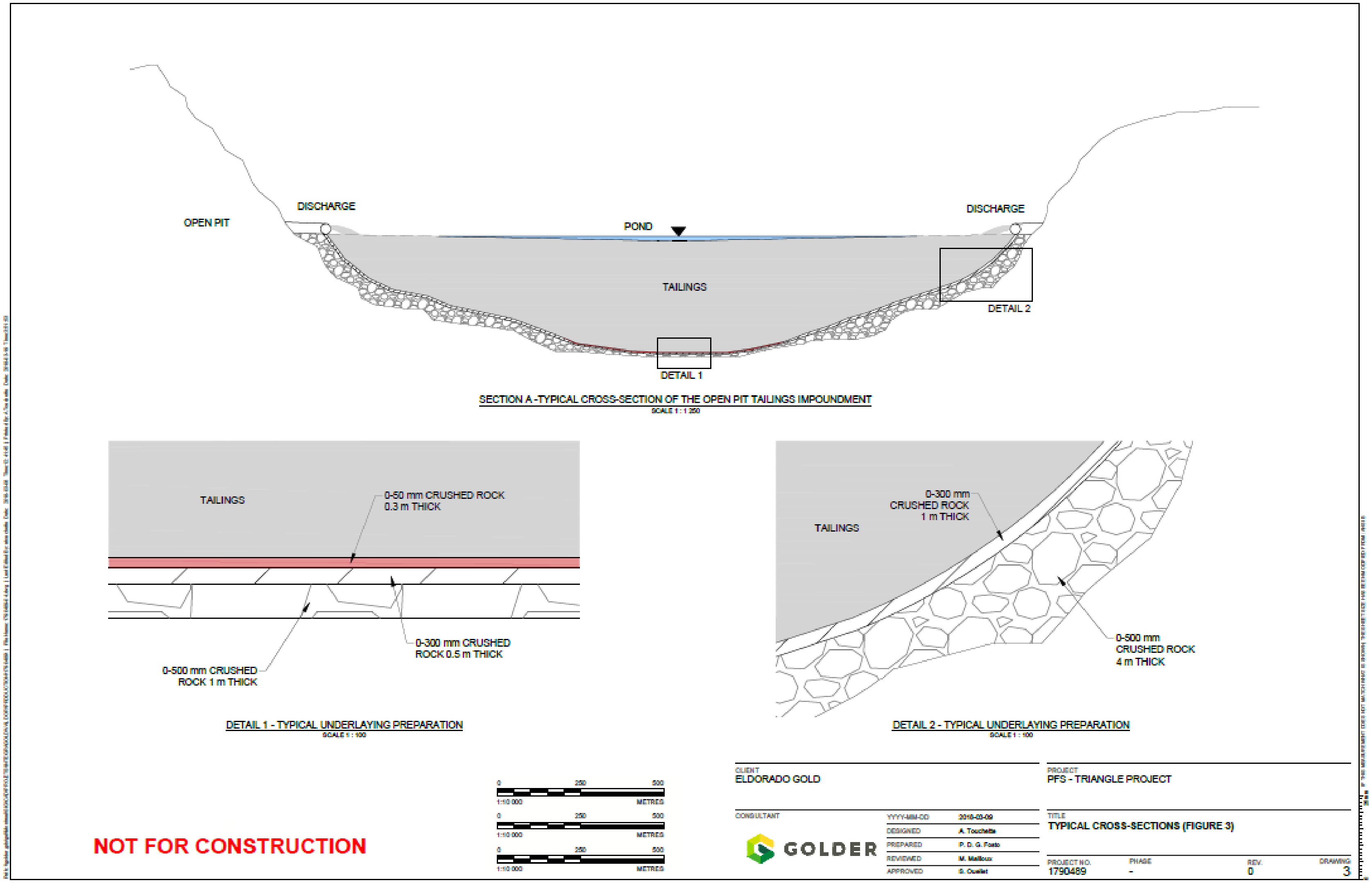
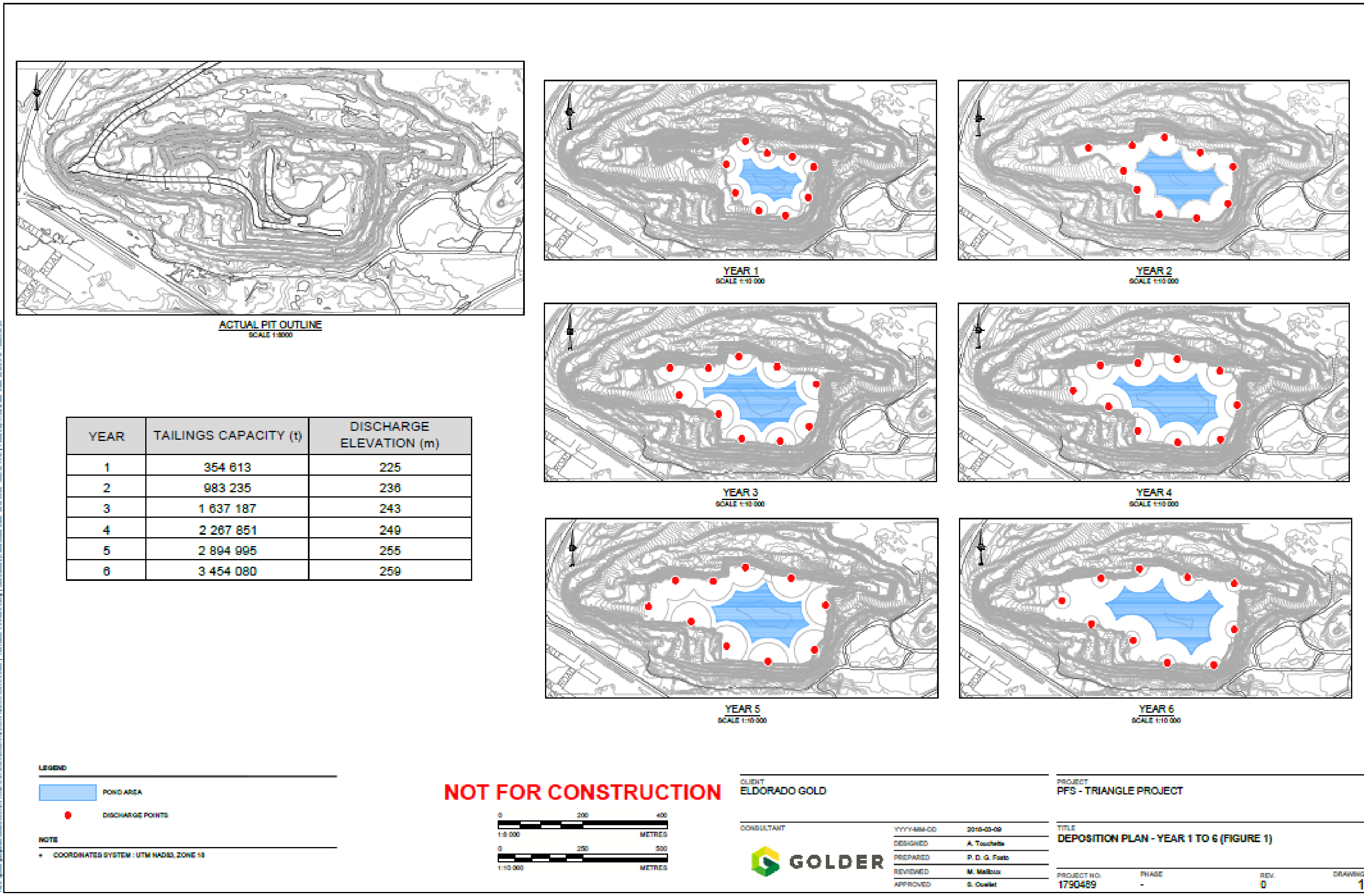


Figure 18.16 - Typical Cross-Section of the Pervious Surround



YEAR	TAILINGS CAPACITY (t)	DISCHARGE ELEVATION (m)
1	354 613	225
2	983 235	236
3	1 637 187	243
4	2 267 851	249
5	2 894 995	255
6	3 454 080	259

Figure 18.17 - Deposition Plan

SECTION • 19 MARKET STUDIES AND CONTRACTS

19.1 MARKET STUDIES

The market for doré is well established and accessible by all new producers. Doré bars produced from Lamaque Project are and will be refined in a certified refinery of which there are many available in the Eastern part of Canada and the gold sold on the spot market.

19.2 CONTRACTS

In June 2016, Integra Gold Corp and Promec Mining Inc.entered into an agreement whereby Promec undertook to carry out an advanced exploration program and related work including the excavation of 1,687 metres of ramp, 544 metres of access drift, 295 metres of miscellaneous development and 513 metres of ore drifting. The contract was valued at $5.94M. The contract covers Promec to supply man power and supervision for rock work and equipment mechanical maintenance and drilling consummmable. Integra was responsible to supply technical services, mining equipment and all other consumable (explosives, rock support, piping, etc). At the time of writing, the work leading to the recovery of a 60,000T bulk sample is completed whereas some pre-production development associated with the ramp up to full production is ongoing. Actual value of the completed work under the contract is 18M$.

As written in the Promec contract, a transition is in process to hire directly by Eldorado Lamaque, selected supervision and hourly personnel from Promec. No commercial dispute has occurred during the contract between Promec and Integra and collaboration level has been excellent.

In August 2017, Eldorado Gold Corporation entered into a custom milling agreement with Monarques Gold Corporation to process up to 55,000 tonnes of bulk sample ore between July 1st 2017 and December 31st 2017. 40,431 tonnes were processed in 2017 under this contract with the balance of the tonnage to extend into early 2018. The contract was extended into early 2018 to process a further 65,000 tonnes between February and December 2018.

At the time of this report, Eldorado is in final negotiation with another Mill to sign another custom milling agreement to process up to 90,000 tonnes of ore in 2018. Material will come from development and LH stoping in C2 zone at Q2, Q3 and Q4 2018.

Material contracts also include an agreement to provide ore haulage services by Fournier & Fils to Camflo, a local contractor specializing in various civil works.

For underground operation constracts are in place for definition drilling (man power, supervision, equipment, material, $/m basis, Orbit Garant), production drilling (man power, supervision, equipment, material, $/m basis, C-Mac), construction work (manpower, supervision, equipment, $/hr basis, SDF), explosives (transport, material, technical support, $/item, Orica) and rock support (transport, material, technical support, $/item, DSI).

For underground equipment, contracts are in place for specific equipment on a unit approach basis based on detailed quotation by Cat-Toromont, Epiroc and Maclean.

At Sigma mill, no major contracts have been finalized up to now for mill refurbishment.

SECTION • 20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 REGULATIONS AND PERMITTING

The Quebec mining industry is subjected to federal and provincial laws and regulations. Both levels of government regulate environmental assessments and operation outputs to the receiving environment. Provincial authorities provide the project description to municipal authorities.

20.1.1 Federal Regulations and Permitting

The potential federal regime of environmental and social assessment (ESA) for the Project is established by the Canadian Environmental Assessment Act 2012 (CEAA 2012). The Regulations Designating Physical Activities list the construction and operation of a gold mine with a production capacity of 600 tonnes per day (tpd) or more as a designated project for which a description must be submitted to the Canadian Environmental Assessment Agency (CEAAg). The same applies to the expansion of an existing gold mine that would result in an increase in the area of the mine operations of 50% or more and a total production capacity of 600 tpd or more.

The Company submitted a preliminary project description to the CEAAg to ensure compliance with the CEAA 2012. Upon review of the preliminary project description, the Company was informed on September, 29 2014 by the CEAAg that the combined Sigma-Lamaque Mine and Mill Complex and the Lamaque South Project which includes the Triangle deposit will not be subject to a federal ESA. This is due to the fact that surface disturbance at Integra's Lamaque South Project accounts for only a small fraction of the combined land package (Lamaque South Project and the Sigma-Lamaque Mine and Mill Complex). As a result of the Company's acquisition of the Sigma-Lamaque Mine and Mill Complex and its integration as part of the Lamaque South Project, the CEAAg considers the Project as an expansion that will result in an increase of less than 50% of the area of the mine operations.

Since November 25, 2013, the federal Fisheries Act prohibes disturbance of fish habitat without authorization only when a project could potentially entail serious harm to fish that are part of a commercial, recreational or Aboriginal fishery, or to fish that support such a fishery. The waters located on the Lamaque South Property do not directly support a commercial, recreational or Aboriginal fishery, nor do the fish species indicated during the baseline survey performed on the Lamaque South Property support such a fishery. Therefore, an authorization pursuant to section 35(2) of the Fisheries Act, will not be required for this Project.

The Metal Mining Effluent Regulations (MMER) pursuant to section 36 of the Fisheries Act, and administered by Environment Canada, will apply in some form. The final effluent quality will be submitted to toxicity testing and to deleterious substances restrictions as the Environmental Effects Monitoring Program will continue to apply; five cycles presently are completed.

There are nuclear probes in the Sigma mill that are registered by the Canadian Nuclear Safety Commission (CNSC). These permits will have to be updated and some employee will have to be trained with CNSC standards.

20.1.2 Provincial Regulations and Permitting

ENVIRONMENT QUALITY ACT:

Key provincial permits will be required during the construction and operation of the mine. The Ministry of Sustainable Development, Environment, and the Fight against Climate Change (known as the "MDDELCC": Ministère du Développement durable, de l'Environnement et de la Lutte contre les changements climatiques) is the Québec entity responsible for environmental protection and the conservation of biodiversity to improve the environmental quality of life.

This department is responsible for the control and enforcement of laws and regulations concerning environmental protection, including the analysis of application to certificates of authorizations and other permits. The department also regulates the prevention or reduction of the contamination of water, air and soil, drinking water quality, measures against climate change, as well as the conservation and protection of wildlife and its habitats.

The potential applicable provincial ESA regime is set out in Chapter I of the Environment Quality Act (EQA) of Québec, which establishes the provisions of general application. Chapter II outlines the particular provisions of the territory covered by the James Bay and Northern Québec Agreement. The Lamaque South Project is located south of that territory so that only Chapter I is of interest for the Project.

The main sections of Chapter I of the EQA associated with obtaining certificates of authorization or other permits are section 22 (most of industrial activities that may contaminate), section 31.1 (environmental and social assessment process), section 32 (drinking water and domestic wastewater) and section 48 (atmospheric emissions). As well, now that the Project encompasses the Sigma-Lamaque Mine and Mill Complex (processing plant with waste rock storage area, tailings impoundment area and associated water treatment facility), the Company will be subject to a de-pollution attestation under section 31.11 of the EQA.

The Company is not required to complete an ESA under section 31.1 of the EQA, as planned underground production is less than the provincial threshold of 2,000 tpd pursuant to the Regulation respecting environmental impact assessment and review.

The Project is subject to section 22 of the EQA, pursuant to which a certificate of authorization (CofA) is required for activities that may result in a change in the quality of the environment. Each activity such as mining and milling may be subject different CofAs. The applications to the MDDELCC will be accompanied by sufficiently comprehensive studies to address the requirements of Directive 019 applicable to the Mining Industry. Directive 019 presents the Québec operational requirements with respect to all phases of the development, operation and closure of a mine. It includes namely, water management and effluent concentration criteria, protection of groundwater, noise and vibration, waste rock and tailings management, atmospheric emissions, hazardous materials management, site reclamation and follow-up environmental monitoring. It also requires a description of the natural and human receiving environments of the mine, potential mitigating measures to reduce environmental effects and possible compensation measures.

Other permits, authorizations, approvals and leases from both the Ministry of Energy and Natural Resources (“MERN”: Ministère de l’Énergie et des Ressources naturelles) and the MDDELCC, Régie du Bâtiment (Quebec Building Agency) and potentially the Ministry of Forests, Wildlife and Parks (“MFFP”: Ministère des Forêts, de la Faune et des Parcs), for various components of the overall project development work are required. These applications will be submitted as part of the ongoing process of developing the site and should therefore not impact the Project critical path schedule.

- These applications may be associated with the following works:
- Wood cutting
- Management of the overburden material to be reused at closure stage;
- Stripping work that could result in disrupting the soil, water or hydraulic regime;
- Mine waste water treatment basins;
- Sinking of ramps and shafts;
- Dewatering and dry maintenance of underground works;
- Storage of hazardous materials;
- High-risk petroleum products containment installation and/or oil separators;
- Ore and waste rock storage;
- Underground backfill;
- Atmospheric emissions purification devices; and
- Site reclamation.

There are established fees to be paid to the government with the applications for these various permits/approvals are normally between $1,000 and $5,000 for each specific request.

The Sigma Mill will also require a de-pollution attestation from the MDDELCC. This document is renewable every five (5) years and identifies the environmental conditions that must be met by the industrial facilities when carrying out its activities.

The attestation compiles all of the environmental requirements related to the operation of an industrial facility already stated in the former CofAs. The operator of an industrial facility must apply for a de-pollution attestation within 30 days following the issuance of the CA under section 22 of the EQA for the operation of its mine project. Once the attestation is issued, annual fees are applicable on mine operation rejects in the environment. The fees are calculated on contaminant load to air and water and tonnage of stored industrial waste on land, sludge, waste rock and mill tailings.

MINING ACT AND ASSOCIATED REGULATIONS:

The application for a mining lease must be accompanied by a survey of the parcel of land involved, a project feasibility study as well as a scoping and market study as regards processing in Québec. Unlike metal concentration which is considered as ore treatment, gold refining is considered as metal processing.

Since 2013, application for a mining lease with a production capacity of 2,000 tpd or less, while being exempt from the formal ESA process, requires a public consultation has been held in the region where the project is located before and the report sent with the application.

The Mining Act also stipulates that a mining lease cannot be granted until a rehabilitation and reclamation plan (or closure plan) is accepted and CofA for mining required under the EQA has been issued. When the closure plan is accepted, proponents have 90 days to make first payment of the security deposit of 100% of the estimated cost of reclamation work. Payments are distributed over 3 years, i.e. 50%, 25% and 25%. Rehabilitation and reclamation work must begin within three years after operations cease The MERN may, exceptionally, require work to commence before this deadline, and it can authorize an extension. An initial extension may be granted for a period not exceeding three years, and for additional periods not exceeding one year.

Since 1982, government authorities do not issue mining concession but recognized active concessions as equivalent to mining leases.

20.1.3 Current Status

After acquisition of the Sigma-Lamaque Mine and Mill Complex, all existing CofAs associated with the property have been transferred into Integra's name and are in good standing with the Québec government, including permits for milling, tailings and waste rock. All mining rights were also transferred to Integra.

MINING RIGHTS

Approximately 80% of the Lamaque South Project (including the Parallel, Fortune Zones and Triangle portal) and 100% of the land recently acquired through the acquisition of the Sigma-Lamaque Mine and Mill Complex are mining concessions recognized by MERN.

An application for a mining lease has been submitted to MERN on June 29, 2017 and the lease has been received in March 2018.

FEDERAL ENVIRONMENTAL REGULATIONS REQUIREMENTS

As mentioned above, CEAAg notified Integra on September 29, 2014 that the project described in the project notice dated August 2014 was not subjected to the federal environmental assessment process. The 2014 project description involved development of the Lamaque South property in order to resume Sigma mill activity and to mine the Parallel and Triangle deposits by underground operations.

However, other federal regulations aiming the environmental protection apply. MMER standards for effluent quality continue to apply since a final effluent is registered for many years; control and reporting activities have been continual. MMER also requires Environmental Effects Monitoring (EEM) studies to be submitted. This program was initiated by Century Mining in 2004 and resumed by Integra in 2015 with the submission of the 5th cycle study plan.

ENVIRONMENTAL EFFECTS MONITORING STUDIES CONDUCTED AT THE SIGMA-LAMAQUE MINE AND MILL COMPLEX

Environmental Effects Monitoring (EEM) studies, as required as part of Schedule 5 of the Metal Mining Effluent Regulations (MMER) under the Fisheries Act of Canada, were conducted in 2004, 2007 and 2012 (Genivar, 2013) at the Sigma-Lamaque Mine and Mill Complex by Century Mining Corporation, the former owner. These studies include the following various components: effluent characterization, sub-lethal toxicity testing, water quality monitoring, and biological monitoring (fish and benthic communities). As per the requirements of the MMER, these various components are to be examined with varying frequencies, depending on the component as well as the history of the results obtained. The three existing studies (covering the first, second and fourth cycles) only partially respond to the requirements of the MMER. It is clear from these reports, and the apparent absence of a report addressing the third cycle of the required EEM studies, that this site has not been in compliance with the MMER in terms of frequency of various designated investigations. It is equally clear that based on the data collected on surface water quality, various parameters surpass certain federal as well as provincial criteria (aluminum, copper, iron and cyanide). As well, there are indications in the reports that there has been possible development of abnormalities in resident fish. Finally, although not required by the MMER, limited sampling of sediments has indicated contamination with copper in excess of Québec MDDELCC criteria. In the future, compliance with sampling frequencies of effluent, surface water and sublethal toxicity testing as specified in Schedule 5 of the MMER will be maintained by Integra. Also, given the lack of uniformity in sampling procedures as demonstrated in the existing reports, a thorough effluent and receiving water sampling program has been established to develop a solid baseline for the future. EEM study procedures indicate that biological monitoring is to be conducted at most every three years if effects on fish and benthos are noted, or every six years if no effects were noted in the previous two studies. Sediment sampling program should also be implemented.

The Company is continuing regulatory monitoring of the quality of the Sigma mine effluent required by the application of the MMER of the Canada Fisheries Act. A reduction in the frequency of monitoring was also recently granted by Environment and Climate Change Canada in 2016. This regulation also provides for monitoring of the quality of the receiving environment and of fish populations and benthos communities in that receiving environment. The fifth cycle report was filed with Environment and Climate Change Canada in June 2016 within the prescribed timeframe, and the analyst's assessment of the findings of this study is dated December 12, 2017.

ENVIRONMENTAL EFFECTS MONITORING STUDIES EXPECTED AT THE SIGMA-LAMAQUE MINE AND MILL COMPLEX

Environmental Effects Monitoring (EEM) studies, as required as part of Schedule 5 of the Metal Mining Effluent Regulations (MMER) under the Fisheries Act of Canada, were conducted in 2004, 2007 and 2012 (Genivar, 2013) at the Sigma-Lamaque Mine and Mill Complex by Century Mining Corporation, the former owner. These studies include the following various components: effluent characterization, sub-lethal toxicity testing, water quality monitoring, and biological monitoring (fish and benthic communities). As per the requirements of the MMER, these various components are to be examined with varying frequencies, depending on the component as well as the history of the results obtained.

The three existing studies (covering the first, second and fourth cycles of MMER-required monitoring) only partially respond to the requirements of the MMER. It is clear from these reports, and the apparent absence of a report addressing the third cycle of the required EEM studies, that this site has not been in compliance with the MMER in terms of frequency of various designated investigations. It is equally clear that based on the data collected on surface water quality, various parameters surpass certain federal as well as provincial criteria (aluminum, copper, iron and cyanide).

In the future, compliance with sampling frequencies of effluent, surface water and sub-lethal toxicity testing as specified in Schedule 5 of the MMER will be maintained by Integra. Also, given the lack of uniformity in sampling procedures as demonstrated in the existing reports, a thorough effluent and receiving water sampling program will be established, as well as one addressing sediments, in order to establish a solid baseline for the future.

The Company is continuing regulatory monitoring of the quality of the Sigma mine effluent required by the application of the MMER of the Canada Fisheries Act. The fifth cycle of MMER-required monitoring was filed with Environment and Climate Change Canada in June 2016 within the prescribed timeframe, and the analyst's assessment of the findings of this study were received in December 2017.

Environment and Climate Change Canada is requesting next cycle study report by June 6, 2019. Study plan shall be approved by them before the study is launched and shall include explanation of the constraints with sampling the remote exposed zone and a discussion of potential effects in the distant area. As field work can only be conducted during no ice seasons, it will have to be conducted during 2018, the study plan shall then be submitted during first quarter 2018 except if an agreement is taken between the federal authorities and the site management to shorten the approval of study plan period.

PROVINCIAL ENVIRONMENTAL REGULATIONS REQUIREMENTS

CERTIFICATES OF AUTHORIZATION

Existing certificates of authorization transferred to Integra from Century with respect to the Sigma-Lamaque property are:

- 7610-08-01-70096-21 – Exploitation of Sigma II open pit mine
- 7610-08-01-70095-23 – Expansion of emergency tailings pond C
- 7610-08-01-70071-25 – Exploitation of Lamaque Mine open pit
- 7610-08-01-70071-26 – Exploitation of Lamaque Mine
- 7610-08-01-70071-27 – Exploration of Underground Lamaque 2000
- 7610-08-01-70095-28 – Exploitation of Sigma Mine and mill including the modification related to the tailings dams increase dated on October 27, 2005
- 7610-08-01-70095-31 – Exploitation of Sigma mine at 2500tpd down to Level 12 including the modification related to referenced mining rights dated on August 6, 2015
- 7610-08-01-70095-32 – Creation of a mine water pond in cell B-9 of tailings impoundment
- 7610-08-01-70095-33 – Usage of material from tailings pond for rehabilitation of waste rock pile.

Existing certificates of authorization issued to Integra are:

- 7610-08-01-70182-20 – Underground water intake to provide industrial water supply to Triangle Zone
- 7610-08-01-70182-21 – Exploration Development of Triangle Zone and relocation of waste pile

- 7610-08-01-70182-22 – Usage of waste rock from Triangle for on site construction workings
- 7610-08-01-70182-23 – Installation of oil-water separator on Mining Concession 375
- 7610-08-01-70182-24 – Connection to municipal services for Triangle Zone
- 7610-08-01-70182-27 – Industrial zone waste pile extension
- 7610-08-01-70182-29 – Exploitation of Triangle Zone
- 7610-08-01-70182-30 – Treatment of mine water from Triangle exploitation
- 7610-08-01-80859-00 – Exploitation of a sandpit – Triangle Zone

With the mining lease, these authorizations allow the exploitation of the Triangle deposit.

The ore will be trucked to Sigma Mill where it will be processed into gold bars. Although gold processing produces almost as much tailings as the plant feed. The average grade of the deposit is around 7gAu/t which means 99,999993% of the material will be directed to the tailings.

TAILINGS MANAGEMENT PERMITTING

Already authorized modification of tailings pond (7610-08-01-70095-28 - modification related to the tailings dams increase dated on October 27, 2005) allows dam increase of 10 feet (3m) with respect to cells B1 and B2. This represents a volume of approximatively 925 000 m3 or about 1 250 000 tm.

The anticipated scenario to dispose all of the tailings entailed by the Triangle exploitation is summarized as follow:

- Milling at Sigma, deposition of 400 000 tm in B1 during Sigma 1 pit preparation to receive tailings.
- Milling at Sigma, deposition in Sigma pit.
- B2-basins could be used as emergency pond.

The detailed deposition concept is described in section 20.4.2.

To ensure this concept complies with current regulations, some verifications shall be done with MDDELCC analyst. As mentioned, cells B1 and B2 are authorized for an increament of 3m by the certificate of authorization 7610-08-01-70095-28 modified on October 27, 2005. However since that time the stability criteria have been modified. The baseline recurrences for a cyanide-laced tailings retention area had increased from 1000 years to 2000 years. Presently, static and dynamic stability criteria as defined in the Mining Reclamation Guide (MERN, November 2017) are not respected.

Before proceeding with the construction granted in 2005, the dams will have to be stabilized. This exercise can be done in two stages, static stabilization, then dynamic stabilization. The two stages of construction can not be performed at the same time; the static stabilization must be consolidated before adding additional weight to the lateral clay foundation in order to prevent slippage.

Although, static stabilization workings can be performed at the same time as the increament. Calculations done by Amec Foster Wheeler (see section 18) to upgrade stability and construct the

3m enhancement of B-1 and B-2 basins demonstrate the deposition of 400 000t of residues in the B-1 basin is possible within the limits authorized by the certificate of authorization of 2005.

The second phase, which consists of dynamic stabilization, will require the construction of a berm, which will encroach outside the Integra terrain. Integra will need to acquire the land and apply for wetland work with a compensation proposal before obtaining the certificate of authorization for completion of Phase 2.

Subsequently, cells B-1, B4 and B9 can be closed and restored; B2 could remain an emergency basin until the end of the activities.

During the deposition period in cell B-1, the Sigma pit will be set up to receive the tailings, in form of paste, mixed with cement. In addition to avoid increasing the footprint of operations, this method will chemically and physically stabilize the tailings produced by the exploitation of the Zone triangle.

Thereby, most of the tailings produced by the exploitation of the Triangle Zone will be stored as cemented paste in the Sigma pit. The pit will be filled up to level 259 m which represent approximately a third of its capacity.

n application for a certificate of authorization should therefore be prepared this Spring for the paste plant including cement mixing plant and the use of the pit as a permanent storage area for cemented paste.

RECLAMATION PLANS

Reclamation plans are discussed in section 20.7

FUEL SUPPLY

Integra holds two licenses with Régie du Bâtiment. First one registers a 50 000L tank on Corporation d'Or Integra name, this permit expires on 2018/08/18. That tank is located at Triangle site and is used for diesel storage. Second one registers a 4800L diesel tank at Sigma site on Or Integra-Quebec name, this permit expires on 2018/04/04. The latter one is expected to be dismantled during Summer 2018.

Gasoline supply for the operations will be provided by local gas station.

All these permits are renewable on demand.

RECENT NON COMPLIANCE NOTICE

There was a non compliance notice issued in May 2017 regarding an exceedance of the Iron criteria in the effluent. The root cause was identified and correction were applied in order to avoid any other offense.

UPGRADE OF THE SIGMA MILL

In order to process ore from Triangle, mill refurbishment will include repairs, equipment replacement and new equipment, instrumentation and control points. The upgrading of the Sigma Mill and associated facilities will include the following works requiring modification of CofAs or permits:

- Modifications to the existing Sigma processing plant including detoxification unit (cyanide removal)
- Upgrading Sigma tailings disposal facilities to recent stability standards
- Sigma tailings disposal facilities effluent control and treatment plant
- Paste plant with cement mixing and in-pit tailings disposal
- De-pollution attestation – in process. The application was submitted by former administration
- Site reclamation (see section 20.6).

20.2 ENVIRONMENTAL BASELINE STUDIES

20.2.1 Lamaque South Property

The environmental baseline study, as described in the section below, had supported various applications for approval, as well as provided important input for the closure plan related to the bulk sample. The information contained in the environmental baseline study was used for the preparation of the operation CofA application for the anticipated mine production at Triangle Zone.

An environmental baseline study of the Lamaque South Property was undertaken on behalf of Integra by Amec Environment & Infrastructure in 2013 (Amec Environment & Infrastructure, 2014). The study reviewed available information across a number of disciplines, including geology and soils, hydrogeology, hydrology, air quality and noise, flora and fauna, socio- economic setting and archaeology.

The Lamaque South Property is located in the Superior Province and Abitibi Subprovince of the Cadillac Deformation Zone. This region is known as a greenstone belt, due to the degree of rock deformation and metamorphism forming greenschist facies. The area around Cadillac, Malartic and Val-d’Or hosted many operating mines in the past.

The on-site surficial deposits are mostly of glacial origin, but the northern portion of the Lamaque South Property is occupied by a tailings impoundment area, the result of earlier mining of a portion of the overall site. Surficial deposits in the other portions of the property include: glacial lake deposits (e.g., mostly stratified sand and silt), marsh deposits (e.g., organic), glaciofluvial deposits (e.g., sand and gravel esker) and glacial deposits (e.g., till).

The Abitibi region is known for the quality and quantity of groundwater contained in its eskers. The Lamaque South Property is partially located within the urban limits of Val-d’Or, where the aquifer potential appears average. Val-d'Or uses the water contained in Harricana’s interlobate moraine for its drinking water supply. This aquifer, which crosses Val-d’Or along a north-south axis, has branches that run adjacent to the Lamaque South Property (depending on the map of surface deposits consulted). No hydraulic connection between the groundwater contained in the moraine and this property, particularly with respect to mine dewatering, has been identified during a hydrogeological study conducted in 2014- 2015 (Richelieu Hydrogéologie, March 2015). It should be noted that when the earlier Sigma and adjacent Lamaque mines were in operation, no concern was identified concerning the town's water supply. Sampling and analysis of groundwater performed in 2013 from eight exploration boreholes indicated that several constituents exceeded both provincial and federal standards, suggesting that mine water would require treatment prior to discharge to the receiving environment.

The Lamaque South Property is located within a large watershed of approximately 710 ha, drained by a minor unnamed stream. The stream is typical of wetland regions with thick aquatic vegetation lining the shores, varying widths and depths, and a bed composed of a mixture of organic material and small stones. The area experiences frequent precipitation events; on average, it receives 914 mm of precipitation, of which about 314 mm falls as snow. Under extreme conditions, the stream draining the property can be expected to collect a peak flow reaching 8 m3/s for a 1-in-100 year rain event.

The Abitibi-Témiscamingue region of western Québec experiences a sub-Arctic, humid continental climate which is characterized by short warm summers and long, cold, snowy winters. No ambient air quality measurements were available for the property or in close proximity to it. Regional National Air Pollution Surveillance stations (55 km to the northeast, 94 km to the west) indicated that ozone and PM2.5 exceeded, at times, provincial and/or federal standards, while SO2 was at acceptable levels. The only greenhouse gas (GHG) emitter registered around the Project site is the Canadian Malartic. The projected GHG emissions from Eldorado's Lamaque South Project are expected to be below the declaration threshold.

The majority of the Lamaque South Property is characterized by organic deposits of variable thickness generally concentrated in low-lying areas. The southern part of the study area is characterized by a mixture of terrestrial and wetland ecosystems. Many wetlands are present on the edge of the tailings impoundment area and they occupy about 50% of the study area. Wetland ecosystems are protected under provincial legislation, and authorization is required prior to their disturbance. Some logging is also present. No "at risk" plant species were observed on this property, and no exceptional forest ecosystem was identified. Finally, no wildlife habitats, as described in the provincial Regulation respecting Wildlife Habitats, are present on the property. Woodland caribou, a vulnerable species in Québec, frequent a so-called "buffer zone" some 5 km from the Lamaque South Property. Woodland caribou is designated as threatened under the federal Species at Risk Act (SARA).

The presence of 12 mammal species was identified, in addition to four bat species. One of the bat species is considered "endangered" by Canada, although it is not protected under provincial legislation. Provincial authorities representative have suggested the creation of some bat nurseries in the site surroundings in which Integra had agreed to participate.

Of the 91-bird species identified on this property, three threatened species pursuant to federal legislation have been identified (Canada warbler, Common nighthawk and Olive-sided flycatcher). While these species are not legally protected by Québec, they are recognized as susceptible to being at risk.

Two species of snakes and five species of amphibians were also observed. None of these species presents an impediment to the development of the Lamaque South Property, as long as certain mitigation measures are observed.

One stream, with two tributaries arising from wetlands on the property, provides habitat for six species of fish and nine taxa of benthic invertebrates. None of the fish species are exploited by commercial, recreational or Aboriginal fisheries.

Should an activity affect a species listed under SARA, an agreement with the Minister or a permit pursuant to section 73 may be required.

20.2.2 Sigma Mill property

The area surrounding North and eastern sides of the tailings impoundment was reassessed in 2017 (Amec, 2017). Wetlands occupy an area of 86.8 hectares and form the dominant ecosystems of the area study. These are mainly stands dominated by black spruce and tamarack, forming swamps and peat bogs. Conditions were identified as favorable for fish habitat; scientific fishing program showed three healthy species dominated by cyprinids. No precarious species were observed during the reconnaissance.

Several wildlife observations have been reported, including 3 species of amphibians, 22 species of birds and 6 species of mammals. This type of habitat could potentially host 2 species of turtles.

20.3 TAILINGS IMPOUNDMENT AREA DESCRIPTION AND CAPACITY

20.3.1 Old Sigma Tailing Storage Facility (TSF)

As mentioned in section 18.23.1, the tailings impoundment is made up of four separate basins: B-1, B-2, B-4, and B-9. A recirculation pond and a polishing pond are located to the east of those four cells. The tailings impoundment is currently under the care and maintenance of the Company.

The elevation of the existing dikes containing these cells are as follows: 320.14 m for B-1; 320.14 m for B-4, and 320.14 m for B-9, while the elevation of the existing dikes containing cell B-2 is 318.31 m. B-4 and B 9 are currently at maximum tailings storage capacity, B-2 is near maximum tailings storage capacity, and B-1 has available remaining storage volume.

As mentioned in chapter 18, the projected tailings production for the Lamaque Gold project is estimated to be 3.8Mt, for seven years of operation (see Table 20.1). The Sigma open pit will be used to store 3.4Mt of the tailings in form of paste fill. The Sigma Tailings Impoundment will be used as a secondary storage facility only. he total tonnage of tailings to be stored in the Sigma Tailings Impoundment is about 0.4 Mt, which is equivalent to 0.29 Mm3 (See Table 20.1 below).

The intent is to use the Sigma TSF during construction of the Sigma Open pit storage facility and later as emergency storage.

Table 20.1: Projected Tailings Storage at the Sigma Tailings Impoundment

Year	Tailings Storage (t/year)	Cumulative Tailings Storage (t)	Tailings Storage Volume (m3/year at 1.4 t/m^3)	Cumulative Tailings Volume (m3 at 1,4 t/m^3)
2018	15,000.00	15,000.00	10,714.29	10,714.29

2019	330,000.00	345,000.00	235,714.29	246,428.58
-	55,000.00*	400,000.00	39,285.71	285,714.29

* 55,000 tons of tailings will be reserved for emergency storage in the Sigma tailings impoundment.

The historical deposition strategy for the Sigma tailings impoundment involved having one active cell for tailings storage, while using two inactive cells for water management and treatment, and keeping the fourth cell ready to be raised. The water management strategy was used by the previous owners in compliance with the requirements of the 2005 Directive 019. However, after the regulatory changes following the update of the Directive 019 in 2012 (higher rainfall events for water management for cyanide tailings), the tailings and water management strategy has been updated since.

The proposed tailings and water management strategy will be similar to that used in the past.The contaminated water will be managed and treated inside the tailings basins (B-1, B-2, B-4, and B-9). The Sigma Tailings Impoundment will be used to store the Directive 019 design flood of the Sigma impoundment area, as well as treat water. In addition, considering that the basins B-4 and B-9 are full, tailings will mainly be deposited in basin B-1.

After determining the water management concept, a high-level fill plan model was developed considering the following limitations:

- Based on stability analysis (Geotechnical report TX17012803-01000-RGE-0001-0 (Amec Foster Wheeler, 2017), “Étude géotechnique et options d’entreposage des résidus et d’eau à court et long terme, Parc à résidus Sigma, Val-d’Or (Québec)”;
- The Sigma Impoundment Area topography dated 2014, (CAD file: RESIDU_MAJESTIC.dwg);
- The Sigma Impoundment Area bathymetry dated March 2015 (SYLVESTRE, JULIEN, LECLERC, CAD file: S27983.dwg);
- The recirculation and polishing basin bathymetries, revised on July 2015 (CORRIVEAU J.L. & ASSOC. INC, CAD file: C9821_PLAN_BATHY_FINAL.dwg).

The study uses the following assumptions as a baseline for the tailings which will be stored in the Sigma Tailings Impoundment:

- The historic tailings-deposition slope has been measured to be 1% sub-aerial and 2% subaqueous, based on the geomatic information available. These values will be used in the current study;
- The average in-place tailings dry density will be 1400 kg/m3, as observed in some gold mines in the Abitibi region;
- The solid content of the tailings slurry has been estimated to be 50% by weight;
- The porosity of the in-place tailings has been estimated to be 30%. It is assumed that this volume will be permanently occupied by the slurry water.

In order to be compliant with the current Directive 019 (MDDELCC, 2012), the Sigma water management infrastructure, for cyanide tailings, must be able to handle a design flood event that is specified as the volume of water generated by a 24-hour rain event with a 2000-year recurrence, combined with a 30-day spring melt event with a 100-year recurrence.

The precipitation data used for this project is presented in Table 20.2.

Table 20.2: Estimated Project Precipitations

Precipitation	Recurrence	Value	Units
24h Rainfall	1 : 1000 years	93.70	Mm
24h Rainfall	1 : 2000 years	99.30	Mm
24h PMP	Undetermined	221.05	Mm
30-days snowmelt	1 : 100 years	521	Mm

The raw 30-day snowmelt, based on estimated data, is 521 mm. This study does not account for climate change and should be reviewed every 5 years. All of these assumptions are significant, and therefore should be confirmed in a future phase of the project, according to the IPCC conclusions.

The proposed water management strategy is decribed in chapter 18 and summarized below:

- The Sigma Tailings Impoundment will served as storage for the Directive 019 design flood of the Sigma impoundment area, as well as water treatment;
- Contaminated water from the paste fill plant or from the mill will be sent to the Sigma Tailings Impoundment to be managed (a maximum flow rate of 80 m3/hour was estimated by WSP);
- Once the treatment process completed, water will be transferred by pumping to the recirculation basin and from there to the mill, or to the polishing pond for water quality control and monitoring before being released to the environment;
- Water from the Triangle zone (infiltration and runoff water), as well as the mine drainage water, is sent to a treatment system call the Mud-Wizard for treatmentand then sent to the Sigma-Lamaque underground mine stope;
- Runoff water from Lamaque Pit is also managed in the underground mine stope;
- The water from Sigma-Lamaque underground mine stope, similarly to the former water management strategy, is transferred by pumping to the polishing pond to be discharged to the environment;
- During the design flood, the Sigma Pit, used as the main tailings storage area, must be able to store its own legislative flood precipitations each Spring;
- During normal operations, the contaminated water from the Sigma open pit can be directed inside the Sigma Tailings Impoundment for treatment.

The design water balance of the tailings impoundment has to be updated as per current regulation and as presented in the newly proposed water management strategy. Other water inputs into the tailings impoundment during the design flood include contaminated water from the paste fill plant or from the mill. The existing polishing pond and recirculation basins have negligible storage capacities, but have however been considered in the water management scheme and will still be used in their present capacity.The design water balance for the tailings impoundment is presented in the table below.

Table 20.3: Design Water Balance for the Sigma Tailings Management Concept*

Water Volume		Unit	Value
Input**	Net Runoff	m^3	188,936
	Net Spring Melt	m^3	991,194
	Water from Concentrator (for 1 month)	m^3	57,600
	Total Input	m3	1,237,730
Output	Water trapped in voids (for 6 months) *	m^3	21,429
	Total Output	m^3	21,429
Water Volume to be stored (Directive 019, 2012)		m^3	1,216,302

* 30% of the slurry water is trapped in voids in the tailings (estimation based on site experience).

** Return of water to the mill is at least equivalent to slurry water

Based on the bathymetric survey provided by Eldorado Gold, the total currently available volume of the existing Sigma Tailings Impoundment is approximately 1.01Mm3 (considering a 1-metre freeboard according to Directive 019, MDDELCC, 2012, and 0.4Mt of tailings to be stored in the Sigma Tailings Impoundment). In order to manage 0.4Mt of tailings while managing the design flood, the Sigma Tailings Impoundment has to be raised, more specifically basin B-2. The mentioned basin, currently at the elevation of 318.50m, will be raised to 320.00m. The projected water storage capacity of the Sigma Tailings Impoundment after the raising of the basins B-2 will be 1.23 Mm3.

An evaluation of the stability of the dykes of the Sigma Tailings Impoundment was performed under static and dynamic loading, based on current regulations (Directive 019, March 2012). The results obtained from the mentioned evaluation have shown that the Sigma Tailings Impoundment does not currently meet the minimal factors of safety.

Under static conditions, all the factors of safety for the basins of the Sigma impoundment were below the minimum specified in Directive 019, for drained and undrained conditions. The same results were observed under dynamic conditions. The details of the analysis and the mechanical parameters of the materials are presented in the report titled “TX17012803-01000-RGE-0001-0, Étude géotechnique et options d’entreposage des résidus et d’eau à court et long terme” (Amec Foster Wheeler, 2017).

Consequently, a stability berm and a roc shell will have to be built on the downstream of all the cells of the impoundment before the reutilization of the Sigma tailings impoundment.

Considering the tailings planning for the Sigma Tailings Impoundment and the stabilization of the basins that will be carried out in phases, the following sequencing/planning is proposed in table 20.4:

Table 20.4: Tailings Planning and Sequence of Construction

Years	Projected Tailings Storage at Sigma Tailings Impoundment*	Required Tailings and Water Management Infrastructure	Complementary Notes
2018	15,000 tons of tailings at basin B-1	Raising of the Basin B-2 of the Sigma Tailings Impoundment, from 318.50 m to 320.00 m of elevation and construction an emergency spillway Construction of the Phase 1 berm around the tailings impoundment. Reconstruction/relocation of the eastern and southern seepage ditch Reconstruction of a portion of the western seepage ditch Lower the Northern and Eastern sections of B-9 from 320.00 m to 318.50 m of elevation and reconstruction/relocation of a portion of the dyke upstream. Reprofiling the median dyke between B-1 and B-2 from 319.50 m to 320.00 m of elevation Reconstruction of the operational spillway between B-2 and B-1, reconstruction of the emergency spillway at B-9 Relocation of the power line southeast of the Sigma Tailings Impoundment HDPE piping works	The raising of B-2 will be 15m upstream from the current alignment The northern dykes of B-1 and B-9, and the Eastern dyke of B-9 will be relocated 15m upstream from their current alignment Slurry water will be drained by gravity to B-2 and pumped to the polishing pond to be discharged to the environment or pumped to the recirculation basin to be returned to the mill Dust control procedure to be implemented at inactive cells B-4 and B-9
2019	330,000.00 tons of tailings at basin B-1	Installation of the mobile treatment plant at the polishing pond	The same water management strategy as above will apply
2020	None	Construction of the Phase 2 berm around the tailings impoundment. Reconstruction/relocation of the northern and western seepage ditches	The same water management strategy as above will apply

* 55,000 tons of tailings will be reserved for emergency storage in the Sigma tailings impoundment.

20.3.2 Sigma pit

Tailings management approach is presented in Sections 18.26. The former Sigma pit offers a capacity of approximately 13,5 million m3 close to the process plant, where the current life of mine plans requirement for 2,6 million m3 at elevation 259 m (planned spillway being at elevation 306.5 m). The tailings management approach selected uses the existing permeability of the open pit to limit the amount of water that would turn into contact water and need treatment. A layer of crushed rock will be installed (a pervious surround, e.g., MEND 2015) to surround the tailings and act as a drain for run-off. Accordingly, most of the water reporting to the open pit over the life of mine will continue to be managed with the existing dewatering system; only the water directly in contact with the tailings surface (direct precipitations and bleed water from tailings) will be managed with a contact water pond and pumped to the mill using a reclaim system.

20.4 TAILINGS MANAGEMENT FACILITY (TMF) OPERATIONS

In this section, tailings management, mine water and process water treatment, and site drainage are briefly summarized.

20.4.1 Tailings Management at the Old Sigma TSF

The projected tailings production for the Lamaque Gold project is estimated to be 3.8 Mt, for seven years of operation. The current concept consists of using the Sigma open pit to store 3.4 Mt of tailings in form of paste fill. The Sigma Tailings Impoundment will be used as a secondary storage facility only, and is expected to hold up to a maximum of 0.4 Mt of tailings.

Considering that the basins B-4 and B-9 are full, tailings will mainly be deposited in basin B-1. A supernatant pond will be maintain in basin B-1 which can be used for sedimentation purposes. The basin B-2, once raised, to elevation 320,00m will served as a treatment basin for cyanide destruction.

The operational strategy of the tailings impoundment currently involves the progression of the tailings deposition from the east to the west in the basin B-1, sequenced in a way to avoid the obstruction of the water flow from basin B-1 to the basin B-2.

20.4.2 Tailings Management at the Sigma pit

Tailings discharged in the Sigma open pit will be under the form of cemented paste backfill having wet and dry densities respectively estimated to 1.85 t/m3 and 1.32 t/m3.

Cement will be added to the tailings; 1.5% cement by weight of dry tailings is estimated for the purpose of the prefeasibility study. This proportion will be confirmed by ongoing tests performed at URSTM. Cement is added to the tailings to prevent for liquefaction, and 1.5% is the typical proportion used to avoid liquefaction in underground mines.

The deposition plan was prepared using Muck 3D, a program developed by MineBridge Software. The deposition plan drawing (see section 18.25.1) shows the yearly development of the tailings in-pit disposal over the life of mine. Deposition is expected to use a single end pipe discharge point system to build and maintain the contact water pond. For the purpose of planning, an average deposition slope of 2% above and below water was assumed, considering that the viscosity of the cemented paste will be controlled and the operating objective is of minimizing as much as possible the amount of water stored in the internal water pond to allow the cemented paste to strengthen. A reclaim pumping system is planned from this pond to the process plant. The cement content and the viscosity of the cemented paste backfill will be adapted, depending on the discharge location in the pit and the vertical position in the stack to limit the bleed water and the potential infiltration in the former underground mine under the pit.

20.4.3 Mine Water and Process Water Treatment

Mine and process water will require treatment prior to discharge to the receiving environment:

Water from the Triangle zone (infiltration and runoff water), as well as the mine drainage water, is sent to a treatment system call the Mud-Wizard for treatment (mainly for total suspended solids), and then sent to the Sigma-Lamaque underground mine stope. Runoff water from Sigma Pit is also managed in the underground mine stope.

The water from Sigma-Lamaque underground mine stope, is transferred by pumping to the polishing pond to be discharged to the environment.

The pumping system at Sigma-Lamaque underground mine allow fluctuation of water level of about 60m. That installation can manage design flood from Sigma pit.

The Sigma tailings impoundment will manage and treat its own water (runoff and slurry water) as well as contaminated water from the mill and the paste fill plant.

With the raising of basin B-2, the current tailings impoundment will have enough capacity to handle 0.4Mt of tailings as well the volume of water of the design flood of D019.

Further flexibility for water management is highly recommended and alternate water treatment system (beside the natural degradation process) should be put in place. A mobile water treatment station can be installed as an added security measure to ensure water quality compliance, before being discharged to the polishing pond and to add more flexibility for the water management of the site.

Based on past projects with similar setting, the mobile water treatment can feasibly be sized to treat an effluent water with 10 mg/L of total cyanides at a flow rate of 5000 m3/d. The mentioned mobile water treatment station, can be installed near the polishing pond and will consist of several units for managing any residual cyanide in the decant water. It will be composed of:

- A pump station to feed the treatment station;
- Ozonator and reactor for oxidation of cyanide, into less toxic species;
- A ferric sulfate system for the treatment of any strong acid cyanides, ferric ions or any other metals that could be released from the treatment process;

- A filtration process with geotubes for the removal of solids from the previous step. Filtered water is collected and discharged to the polishing pond.

The design of this mobile water treatment station can be further details in a future phase of this project. An investment and operation cost estimation for this type of mobile treatment station is presented in chapter 21.

20.5 WASTE ROCK MANAGEMENT

The waste rock from the Triangle will be disposed of on-site in a dedicated storage pile located near the mine portal at Triangle, with control and capture of run-off from the waste rock.

Run-off from the waste rock pile will be collected by collection ditches. This water will be sampled, then pumped and conveyed to an underground stope (LAM-7-76-140). Underground, the water from Lamaque South Project will follow the flow of mine water from the Sigma mine where it will be re-pumped from underground to surface and will undergo the same treatment methods.

This strategy is included in the C of A relating to the Triangle Zone exploitation (7610-08-01-70182-29)

20.6 CONSULTATION ACTIVITIES – SOCIO-ECONOMIC SETTING

20.6.1 Socio-Economic Setting

An investigation of socio-economic information was carried out for the Lamaque South Project as part of the environmental baseline study referenced above. This investigation relied solely upon information available in the public domain and did not collect any primary information.

The local study zone and the regional study zone considered for the socio-economic baseline are broader than that typically used for the biophysical baseline studies. As such, it can be used for both the Lamaque South Property and the Sigma-Lamaque Mine and Mill Complex, now known as the Lamaque South Project.

The Lamaque South Project is located in the Abitibi-Témiscamingue Administrative Region (08) in the Vallée-de-l'Or regional county municipality (RCM, or MRC in French). The Project area falls entirely within the territory of the municipality of Val-d'Or. Responsibility for land use planning is divided between the MERN, the RCM of La Vallée-de-l'Or and the municipality of Val-d'Or.

According to the Val-d'Or zoning plan, the Lamaque South Project is located in an area zoned as "natural resources", within which mining operations are a conforming use. The northwest portion of the Lamaque South Property, where no mining operation is planned, is located within the Val-d'Or urban perimeter, a zone mainly designated and used for residential and commercial purposes, where no extractive or high-impact industries are permitted. Also, wood harvesting rights are present in the southeastern portion of the Project.

The Public Land Designation plan of the Abitibi-Témiscamingue region did not identify any protected designated zones in or adjacent to the Lamaque South Property.

The Project area is part of the area for which the Algonquin Anishinabeg Nation Tribal Council asserts land claims. The Nation Anishnabe du Lac Simon is potentially affected by the Project.

An Archaeological Potential Study was conducted and revealed no areas of interest on the Lamaque South Property.

Two sites of historical interest (the Bourlamaque Mining Village and the historical former Lamaque mine) are located within the Sigma-Lamaque Mine and Mill Complex property that is part of the Lamaque South Project. With the integration of the Sigma-Lamaque Mine and Mill Complex and the Lamaque South Property, the Forestel Hotel is now located inside the Lamaque South Property. Also, the Trans-Québec network of snowmobile trails, which is a regional recreational focus area, overlaps with the southern portion of the Lamaque South Property. Operations and truck movements, as well as vibrations from blasting, could be an issue for the Forestel Hotel, the two historical sites, residential areas adjacent to the Project and snowmobile track activities.

Since September 2013, the Company has initiated a proactive information and consultation strategy with its stakeholders. It has conducted four phases of information and public consultation with approximately 40 stakeholders, reaching more than 600 persons.

20.6.2 Consultation Activities

As documented on the Lamaque website (www.eldoradogoldlamaque.com) a consultation committee and a follow-up committee were created respectively in 2014 and 2015, the latter being the logical evolution of the first.

Information and consultation meetings regarding the Lamaque South Project were hold from September 2013 to January 2015. During this period a consultation committee was active to ensure every interested party would be involved.

The information-consultation meetings began late 2013 and were intended to present with transparency the Lamaque South Project as well as to gather concerns regarding Integra's current and future activities. There were private and public meetings.

Private meetings first took place in small focused groups; then these meetings were open to residents of the village Bourlamaque and a sector of the Sigma district both located in the close vicinity of the project.

Public meetings were open to everyone and advertised through various channels of communication, local radio spots, local newspaper ads and mass mailing. The meetings revealed that dialogue was initiated early on in the Project’s development and that Integra took its information and consultation activities seriously. The Project’s location and planned underground operations reassured residents. Other concerns were mainly about noise, blast vibrations, traffic and road safety, dust, community benefits and mine site restoration. Integra committed to make a social contribution to education and leave a positive legacy for the community. In June 2014, during a public meeting, a Follow-up committee was created to bridge with the consultation committee. A report summarizing 2015-2016 activities is published on the Lamaque website as minutes of March,8 2017 meeting.

20.6.3 Consultation with Aboriginal Community

Integra met with members of the Algonquin community of Lac Simon three times to introduce them to the Project and to collect their concerns. However, they chose to participate actively in the Consultation process and one representant of Lac Simon community is present on the Follow-up Committee. More meetings, exchanges have taken place to continue along the same momentum and develop a partnership, with the training centre (CFP) associated with the local Schoolboard (CSOB), to train young Anishnabe as operators.

20.7 RECLAMATION

20.7.1 Reclamation -Triangle Zone

Buildings and other infrastructure will be dismantled and demolished at the end of mining activities. Management of dismantled materials will be done in compliance with the Guide des bonnes pratiques sur la gestion des matériaux de démantèlement, published by the MDDELCC. Non-hazardous residual materials will be managed by disposal at approved landfill sites nearby.

In May 2015, the provincial CofA was received for the underground exploration decline development at Triangle Deposit (7610-08-01-70182-21 – Exploration Development of Triangle Zone). Issuance of this C of A was conditional with the acceptance of a closure plan specific to the development project and securement of a bond which represent total cost estimated for closure of the exploration development project. In 2015, a bond of 665 410$ was issued to secure theTriangle Bulk Project environmental disturbances. Now that the Triangle deposit is to become a mining operation, the reclamation work related to the bulk Project will not be carried out and the Closure Plan have been updated. According to MERN files, as of February 28, 2018 an amount of 665 410$ is secured regarding the Triangle Project.

The recent version of reclamation plan which deals with closing the Triangle operation was submitted to MERN on November 2017 and approved on February 28, 2018. The total financial guarantee is $1 918 600. According to MERN files, as of February 28, 2018, an amount of 665 410$ is secured regarding the Triangle Bulk Project is considered as part of first payment.

According to the MERN correspondence dated on February 28, 2018, the residual ($1 253 190) will have to be secured as the following on approval of the plan:

- Within 90 days of Feb 28, 2018 : 293 890$ - paid on March 12, 2018
- Feb 28, 2019: 479 650$
- Feb 28, 2020: 479 650$

20.7.2 Reclamation - Sigma-Lamaque Mining Complex

A rehabilitation and reclamation of the land is required under section 232.1 of the Mining Act (M-13.1 A) for the Sigma-Lamaque Mining Complex and for the Lamaque South Project. A plan for that purpose must be prepared according to the new Guide de préparation du plan de réaménagement et de restauration des sites miniers au Québec (translated as Guide for the preparation of a redevelopment and restoration plan for mining sites in Québec (November 2017)). The closure plan must be reviewed every five years, but significant changes to a project might also trigger the need for an update, at the request of MERN.

The closure plan for the Sigma-Lamaque Mining Complex is currently being reviewed to include the new approach related to the tailings management, as described in the current pre-feasibility study. The revised closure plan for this property will be submitted in early April 2018. This updated closure plan will replace the previous closure plan submitted in January 2010 by Century Mining and approved in May 2012.

The associated guarantee must cover not only reclamation costs associated with the accumulation areas (waste rock piles, tailings impoundment, polishing and recirculation ponds), but all costs for the entire mine site and associated infrastructure.

Buildings and other infrastructure will be dismantled and demolished at the end of mining activities. Management of dismantled materials will be done in compliance with the Guide des bonnes pratiques sur la gestion des matériaux de démantèlement, published by the MDDELCC. Non-hazardous residual materials will be managed by disposal at approved landfill sites nearby.

Waste rock piles are generally considered safe for the long term, according to geochemistry data obtained from samples submitted for analysis. Eldorado will investigate the possibility of reutilizing the material for backfilling of galleries, or Eldorado Gold might choose to use it for its civil needs at surface or to offer the material to local contractors for civil works and thus reduce the use of borrow-pit material for construction. The impacted area will then be managed in keeping with the requirements of the EQA and backfilled or leveled with overburden and top soils (kept in a proper storage area during the mine's life) to re-establish vegetation on both sites. A natural water regime will be re-established on the sites. It is currently assumed that the pit will be converted to an artificial lake at site closure.

Considering the estimated amount of mine tailings (3.4 Mt) that will be stored in the Sigma open pit, the pit will be only partially filled. Closure of the Sigma open pit, used as tailings storage under the form of cemented paste backfill, will start when the pumping system maintaining the groundwater level below the pit bottom will stop. Time to flood the open pit to the planned spillway elevation of 306.5 m will be estimated during the next stage of the project. During post-closure, surface runoff and precipitations reporting to the pit will flow through this spillway conveying the overflowing water from the northwest side of the open pit to the channel along Highway 117 (see Figures 20.1 and 20.2). During open pit flooding and post-closure periods, the pervious surround (see Section 18.23) will limit groundwater contact with the tailings mass. The necessity of having a pervious surround will depend on the tailings potential impacts on groundwater quality downstream of the pit. Potential impacts will depend on tailings characteristic, site hydrogeological conditions, and compliance point and criteria. Evaluation of potential impacts and closure requirements for tailings in-pit disposal will be carried out at the next stage of the project. Once completly flooded and considering the current life of mine and deposition plan, the tailings stored in the former Sigma open pit would have approximately 47 m of water cap. In such conditions, the formation of meromixis within the pit lake can be anticipated, but water quality and hydrodynamic modelling will need to be performed during the next stage of the project to complete the closure design. In the case of fully mixed conditions, a layer of non acid generating and non metal leachable rock can be added on the top of tailings; this is not accounted in the closure budget at this time. Once the pit lake hydrodynamic model is completed, an updated version of the closure plan for the Sigma property will be issued, which is expected in 2023.

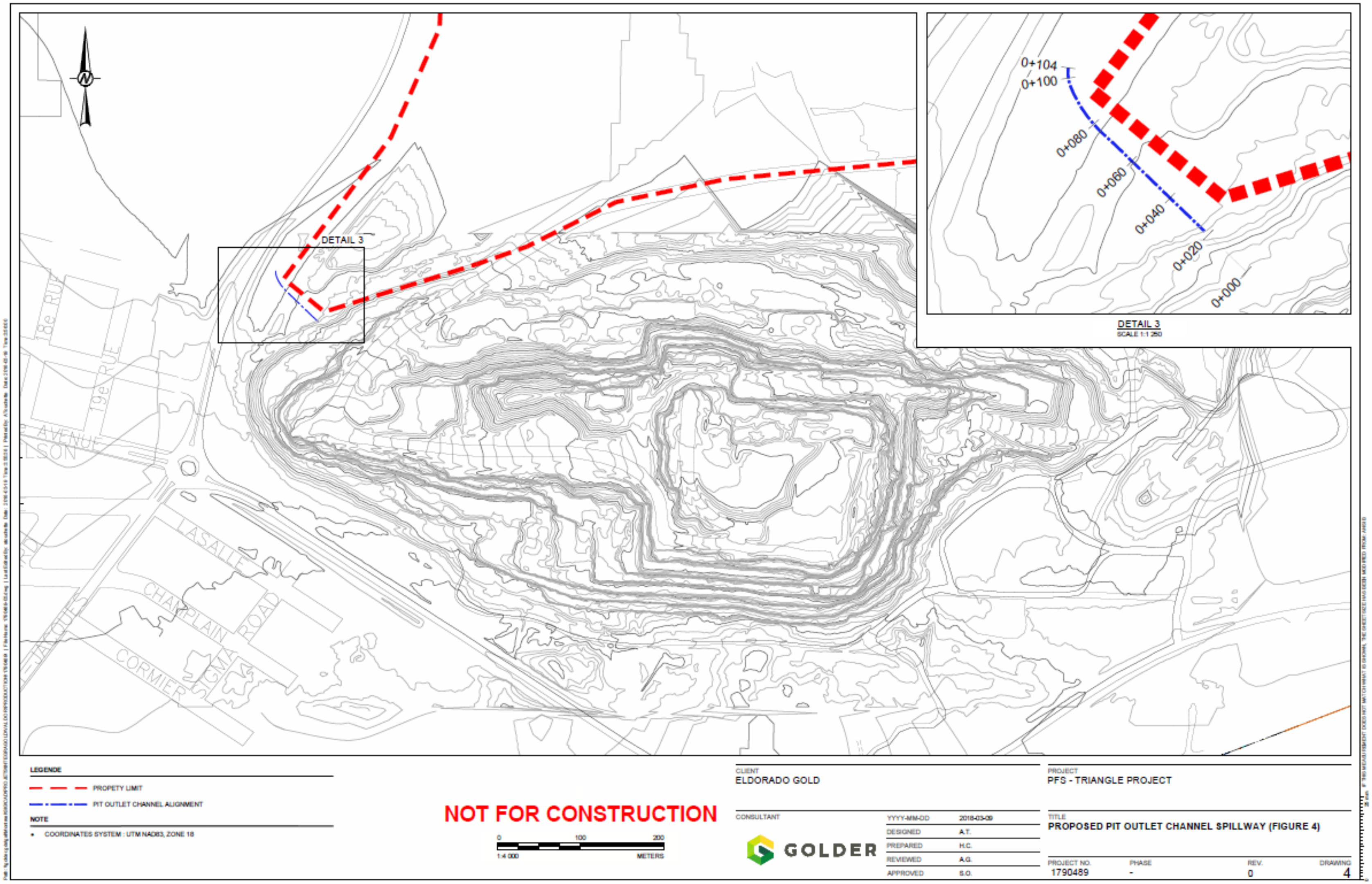


Figure 20.1 – Proposed Pit outlet Channel Spillway

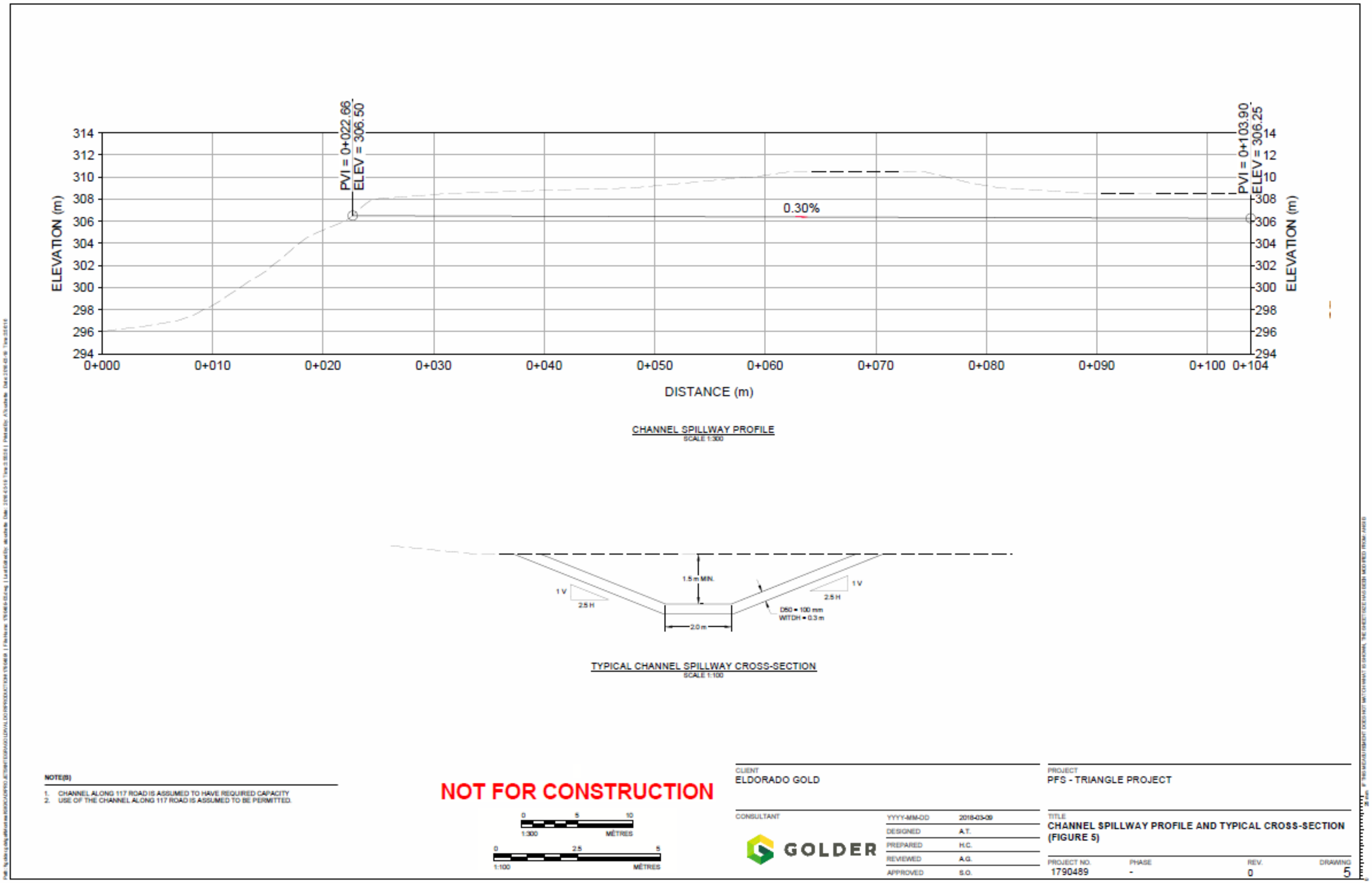


Figure 20.2 – Channel Spillway Profile and Typical Cross-Section

Recreational activities could be eventually considered following restoration of the site. An emergency spillway for the artificial lake has been added to the reclamation costs for the Sigma-Lamaque Mining Complex in the attached table.

Following site closure, an environmental site assessment will be triggered by section 31.51 of the EQA, because the industrial activities at the site (NAICS code: Gold and Silver Ore Mining or Processing – 21222) are listed in Schedule III of the Land Protection and Rehabilitation Regulation (Q-2, r.37 (RPRL)). Therefore, a site characterization study (Phases I and II Environmental Site Assessment) is required under a provision of Division IV.2.1. of Chapter I of the EQA. The environmental site characterization at site closure will have to be performed in the six months following the cessation of activities and according to the MDDELCC's guides and specifications. Estimated costs for such a study appear in the attached tables for each site. Should any identified contaminants exceed the applicable limits stated in the RPRL, a rehabilitation plan would have to be submitted to the MDDELCC. Following its approval, the Company would have to conduct rehabilitation works in compliance with the plan and in a manner compatible with future site utilization.

20.7.3 Monitoring Program after Closure - Sigma-Lamaque Mining Complex

A monitoring program following site closure will be conducted periodically for the evaluation of the integrity of existing structures. This will consist of visual inspections of any dikes (including the Sigma tailings impoundment area), waste rock piles, crown pillars and any anomalies that could jeopardize stability. Particular attention will be paid to water exit points to note any signs of erosion.

An environmental monitoring program will be conducted during post-mining and post-closurerestoration periods for both sites. It is expected that the post-mining period for the Lamaque South Project will be one year compared to an estimated period of three years for the Sigma-Lamaque Mine and Mill Complex will be three years.

For the Lamaque South Project, there should be no effluent following the end of activities, and thus no post-closure effluent monitoring is planned. At the Sigma-Lamaque Mine and Mill Complex, an environmental follow-up of the mine effluent (exit of the decantation pond) and of the runoff water/pit is planned for a period of five years. After such a period, the abandonment by Eldorado Gold of the environmental monitoring program could be possible under certain conditions, including if the results of the monitoring program fulfill the requirements of Québec's Directive 019.

Agronomic monitoring will be undertaken after closure, in the areas of waste rock piles, tailings impoundment, buildings, storage areas, filled ditches and other facility footprints. This program will consist of annual visits and will focus on an assessment of the percentage of vegetation recovery in selected sample plots and, if required, recommendations regarding amendments (fertilization and re-seeding) to be applied to specific areas. It is expected that the vegetation will become self-sustaining after two or three years.

For the Sigma-Lamaque Mine and Mill Complex, the reclamation costs had been estimated at $C6.05M related to the financial guarantee required by the MERN, including the mill and tailings facilities. Details are shown in Table 20.5 That cost excludes an estimated amount (provision) related to potential contaminated sludge/sediments (section 3.4) within polishing and recirculation ponds and soil remediation related to former mining activities (section 5.2). As per the Guide for the preparation of a redevelopment and restoration plan for mining sites in Québec revised in November 2017, costs related to the remediation of contaminated land are now excluded from the financial guarantee to be provided. This explains why two columns are supplied in Table 20.5. The column entitled ‘Total estimated amount’ ($C6.8M) considers the provision related to contaminated sludge and soils that Eldorado should consider for the restoration of the site.

For the Sigma open pit, considering the two monitoring holes installed near Bedard Dyke (West extend of the open pit), five new monitoring holes will be installed during operations to sample and monitor groundwater quality. Figure 20.3 shows the location of the proposed new monitoring holes. Three of these new monitoring holes are located near the proposed position of the spillway, one on the south side of the open pit near Highway 117, and the fifth in the area of the process plant. These facilities will be maintained operational in place as long as agreed with regulation authorities

The financial guarantee currently in place for the Sigma-Lamaque Mine and Mill Complex is C$2.5M. That amount was in place at the acquisition of the Complex.

Table 20.5 – Cost Estimate for Restoration and Financial Guarantee - Sigma Mine Property, Eldorado Gold

	Work related to:	Quantity	Unit Cost	Total estimated amount	Financial Guarantee
			in 2018 CAD dollars		
1.0	*Building and Surface Infrastructure (1)*			1,240,000 $	1,240,000 $
	Sub-total :			1,240,000 $	1,240,000 $
2.0	*Accumulation Areas - Tailing ponds*				
	2.1 Evacuation of water	1	200,000 $	200,000 $	200,000 $
	2.2 Re-vegetation	160.0 ha	9,500 $	1,520,000 $	1,520,000 $
	Sub-total :			1,720,000 $	1,720,000 $
3.0	*Polishing & Recirculation Ponds*				
	3.1 Characterization of sludges (bottom of ponds)	1	8,000 $	8,000 $	8,000 $
	3.2 Conversion into a wetland at mine closure	1	500,000 $	500,000 $	500,000 $
	3.3 Surface water monitoring after conversion	5 years	6,000 $	30,000 $	30,000 $
	3.4 Volume of contaminated sludges to be disposed of offsite - cost for provision (2)	1	100,000 $	100,000 $	-
	Sub-total :			638,000 $	538,000 $
4.0	*Open pit*				
	4.1 Re-vegetation on the edge of the open pit	Estimate	150,000 $	150,000 $	150,000 $
	4.2 Flattening the pit slopes	Estimate	200,000 $	200,000 $	200,000 $
	4.3 Emergency spillway	Estimate	50,000 $	50,000 $	50,000 $
	Sub-total :			400,000 $	400,000 $
5.0	*Surface Infrastructure & other areas*				
	5.1 Environmental site investigation (Phase I & II)		-	180,000 $	180,000 $
	5.2 Soil remediation (unknown if required) (3)		-	500,000 $	-
	5.3 Re-establishment of vegetation	23.7 ha	17,000 $	402,900 $	402,900 $
	Sub-total :			1,082,900 $	582,900 $
6.0	*Monitoring Program for post mining - 3 years (4)*				
	6.1 Environmental follow-up of the mine effluent	42 visits	400 $	16,800 $	16,800 $
	6.2 Agronomical monitoring (former waste rock piles, tailing ponds)	2 years	8,000 $	16,000 $	16,000 $
	6.3 Monitoring well installation for groundwater	5	3,000 $	15,000 $	15,000 $
	6.4 Groundwater follow-up for 9 monitoring wells (waste pile & tailing areas, area with surface infrastructure)	6 visits	4,500 $	27,000 $	27,000 $
	6.5 Environmental follow-up of runoff water - open pit	21 visits	800 $	16,800 $	16,800 $
	6.6 Geotechnical Integrity of structure	7 visits	6,000 $	42,000 $	42,000 $
	Sub-total :			133,600 $	133,600 $
7.0	*Monitoring Program for post restoration - 5 years (4)*				
	7.1 Environmental follow-up of the mine effluent	30 visits	400 $	12,000 $	12,000 $
	7.2 Agronomical monitoring (former waste rock piles, tailing ponds)	5 years	8,000 $	40,000 $	40,000 $
	7.3 Environmental follow-up of runoff water - open pit	30 visits	800 $	24,000 $	24,000 $
	7.4 Groundwater follow-up for 9 monitoring wells (waste pile & tailing areas, area with surface infrastructure)	10 visits	4,500 $	45,000 $	45,000 $
	7.5 Geotechnical Integrity of structure	5 years	10,000 $	50,000 $	50,000 $
	Sub-total :			171,000 $	171,000 $
	Sub-total - all items :			**5,385,500 $**	**4,785,500 $**
8.0	*Supervision & contingency*				
	8.1 Supervision (10%)		10%	538,550 $	478,550 $
	8.2 Contingency (15 %) (5)		15%	888,608 $	789,608 $
		Total (6):		**6,812,658 $**	**6,053,658 $**

Notes:

(1) The cost estimate is limited to above-ground buildings. This estimate was provided by Legault Metal inc. to CMC in April 2012. Cost exclusions related to: hazardous waste removal and disposal, costs related to transporting waste material at licensed facilities if over a 10km radius from the site; cement crushing (to be used as on-site filling). Metal scrap will be reimbursed at 60$/metric ton. Inflation rate of 2 % had been considered for the period 2012-2018.

(2) No information exists about the quality of sludges/sediments within the two ponds. This amount is a provision and does not need to be considered in the financial guarantee.

(3) A provision had been considered ($ 500 000) at mine closure, considering the former mining activities on the property (back to 1934). No information is currently available about the presence of contaminated soils on This amount does not need to be considered in the financial guarantee.

(4) Frequency of site visits is according to *Directive 019*.

(5) Contingency item according to the minimum as stated in the « Guide de préparation du plan de réaménagement et de restauration des sites miniers au Québec » (November 2017).

(6) An updated restoration plan will be prepared and submitted by Integra Gold for approval to the MERN in the first half of 2018 for this property.

Updated on Feb 2018

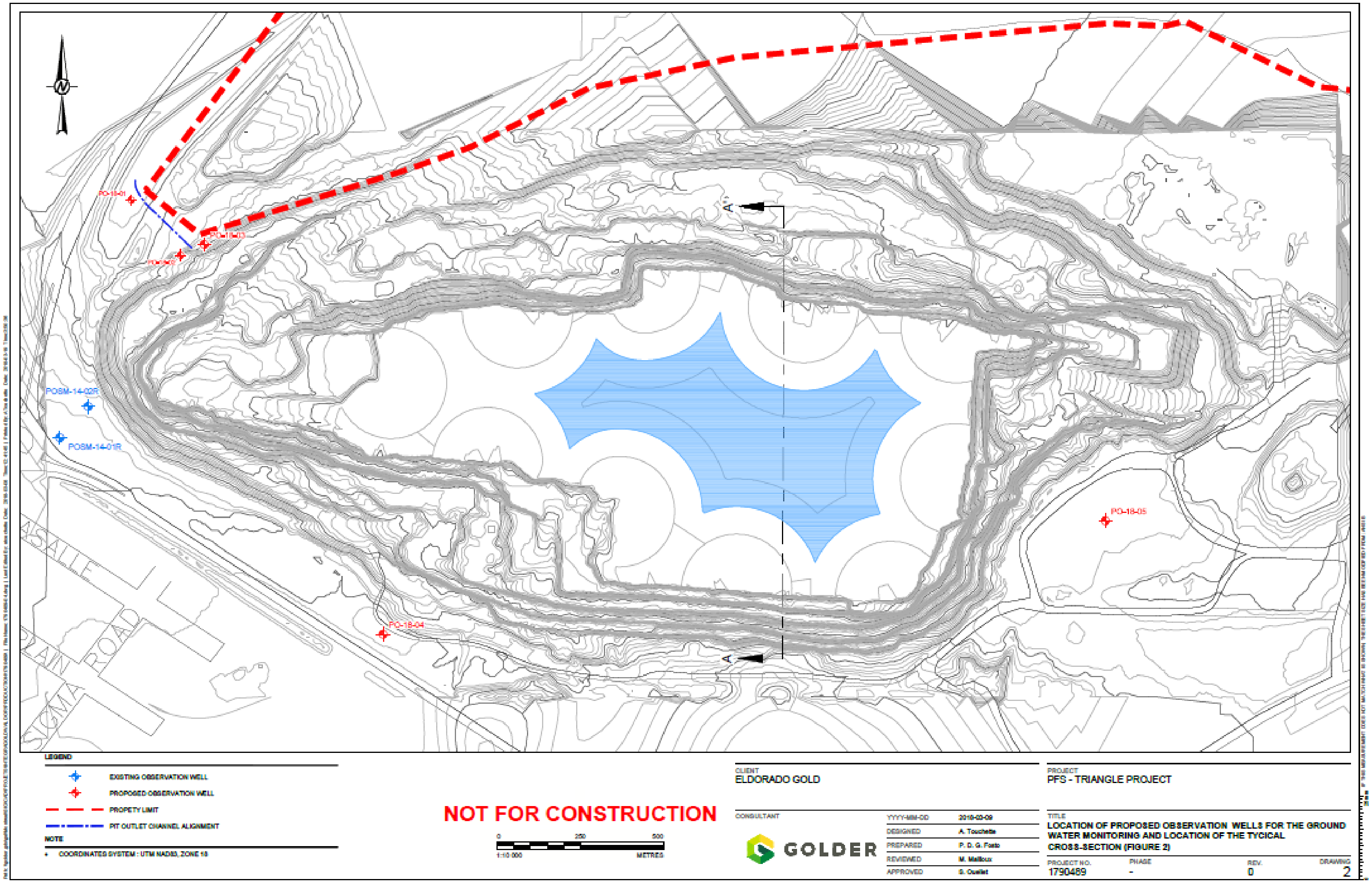


Figure 20.3 – Location of Proposed Observation Wells for the Ground Water Monitoring

SECTION • 21 CAPITAL AND OPERATING COSTS

Cost evaluations were completed in 2018 using Q1 Canadian dollars ($CAD). Costs were converted to US$ for economic modelling and reporting. For items quoted in $US, an exchange rate of 1.30 $CAD = 1.00$US was used.

For most of the mining equipment and also for some fixed equipment (generators, compressors), a lease to buy approach was used with terms fixed at 48 months and interest rate at 7.0%.

Salaries for Eldorado staff and hourly employees currently employed were based on their actual 2018 salaries and planned bonuses (a percentage according to level for staff; an hourly bonus for hourly workers). The salaries of future employees were evaluated considering actual competitive market conditions in the Val-d'Or area. For various benefits, a fringe of 40% was added.

21.1 CAPITAL COSTS

Pre-production capital costs include completion of surface infrastructure at Triangle (site preparation, roads, electric and water lines, buildings); refurbishment of existing office buildings at the Sigma site; refurbishment of Sigma mill and tailings facility; mobile equipment; development and capitalized operating costs; and owner costs (closure plan costs in line with required financial guarantees, company staff and indirect costs).

Pre-production capital costs are low given that there is an existing processing plant and tailings facility, major infrastructure at Triangle is already in place, and underground development has reached Level 202.

Pre-production is anticipated to take place over a 15-month period with most of the capital expenditures allocated to ramp development, mill refurbishment, tailings expansion, development of mineralized zones, and mining at the proposed mining rate and mill throughput.

Contingencies are included in the costs shown in Table 21.1. For sustaining capital development, a growth factor of 10% is included in the scheduled meters. A 10% contingency has been applied on Triangle and Sigma infrastructure. Contingencies of 18% for mill refurbishment at Sigma, and 15% for the new paste plant at Sigma were applied.

Table 21.1 – Capital Cost Estimate ($CAD M)

Description	Construction	Sustaining	Total	% Contingency
Sustaining Development	22.8	71.4	94.2	10%
Equipment	14.1	42.0	56.1	0%
Maintenance	8.2	31.5	39.8	0%
Underground Construction	1.6	4.5	6.1	0%
Underground Ancillary Equipment	4.6	6.4	11.0	0%
Triangle Constructions and Ancillaries	29.2	18.1	47.3	10%
Sigma Mill	56.1	8.8	64.9	18%
Sigma Paste Plant	9.7	21.7	31.4	15%
Tailing Residue Area and Other Sigma Site Costs	7.3	12.2	19.5	10%
Mine Restoration Costs	2.1	3.4	5.5	10%
Royalties Buy Out	3.0	0.0	3.0	0%
Salvage Value	0.0	-14.5	-14.5	100%
Total	**158.8**	**205.5**	**364.3**	

21.1.1 Type and Class of Cost Estimate

The capital cost estimate pertaining to this Prefeasibility Study is meant to form the basis for an overall project budget authorization and funding, and as such, forms the “Control Estimate” against which, subsequent phases of the Project will be compared to and monitored. The estimate meets the definition of an AACE Class 4 estimate. The accuracy of the capital cost estimate developed in this Study is qualified as - 30% / + 30%. Generally, engineering and project definition are developed to an approximate maturity of 10%.

The cost estimate classification system maps the phases and stages of asset cost estimating, and aids in achieving the following:

- A common basis of the concepts involved with classifying project cost estimates, regardless of the type of facility, process or industry that the estimates relate to.
- Fully defines and correlates the major characteristics used in classifying cost estimates, so that companies may unambiguously determine how their practices compare to the guidelines.
- Allows for the measurement of degree of project definition and degree of engineering completion as the primary characteristic to categorize estimate classes.
- Reflects generally accepted practices in the cost estimation profession.

21.1.2 Labour

21.1.2.1 Labour Rates

All estimated costs for labor are based on ten (10) hours per day, five (5) days per week, for a total of fifty (50) hours worked per week. There is no allowance for evening or night shifts.

These average crew rates are built from three (3) components as follows:

- The direct costs are derived from the Québec construction collective agreement for the industrial sector applicable for the period from 2014 to 2017, indexed for 2018 as per the new agreement. The crew rates include a mix of skilled, semi-skilled and unskilled labours for each trade, as well as the fringe benefits on top of the gross wages. Supervision by the foremen and surveyors is built into the direct costs; the rates are calculated as per the annex B (Industrial).
- The indirect costs include small tools, consumables, supervision by the general foremen, contractor management team, contractor on-site temporary construction facilities, mobilization / demobilization, as well as contractor overhead and profit. The costs related to room and board was calculated for 25% of mechanical, electrical and automation workers and are included in the indirects; all of the other discipline workers were assumed to be all available locally.
- Contractor construction equipment, required by each trade to accomplish their tasks, are estimated based on the tariff proposed by “La Direction Générale des Acquisitions du Centre de Services Partagés du Québec”, detailed inside edition dated April 1, 2017. The fuel (diesel) associated with the construction equipment in the present estimate is estimated at $1.00/litre (excluding taxes).

21.1.2.2 Productivity Factor

Project construction performance is an important concern of project owners, constructors, and cost management professionals. Project cost and schedule performance depend largely on the quality of project planning, work area readiness, preparation and the resulting productivity of the work process made possible in project execution. Labour productivity is often the greatest risk factor and source of cost and schedule uncertainty to owners and contractors alike.

The two (2) most important measures of labor productivity are:

- The efficacy of labour used in the construction process; and
- Their relative efficiency in doing what is required at a given time and place.

Important factors affecting productivity on a construction site include, but are not limited to, the following:

Table 21.2 – Factor Affecting Productivity

Factors	
Site location	Weather conditions
Extended overtime	Work over scattered areas
Access to work area	Worker accommodations
Height – scaffolding	Work complexity
Availability of skilled workers	Supervision
Labour turnover	Project schedule pressure
Health and safety considerations	Fast-track requirements

21.1.3 Pre-commercial Revenue and Production Costs

Pre-commercial production costs include all costs related to operations during the pre-production period. These costs include costs such as pre-production mining, underground mine services, maintenance, surface support, underground indirects, processing costs and G&A. Operating costs incurred during the pre-production period were capitalized.

During the 15-month pre-production period, it is anticipated that 62,900 ounces of gold will be produced, providing revenue of $CAD 104.5M. The pre-production revenue will be used to offset pre-production capital expenditures.

21.1.4 Mobile Equipment

Mobile equipment includes all surface and underground equipment. No contingency has been applied to firm quotes obtained from equipment suppliers. The equipment fleet include extra equipment in order to respect expected availability for a typical mining operation. For the economic evaluation, it was assumed that equipment are under a formula of lease-to-buy on a 48 month term at 7% interest rate. For smaller equipment like tractor, 100% of the cost is applied when the equipment is added to the fleet.

Residual value was limited to 25% of original value.

21.1.5 Mill Refurbishment

The capital expenditure covers both direct and indirect costs for refurbishment and replacement of installed equipment, as well as costs to modify the plant to the new design. Equipment is added in areas such as crushing, ore storage, grinding, gravity concentration, thickening, leaching, carbon in pulp, desorption and regeneration, refinery, detoxification, laboratory, reagents, detoxification, fire protection and detection, and process air. Capital expenditures for the overall plant including HVAC, building repairs, lighting and electrical, control and instrumentation are cover in area “General”.

21.1.5.1 Rehabilitation Costs

The processing plant rehabilitation costs were developed by a professional estimator based on process flow sheets, material take-offs (MTO), site quantity surveys and inspections.

The mill rehabilitation costs have been estimated using budgetary proposals obtained from vendors for most process equipment or equipment replacement parts. Labour rates have been estimated, as previously described in this section.

The following methodology was used to estimate the direct costs of the mill rehabilitation project:

- Site Works – Earthwork quantities were estimated from drawings and topographical data, no geotechnical information was available.
- Concrete – Preliminary design sketches were used to develop the concrete and embedded steel quantities.
- Architectural – Plant wall siding upgrade quantities are estimated from general arrangement drawings.
- Mechanical and Process Equipment – A detailed equipment list was developed with equipment sizes, capacities, motor power. The extent of repairs required for mechanical equipment is based on visual inspection of each equipment by specialized personnel, suppliers inspector's report and/or contractor's reports that came to site for static inspections.
- Fire Protection – Quotes were obtained from fire protection specialists. Scope uncovered in quotes was added based on MTO and cost estimated based on BBA reference projects.
- Piping – MTO's were taken from a survey done on site, by engineers from BBA, and adjusted according to the estimated requirements of piping, supports and valves to be replaced. Manual valves larger than 8" will be refurbished while smaller ones are expected to be replaced. Lengths for each line to be insulated were also determined from the survey.
- Electrical Equipment – An equipment list was developed with capacities and sizing from the single line diagrams identified from field surveys and changes developed in this Study.
- Electrical Bulk Quantities – MTO's were estimated based on BBA experience and reference projects. Site electrical includes the main electrical substation, all infrastructure to connect to the local power grid, and distribution from the main substation to the various electrical rooms located throughout the site facilities.
- Automation – MTO's were taken from a survey done on site by BBA personnel.

The pricing and unit costs used in this estimate were based on a combination of budgetary quotes and/or data obtained from similar projects.

- Civil – Pricing is based on quotes received from a local contractor, or references on recent data from similar projects
- Concrete – Unit rates, including formwork and rebar, were estimated with recent costs from similar projects overseen by BBA.

- Steelworks – Material priced from the current steel market value benchmarked with current projects. Labour costs are estimated from BBA's historical data.
- Architectural – Pricing is based on BBA's references and recent data from similar projects.
- Plant Equipment – For process and mechanical equipment packages, equipment data sheets and summary specifications were prepared, and budget pricing was obtained from vendors. For packages with low monetary value, pricing was obtained from BBA's recent project data, when available.
- Piping – Material pricing for carbon steel and rubber-lined piping was based on BBA's references on recent data from similar projects.
- Electrical & Instrumentation Bulks – Pricing of bulks were based on BBA's references on recent data from similar projects.
- Electrical Equipment – For all major electrical equipment and components, budget pricing obtained from Vendors. For electrical equipment of lower value, BBA's historical data was used.

The refurbishment costs were estimated based on information available including:

- Project scope description;
- Plant production / facility capacity and description;
- Preliminary process and mechanical equipment list;
- Preliminary Single Line diagrams
- Field survey and inspection for the quantification of the following elements:
  - Piping;
  - Manual Valves;
  - Motors;
  - Control Valves; and
  - Instruments.
- Process equipment refurbishing requirements from supplier inspections and quotations received.

The transfer conveyor building, the moving of the crushed ore silo, the catwalk and covers for conveyor #8, the acoustic enclosure for the impact hammer, and the plant overall fire detection, was evaluated independently by Concept DB and included in the estimate.

21.1.5.2 Existing Equipment Replaced or New Equipment

The capital expenditure to replace existing equipment and procure new equipment was estimated based on data from budget quotations from suppliers (multiple or single source), historical data and internal estimations based on process flow sheets, material take-offs (MTO), and site quantity surveys and inspections. Breakdown of the mechanical equipment cost sources is shown below:

Table 21.3 - Equipment Cost Breakdown (Mill - New Equipment)

Source	CAPEX Percentage
Firm quotations	0 %
Budget quotations	26 %
Budget quotations – single source	49 %
Historical data	7 %
Internal estimation	18 %

The mechanical repair and/or installation hours were estimated based on experience and in some cases were validated with a contractor. The estimate is based on the assumption that 10,000 hours will be provided by Eldorado maintenance personnel and the balance by external contractors.

Mill refurbishment indirect costs include engineering, procurement, management and construction (12%). Equipment for construction, mobilization and demobilization (2%) have also been considered. First fill, transportation and spare parts (5%) are covered under another line in the indirect cost. Finally, the commissioning (2%) and the contingency (18%) on the direct cost were considered under two separate lines. The owner costs were considered at 0% as the owner will consider it under its capital expenditure.

21.1.6 Paste Plant

The capital expenditure to replace existing equipment and procure new equipment was estimated based on data from budget quotations from suppliers (multiple or single source), historical data and internal estimation and based on process flow sheets, material take-offs (MTO), and site quantity surveys and inspections. A mechanical equipment costs breakdown by source is shown below:

Table 21.4 - Equipment Cost Breakdown (Paste Plant)

Source	CAPEX Percentage
Firm quotations	0 %
Budget quotations	53 %
Budget quotations – single source	1 %
Historical data	12 %
Internal estimation	34 %

The installation hours were estimated based on previous experience.

Mill refurbishment indirect costs include engineering, procurement, management and construction (12%). Equipment for construction, mobilization and demobilization (2%) have also been considered. First fill, transportation and spare parts (5%) are covered under another line in the indirect cost. Finally, the commissioning (2%) and the contingency (15%) on the direct cost were considered under two separate lines. The owner cost was considered at 0% as the owner will consider it under its capital expenditure.

21.1.7 Tailings Management Facility

Most of the tailings produced over the LOM will be managed on the form of cemented paste backfill discharged in the former Sigma open pit. Capital cost estimate for in-pit disposal was prepared by Golder Associates Ltd.

Operating cost for in-pit disposal is mainly related to the paste plant operation and maintenance, cement added to the tailings mixture (1.5% of the dry tailings tonnage) and construction of the pervious surround resulting in a 6.24$/t of ore.

21.1.8 Tailings Impoundment

The quantities of materials for the construction works necessary to store 0.4 Mt in the Sigma Tailings Impoundment and stabilize dykes as per current regulations, were calculated in order to develop the required investments costs. This cost estimation includes as well the cost for water treatment infrastructure.

This cost estimate was prepared using conservative assumptions, since the study is currently at a prefeasibility level; a 25% contingency is assumed. The following elements are not included in the preliminary cost estimation:

- Environmental permitting;
- Investments costing for electricity, and mechanical;
- Purchase of a portion of lands outside of the property;
- Compensation for wetlands losses.

21.1.9 Triangle Electrical & Communication

Electrical and communication includes material for underground power distribution and transformation, cables and connectors, communication, and instrumentation. A 10% contingency has been applied. Manpower is included in operation cost.

21.1.10 Triangle Ventilation

Ventilation includes main fans installation underground and secondary fans required for underground production. A 10% contingency has been applied.

21.1.11 Triangle Pumping

Pumping includes secondary and main pumping station. Pumping also includes cost for primary treatment of mine water prior to pumping to Sigma. A 10% contingency has been applied.

21.1.12 Surface Infrastructure

Surface infrastructure includes completion of site preparation, modular and permanent buildings, surface permanent ventilation system, electrical substation and the water management and distribution. A 10% contingency has been applied.

All costs were estimated using actual costs from Triangle, budget quotes provided by suppliers and based on existing comparable projects.

21.1.13 Underground construction

Underground construction costs at Triangle include ventilation doors, lunch room and refuges, powder room, garage, and a 10% contingency has been applied.

21.1.14 NSR payment Cost

The NSR Royalties will be bought out for $CAD 3.0 M during the pre-production period.

21.1.15 Salvage Value

A salvage value was estimated for some mining equipment, electrical, pumping and ventilation equipment. For the mobile equipment, it was limited to 25% of the original value; for other equipment, only 10% of the original value has been applied.

The salvage value is established at $CAD 6.7 M in 2024 and $CAD 8.4M in 2025.

21.1.16 Closure Plan

Based on evaluations the LOM reclamation costs for the Sigma site, including the mill and tailings facilities, have been estimated at $CAD 6.8M. There is a financial guarantee of $CAD 2.6M in place for the project, passed down from the past operator.

Reclamation costs for Triangle were estimated at $CAD 1.9M. There is a financial guarantee of $CAD 0.7M in place for the project.

21.2 OPERATING COSTS

Operating costs are estimated in 2018 Canadian dollars with no allowance for escalation. All costs in this section are expressed in Canadian dollars unless otherwise indicated. The total operating cost and average unit operating costs are summarized in Table 21.5. The overall unit operating cost is $152.52 per tonnes of milled ore.

Operating costs are summarized below for the production period.

Table 21.5 – Summary of total operating costs

Description	Total Cost (C$M)	Unit Cost ($CAD) ($/t)
Mining	324.1	85.10
Milling (including ore transport and refining)	156.1	40.97
G&A	100.8	26.45
Total	**580.9**	**152.52**

Notes: Operating costs are shown life of mine (including pre-production operating costs).

21.3 MINING COSTS

21.3.1 Definition Drilling

Definition drilling estimates are based on actual costs of drilling completed at Triangle in 2017 on a per tonnage basis of indicated resources defined. The costs including definition drilling by a contractor and sampling in 2017 was 4.60$/t.

For the preproduction and production periods, definition drilling averaged 4.74$/t. In the first years, definition drilling is up to 10.00$/t as more tonnage has to be defined with costs going down as the project progresses.

21.3.2 Stope preparation

The unit cost for stope preparation averages 3.05$/t based on tonnage milled assigned to production. Stope preparation is associated to development costs and include material (explosive, ground support, installed piping) and manpower (for development and transport to surface).

21.3.3 Stope production

The unit cost for stope preparation averages 28.27$/t based on tonnage milled assigned to production. Stope production covers all costs (material and manpower) associated directly to LH mining (production drilling, blasting, cables bolting, mucking, backfilling), room and pillar mining and transportation of ore to surface (manpower). For LH mining, contractor costs were used for drilling.

The preparation is associated to development costs and include material (explosive, ground support, installed piping) and manpower (for development and transport to surface).

The staff and associated salaries include administration, technical services, site security, mechanical and electrical personnel. Salaries were evaluated based on experience and other projects near Val-d'Or. To account for benefits, 33% was added, and depending on the job, bonuses of 5% to 20% were also included. The estimate of the department's general operating cost was based on a comparable mine operating budget. The average cost for Eldorado staff is $20.88 per tonne milled.

21.3.4 Underground services

The unit cost for underground services averages 10.06$/t based on tonnage milled assigned to production. Underground services coves all costs associated to supervision underground, manpower for transporting material underground, ramp maintenance and general maintenance underground. For supervision, a ratio of 12 workers or less per supervisor was applied. All personal safety equipment for mine workers is included in underground services.

21.3.5 Mechanical maintenance

The unit cost for mechanical maintenance averages 14.17$/t. Mechanical maintenance covers all costs associated to maintenance of mobile equipment underground including manpower (supervision, preventive maintenance, and hourly workers), consumables, tire, lube and diesel.

Mechanical maintenance costs are mostly based on unit cost ($/drilled meter for jumbo and bolter; $/hour for other equipment) coming from actual equipment unit costs from a similar operation and validated by InnovExplo. Based on planned development, planned production, and 2017 Triangle data, a projection of the required meters drilled (jumbo and bolter) and hours of equipment (LHD, truck, scissor, tractor, etc) has been developed on a quarter basis. For each piece of equipment, mechanical maintenance costs were calculated based on unit costs applicable.

Mechanical maintenance costs also included tools and general expenses for maintenance department.

21.3.6 Electrical maintenance

The unit cost for electrical maintenance averages 5.15$/t. Electrical maintenance covers all costs associated to maintenance of electrical installation underground including manpower (supervision, preventive maintenance, hourly workers) and pieces. The costs for the maintenance of ventilators and pumps are included in electrical department.

21.3.7 Surface support

The unit cost for surface support averages 7.16$/t. Surface support covers all costs associated to maintenance of road, yard, surface infrastructure, buildings (manpower and material) and energy (natural gas and electricity).

Energy consumption has been evaluated on planned ventilation for natural gas (air heating system) and on planned demand for electricity (surface and underground).

21.3.8 Engineering, technical studies and geology

For engineering (3.50$/t) and geology (3.16$/t), the evaluated costs is mostly associated to required manpower to support operation. General expenses are for respective department are include. For geology department, expenses for required assay to do grade control are included based on a unit cost of 1.50$/t. Costs for technical studies are associated to required studies in support of planned permitting process.

21.3.9 Sigma operational costs

Sigma operational costs (1.61$/t) are associated to all required expenses to support the site as road, yards and building maintenance, underground maintenance (dewatering system) and energy (natural gas and electricity).

21.3.10 Milling

Milling costs include reagents, grinding media, plant maintenance materials, vehicle fuel and maintenance, laboratory services, energy (electricity and natural gas), and manpower required for operation of the Sigma mill. Milling costs for an approximate average production rate of 575,000 tpa are estimated at 32.69 $/t for the C2 zone and 28.03 $/t for the C4 zone. The cost difference between the C2 and C4 zones is mainly due to the use of lead nitrate for the C2 zone. For the C4 zone, addition of lead nitrate did not improve recovery thus is not added.

Reagent and grinding media consumptions were reported in section 17.7.2. Budget quotes were obtained from suppliers for reagent and grinding media unit prices.

Budget quotations were received for crusher and mill liners and change frequency was based on experience. Other maintenance materials were estimated per major equipment based on experience, with allowances added for general materials and per plant area for lubricants and miscellaneous mechanical, piping, electrical and instrumentation materials.

The operating cost estimate assumes that gold assays will be measured at an external laboratory. A revised budget quotation was obtained to estimate this cost.

A loader will be required at the Sigma site to transfer ore from the pile to the crusher. A rental cost, including the operator for the loader, has been included in the estimate. Maintenance and fuel costs for other vehicles (small loader and backhoe) have also been included.

Estimation of electrical energy consumption was reported in section 17.7.1. The electricity cost was calculated based on Hydro Quebec's current L tariff (12.87$/kW/month plus 3.27¢/kWh). The natural gas cost (used for heating) was estimated based on the Sigma plant gas billing for March 2017 to February 2018.

Required plant personnel are presented in Section 17.6. Salaries, expected bonuses and percentage of fringe benefits were estimated to be 40%.

21.3.11 Paste Plant

The costs for the operation of the Sigma paste plant (6.31$/t) include reagents, maintenance materials, energy and manpower. The binder cost represents more than 40% of the total operating cost. The annual binder consumption was evaluated according to Golder recommendations based on the final paste recipe and mass balance. A request for proposal was made to a local supplier in order to obtain a cost estimate.

Flocculant consumption is estimated based on mass balance and results obtained from SGS and Pocock liquid-solid separation testwork. A budget quotation was obtained in order to estimate the cost.

Maintenance materials were calculated from WSP database and experiences for similar equipment.

Electrical energy consumption was estimated based on the electrical load list.

The electricity cost was calculated based on Hydro Quebec's current L tariff rates. The paste plant genset rental cost and its fuel consumption are included in the operating cost estimate.

Manpower was estimated based on the work of 2 maintenance personnel and 4 operators employees working on two 12 hours, day and night. Supervision, electrical maintenance and operator replacement will be done by Sigma Mill staff. Salaries and benefits were developed from collective agreements and in-house data.

21.3.12 Tailings Impoundment

The operating costs for the Sigma Tailings Impoundment include manpower costs required for annual monitoring of the effluent water quality and the underground water based on the current regulations, inspection and maintenance of the Sigma Tailings Impoundment, and running costs for the mobile water treatment plant, including electricity costs, chemical costs, and maintenance costs.

Unit rates of items/activities already performed in past projects of the region were used in order to estimate the operating costs for the Sigma Tailings Impoundment over the projected life of mine. The salaries used for the manpower costs were evaluated based on experience and other projects near Val-d'Or. To account for benefits, 33% was added, and bonuses of 10% were also included.

Based on the preliminary cost analysis, the operating costs for the Sigma Tailings Impoundment will be $4.4 million over the projected life-of-mine.

21.3.13 Transportation Cost

The ore material will have to be rehandled from a pad located near Triangle portal entrance while the material is being custom milled (Camflo or Westwood) or milled to Sigma. Transportation costs to Camflo are estimated at $3.67/t and to Westwood at $8.00/t. At Sigma, transportation costs were evaluated to $3.00/t.

21.3.14 G&A

The G&A costs cover administration, procurement, human resources, health and safety, information technology (IT), and environment. For each department, costs included manpower and associated expenses.

Administration manpower included all heads of department and associated manpower for accounting. Major expenses are associated with taxes (municipal and scholar) and insurances.

Human resources include major expenses associated to recruitment and consultant fees. In health and safety, major expenses are for site security (keeper at Sigma and Triangle; 24H, 7 days), medical examinations, and consultant fees. IT department major expenses are mostly associated to software licenses. Environment department major expenses are associated to disposition of waste and water sampling to meet effluent regulation obligation.

SECTION • 22 ECONOMIC ANALYSIS

22.1 SUMMARY

The economic analysis for the Project based on US$1,300/oz Au indicates an after-tax net present value (NPV) of US$211 M, using a discount rate of 5%.and yields an internal rate of return (IRR) of 35% on an after-tax basis. Payback of the initial capital is achieved in 3.3 years from the start of commercial production.

At the mineral reserve metal price of US$1,200/oz Au, the Project shows positive economics. The after-tax IRR is 25.9 % and the NPV is estimated to be US$150.3 M using the 5% discount rate, with a calculated payback period of 3.9 years.

The economic model was subjected to a sensitivity analysis to determine the effects of changing metal prices and capital and operating expenditures on the Project financial returns. This analysis showed that the project economics are robust, and are most sensitive to metal prices.

22.2 CAUTIONARY STATEMENT

The results of the economic analysis are based on forward-looking information that are subject to a number of known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those presented here.

Forward-looking statements in this section include, but are not limited to, statements with respect to:

- Future prices of gold;
- Currency exchange rate fluctuations;
- Estimation and realization of mineral reserves; and
- Estimated costs and timing of capital and operating expenditures.

22.3 METHODS, ASSUMPTIONS AND BASIS

The economic analysis is based on the mineral reserves as outlined in Section 15, the mining methods and production schedule as outlined in Section 16, the recovery and processing methods in Section 17, and the capital and operating costs as outlined in Section 21.

The Project case metal price used in the economic model is US$1,300/oz Au, which represent Eldorado's view of long-term metal pricing

The economic analysis evaluates revenue, expenditures, taxes, and other factors applicable to the Project.

The cash model has been prepared on a yearly life of mine basis.

The cash flow model was prepared based on the following assumptions:

- All costs are reported in United States Dollars (USD) unless otherwise stated;
- 100% equity ownership;

- No cost escalation beyond Q1 2018;
- No provision for effects of inflation;
- Constant Q1 2018 USD analysis;
- Exploration costs are deemed outside of the Project;
- Any additional project study costs have not been included in the analysis;
- Exchange rate of 1 US$ = 1.30 CAN$ for 2018 and 2019;
- Exchange rate of 1 US$ = 1.25 CAN$ from 2020 onwards; and
- Royal Mint fees of $3 per ounce.

22.4 CASH FLOW

The cash flow model for the Triangle Project is summarized in **Table 22.1** and presented in **Table 22.2.**

Table 22.1 – Cash Flow Analysis Summary

Parameter	Units	Results
Peak Milling Rate	ktpa	600
Mine Life	year	7
Average Grade	g/t	7.30
Average Recovery Rate	%	94.5%
Average Annual Gold Production	Koz	117
Peak Gold Production	Koz	135
Average Cash Costs	US$/oz	516
Average All-in Sustaining Costs	US$/oz	717
Estimated Capital Expenditure	US$ M	284
Initial Capital Costs (to commercial production)	US$ M	122
Capitalized Operating Costs (pre commercial production)	US$ M	57
Proceeds from Pre-Commercial Gold Sales	US$ M	(80)
Sustaining Capital	US$ M	162
Gold Price Assumption Used in Financial Analysis	US$/oz	1,300
NPV-5% (after tax)	US$ M	211
IRR (after tax)	%	35%
Payback Period	year	3.3

Notes: • *Total Cash Cost: Direct operating cost, refining and royalties*
• *All-in Sustaining Cost: Direct operating cost, refining, royalties, sustaining capital, reclamation and salvage*

Table 22.2 – Triangle Project Cash Flow Model

MINE SCHEDULE	UNIT	TOTAL	2018	2019	2020	2021	2022	2023	2024	2025
ECONOMIC ASSUMPTIONS										
Gold Price	US$/oz	1,300	1300	1300	1300	1300	1300	1300	1300	1300
Exchange Rate	CAD/USD	1.26	1.30	1.30	1.25	1.25	1.25	1.25	1.25	1.25
MINING SCHEDULE										
Ore Mined	000 t	3,808.7	210	440	563	562	585	592	574	283
Gold Grade	g/t Au	7.30	8.01	7.69	6.41	7.52	7.11	7.38	7.54	7.18
Contained Gold	Koz	893.4	54.2	108.8	116.0	135.7	133.7	140.5	139.1	65.3
PROCESSING SCHEDULE										
Ore Processed	000 t	3,808.7	156	470	560	560	580	600	585	298
Gold Grade	g/t Au	7.30	8.03	7.71	6.43	7.52	7.11	7.38	7.54	7.20
Contained Gold	Koz	893.4	40.3	116.5	115.7	135.3	132.5	142.4	141.8	68.9
Gold Recovery	%	94.5%	91.7%	93.8%	94.1%	94.5%	94.8%	95.0%	95.1%	95.2%
Gold Production	Koz	844.4	36.9	109.3	108.9	127.9	125.7	135.4	134.8	65.5
Pre-Commerical Gold Production	Koz	62.9	36.9	26.0						
Commerical Gold Production	Koz	781.5	-	83.3	108.9	127.9	125.7	135.4	134.8	65.5
REVENUE										
Gold Sales	US$mln	1,097.7	48.0	142.0	141.5	166.3	163.3	176.0	175.3	85.2
Royalty Expense	US$mln	11.8	1.0	1.8	1.4	1.7	1.6	1.8	1.8	0.9
Net Revenue	US$mln	1,085.9	47.1	140.3	140.1	164.7	161.7	174.2	173.5	84.4
Pre-Commercial Revenue	US$mln	80.4	47.1	33.3						
Commercial Revenue	US$mln	1,005.5	-	107.0	140.1	164.7	161.7	174.2	173.5	84.4
OPERATING COSTS										
Mining Cost	C$mln	324.1	30.8	44.3	43.0	42.7	47.3	48.8	46.4	20.7
Processing Cost	C$mln	156.1	11.3	20.7	23.2	22.3	22.2	22.2	21.8	12.4
G&A Cost	C$mln	100.8	13.8	14.3	13.4	13.4	13.3	13.2	12.1	7.2
Total Production Costs	C$mln	580.9	55.9	79.3	79.6	78.4	82.9	84.3	80.4	40.2
Pre-Commercial Production Costs	C$mln	74.1	55.9	18.2	-	-	-	-	-	-
Commercial Production Costs	C$mln	506.8	-	61.1	79.6	78.4	82.9	84.3	80.4	40.2
Total Production Costs	US$mln	460.6	43.0	61.0	63.7	62.8	66.3	67.4	64.3	32.2
Pre-Commercial Production Costs	US$mln	57.0	43.0	14.0						
Commercial Production Costs	US$mln	403.6	-	47.0	63.7	62.8	66.3	67.4	64.3	32.2
CASH COST PER OUNCE										
Mining Cost	US$/oz	288		312	316	267	301	289	275	252
Processing Cost	US$/oz	85		100	99	84	85	78	72	88
G&A Cost	US$/oz	143		146	170	140	141	131	129	151
C1 - Cash Operating Cost	US$/oz	516		558	585	490	527	498	477	491
Royalty Cost	US$/oz	13		16	13	13	13	13	13	13
C2 - Total Cash Cost	US$/oz	529		574	598	503	540	511	490	504
Sustaining Capital Cost	US$/oz	188		570	422	291	145	59	(24)	(98)
C3 - Sustaining Cash Cost	US$/oz	717		1,145	1,020	794	685	570	466	405
CAPITAL EXPENDITURES										
CONSTRUCTION CAPITAL (to commercial production)										
Differed development mining	C$mln	22.8	17.6	5.3	-	-	-	-	-	-
Mine equipment	C$mln	14.1	10.7	3.4	-	-	-	-	-	-
Maintenance Mechanical	C$mln	8.2	6.1	2.1	-	-	-	-	-	-
Underground construction	C$mln	1.6	1.3	0.3	-	-	-	-	-	-
Underground construction	C$mln	4.6	4.2	0.4	-	-	-	-	-	-
Buildings, equipment, …	C$mln	29.2	28.0	1.2	-	-	-	-	-	-
Sigma Mill	C$mln	56.1	50.9	5.2	-	-	-	-	-	-
Sigma Paste Plant	C$mln	9.7	3.1	6.6	-	-	-	-	-	-
Sigma - Others (eg. Tailings)	C$mln	7.3	7.0	0.3	-	-	-	-	-	-
Mine Restoration	C$mln	2.1	1.4	0.6	-	-	-	-	-	-
NSR buy-out	C$mln	3.0	-	3.0	-	-	-	-	-	-
Total Development	C$mln	158.8	130.4	28.4	-	-	-	-	-	-
SUSTAINING CAPITAL										
Differed development mining	C$mln	71.4	-	17.9	20.4	19.0	9.1	4.4	0.6	-
Mine equipment	C$mln	42.0	-	9.9	12.4	11.3	6.6	1.5	0.4	-
Maintenance Mechanical	C$mln	31.5	-	6.6	10.0	9.7	3.9	1.3	-	-
Underground construction	C$mln	4.5	-	0.7	2.0	0.9	0.5	0.4	-	-
Underground construction	C$mln	6.4	-	0.8	2.7	2.4	0.3	0.1	0.1	-
Buildings, equipment, …	C$mln	18.1	-	13.3	1.8	1.5	0.9	0.4	0.1	-
Sigma Mill	C$mln	8.8	-	7.3	-	-	-	0.9	0.7	-
Sigma Paste Plant	C$mln	21.7	-	19.3	2.0	0.0	0.1	0.3	-	-
Sigma - Others (eg. Tailings)	C$mln	12.2	-	1.8	5.9	1.7	1.4	0.8	0.7	-
Mine Restoration / Salvage	C$mln	3.4	-	3.4	-	-	-	-	-	-
Salvage Value	C$mln	-14.5	-	-	-	-	-	-	(6.5)	(8.1)
Total Sustaining Capital	C$mln	205.5	-	81.0	57.4	46.5	22.7	9.9	(4.0)	(8.1)
TOTAL CAPITAL										
Construction Capital	C$mln	158.8	130.4	28.4	-	-	-	-	-	-
Sustaining Capital	C$mln	205.5	-	81.0	57.4	46.5	22.7	9.9	(4.0)	(8.1)
Total Capital Cost	C$mln	364.3	130.4	109.4	57.4	46.5	22.7	9.9	(4.0)	(8.1)
Construction Capital	US$mln	122.1	100.3	21.8	-	-	-	-	-	-
Sustaining Capital	US$mln	161.9	-	62.3	45.9	37.2	18.2	7.9	(3.2)	(6.5)
Total Capital Cost	US$mln	284.1	100.3	84.1	45.9	37.2	18.2	7.9	(3.2)	(6.5)
CASH FLOW										
EBITDA	US$mln	601.9	-	60.0	76.4	101.9	95.4	106.8	109.2	52.2
Construction Capital	US$mln	-122.1	(100.3)	(21.8)	-	-	-	-	-	-
Sustaining Capital	US$mln	-161.9	-	(62.3)	(45.9)	(37.2)	(18.2)	(7.9)	3.2	6.5
Pre-Commercial Production Costs	US$mln	-57.0	(43.0)	(14.0)	-	-	-	-	-	-
Pre-Commercial Revenue	US$mln	80.4	47.1	33.3	-	-	-	-	-	-
Pre-tax Cash Flow	US$mln	341.2	(96.2)	(4.8)	30.4	64.7	77.3	98.9	112.4	58.6
Tax Expense	US$mln	-47.3	-	(4.2)	(3.0)	(6.1)	(6.6)	(10.1)	(13.0)	(4.4)
After-tax Cash Flow	US$mln	293.9	(96.2)	(9.1)	27.5	58.6	70.7	88.8	99.4	54.3
Cumulative After-tax Cash Flow	US$mln		(96.2)	(105.3)	(77.8)	(19.2)	51.4	140.2	239.7	293.9
ECONOMICS										
Net Present Value (5%)	US$mln	211								
Internal Rate of Return (IRR)	%	35.0%								
Payback Period	Years	3.3								

22.5 TAXES

In 2013 the province of Quebec introduced a mining tax based on the operator's profitability. Profitability is calculated by dividing the operator's annual profit by the gross value of the operator's annual output. A progressive rate structure is then applied based on the operator's profit margin.

Table 22.3 –Quebec Mining Tax Rates

Profit Margin	Applicable Tax Rate
0% - 35%	16.0%
35% - 50%	22.0%
50% - 100%	28.0%

Annual profit is calculated by subtracting the following allowances from the gross value of the mine's annual output:

- Direct operating costs;
- Royalties;
- Depreciation;
- Post-production development allowance;
- Processing allowance;
- Additional depreciation allowance;
- Additional allowance for a northern mine; and
- Additional allowance for a mine situated in Northern Québec.

The Company expects that through the use of intercompany financing and existing tax losses that the Company will not pay any current income tax on the profits from the project.

22.6 ROYALTIES

The Lamaque Property is made up of several individual mining concessions that are subject to a 2% royalty in favour of various parties. Each of the royalty agreements provide the Company with a buyout provision to reduce the royalty rate to 1%. For purposes of the economic analysis, a buyout amount of C$3.0 million has been included in the capital costs of the project and a royalty rate of 1% has been applied to all commercial ounces.

22.7 CLOSURE AND SALVAGE VALUES

The basis of life of mine reclamation costs and financial guarantees for the Sigma site and Triangle are described in Section 21.

Salvage value was estimated for the mining equipment and electrical, pumping and ventilation equipment. For the mobile equipment, salvage value was limited to 25% of the original value. For other equipment, 10% of the original value has been applied.

22.8 THIRD PARTY INTERESTS

Eldorado Gold is the 100% owner of the Lamaque property.

22.9 SENSITIVITY ANALYSIS

The parameters for the sensitivity analysis were chosen based on their potential impact on the outcome of the economic evaluation. Key economics were examined by running cash flow sensitivities against:

- Capital cost (CAPEX);
- Operating cost (OPEX); and
- Metal prices.

Sensitivity calculations were performed on the Project's after-tax NPV (5%) and IRR by applying a range of variation (-10% to +10%) to metal prices, capital costs, and operating costs.

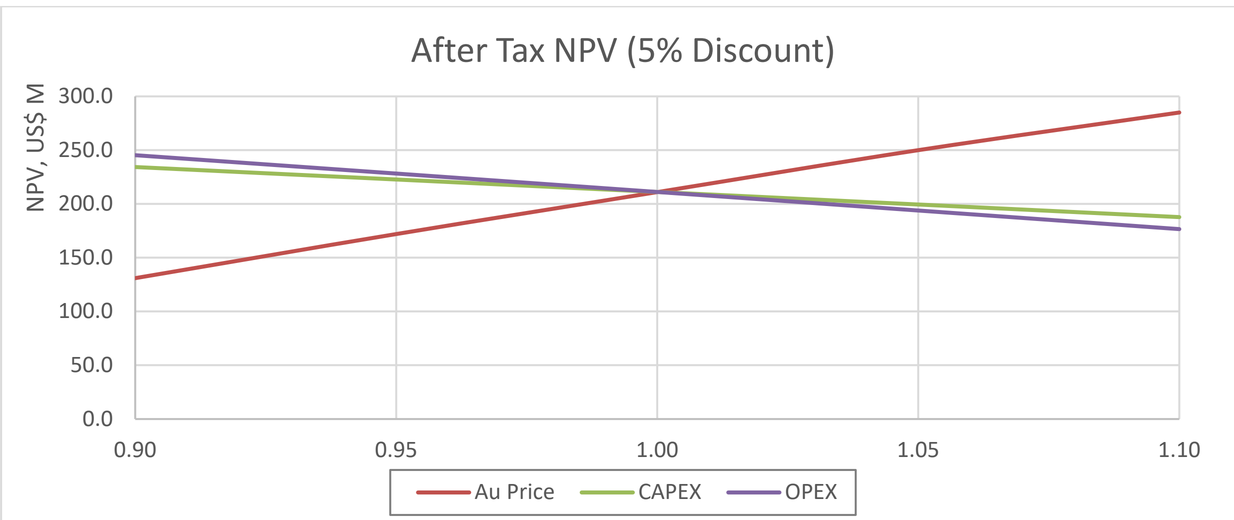


Figure 22.1 - Sensitivity Analysis – NPV @ 5% Discount

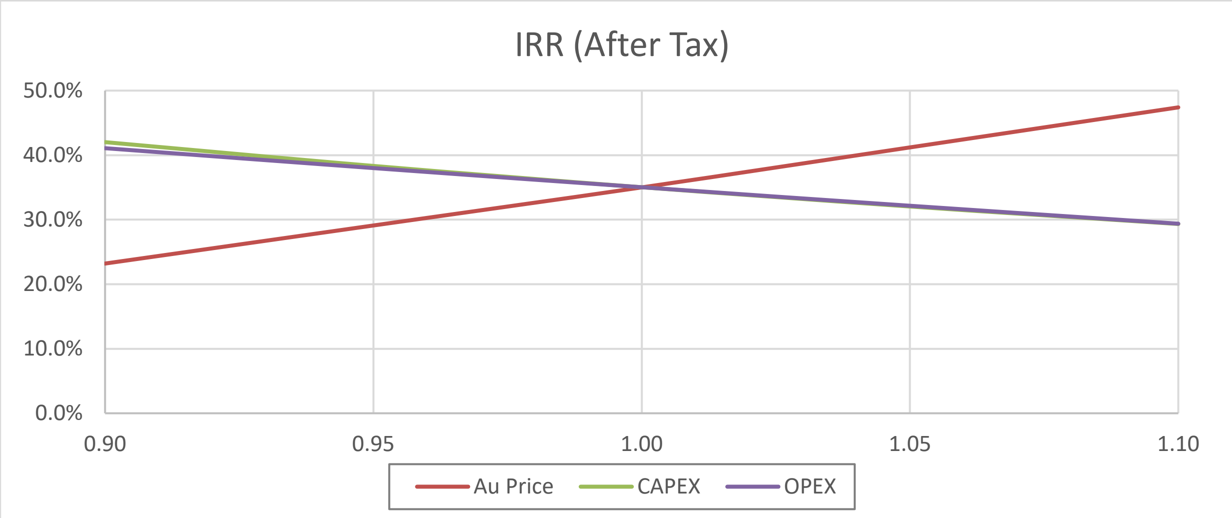


Figure 22.2 - Sensitivity Analysis – IRR

Table 22.4 – Summary of Sensitivity Analysis

	Au Price		CAPEX		OPEX	
Factor	**NPV (5%)**	**IRR**	**NPV (5%)**	**IRR**	**NPV (5%)**	**IRR**
0.90	131.0	23.2%	234.3	42.0%	245.3	41.1%
0.95	172.0	29.1%	222.7	38.3%	228.2	38.0%
1.00	211.0	35.0%	211.1	35.0%	211.1	35.0%
1.05	250.0	41.2%	199.5	32.0%	193.9	32.2%
1.10	285.0	47.4%	187.8	29.3%	176.6	29.4%

The cash flow has been discounted to 5% and is considered reasonable. **Table 22.5** presents the summary of the Project NPV for varying discount rates.

Table 22.5 – NPV Discount Rate Summary

Discount Rate	After-tax NPV Value (US$ M)
0%	293.9
5%	211.1
7%	184.4
10%	149.6

SECTION • 23 ADJACENT PROPERTIES

This item was updated from the technical report and 2016 resource estimate update prepared by Poirier et al. (2016).

The Lamaque Project is located in a historical mining district with multiple past gold producers. The area is still a major mining centre with considerable exploration activities. Several Canadian exploration and mining companies are actively working in the area. Figure 23.1 shows the location of the various properties in the vicinity of the Lamaque Project.

23.1 GLOBEX MINING CORPORATION

Contiguous with and east of the Sigma-Lamaque Project is the Sigma East Property, owned by Globex Mining Corporation ("Globex"). The property consists of 15 claims, and hosts extensions of the geological features found on the Sigma-Lamaque Property. Historical drilling revealed a number of gold mineralized zones, known as the Cisaillement E-Q (Sud, Zone S) zones. There has been no exploration work on this project to our knowledge since 2014.

23.2 ALEXANDRIA MINERALS CORPORATION

Alexandria Minerals Corporation ("Alexandria") owns a land package consisting of 462 claims (15,162 ha) adjacent and south of the Lamaque South Project. The property extends from Lamaque South to the Sigma II Project and beyond, covering the prolific gold-bearing Larder Lake–Cadillac Fault Zone. Alexandria has been focused on their Orenada Zone #2, which is located some 2km south-east of the Triangle zone. Alexandria is currently working on an updated resources estimate for this deposit.

23.3 QMX GOLD CORPORATION

QMX Gold Corporation ("QMX"; previously Alexis Minerals Corporation) owns both a gold project contiguous with and north of the Lamaque Project, and a mine that closed in 2015 called Lac Herbin. In 2017, QMX has re-started their exploration program on their large land package, which also extends from immediately east of Lamaque to near Sigma II. They have conducted a regional airborne MAG survey and have been drill testing numerous targets in the area. They have had recent success east of the old Louvicourt mine on the Bonnefond target, where early drilling results is indicating a series of gold bearing quartz tourmaline veins associated with a cylinder shape porphyritic diorite intrusion.

23.4 HARRICANA RIVER MINING CORPORATION

Harricana River Mining Corporation ("Harricana") is a private company with more 100 contiguous mineral claims adjacent to and west of the Lamaque Project. The company has two shafts from past gold producers, called the Hydro Zone and the New Harricana Zone. The last drill program was conducted in 2009, but no results are publicly available.

23.5 GOLD POTENTIAL FROM ADJACENT PROPERTIES

InnovExplo has not verified the above information about mineralization on adjacent properties around the Lamaque Project. The presence of significant mineralization on these properties is not necessarily indicative of similar mineralization on the Lamaque Project.

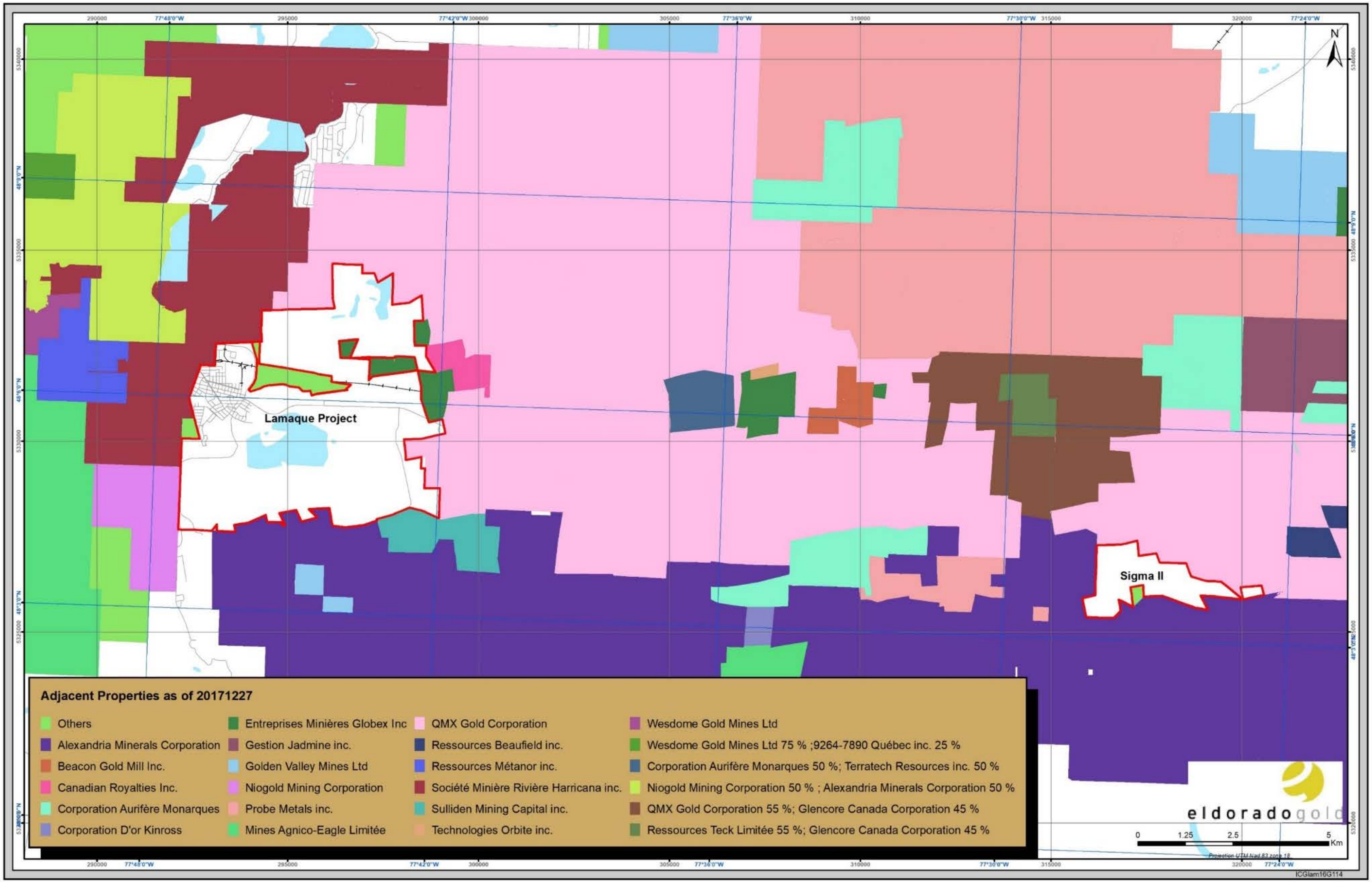


Figure 23.1 – Location map of Adjacent Properties

SECTION • 24 OTHER RELEVANT DATA AND INFORMATION

24.1 LIFE OF ASSET STRATEGY

Eldorado Gold Corporation endeavours to maximise the value of the Lamaque project by adding to its existing reserve base, thereby extending mine life and potentially increasing the rate of gold production.

Key to expanding the reserve base are:

- Conversion to reserves of mineral resources not currently in the mine plan through infill drilling.
- Expansion of the resource base through exploration.

The focus at Lamaque since the acquisition in July 2017 has been on infill drilling the upper portion of the Triangle deposit to quantify and declare the maiden reserves. Eldorado Gold Corporation will continue to infill drill resources to add to its existing reserve base. 58,500 meters of infill drilling are planned for 2018, and a provision to maintain this level of infill drilling in subsequent years have been factored into the operational cost model.

The Lamaque property hosts significant Indicated resources in the Plug 4 and Parallel zones that have not been incorporated into the current reserves. Requirements for the recovery of these zones will be evaluated in detail over the course of 2018, alongside the evaluation of an option to develop an underground ramp to haul ore from Triangle to the Sigma mill, which would pass in close proximity to the Plug 4 and Parallel zones. The underground ramp is expected to reduce ore transport costs, improve access, and could serve as an excellent exploration drill platform for the Plug 4 and Parallel zones as well as new targets along its route.

Eldorado Gold Corporation conducted a preliminary assessment of the impact to the mine plan and economics from a scenario that includes reserves formed from Indicated resources in the Plug 4 and Parallel zones, and also including reserves formed from a partial conversion of Inferred resources in the Triangle deposit, Plug 4 and Parallel zones. The results of the assessment suggest that the mine life would be extended by several years at an increased rate of gold production leading to a 50% increase in value. Accordingly, current life-of-mine planning incorporates the judicious placement of development infrastructure to recover proven and probable reserves while locating this infrastructure in a manner that would avoid sterilisation of these additional resources.

Expansion of the resource base in 2018 is has been initiated through 34,000 metres of exploration drilling that will test deeper “C” zones at Triangle and numerous other targets on the property. Recent exploration drilling has returned mineralised intercepts in the “C” zones to a depth of 1500 m, progressively defining zones C6 through C10 at elevations commensurate with the vertical extent of mining at the historic Sigma and Lamaque mines, which are adjacent to and in close proximity to the Triangle zone.

Figure 24.1 illustrates the relative locations of the Indicated and Inferred resources that are not part of the current PFS reserves against the backdrop of planned development infrastructure at Triangle. The proposed haulage ramp between the Sigma mill and the Triangle zones has been included. Deep exploration at Triangle can be referenced against the vertical extent of mining at the historic Sigma and Lamaque mines.

Figure 24.2 shows the planned exploration drilling for 2018 that targets deeper resources at the Triangle zones. A preliminary evaluation of material handling options to recover deeper resources will be undertaken in the near future, and will include various shaft/winze scenarios that will be conceivably integrated with the proposed ramp between the Triangle zone and the Sigma mill, thereby providing a seamless and cost-effective system for the recovery of resources associated with these zones.

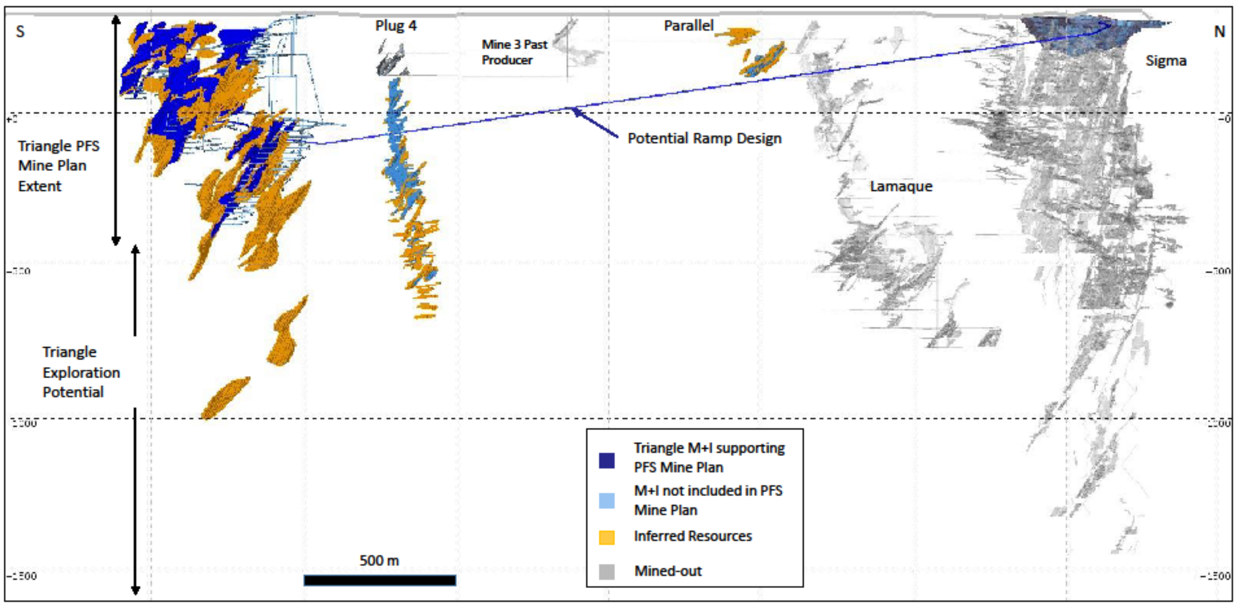


Figure 24-1- Resources and Exploration Potential

Notwithstanding the life of asset strategy, Eldorado Gold Corporation cautions that Inferred resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and that the Inferred resources included in this preliminary assessment do not yet have demonstrated economic viability.

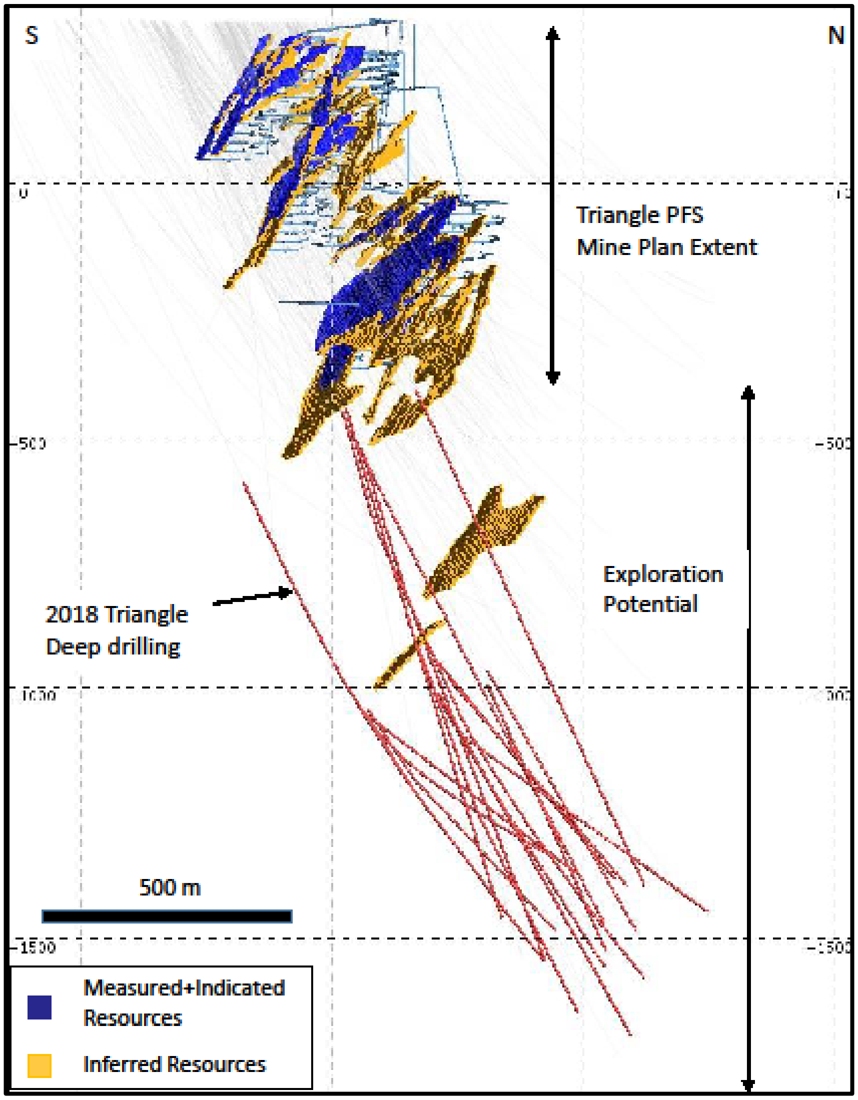


Figure 24.2 Planned Deep Exploration Drilling At Triangle - 2018

24.2 PROJECT RISKS AND OPPORTUNITIES

This section highlights the main project risks and opportunities.

24.2.1 Project Risks

- The Company received the Certificate of Authorization, closure plan and the mining lease for the Triangle deposit in the first quarter 2018. The only outstanding permit is for ore processing at the Sigma mill, which is expected to be received during the third quarter 2018;
- The thickness and continuity of each vein may vary over the expected distances;
- Encountering more water infiltration than expected in current plan;

- High pH operations, which could lead to a higher acid washing of the carbon. Further testing during the detailed engineering phase will allow to properly mitigate this situation.

24.2.2 Project Opportunities

- The Company is evaluating an option to build an underground ramp to haul ore from Triangle to the Sigma mill, while passing through the Plug 4 and Parallel ore zones. The underground ramp is expected to reduce ore transport costs, improve access, and could serve as an excellent exploration drill platform for the Plug 4 and Parallel deposits as well as new targets along its route;
- Increasing automation in operations could yield opportunities in lower operating costs and increases in production;
- More tests without pre-aeration should be conducted and the PEA trade-off study for inclusion or not of a pre-aeration tank redone as the difference in cyanide consumption no longer justifies the addition of a pre-aeration tank. The addition of the tank would need to be justified based on added recovery, and potentially on beneficial effect on cyanide destruction.
- Opportunity for further extension of the known ore zones in Triangle. Also opportunity to add other ore zones (Plug 4, Parallel etc) that have known resources on them but are not included in the current mine plan.

SECTION • 25 INTERPRETATION AND CONCLUSIONS

The results of this PFS demonstrate that the Lamaque Mine Project warrants development due to its positive, robust economics. To date the qualified persons are not aware of any fatal flaws on the Lamaque project and the results are considered sufficiently reliable to guide Eldorado management in a decision to further advance the Project. With the exception of those outlined in this report in section 24, the report authors are unaware of any unusual or significant risks or uncertainties that would affect project reliability or confidence based on the data and information made available.

It is concluded that the work completed in the prefeasibility study indicate that the exploration information, mineral resource and mineral reserve estimates and Project economics are sufficiently defined to indicate the Project is technically and economically viable.

For these reasons, the recommended path going forward to continue to focus on obtaining environmental permit/approvals, while concurrently advancing key development activities that will reduce Project execution time.

25.1 MINERAL RESOURCE AND MINERAL RESERVE

Information and data contained in, or used in the preparation of mineral resource and mineral reserve update was obtained from historic data obtained from Integra Gold verified and supplemented by information from a number of surface diamond drill campaign undertaken by Integra Gold and now subsequently Eldorado Gold. The mineral resource and mineral reserve are consistent with the CIM definitions referred to in NI 43-101. It is the opinion of the qualified persons that the information and analysis provided in this report is considered sufficient for reporting mineral resources and mineral reserves.

Results of drilling indicate the ore body is open at depth and indications are that inferred resources could be converted to indicated resources with further exploration. Eldorado considers this an opportunity to the Project, further exploration at depth should be completed during operations.

25.2 METALLURGY

Significant metallurgical test work and analysis has been completed to confirm the process designs and substantiate stated recoveries. WSP and Eldorado has reviewed and validated historic data obtained from Integra Gold and completed additional confirmatory test work. The qualified persons have a high degree of confidence in the process designs and the stated recoveries.

Two laboratory test work programs were carried out on ore samples from the Triangle zone to characterize the hardness and the achievable gold recovery using the Sigma Mill. The first series was conducted by SGS in 2016 (refer to Chapter 13 for report citation). The second series was conducted in 2017 by Bureau Veritas (BV). In addition, three bulk samples were processed through the Camflo mill.

The SGS tests, conducted on composite samples from the Triangle zone, included chemical and mineralogical analyses, acid generation potential tests, grinding tests, gravity concentration and leaching of gravity tails, cyanide destruction tests as well as a gold deportment study on the cyanidation tailings.

- The Triangle composites were found to be non-acid generating.
- The Bond rod mill work index, Bond ball mill work index and Abrasion index were measured and ranged from 16.7 to 17.3 kWh/t, 15.5 to 16.4 kWh/t, and 0.247 to 0.262g respectively.
- For the Triangle Blend composite, gravity recovery was 30.8% at 80% passing 71 µm and around 35% at 80% passing 49 µm.
- The gold extraction from cyanidation of gravity tails ranged from 87.6% to 90.2% at 80% passing 75 µm without pre-aeration for a retention time between 84 and 96 hours. Grinding to 49 µm increased cyanidation recovery to between 91.7 and 92.6%.
- It was demonstrated that a pre-aeration step before the leach circuit significantly reduces cyanide consumption and improves leach kinetics.
- Cyanide destruction testwork showed that the addition of 6.92 g SO_2/g CN_{WAD}, 3.23 g CaO/g CN_{WAD} and 0.08 g Cu/g CN_{WAD} were necessary to achieve a final concentration of < 1 mg/L CN_{WAD}.
- The microscopic gold deportment study conducted on cyanidation tailings showed that the main unleached gold mineral is calaverite ($AuTe_2$), with moderate amounts of petzite (Ag_3AuTe_2) and native gold, accounting for 65%, 19% and 15%, respectively.

The BV tests were mainly conducted on C2 and C4 coarse assay rejects zone composites as well as a Master composite. C2 and C4 core samples were used for grinding testwork. Variability tests were conducted on ten samples, one from the C1 zone, four (4) from the C2 zone, four (4) from the C4 zone and one for the C5 zone. In addition, 3-day Camflo rod mill feed composites from the first Camflo bulk sample campaign were also tested.

Tests included chemical analyses, comminution testwork, gravity concentration E GRG tests, multiple whole ore cyanidation tests, carbon gold loading testwork, cyanide destruction tests as well as thickening, rheology and filtration testwork.

Average recoveries during the three Camflo bulk sample campaigns were 95.9, 94.4 and 94.6% for the first, second and third campaigns respectively, with daily recoveries ranging from 94.7 to 97.4, 92.3 to 98.1 and 92.6 to 95.9%.

25.3 MILLING

This PFS assumes the Sigma Mill will process an average 575,000 tonnes per year, with toll milling at the Camflo and Westwood mills in 2018, Sigma mill start-up in Q4 2018 and planned processing of about 440,000 tonnes in 2019.

The Sigma Mill is composed of crushing, grinding, gravity concentration, leach and carbon-in-pulp (CIP) circuits, as well as elution, carbon regeneration and refinery areas. The crushing circuit includes a jaw crusher, cone crusher and a screen which needs to be replaced. The grinding circuit is composed of a primary rod mill and two ball mills. The circuit will be rearranged to use one ball mill as the secondary grinding step and the other as a tertiary grinding step, with each mill in closed circuit with its own cyclone, to be able to achieve a 50 µm grind size.

In 2019, installation of a new gravity concentration circuit is planned. This circuit will include two new vibrating screens, two new gravity concentrators and an intensive cyanidation skid with its dedicated electrolysis cell to be installed in the refinery. In the meantime, for the 2018 start-up one of the existing gravity concentrators will be refurbished and the existing shaking table will be used. For the time being, a new building extension to the north of the grinding building is planned for the installation of the new gravity circuit.

A pre-aeration tank is planned to be added before the leach circuit in 2019. The five leach tanks provide 52 hours of retention time before the slurry goes on to the 7 tank CIP circuit, four of which need to be replaced, for a combined leach and CIP retention time of 66 hours.

In 2019, installation of a new cyanide recovery thickener is planned. By then the paste plant will have started operation (planned for Q3 2019). Paste plant thickener overflow will be mixed with the CIP tailings prior to feeding the cyanide recovery thickener to enable cyanide recovery to the overflow. The underflow from the thickener will go on to cyanide destruction. Prior to installation of the cyanide recovery thickener, CIP tailings will go directly to cyanide destruction where sodium metabisulfite will be used as SO_2 source to reduce cyanide concentrations to environmentally acceptable limits.

A few reagent systems will be added to the plant, namely lead nitrate, milk of lime and sodium metabisulfite. Copper sulphate tanks will be moved and equipped with new pumps. The lead nitrate system will be installed in the former SAG building whereas milk of lime, sodium metabisulfite and copper sulphate systems will be installed in new building extensions to the south of the plant. In order to prepare the plant for potential future compliance to the International Cyanide Management Code, the cyanide tank will also be moved to a new building extension.

New low pressure air compressors will also be installed to feed the pre-aeration, leach, CIP and cyanide destruction tanks. For this the existing compressor building will be extended on each side.

Table 25.1 below presents the expected gold recoveries achievable in the Sigma mill for each zone, based on testwork results obtained thus far.

Table 25.1 – Expected Recoveries

Zone	Expected Gold Recovery (%)
C2	93.8
C4	95.3

In addition to the mill modifications and equipment replacements, almost all major equipment within the plant require some degree of refurbishing. This includes crushers, conveyors and feeders, grinding mills, the existing cyclone, screens, agitators, pumps and the regeneration kiln. Plant general infrastructure, such as piping systems, electrical system, instrumentation and control systems, the plant building itself, building ventilation, dust collection, spill containment areas and safety shower circuit, also need upgrades to meet current standards. These refurbishment works represent more than two thirds of the mill related capital costs.

25.4 TAILINGS IMPOUNDMENT AREA

The current plan consists of using the Sigma open pit, to store all tailings from the current LOM plan which is approximately 3.4 Mt of the tailings in form of paste fill. The Sigma Tailings Impoundment will be used as a secondary storage facility only, and is expected to hold up to a maximum of 0.4 Mt of tailings. Based on our assessment, the Sigma Tailings Impoundment has to be raised, in order to store all the 0.4 Mt of tailings volumes, while managing the design flood and other water inputs from the mill and the paste fill plant. A mobile water treatment station can be installed as an added security measure to ensure water quality compliance, and to add more flexibility for the water management of the site

The current status of the Sigma Tailings Impoundment does not meet the safety criteria recommended by Directive 019 under static and dynamic conditions. Moreover, before the current Sigma Tailings Impoundment could be used to store water and tailings as planned, all the basins need to be stabilized. The construction of the berms required to stabilize the Sigma impoundment's existing dykes under static and dynamic conditions will be performed in two (2) phases because of the following constraints:

- The footprint of the northern section of the stability berm exceeds Eldorado Gold's property limits. That portion of lands outside the property, which belong to the Corporation de développement industriel de Val d'Or (CDIVD), will need to be purchased;
- The portion of lands outside of the property is also considered as wetlands, which could delay significantly the permitting process. Compensation for wetlands losses will be required;
- The completion of the projected berm in one construction season could possibly overload the soft clay foundation, causing local failure as well as health and safety issues during construction.

This approach would allow for an additional time for Eldorado Gold to properly purchase the required lands from CDIVD, for permitting and for financial compensation for the loss of the mentioned wetlands. Furthermore, staged construction of the stability berm, planned over two phases, would allow time for the clay foundation to consolidate and thus gain strength, limiting the possibility of local failure during construction. These gains in strength of the foundation were not accounted for in stability analyses and could help in further optimizing the final configuration of the berm at Phase 2.

Further stress-strain modelling will be carried out for the phase 2 construction during the detailed design phase, while taking the considerations of the clay foundation following the planned stage construction. The impact of the phase 2 construction on the existing railway on the southside of the tailings impoundment, will also be assess to be certain that the final required berm will not cause any significant deformation of the railway.

25.5 ENVIRONMENT

Federal Regulations and Permitting: The Canadian Environmental Assessment Agency advised that the Lamaque South Project is considered as an extension of the Sigma-Lamaque Mine and Mill

Complex. Given that the extension represents an increase of less than 50% in the area of the mine operations, it is not subject to a federal environmental assessment.

25.6 CAPITAL AND OPERATING COST AND FINANCIAL MODEL

The accuracy of the capital and operating cost estimates is consistent with the standards outlined by the AACE. The economic model has been built from first principles and includes all relevant data, the qualified persons have a high level of confidence in the stated economic performance of the Project.

Eldorado's forecasts of costs are based on a set of assumptions current as at the date of completion of this technical report. The realized economic performance achieved on the Project may be affected by factors outside the control of Eldorado, including but not limited to mineral prices and currency fluctuations.

SECTION • 26 RECOMMENDATIONS

Due to the positive results of the 2017 underground bulk sampling program, completion of permitting for Lamaque production and PFS robust economics, it is recommended to expediently advance the Lamaque Project to pre-production followed by production including the Sigma mill refurbishment.

The recommended development path will include efforts to obtain the remaining permit approval for Sigma mill added construction and tailing management. It is also recommended that Eldorado concurrently advance key activities that will reduce Project execution time. This includes mostly for Sigma mill, initiating basic and detailed engineering work to finalize engineering designs, prepare work packages for procurement and the procurement of long lead time equipment.ue to the positive, robust economics, all of which have been included in the Capital Cost Estimate in Chapter 21.

Below is a description of the recommended steps in the continued advancement of the Triangle Mine Project. Table 26.1 summarizes each item and its estimated cost.

Table 26.1 – Proposed work program and budget

	Item	Cost (CAD$)
26.1	Undertake an economical study to reevaluate the potential of the Parallel Zone resources	60,000
26.2	Undertake a trade-off studies for alternate method of ore transportation, mining method and backfill method	150,000
26.3	Continue exploration on the Triangle Mine Project with 34,000m.	5,000,000
26.4	Undertake technical study to evaluate underground ramp from Sigma to Triangle to serve as access to Parallel zone, exploration drilling base and ore transportation between Triangle and Sigma.	150,000
26.5	Complete permitting works for Sigma mill and tailing management	500,000
	Total	$5,860,000

26.1 UNDERTAKE A TRADE-OFF STUDY TO REEVALUATE THE POTENTIAL OF THE PARALLEL ZONE RESOURCES

A revision of the block model of resources as well as a revised mine plan is recommended using different cut-off grades derived from alternate mining methods. This study should include but not limited to:

- An Integration of results from to date completed technical studies;
- An Integration of data obtained from the ongoing bulk sample on Triangle;
- A revised mining plan using revied block model;
- Consider alternative method of access to Parallel future infrastructure and adapt the mine plan and mining sequence accordingly to limit the capital requirements, and optimize the IRR and NPV of the project;
- An updated economic evaluation of capital and operating expenditures;

26.2 UNDERTAKE A TRADE-OFF STUDIES FOR ALTERNATE METHOD OF ORE TRANSPORTATION, MINING METHOD AND BACKFILL METHOD

The goal of these studies is to evaluate the best long-term strategy to transport the ore and optimized the mining and backfilling approach. These studies shall provide a thorough insight of the course of action to follow if the mining strategy should be changed so to offer an operational and financial gain over the current operating strategy.

The study shall include but not limited to the following:

- Make a preliminary selection of the applicable technologies;
- Definition of the Base Case using the latest PEA / PFS and current operating data and align the selected options with the actual mine plan;
- Definition of the comparison options;
- Validation/estimation of the production improvement globally and for each ore zone (automation, haulage / conveyance apparatus, mining method, muck size, backfill type, etc);
- For each option, prepare the design model (scoping level);
- Summary presentation with comparison table of the options vs the base case.

26.3 CONTINUE EXPLORATION ON THE TRIANGLE MINE PROJECT

Exploration objective is to grow the resources base, identify and test new exploration targets on the property. In 2018, some 34,000m of surface diamond drilling is proposed. The exploration program has the following objectives and consist in;

- Drill testing the extension at depth of the Triangle Deposit, below the C7 Zone to a vertical depth of 1800m with the objective to intersect gold mineralized structures similar to the ones hosting the current resources at Triangle and define the potential for the area to host a significant gold resources.
- Drill test potential extensions of the mineralized structures at Parallel Zone and identify new zones in the area that could grow the resources.
- Define the potential to host significant gold mineralization associated with sub-vertical shear structures associated with the various intrusions in the SW area of the project.
- Generate new exploration targets on the property showing potential to host significant mineralization from the compilation and interpretion of all historical and current data and based on our understanding of the regional and local ore-controls.

26.4 UNDERTAKE TECHNICAL STUDY TO EVALUATE UNDERGROUND RAMP FROM SIGMA TO TRIANGLE

Evaluate to a preliminary economic evaluatiuon level, a possible underground ramp between Sigma open pit and Triangle planned underground opening

Integrate the ramp as a possible access to Parallel and Plug 4 zone resources and also to serve as exploration drilling platform for other potential zones (Mine No 3, Plug 5, Fortune).

26.5 COMPLETE PERMITTING WORKS FOR SIGMA MILL AND TAILING MANAGEMENT

- Submit the authorizations documentation under section 22 of the EQA for ten CofA related to the technological upgrade of the Sigma mill, it's TSF, effluent treatment and inpit deposition. This permitting process is also subject to the acquisition of lands on the north side of the TSF Sigma and legal compensation to the crown for encroachment on 6.6 hectares of wetlands.
- Pursue rehabilitation and restoration plans process as stated in regulations, i.e. complete payments and reviews when required.

SECTION • 27 REFERENCES

AMEC Foster Wheeler Environment & Infrastructure, 2014. Étude de base sur l'environnement et le milieu socio-économique – projet aurifère Lamaque, Val-d'Or (QC). Report prepared for Integra Gold Corp. 151 p. and appendices.

Amec Foster Wheeler Environment & Infrastructure, 2017. Preliminary Economic Assessment – Lamaque Gold – Tailings management. 28 pages and 3 appendices.

Ayer, J., Amelin, Y., Corfu, F., Kamo, S., Ketchum, J.F., Kwok, K., and Trowell, N.F., 2002a. Evolution of the Abitibi greenstone belt based on U-Pb geochronology: Autochthonous volcanic construction followed by plutonism, regional deformation and sedimentation: Precambrian Research, v. 115, p. 63–95.

Ayer, J.A., Ketchum, J., and Trowell, N.F., 2002b. New geochronological and neodymium isotopic results from the Abitibi greenstone belt, with emphasis on the timing and the tectonic implications of Neoarchean sedimentation and volcanism: Ontario Geological Survey Open File Report 6100, p. 5-1–5-16.

Ayer, J.A., Trowell, N.F., Amelin, Y., and Corfu, F., 1998. Geological compilation of the Abitibi greenstone belt: Toward a revised stratigraphy based on compilation and new geochronology results: Ontario Geological Survey Miscellaneous Paper 169, p. 4-1–4-14.

Barnicoat, A. C., Fare, R. J., Groves, D. I., McNaughton, N. J., 1991. Synmetamorphic lode–gold deposits in high-grade Archean setting. Geology, v. 19, pp. 921–924.

Bateman, R., Ayer, J.A., and Dubé, B., 2008. The Timmins-Porcupine gold camp, Ontario: Anatomy of an Archean greenstone belt and ontogeny of gold mineralizations. Economic Geology, v. 103, p. 1285–1308.

Beaucamp, C., 2010. Origine métasomatique et contrôle structural de la minéralisation aurifère du secteur minier de Marban, canton de Dubuisson, Val-d'Or, Abitibi, Québec. Unpub. M.Sc. thesis, Montréal, Québec, Canada. Université du Québec à Montréal. 65 pages.

Beauregard, A.J. and Gaudreault, D. 2013. NI 43-101 Technical Work Report 2012 on the Lamaque property prepared by Geologica Groupe-Conseil Inc. for Integra Gold Corp. 2,777 pages (GM 67240).

Beauregard, A.J. and Gaudreault, D., 2013. Work Report on the MacGregor property prepared by Geologica Groupe-Conseil Inc. for Integra Gold Corp. 77 pages (GM 67569).

Beauregard, A.J. and Gaudreault, D., 2015. 2015 Fieldwork Report on the Lamaque South property over the Mining Concessions prepared by Geologica Groupe-Conseil Inc. for Integra Gold Corp. 1,938 pages (GM 69314).

Beauregard, A.J. and Gaudreault, D., 2015. Diamond drill hole PRS-14-01 on the Lamaque South property prepared by Geologica Groupe-Conseil Inc. for Integra Gold Corp. 11 pages (GM 68789).

Beauregard, A.J. and Gaudreault, D., 2016. 2015 Fieldwork Report on the Aumaque property prepared by Geologica Groupe-Conseil Inc. for Integra Gold Corp. 93 pages (GM 69236).

Beauregard, A.J. and Gaudreault, D., 2016. 2015 Fieldwork Report on the Sigma-Lamaque property prepared by Geologica Groupe-Conseil Inc. for Integra Gold Corp. 177 pages (GM 69237).

Beauregard, A.J., Gaudreault, D. and D'Amours, C. 2014. NI 43 101 Technical Report on the Lamaque property. Technical report prepared by Geologica Groupe-Conseil Inc. for Integra Gold Corp. 101 pages.

Beauregard, A.J., Gaudreault, D. and D'Amours, C., 2012. NI 43 101 Technical Report on the Lamaque property (Amended Sept. 2012). Technical Report on the Lamaque property, for Integra Gold Corp.112 pages.

Beauregard, A.J., Gaudreault, D. and D'Amours, C.. 2011. NI 43 101 Technical Report on the Lamaque property, for Integra Gold Corp. 90 pages.

Beauregard, A.J., Gaudreault, D. and D'Amours, C.., 2013. NI 43 101 Technical Report on the Lamaque property. Technical report prepared by Geologica Groupe-Conseil Inc. for Integra Gold Corp. 354 pages.

Beauregard, A.J., Gaudreault, D. and Guillaume. L., 2015. 2014 Fieldwork Report on the Lamaque South property over the Mining Concessions prepared by Geologica Groupe-Conseil Inc. for Integra Gold Corp. 1,433 pages (GM 68624).

Beauregard, A.J., Gaudreault, D. and Guillaume. L., 2015. 2014 Fieldwork Report on the Aumaque property prepared by Geologica Groupe-Conseil Inc. for Integra Gold Corp. 59 pages (GM 68822).

Beauregard, A.J., Gaudreault, D. Thibault, D., Blaize, B. and Boily, J., 2014. 2013 Fieldwork Report on the Lamaque South property (Triangle and South triangle Zones) prepared by Geologica Groupe-Conseil Inc. for Integra Gold Corp. 1,162 pages (GM 68213).

Bédard, P. 1979. Compagnie minière Lamaque Limitée. Gac-MAC Joint Annual Meeting, Excursion A2 Guidebook, pp. 59-65.

Bédard, P. 2014. Étude de Faisabilité infrastructure réseau, N/Réf: 121043 -0000- 9M- ER- 0001-_0, prepare par Stavibel, 9 janvier 2014, 11 p.

Bégin, P. 2005. Étude de suivi des effets sur l'environnement aquatique. Rapport d'interprétation du 1er cycle – Suivi initial. Rapport de GENIVAR Groupe Conseil inc. pour Mine Sigma, Century Mining Corporation, à Val-d'Or et soumis à Environnement Canada, Région du Québec. 50 p. et annexes.

Benn, K., and Peschler, A.P., 2005. A detachment fold model for fault zones in the Late Archean Abitibi greenstone belt: Tectonophysics, v. 400, p. 85–104.

Benn, K., Miles, W., Ghassemi, M. R., Gillet, J., 1994. Crustal structure and kinematic framework of the north-western Pontiac Subprovince, Québec: an integrated structural and geophysical study. Canadian Journal; of Earth Sciences, Vol. 31, pages 271-281.

Blecha, Matthew, 1985, Proposed Exploration of the Villemaque property Bourlamaque Township. Québec. Report #947T. Teck Explorations Limited.

Blecha, Matthew, Aug. 1985, Report on the First Program on the Lamaque property for the Golden Pond Joint Venture. Report #950T. Teck Explorations Limited.

Blecha, Matthew, Jun. 1986, Report on the 1985/86 Exploration of the Lamaque property Val-d'Or, Québec. Report #959T. Teck Explorations Limited.

Blecha, Matthew, Mar. 1986, Teck-Golden Pond Joint Venture Proposed Exploration and Development at Lamaque. Report #956T. Teck Explorations Limited.

Boileau, Pierre, 2004, Induced Polarization Survey executed on the Lamaque Project on behalf of Kalahari Resources Inc.

Bucci, L. A., Hagemann, S. G., Groves, D. I., Standing, J. G., 2002. The Archean Chalice gold deposit: a record of complex, multistage, high temperature hydrothermal activity and gold mineralisation associated with granitic rocks in the Yilgarn Craton, Western Australia. Ore Geology Reviews, v. 19, pp. 23-67.

Bucci, L. A., McNaughton, N. J., Fletcher, I. R., Groves, D. I., Kositcin, N., Stein, H. J., Hagemann, S. G., 2004. Timing and duration of high temperature gold mineralization and spatially associated granitoid magmatism at Chalice, Yilgarn Craton, Western Australia. Economic Geology, v. 99, pp.1123-1144.

Burrow, D.R., and E.T. Spooner, 1989, Relationship between Archean Gold Quartz Vein-Shear Zone Mineralization and Igneous Intrusions in the Val-d'Or and Timmins Areas. Abitibi Subprovince. Canada. Econ Geol. Monograph 6. pp. 424-444.

Campiglio C., 1977. Batholite de Bourlamaque. Ministère de l'Énergie et des Ressources du Québec. ES-026. 211 p.

Chown, E. H., Daigneault, R., Mueller, W., and Mortensen, J., 1992. Tectonic evolution of the Northern Volcanic Zone of Abitibi Belt. Canadian Journal of Earth Sciences, v. 29, pp. 2211-2225.

Chown, E. H., Hicks, J., Phillips, G. N., and Townend, R.,1984. The disseminated Archaean Big Bell gold deposit, Murchison Province, Western Australia; an example of premetamorphic hydrothermal alteration. In: Foster, R. P. (ed). Gold'82; the geology, geochemistry and genesisof gold deposits. Balkema, Rotterdam, pp 305–324.

Christopher, A., 1993, The Lamaque property Exploration Potential for Base Metals Report No. 1235NB. Teck Exploration Ltd.

Colvine, A. C., 1989. An empirical model for the formation of Archean gold deposits: products of final cratonization of the Superior Province, Canada. In: Keays, R. R., Ramsay, R., Groves, D. I., (eds). The Geology of gold deposits: the perspective in 1988. Economic Geology Monograph 6, pp. 37–53.

Corfu, Fernando, 1993, The Evolution of the Southern Abitibi Greenstone Belt in light of Precise U-Pb Geochronology. Econ. Geol. 88. pp. 1323-1340.

Costmine – division of INFOMINE USA INC., 2013 – Mining Cost Service, Volume 1 and Volume 2; Publisher: Jennifer B. Leinart.

Couture, J.-F., Pilote, P, Machado, N., and Desrochers, J.-P., 1994. Timing of gold mineralization in the Val-d'Or camp, southern Abitibi belt: Evidence for two distinct mineralization events. Economic Geology, v. 89, p. 1542-1551.

Daigneault, R., 1996. Couloirs de déformation de la Sous-Province de l'Abitibi. Ministère des Ressources naturelles du Québec, 114 pages. MB 96-33.

Daigneault, R., Mueller, W.U., Chown, E. H., 2002. Oblique Archean subduction: accretion and exhumation of an oceanic arc during dextral transpression, Southern Volcanic Zone, Abitibi Subprovince, Canada. Precambrian Research 115: 261–290.

Daigneault, R., Mueller, W.U., Chown, E.H., 2004. Abitibi greenstone belt plate tectonics: the diachronous history of arc development, accretion and collision. In Eriksson, P.G., Altermann, W., Nelson, D.R., Mueller, W.U., Catuneanu, O. (Eds.). The Precambrian Earth: Tempos and Events, Series: Developments in Precambrian geology, vol. 12, Elsevier, pages. 88–103.

Davis, Donald W., 2002, U-Pb geochronology of Archean metasedimentary rocks in the Pontiac and Abitibi subprovinces. Québec. Constraints on timing. provenance and regional tectonics. Precambrian Research 115. pp. 97-117.

Davis, W.J., Machado, N., Gariépy, C., Sawyer, E.W., and Benn, K., 1995. U-Pb geochronology of the Opatica tonalite-gneiss belt and its relationship to the Abitibi greenstone belt, Superior Province, Québec. Canadian Journal of Earth Sciences, 32: 113-127.

Derry, Michener, Booth & Wahl, 1988, Economic Evaluation of the Lamaque Mine. Val-d'Or, Québec prepared for Tundra Gold Mines Ltd.

Desrocher, J.-P., and Hubert, C., 1996. Structural evolution and early accretion of the Archean Malartic composite block, southern Abitibi greenstone belt, Quebec, Canada: Canadian Journal of Earth Sciences, v. 33, p. 1556-1569.

Desrochers, J.-P, Hubert, C, and Pilote, P., 1993. Géologie du secteur du lac De Montigny (phase 3), region de Val-d'Or. Ministère de l'Énergie et des Ressources, Québec, 47 pages. MB 93-15.

Dimroth, E, Imrech, L., Rocheleau, M., Goulet, N., 1982. Evolution of the south-central part of the Archean Abitibi Belt, Quebec. Part I: stratigraphy and paleostratigraphic model. Canadian Journal of Earth Sciences, Vol. 19, pages 1729-1758.

Dimroth, E, Imrech, L., Rocheleau, M., Goulet, N., 1983b. Evolution of the south-central part of the Archean Abitibi Belt, Quebec. Part III: plutonic and metamorphic evolution and geotectonic model. Canadian Journal of Earth Sciences, Vol. 20, pages 1374-1388.

Dimroth, E. - Imreh, L. - Rocheleau, M. - Goulet, N., 1983a. Evolution of the south-central part of the Archean Abitibi Belt, Québec. Part II: Tectonic évolution and geochemical model. Canadian Journal of Earth Science; volume 20, pages 1355-1373.

Dubé, B., and Gosselin, P., 2007. Greenstone-hosted quartz–carbonate vein deposits. In: Goodfellow, W. D. (ed). Mineral deposits of Canada: a synthesis of major deposit-types, district metallogeny, the evolution of geological provinces, and exploration methods. Geological Association of Canada, Mineral Deposits Division, Special Publication 5, pp. 49–73.

Eisenlohr, B. N., Groves, D. I., and Partington, G. A., 1989. Crustal-scale shear zones and their significance to Archean gold mineralization in Western Australia. Mineralium Deposita, v. 24(1), pp.1–8.

Feng, Rui, Jianzhong & Robert Kerrich, 1993, Noble Metal Abundances and Characteristics of Six Granitic Magma Series. Archean Abitibi Belt. Pontiac Subprovince: Relationship to Metallogeny and Overprinting of Mesothermal Gold Deposits. Econ. Geol. 88. pp. 1376-1401.

Foxford, K.A., Nicholson, R., Hebblethwaite, R.P.B., and Polya, D.A., 1995. Conceptual methods for modelling systems of mineralized echelon veins: Examples from southern England and Portugal: Exploration and Mining Geology, v. 4, p. 285-296.

Gaboury, D., Carrier, A., Crevier, M., Pelletier, C., and Sketchley, D.A., 2001. Predictive distribution of fault-fill and extensional veins: Example from the Sigma gold mine, Abitibi subprovince, Canada. Economic Geology, v. 96, p. 1397-1405.

Gagnon, Adam-Pierre H., Concept DB ingénierie, Réparation de la toiture existante et structure & enceinte antibruit au grizzly, Usine Sigma, Val-d'Or, Étude budgétaire, Révision 0.1, 24 November 2014.

Gagnon, Adam-Pierre H., Concept DB ingénierie, Réparation de la toiture existante et structure & enceinte antibruit au grizzly, Usine Sigma, Val-d'Or, Précisions pour étude budgétaire, Révision 0.1, 8 December 2014.

Garofalo, P., 2000. Gold precipitation and hydrothermal alteration during fluid flow through the vein network of the mesothermal gold deposit of Sigma (Abitibi belt, Canada): Unpublished Ph.D. thesis, Zurich, Swiss Federal Institute of Technology, 246 pages.

Gebre-Mariam, M., Hagemann, S. G., Groves, D. I., 1995 A classification scheme for epigenetic Archean lode–gold deposits. Minerilum Deposita, v.30(5), pp. 408–410.

GENIVAR. 2008. Rapport d'interprétation du deuxième cycle des ESEE de la mine Sigma. 39 p. et annexes.

GENIVAR. 2013. Rapport d'interprétation de quatrième cycle des ESEE; Century Mining. Corporation – Mine Sigma. 49 p. et annexes.

Golder Associates Limited, August 2014, Crown Pillar Stability Assessment in Support of the Prefeasibility Study for the Lamaque South Project, Ref. # 005-14-1221-0002-4000-RA-RevA.

Golder Associates Limited, August 2016, Rock Mass Classification Update of Triangle Zone and Mine Design of C2 Zone, Ref. # 003-1542161-TM-REV0.

Golder Associates Limited, December 2013, PEA Level Review of Geomechanics and Hydrology – DRAFT, Integra Gold – Lamaque Project, Ref. # 003-12-1221-0123-RA-RevD.

Golder Associates Limited, July 2014, Geomechanical Investigation and Design for the Pre-Feasibility Study for the Lamaque South Project, Ref. # 002-14-1221-0002-RevA.

Golder Associates Limited, November 2016, Mine Design of C4 Zone, Ref. # 004-1542161-TM.

Goutier, J., 1997. Géologie de la région de Destor: Ministère des Ressources naturelles du Québec 37 pages. RG 96-13.

Goutier, J., and Melançon, M., 2007. Compilation géologique de la Sous-province de l'Abitibi (version préliminaire): Ministère des Ressources naturelles et de la Faune du Québec.

Gouvernement du Québec. Mining Act, L.R.Q., Chapter M-13.1.

Gouvernement du Québec. Règlement sur la protection et la réhabilitation des terrains, c. Q-2, r. 37.

Gouvernement du Québec. Règlement sur les substances minérales autres que le pétrole, le gaz naturel et la saumure, c. M-13.1, r. 2.

Groves, D. I., Goldfard, R. J., Grebre-Mariam, M., Hagemann, S. G., Robert, F., 1998. Orogenic gold deposits: a proposed classification in the context of their crustal distribution and relationship to other gold deposit types. Ore Geology Reviews, v.13, pp. 7–27.

Groves, D. I., Goldfard, R. J., Robert, F., Hart, C. J. R., 2003. Gold deposits in metamorphic belts: overview of current understanding, outstanding, problems, future research, and exploration significance. Economic Geology, v. 98, pp. 1–29.

Groves, D. J., 1993. The crustal continuum model for late Archean lode–gold deposits of the Yilgarn Block, Western Australia. Mineralium Deposita, v. 28, pp. 366–374.

Gunning, H.C. and Ambrose, J.W., 1940. Malartic area, Québec. Geological Survey of Canada, Memoir 222. 142 pages.

Hanes, J.A., Archibald, D.A., Hodgson, C.J., and Robert, E, 1992. Dating of Archean auriferous quartz vein deposits in the Abitibi greenstone belt, Canada: 40Ar/39Ar evidence for a 70- to 100-

m.y. time gap between plutonism-metamorphism and mineralization. Economic Geology, v. 87, p. 1849-1861.
Hansen, Jens E., 1988, Tundra Gold Mines Limited, Lamaque Mine Gold Exploration. Targets from Processed Ground Magnetic Data.
Harris, G., Lewis, W.J., Mukhopadhyay, D. K., Patrick, G. and Lattanzi, C. R., 2011 Technical review of the Mining Plan/Operations and audit of the Resource and Reserve Estimates for the Lamaque Mine Project.
Hester, Brian, 1988, Lamaque Feasibility Study Geology and Reserves. DMBW. Inc.
Hodgson, C. J.,1993. Mesothermal lode–gold deposits. In: Kirkham, R. V., Sinclair, W. D., Thorpe, R. I., Duke, J. M., (eds). Mineral deposits modelling. Geological Association of Canada, Special Paper 40, pp. 635–678.
Hugon, Hervè, 1988, Structural Interpretation of the No. 35 Shears and No. 4 Plug Teck-Tundra Joint Venture property. Val-d'Or, Québec.
Imreh, L., 1976. Nouvelle lithostratigraphie à l'ouest de Val-d'Or et son incidence gîtologique. Ministère des Richesses Naturelles, Québec. 73 pages. DP-349.
Imreh, L., 1980. Variation morphologique des coulées méta-ultramafiques du sillon archéen de La Motte-Vassan. Precambrian Research, vol. 12, p. 3-30.
Imreh, L., 1984. Sillon de La Motte-Vassan et son avant-pays méridional: Synthèse volcanologique lithostratigraphique et gitologique. Ministère de l'Énergie et des Ressources du Québec. 72 pages. MM 82-04
Integra Gold, 2015. Plan de restauration pour des travaux d'exploration par voies souterraines – Propriété Lamaque Sud, Zone Triangle. 60 pages and 12 appendices.
J. Rivoirard, 2013; Jacques Rivoiradr, "A Top-Cut Model for Deposits with Heavy-Tailed Grade Distribution", Math Geosci, (2013) 45:967-982.
J. Rivoirard, 2013; Jacques Rivoiradr, "A Top-Cut Model for Deposits with Heavy-Tailed Grade Distribution.", Math Geosci, (2013) 45:967-982.
J.Rivoirard et al., 2012, "A Top-Cut Model for Deposits with Heavy-Tailed Grade Distribution."; Math Geosci (2013) 45:967-982.
Jemielita, R. A., Davis, D. W., Krogh, T. E., and Spooner, E. T. C., 1989. Chronological constraints on the origin of Archean lode gold deposits in the southern Superior Province from U-Pb isotopic analyses of hydrothermal rutile and titanite. Geol. Soc. America Ann. Mtg., Abstracts with Programs, 21 : A351.
Jemielita, R.A., Davis, D.W., and Krogh, T.E., 1990, U/Pb evidence for Abitibi gold mineralization postdating greenstone magmatism and metamorphism. Nature, v. 346, p. 831–834.
Jolly, W. T., 1978. Metamorphic history of the Archean Abitibi Belt. In Metamorphism in the Canadian Shield. Geological Survey of Canada, Paper 78-10, pp. 63-78.
Journeaux, Bédard & assoc. Inc., 2008. Note technique – Rehaussement Parc à résidus Mine Sigma (projet S-07-2018 (27 février 2008)), 7 pages et 2 annexes
Karvinen, William., 1985, Geology of the Lamaque Mine property. Val-d'Or, Québec.
Kent A. J. R., and McDougall. I.,1996. 40Ar–39Ar and U–Pb age constraints on the timing of gold mineralization in the Kalgoorlie gold field, Western Australia. Economic Geology 91, 795–799.

Kerrich, R., Goldfarb, R. J., Groves, D. I., Garwin, S., 2000. The geodynamics of the world-class gold deposits: characteristics, space–time distribution and origins. In: Hagemann, S. G., Brown, P. E. (eds). Gold in 2000, Reviews in Economic Geology v. 13, pp. 501–552.

Knight J.T., Groves D.I., Ridley J.R., 2003. District-scale structural and metamorphic controls on Archean lode–gold mineralization in the amphibolite facies Coolgarlie goldfield, Western Australia. Mineral Deposita 28, 436–456.

Kolb J., Hellmann, A., Rogers, A., Sinderin, S., Vennemann, T., Böttcher, M. E., Meyer, M. F., 2004. The role a transcrustal shear zone in orogenic gold mineralization at the Ajjanahalli Mine, Dharwar Craton, South India. Economic Geology, v.99, pp. 743–759.

Kolb, J., Rogers, A., and Meyer, M., 2005. Relative timing of deformation and two-stage gold mineralization at the Hutti Mine, Dharwar Craton, India. Mineralium Deposita, v. 40, pp. 156–174.

Lang, B. (1994) Span design for entry type excavations. MASc Thesis, University of British Columbia.

Latulippe, M., 1976. Excursion géologique Val-d'Or-Malartic. Ministère des Richesses Naturelles, Québec, 124 pages. DPV-367.

Lewis, W.J., Harris, G., Mukhopadhyay, D. K., Patrick, G. and Lattanzi, C. R., 2011 Technical review of the Mining Plan/Operations and audit of the Resource and Reserve Estimates for the Lamaque Mine Project. Technical report prepared by MICON International Ltd for Century Mining Corp., 223 pages.

Louvicourt Mining Management Co. Ltd., 1988, Evaluation of the Teck Corporation-Lamaque property in Val-d'Or.

Ludden, J.N., Hubert, C., and Gariépy, C., 1986. The tectonic evolution of the Abitibi greenstone belt of Canada: Geological Magazine, v. 123, pp. 153-166.

McCuaig, T.C., and Kerrich, R., 1998. P–T–t–deformation–fluid characteristics of lode gold deposits: evidence from alteration systematics. Ore Geology Reviews, v. 12:381–453.

MERQ-OGS, 1984, Lithostratigraphic map of the Abitibi subprovince: Ontario Geological Survey and Ministère de l'Énergie et des Ressources, Québec, Map 2484 and DV 83–16.

Ministère de l'Energie et des Ressources, Government of Québec, 1990, Carte Géologiques des Gites Métalliferes des Districts de Rouyn-Noranda et de Val-d'Or.

Ministère de l'Environnement et de la Faune du Québec (MDDELCC), 1998. Politique de protection des sols et de réhabilitation des terrains contaminés.

Ministère des Ressources Naturelles du Québec and the Ministère de l'Environnement et de la Faune du Québec, 1997. Guidelines for Preparing a Mine Site Rehabilitation Plan and General Mine Site Rehabilitation Requirements, 1997.

Ministère du Développement durable, de l'Environnement et des Parcs (MDDELCC), 2012 – Directive 019 sur l'industrie minière, mars 2012.

Morasse, S., Wasteneys, H.A., Cormier, M., Helmstaedt, H., and Mason, R., 1995. A pre-2686 Ma intrusion-related gold deposit at the Kiena mine, Val-d'Or, Québec, southern Abitibi subprovince. Economic Geology, v. 90, p. 1310-1321.

Morin, D., Jébrak M., Beaufort, D., Meunier, A., 1993. Metamorphic evolution of the late Archean Cadillac Tectonic Zone, McWatters, Abitibi belt, Quebec. Journal Metamorphic Geology, v.11, pp. 121–135.

Mueller, A. G., Campbell, I. H., Schiotte, L., Sevigney, J. H., Layer, P. W, 1996b. Constraints on the age of granitoid emplacement, metamorphism, gold mineralization, and subsequent cooling of the Archean greenstone terrane at Big Bell, Western Australia. Economic Geology, v. 91, pp. 896-915.

Mueller, W. U., Daigneault, R., Mortensen, J, Chown, E. H., 1996a. Archean terrane docking: upper crust collision tectonics, Abitibi Greenstone Belt, Quebec, Canada. Tectonophysics 265:127–150.

Muir, T. L., 2002. The Hemlo gold deposit, Ontario, Canada: principal deposit characteristics and constraints on mineralization. Ore Geology Reviews, v. 21, pp.1–66.

Neumayr, P., and Hagemann, S.G., 2002. Hydrothermal fluid evolution within the Cadillac Tectonic Zone, Abitibi greenstone belt, Canada: Relationship to auriferous fluids in adjacent second- and third-order shear zones. Economic Geology v. 97:1203–1225.

Neumayr, P., Cabri, L. J., Groves, D, I., Mikucki, E. J., and Jackman, J. A., 1993. The mineralogical distribution of gold and relative timing of gold mineralization in two Archaean settings of high metamorphic grade in Australia. Canadian Mineralogist, v. 31, pp. 711–725.

Neumayr, P., Hagemann, S. G., Banks, D. A., Yardley, B. W.D., Couture, J.-F., Landis, G. P., Rye, R., 2007. Fluid chemistry and evolution of hydrothermal fluids in an Archean transcrustal fault zone network: the case of the Cadillac Tectonic Zone, Abitibi greenstone belt Canada. Canadian Journal of Earth Sciences, v. 44, pp.745–773.

Neumayr, P., S.G. Hagemann and J.-F. Couture, 2000, Structural setting, textures and timing of hydrothermal vein systems in the Val-d’Or camp. Abitibi. Canada: Implications for the evolution of transcrustal. second- and third-order fault zones and gold mineralization. Can. J. of Earth Sci. 37. pp. 95-114.

Norman Lecuyer, Madon, Z. C., 1987, Carte Geologique. Drawing No. 6050. sheets 1 to 6. 1 inch to 200 feet. Corporation Teck-Division Lamaque.

Norman, G.W.H., 1947, Dubuisson, Bourlamaque, Louvicourt: Geological Survey of Canada Paper 47-20, p. 39-60.

Olivo, G. R., and Williams-Jones, A. E., 2002. Genesis of the Auriferous C Quartz-Tourmaline Vein of the Siscoe Mine, Val-d’Or District, Abitibi Subprovince, Canada: Structural, Mineralogical and Fluid Inclusion Constraints. Economic Geology, vol. 97, pp. 929–947.

Olivo, G. R., Chang, F. C., Kyser, T. K., 2006. Formation of the Auriferous and Barren North Dipper Veins in the Sigma Mine, Val-d’Or, Canada: Constraints from Structural, Mineralogical, Fluid Inclusion, and Isotopic Data. Economic Geology, v. 101, pp. 607-631.

Olivo, G. R., Isnard, H., Williams-Jones, A. E., and Gariépy, C., 2007. Pb-Isotope Compositions of the Pyrite from the C Quartz-tourmaline Vein of the Siscoe Gold Deposit, Val-d’Or, Quebec: Constraints on the Origin and Age of the Gold Mineralization. Economic Geology, v. 102, pp. 137-146.

Paquette, David, Concept DB ingénierie, Étude budgétaire – Réparation toit/structure usine & enceinte anti-bruit Grizzly, January 9, 2017.

Patton, T.C., 1988, Summary Report on the Lamaque property Val-d’Or, Québec for Tundra Gold Mines Limited.

Penczak, R. S., and Mason, R., 1999. Characteristics and origin of Archean premetamorphic hydrothermal alteration at the Campbell gold mine, northwestern Ontario, Canada. Economic Geology, v. 94(4), pp. 507–528.

Perrault, Guy, Pierre Trudel and Paul Bedard, 1984, Auriferous Halos Associated with the Gold Deposits at Lamaque mine. Québec. Econ Geol. 79. pp. 227-238.

Phillips, G. N., and De Nooy, D., 1988. High-grade metamorphic processes which influence Archean gold deposits, with particular reference to Big Bell, Australia. Journal of Metamorphic Geology, v. 6, pp. 95–114.

Phillips, G. N., and Powell, R. 2009. Formation of gold deposits: review and evaluation of the continuum model. Earth Sciences Reviews, v. 94, pp.1–21.

Pilote, P, Mueller, W., Lavoie, S., and Riopel, P, 1999, Géologie des Formations Val-d'Or, Héva et Jacola - nouvelle interprétations du Groupe de Malartic. Ministère des Ressources Naturelles du Québec. 19 pages. DV 99-03.

Pilote, P, Mueller, W., Scott, C., Lavoie, S., Champagne, C., and Moorhead, J., 1998b, Volcanology of the Val-d'Or Formation of the Malartic Group, Abitibi subprovince: Geochemical and geochronological constraints. Ministry of Natural Resources, Quebec. 48 pages. DV 98-05.

Pilote, P. - Moorhead, J. - Mueller, W., 1998a. Développement d'un arc volcanique, la régiond de Val-d'Or, Ceinture de l'Abitibi -Volcanologie physique et évolution métallogénique. Association géologique du Canada - Association minéralogique du Canada, Réunion annuelle, Québec 1998; Guide d'excursion A2, 85 pages.

Pilote, P. - Scott, C. - Mueller, W. - Lavoie, S. -Riopel, P., 1999 - Géologie des formations Val-d'Or, Héva et Jacola - nouvelle interprétation du bloc de Malartic. Ministère des Ressources naturelles, Explorer au Québec: Le défi de la connaissance, Séminaire d'information sur la recherche géologique, Programmes et résumés. Page 19. DV 99-03.

Pilote, P., 2015a. Géologie – Val-d'Or. Carte de compilation géoscientifiques, scale: 1: 20,000. Direction générale de Géologie Québec. Ministère des Ressources naturelles du Québec. CG-32C04A-2015-01.

Pilote, P., 2015b. Géologie – Lac de Montigny. Carte de compilation géoscientifiques, scale: 1: 20,000. Direction générale de Géologie Québec. Ministère des Ressources naturelles du Québec. CG-32C04C-2015-01.

Pilote, P., Couture, J-F, Desrochers, J-P, Machado, N., and Pelz, P., 1993. Minéralisations aurifères multiphasées dans la region de Val-d'Or: l'exemple de la mine Norlartic: Ministère de l'Énergie et des Ressources, Québec, DV 93-03, p. 61–66.

Pilote, P., Lacoste, P., David, J., Daigneault, R., McNicoll, V., and Moorhead, J., 2015c. La région de Val-d'or – Malartic: volcanologie et évolution métallogénique. In Stratigraphy, volcanology and metallogenic evolution of Val-d'Or – Malartic (Québec), Kirkland Lake, and Timmins (Ontario) areas, Abitibi Subprovince. Edited by P. Pilote and S. Préfontaine. Ministère de l'énergie et des Ressources naturelles du Québec and Ontario Geological Survey. 147 pages.

Pilote, P., Lacoste, P., Moorhead, J., Daigneault, R., McNicoll, V., and David, J, 2014. Géologie de la Région de Malartic. Présentation Oral. Ministère de l'Énergie et des Ressources naturelles du Québec. http://quebecmines.mrn.gouv.qc.ca/2013/ programme/pdf/s02_01_pilote_conf_fr.pdf.

Pilote, P., Moorhead, J., and Mueller, W., 2000, Partie A. Dévelopment d'un arc volcanique, La région de Val-d'Or, ceinture de l'Abitibi: volcanologie physique et évolution métallogénique. Ministère des Ressources Naturelles du Québec. 110 pages. MB 2000-09.

Pilote, P., Mueller, W., Moorhead, J, Scott, C., and Lavoie, S., 1997, Geology, volcanology and lithogeochemistry of the Val-d'Or and Heva Formations, Val-d'Or district, Abitibi subprovince. Ministry of Natural Resources, Quebec, 47 pages. DV 97-01.

Poirier, S. and Tremblay, A., 2013, Review of custom milling options – Lamaque Property Project, February 18, InnovExplo.

Poirier, S., Roy, L., D'Amours, C., Gaudreault, D., Bergeron, S., Caron, M., 2014. NI 43 101 Technical Report and Preliminary Economic Assessment for the Lamaque Project (according to National Instrument 43-101 and Form 43-101F1), Bourlamaque Township, Province of Québec, Canada. Technical report prepared by InnovExplo for Integra Gold Corp., 257 pages.

Poirier, S., Roy, L., D'Amours, C., Gaudreault, D., Bergeron, S., Ilieva, T., Utiger, M., 2015a. Technical Report and Updated Preliminary Economic Assessment for the Lamaque Project (according to National Instrument 43-101 and Form 43-101F1), Bourlamaque Township, Province of Québec, Canada. Technical report prepared by InnovExplo for Integra Gold Corp., 354 pages.

Poirier, S., Roy, L., D'Amours, C., Gaudreault, D., Bergeron, S., Ilieva, T., Utiger, M., 2015b. Technical Report and Mineral Resource Estimate for the Lamaque Project (according to National Instrument 43-101 and Form 43-101F1), Bourlamaque Township, Province of Québec, Canada. Technical report prepared by InnovExplo for Integra Gold Corp., 362 pages.

Poirier, S., Roy, L., D'Amours, C., Gaudreault, D., Bergeron, S., Ilieva, T., Utiger, M., 2015c. Technical Report and Fall 2015 Mineral Resource Estimate Update for the Lamaque Project (according to National Instrument 43-101 and Form 43-101F1), Bourlamaque Township, Province of Québec, Canada. Technical report prepared by InnovExplo for Integra Gold Corp., 386 pages.

Poirier, S., Roy, L., Turcotte, B., D'Amours, C., Gaudreault, D., Bergeron, S., Utiger, M, 2016. NI 43 101 Technical Report and 2016 mineral resource estimate update on the Lamaque Project, Bourlamaque Township, Province of Québec, Canada. Technical report prepared by InnovExplo for Integra Gold Corp., 429 pages.Poulsen, K.H., Robert, F., and Dubé, B., 2000. Geological Classification of Canadian Gold Deposits: Geological Survey of Canada, Bulletin 540, 106 p.

Powell W. G. and Pattinson D. R. M., 1997. An exsolution origin for low temperature sulphides at the Hemlo gold deposit, Ontario, Canada. Economic Geology 92, 569–577.

Powell W.G., Pattinson D. R. M., Johnston P., 1999. Metamorphic history of the Hemlo deposit from Al2SiO5 mineral assemblages, with implications for the timing of mineralization. Canadian Journal of Earth Sciences, 36, 33–46

Powell, W. D., Carmichael, D. M., and Hodgon, C. J., 1993. Thermobarometry in a subgreenschist to greenschist transition in metabasite of the Abitibi greenstone belt, Superior Province, Canada. Journal of Metamorphic Geology, Vol. 11, pages 165-178.

Ravenelle, J.-F. and Nagy, C., October 2013, Internal report – No.4 Plug Structural Geology Study and Three-Dimensional Modelling – SRK Consulting.

Richelieu Hydrogéologie, 2015. Integra Gold Corp – Propriété Lamaque : Étude hydrogéologique – Projet de mise en valeur souterraine de la Zone Triangle. 32 pages and 8 appendices.

Ridley, J., Groves, D. I., and Knight, J. T., 2000. Gold deposits in amphibolite and granulite facies terranes of the Archean Yilgarn Craton, Western Australia: evidence and implications of synmetamorphic mineralization. In: Spry, P. G., Marshall, B., Vokes, F. M., (eds). Metamorphosed and metamorphogenic ore deposits. Society of Economic Geologists, Reviews in Economic Geology 11, pp. 265-290.

Risto, R. W., Beauregard, A. J., and Gaudreault, D., 2004, Technical report on the Lamaque property, Val-d'Or, Québec for Kalahari Resources Inc. by Watts, Griffis and McOuat Limited. 138 pages.

Rivoirard, J., 2013. A Top-Cut Model for Deposits with Heavy-Tailed Grade Distribution. Math Geosci, (2013) 45:967-982.

Robert, F, Brown, A.C., and Audet, A.J., 1983. Structural control of gold mineralization at the Sigma mine, Val-d'Or, Quebec. Canadian Institute of Mining and Metallurgy Bulletin, v. 76, No. 850, p. 72-80.

Robert, F., 1983. Etude du mode de mise en place des veines auriferes de la mine Sigma, Val-d'Or, Quebec: Unpublished Ph.D. thesis, Ecole Polytech¬nique, University of Montreal, 295 pages.

Robert, F., 1989. Internal structures of the Cadillac Tectonic Zone, southeast of Val-d'Or, Abitibi greenstone belt, Québec. Canadian Journal of Earth Sciences, v. 26, p. 2661-2675.

Robert, F., 1990. Structural setting and control of gold-quartz veins of the Val-d'Or area, southeastern Abitibi subprovince. In Gold and Base Metal Mineralization in the Abitibi Subprovince, Canada, with Emphasis on the Quebec Segment, Ho, S.E., Robert, F, and Groves, D.I. (eds.), University of Western Australia Publication 24, p. 167-209.

Robert, F., 1990a. Structural setting and control of gold-quartz veins of the Val-d'Or area, southeastern Abitibi subprovince: University of Western Australia Geology Department and University Extension Publication, v. 24, p. 167–209.

Robert, F., 1990b. An overview of gold deposits in the eastern Abitibi belt: Canadian Institute of Mining and Metallurgy Special Volume 43, p. 93–106.

Robert, F., 1994. Vein fields in gold districts: The example of the Val-d'Or, southeastern Abitibi subprovince, Quebec. Canada Geological Survey Current Research 1994-C, p. 295-302.

Robert, F., 1994. Vein fields in gold districts: The example of Val-d'Or, southeastern Abitibi Subprovince. Geological Survey of Canada, Current Research, v. 1994c, p. 295–302.

Robert, F., 1996. A pre-2686 Ma intrusion-related gold deposit at the Kiena mine,Val-d'Or, Québec, southern Abitibi subprovince - A discussion. Economic Geology, v. 91, p. 803-807.

Robert, F., and Brown, A.C., 1986a. Archean gold-bearing quartz veins at the Sigma mine, Abitibi greenstone belt, Quebec. Part I. Geological relations and formation of vein system: Economic Geology, v. 81, p. 578-592.

Robert, F., and Brown, A.C., 1986b. Archean gold-bearing quartz veins at the Sigma mine, Abitibi greenstone belt, Quebec. Part II. Vein paragenesis and hydrothermal alter¬ation: ECONOMIC GEOLOGY, v. 81, p. 593-616.

Robert, F., and Poulse, K. H., 2001. Vein formation and deformation in greenstone gold deposits. Structural controls on ore genesis, Richards J. P. and Tosdal R. M., eds., Reviews in Economic Geology Volume 14, p. 111-156.

Robert, F., Boullier, A.-M., and Firdaous, K., 1995. Gold-quartz veins in metamorphic terranes and their bearing on the role of fluids in faults: Jour¬nal of Geophysical Research, v. 100, p. 12861-12879.

Robert, F., Poulsen, K. H., and Dubé, B., 1994. Structural analysis of lode gold deposits in deformed terranes. Geological Survey of Canada. Open File 2850. 140 pages.

Robert, F., Poulsen, K.H., Cassidy, K.F., and Hodgson, C.J., 2005. Gold metallogeny of the Superior and Yilgarn cratons, In: Economic Geology 100th Anniversary Volume, (ed.) J.W. Hedenquist, J.F.H. Thompson, R.J. Goldfarb, and J.P. Richards; Society of Economic Geologists, Littleton, Colorado, p. 1001−1033.

Robert, Francois and Alex C. Brown, 1984, Progressive Alteration associated with gold-quartz-tourmaline veins at the Sigma Mine. Abitibi greenstone Belt. Quebec. Econ Geol. 79. pp. 393-399.

Robert, Francois and Alex C. Brown, 1986. Archean Gold Forming Quartz Veins at the Sigma Mine. Abitibi Greenstone Belt. Québec: Part I. Geologic Relations and Formation of the Vein System. Econ Geol. 81. pp. 578-592.

Robert, Francois and Alex C. Brown, 1986b ,Archean Gold Forming Quartz Veins at the Sigma Mine. Abitibi Greenstone Belt. Québec: Part II. Vein Paragenesis and Hydrothermal Alteration. Econ Geol. 81. pp.593-616.

Roulston D., Johnston H., ALS Metallurgy Kamloops, Metallurgical Testwork on the Lamaque Deposit - Lamaque Project, Report KM3569, April 3, 2013.

Roulston D., Johnston H., ALS Metallurgy Kamloops, Metallurgical Testwork on the Lamaque Deposit - Integra Gold Corporation, Report KM4025, January 16, 2014.

Roulston D., Shouldice T., ALS Metallurgy Kamloops, Metallurgical Testwork on the Lamaque Deposit - Integra Gold Corporation, Report KM3876, May 29, 2013.

Roulston, D, and Johnston, H., 2013, ALS Metallurgy - Metallurgical test work on the Lamaque Deposit – Lamaque Project, KM3569, April 3, 2013.

Roulston, D, and Johnston, H., 2014, ALS Metallurgy - Additional Cyanidation Test Work on the Lamaque Deposit – Lamaque Project, KM4232, March 5, 2014.

Roulston, D, and Johnston, H., 2014, ALS Metallurgy - Metallurgical test work on the Lamaque Deposit – Lamaque Project, KM4025, January 16, 2014.

Roulston, D, and Shouldice, T., 2013, ALS Metallurgy - Metallurgical test work on the Lamaque Deposit – Lamaque Project, KM3876, May 29, 2013.

Sansfaçon, R - Hubert, C, 1990 - The Malartic gold district, Abitibi belt, Quebec: Geological setting, structure and timing of gold emplacement at Malartie Gold Fields, Barnat, East Malartie, Canadian Malartic and Sladen Mines. In La ceinture polymétallique du nord-ouest québécois, Éditeurs: M. Rive, P. Verpaelst, y. Gagnon, J.M. Lulin, G. Riverin et A. Simard, L'institut canadien des mines et de la métallurgie, Volume Spécial 43, pages 221-235.

Sauvé, P, Imreh, L., and Trudel, P., 1993. Description des gîtes d’or de la région de Val-d’Or, Ministère de l’Énergie et des Ressources du Québec. MM-91-03. 178 p.

Sauvé, P., Imreh, L., and Trudel, P., 1993. Description des gîtes d’or de la région de Val-d’Or. Ministère de l’Énergie et des Ressources du Québec. 198 pages. MM 91-03.

Sawyer, E. W., and Benn, K., 1993. Structure of the high-grade Opatica Belt and adjacent lowgrade Abitibi Subprovince, Canada: An Archean mountain front. Journal of Structural Geology, v.15, p. 1443-1458.

Scammell, D.R., & S. Frostad., Nov 1988, Proposed Surface Drilling Program Lamaque property Teck-Tundra/Teck-Golden Pond Joint Ventures. Report #999T. Teck Explorations Limited.

Scammell, D.R., Aug. 1988, Teck-Tundra Joint Venture report on the Geology and Reserves of the #35 Vein (No. 5 Plug) Bourlamaque Township, Québec. Report 996T. Teck Explorations Limited.
Scammell, D.R., Aug. 1989, Tundra/Golden Pond-Teck J.V. Report on the Base Metal Exploration Proposal for the Lamaque property. Report #1024T. Teck Explorations Limited.
Scammell, D.R., Dec. 1988, Report on the Lamaque Main Mine and No. 2 Mine. Val-d'Or, Québec for Tundra Gold Mines Limited. Scammell & Associates Inc.
Scammell, D.R., Dec. 1988, Report on the Lamaque Main Mine and No. 2 Mine. Val-d'Orm Québec for Tundra Gold Mines Limited. Scammell & Associates Inc.
Scammell, D.R., Jan. 1988, Report on the No. 4 Plug Lamaque property. Val-d'Or, Québec, Report #: 1004T. Teck Explorations Limited.
Scammell, D.R., Mar. 1989, Golden Pond/Teck – Tundra/Teck J.V. report on the Drilling of the Magnetic Anomalies (Phase I) and other Targets. Report #1009T. Teck Explorations Limited.
Scammell, D.R., May 1989, Preliminary Drill Report on the West Plug Lamaque property. Val-d'Or, Québec. Report #10118T. Teck Explorations Limited.
Scammell, D.R., May 1989, Report on the No. 4 Plug Lamaque property. Val-d'Or, Québec. Volume I of II. Report #1012T. Teck Explorations Limited.
Scammell, D.R., May 1989, Teck-Tundra J.V. Lamaque-Sigma Drilling Proposal Lamaque Mine. Val-d'Or, Québec. Report #1017T, Teck Explorations Limited.
Scammell, D.R., Oct. 1988, Teck-Golden Pond Venture Summary of the Fourth Program Lamaque property. Report #997T. Teck Explorations Limited.
Scammell, D.R., Oct. 1988, Teck-Tundra Joint Venture Report on the Geology and Reserves of the North Shear and South Shear. No. 5 Plug. Lamaque property. Report #998T. Teck Explorations Limited.
Scott, C R., Mueller, W. U., and Pilote, P., 2002, Physical volcanology. stratigraphy. and lithogeochemistry of an Archean volcanic arc: evolution from plume-related volcanism to arc rifting of SE Abitibi Greenstone Belt. Val-d'Or, Canada. Precambrian Research 115. pp. 223-260.
SEDAR web site, Integra Gold Corporation. http://sedar.com
Simard, M., Gaboury, D., Daigneault, R., and Mercier-Langevin, P., 2013. Multistage gold mineralization at the Lapa mine, Abitibi Subprovince: insights into auriferous hydrothermal and metasomatic processes in the Cadillac–Larder Lake Fault Zone. Mineralium Deposita, Volume 48, Issue 7, pp. 883-905.
Smith, D. S.,1996. Hydrothermal alteration at the Mineral Hill mine, Jardine, Montana: a lower amphibolite facies Archean lode gold deposit of probable synmetamorphic origin. Economic Geology, v. 91, pp. 723-750.
Stantec, May 2015, Memo – Lamaque gold project shaft estimate – 3 options, File #169515511
Sutcliff, R.H., C.T. Barrie, D.R. Burrows & G.P. Beakhouse, 1993, Plutonism in the Southern Abitibi Subprovince: A Tectonic and Petrographic Framework. Econ. Geol. 88. pp. 1359-1375
Taner, Mehmet F. & Pierre Trudel, 1991, Gold Distribution in the Val-d'Or Formation and model for the formation of the Lamaque-Sigma mines. Val-d'Or, Québec. Can J. Earth Sci. 28. pp. 706-720.
Thompson, P. H., 2003. Toward a new metamorphic framework for gold exploration in the Red Lake greenstone belt. Open File Report 6122, Ontario Geological Survey, 52 pages.

Thurston, P.C., and Chivers, K.M., 1990, Secular variation in greenstone sequence development emphasizing Superior province, Canada: Precambrian Research, v. 46, p. 21–58.

Thurston, P.C., Ayer, J.A., Goutier, J., and Hamilton, M.A., 2008, Depositional gaps in the Abitibi greenstone belt stratigraphy: A key to exploration for syngenetic mineralization. Economic Geology, v. 103, p. 1097−1134.

Tomkins A. G., and Mavrogenes J. A,, 2001. Redistribution of gold within arsenopyrite and löllingite during pro- and retrograde metamorphism: application to timing of mineralization. Economic Geology 96, 525–534.

Tomkins A. G., Pattinson D. R. M., Zaleski, E., 2004 The Hemlo gold deposit, Ontario: an example of melting and mobilization of a precious metal–sulfosalt assemblage during amphibolite facies metamorphism and deformation. Economic Geology 99,1063–1084.

Trudel, P., and Sauvé, P., 1992. Synthèse des caractéristiques géologiques des gisements d'or du district des gisements d'or du district de Malartic. Ministère de l'Énergie et des Ressources du Québec. 103 pages. MM 89-04.

Val-d'Or SAGAX, 1997, Image 2D for Teck Exploration Ltd. Lamaque Project Bourlamaque Township, Québec 97-N277. (Boileau. Pierre).

Verret F.-O. and Lascelles D., SGS Minerals Services, An investigation into the metallurgical response of samples from the Triangle zone of the Lamaque Sud Deposit, Project 15591-001, Final Report, Rev1, October 26, 2016.

Williams, B.R., 1988, Lamaque No. 10 Vein Mining No.1 Vein with No. 10 Vein. Teck Corporation.

Wilson, H.S., 1948. Lamaque Mine in Structural Geology of Canadian Ore Deposits A Symposium. Canadian Institute of Mining and Metallurgy, pp. 882 891.

Witt, W. K., 1993. Regional metamorphic controls on alteration associated with gold mineralization in the Eastern Goldfields province, Western Australia: implications for the timing and origin of Archean lode–gold deposits. Geology, v. 19, pp. 982–985.

Witt, W. K., and Vanderhor, F., 1998. Diversity within a unified model for Archean gold mineralization in the Yilgarn Craton of Western Australia: an overview of the late-orogenic, structurally controlled gold deposits. Ore Geology Reviews, v. 13, pp. 29–64.

Wong, L., Davis, D.W, Krogh, T.E., and Robert, E, 1991. U-Pb zircon and rutile chronology of Archean greenstone formation and gold mineralization in the Val-d'Or region, Quebec. Earth and Planetary Science Letters, v. 104, p. 325-336.

SECTION • 28 CERTIFICATE OF AUTHORS AND DATE AND SIGNATURE PAGE

The effective date of this report entitled "Technical Report, for the Lamaque Project, Quebec, Canada'' is March 21st, 2018. It has been prepared by Colm Keogh, P.Eng., Stephen Juras, PhD, P.Geo., Jacques Simoneau, P.Geo., Marianne Utiger, P. Eng., and François Chabot, P.Eng., each of whom are qualified persons as defined by NI43-101.

Signed this 29th day of March 2018.

"Signed"
Colm Keogh

Colm Keogh, P.Eng.
Manager, Mine Engineering (Underground)
Eldorado Gold Corporation

"Signed and Sealed"
Stephen Juras

Stephen Juras, PhD, P.Geo.
Director, Technical Services
Eldorado Gold Corporation

"Signed and Sealed"
Jacques Simoneau

Jacques Simoneau, P. Geo.
Exploration Manager
Eldorado Gold Corporation

"Signed"
'Marianne Utiger

Marianne Utiger, P. Eng.
Process Engineer
WSP Canada Inc.

"Signed"
'François Chabot

François Chabot, P.Eng.
Operations Manager
Eldorado Gold Corporation

CERTIFICATE OF QUALIFIED PERSON

Colm Keogh, P. Eng.
1188 Bentall 5, 550 Burrard St.
Vancouver, BC
Tel: (604) 687-4018
Fax: (604) 687-4026
Email: colmk@eldoradogold.com

I, Colm Keogh, am a Professional Engineer, employed as Manager, Mine Engineering (Underground) of Eldorado Gold Corporation and residing at 1107 Miller Road, Bowen Island in the Province of British Columbia.

This certificate applies to the technical report entitled ''Technical Report, for the Lamaque Project, Quebec, Canada'' with an effective date of March 21st, 2018.

I am a member of the Engineers & Geoscientists British Columbia (formerly the Association of Professional Engineers and Geoscientists of British Columbia) and also a member of l'Ordre des ingénieurs du Québec. I graduated with a Bachelor of Applied Science degree (Mining) from the University of British Columbia in Vancouver, British Columbia in 1988.

I have practiced my profession continuously from 1988 until 2001 and again from 2007 until present. I have been involved in mining engineering at underground base metal and precious mining operations in Canada, Ireland, Greece and Turkey. I have further provided mining engineering consultant services to underground base metal and precious metal properties in the United States, Peru, Brazil, Mexico, Mali, Canada and Ireland.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43-101.

I have visited the Lamaque Project on numerous occasions with my most recent visit occurring on February 27-28, 2018.

I was responsible for reviewing matters related to the estimation of Mineral Reserves, mining methods, project infrastructure, capital and operating costs, and economic analysis. I am responsible for sections 15, 16, 18, 19, 21, 22 and 24 and co-author of items 1 and 25 to 27.

As a third party engineering consultant, I have had prior involvement with the property that is the subject of this technical report.

I am not independent of Eldorado Gold Corporation as defined in Section 1.5 of National Instrument 43-101.

I have read National Instrument 43-101 and Form 43-101FI and the items for which I am responsible in this report entitled, Technical Report, for the Lamaque Project, Quebec, Canada'', Quebec, Canada, with an effective date of March 21, 2018, has been prepared in compliance with same.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the items of the technical report for which I am responsible contain all scientific and technical information that is required for disclosure.

Dated at Vancouver, British Columbia, this 29th day of March, 2018.

"Signed"

Colm Keogh

____

Colm Keogh, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

Stephen J. Juras, Ph.D., P.Geo
1188 Bentall 5, 550 Burrard St.
Vancouver, BC
Tel: (604) 601-6658
Email: stevej@eldoradogold.com

I, Stephen J. Juras, am a Professional Geoscientist, employed as Director, Technical Services, of Eldorado Gold Corporation and reside at 9030 161 Street in the City of Surrey in the Province of British Columbia.

This certificate applies to the technical report entitled *Technical Report for the Lamaque Project, Quebec, Canada*, with an effective date of March 21st, 2018.

I am a member of the Engineers & Geoscientists British Columbia (formerly the Association of Professional Engineers and Geoscientists of British Columbia) and also hold a Special Authorization from the Ordre de Géologues du Quebec for the purposes of evaluating the Lamaque Project mineral resources. I graduated from the University of Manitoba with a Bachelor of Science (Honours) degree in geology in 1978 and subsequently obtained a Master of Science degree in geology from the University of New Brunswick in 1981 and a Doctor of Philosophy degree in geology from the University of British Columbia in 1987.

I have practiced my profession continuously since 1987 and have been involved in: mineral exploration and mine geology on gold, copper, zinc and silver properties in Canada, United States, Brazil, China, Greece and Turkey; and ore control and resource modelling work on gold, copper, zinc, silver, platinum/palladium and industrial mineral properties in Canada, United States, Mongolia, China, Brazil, Turkey, Greece, Romania, Peru and Australia.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43-101.

I have visited the Lamaque Project on numerous occasions with my most recent visit occurring on January 23-25, 2018.

I was responsible for reviewing matters related to data verification and directing the mineral resource estimation and classification work for the Lamaque Project in Quebec, Canada. I am responsible for the preparation or supervising the preparation of items 12 and 14 in the technical report.

I have not had prior involvement with the property that is the subject of this technical report.

I am not independent of Eldorado Gold Corporation in accordance with the application of Section 1.5 of National Instrument 43-101.

I have read National Instrument 43-101 and Form 43-101FI and the items for which I am responsible in this report entitled, *Technical Report for the Lamaque Project, Quebec, Canada*, with an effective date of March 21st, 2018, has been prepared in compliance with same.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the items of the technical report that I was responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading

Dated at Vancouver, British Columbia, this 29th day of March, 2018.

"Signed and Sealed"

Stephen J. Juras

____

Stephen J. Juras, Ph.D., P.Geo.

CERTIFICATE OF QUALIFIED PERSON

Jacques Simoneau, P.Geo
300 3ième Avenue Est,
Val-d'Or, Québec, J9P 0J6
Tel: (819) 874 -3100
Email: jacques.simoneau@qc.eldoradogold.com

This certificate applies to the technical report prepared for Eldorado Gold Corporation – Lamaque ("Eldorado") entitled ''Technical Report, for the Lamaque Project, Quebec, Canada'' signed on March 29th, 2018 (the "Technical Report") and effective date March 21st, 2018.

I, Jacques Simoneau, P.Geo. (OGQ No. 737) do hereby certify:

I am a Professional Geologist, employed as an Exploration Manager at Eldorado Gold Corporation – Lamaque;

This certificate applies to the technical report entitled ''NI 43-101 2018 Prefeasibility Study Update for the Lamaque Project'' (the "Technical Report" or the "report"). The effective date of the report is March 21st, 2018, and the signature date is March 29th, 2018.

I am a graduate of Geology from Université de Montréal (1988). I am a member in good standing of the Ordre des Géologues du Québec (OGQ No. 737). I have more then 25 years of relevant experience in exploration geology, most of it related to gold exploration on projects similar to the Triangle Project.

I have read the definition of "qualified person" ("QP") set out in National Instrument 43-101/Regulation 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a QP for the purposes of NI 43-101.

I have been working full time on the Lamaque Project, including the Triangle Project, since February 2015, first with Integra Gold and now for Eldorado Gold, July 2017.

I am the author of items 4, 5, 6, 7, 9, 10, 11, 12 and 23, and co-author of items 1 and 25 to 27 of the report titled "Technical Report, for the Lamaque Project, Quebec, Canada" (the "Technical Report" or the "report"), prepared for Eldorado Gold. The effective date of the report is March 21, 2018, and the signature date is March 29, 2018.

I am not an independent of Eldorado Gold Corp., as defined by section 1.5 of NI 43-101.

I have read NI 43-101 and the items of the Technical Report that I am responsible for have been prepared in compliance with that instrument.

As of the date of this certificate, to the best of my knowledge, information and belief, the items of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed on this 29th day of March 2018.

"Signed and Sealed"

Jacques Simoneau

____

Jacques Simoneau, P.Geo.

CERTIFICATE OF QUALIFIED PERSON

Marianne Utiger, P.Eng
1600 René-Lévesque Blvd W., 16th Floor,
Montréal, Québec
Tel: (438) 843-7671
Email: Marianne.Utiger@wsp.com

This certificate applies to the technical report prepared for Eldorado Gold Corporation – Lamaque ("Eldorado") entitled "Technical Report, for the Lamaque Project, Quebec, Canada'' signed on March 29, 2018 (the "Technical Report") and effective date March 21st, 2018.

I, Marianne Utiger, Eng., of Montreal, Québec do hereby certify:

I am a Process Engineer with WSP Canada Inc. with a business address at 1600 René-Lévesque Blvd W., 16th Floor, Montréal, Québec H3H 1P9.

This certificate applies to the technical report entitled ''NI 43-101 2018 Prefeasibility Study Update for the Lamaque Project'' (the "Technical Report" or the "report"). The effective date of the report is March 21st, 2018, and the signature date is March 29th, 2018.

I am a graduate of Chemical Engineering from École Polytechnique de Montréal (1996). I am a member in good standing of the Ordre des Ingénieurs du Québec (OIQ No. 121018). I have 17 years of experience in process engineering in the metallurgical, mineral processing and chemical industries, including several years working on gold ore processing projects.

I have read the definition of "qualified person" ("QP") set out in National Instrument 43-101/Regulation 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a QP for the purposes of NI 43-101.

I visited the Sigma Mill on January 25, 2018 accompanied by Gabriel Belley of WSP.

I am the author of items 13 and 17 and co-author of sections 1, 2, 3, 25, 26 and 27 of the report titled "Technical Report for the, Lamaque Project, Quebec, Canada" (the "Technical Report" or the "report"), prepared for Eldorado Gold. The effective date of the report is March 21, 2018, and the signature date is March 29, 2018.

I am independent of Eldorado Gold Corp., as defined by section 1.5 of NI 43-101.

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

I have read NI 43-101 and the items of the Technical Report that I am responsible for have been prepared in compliance with that instrument.

As of the date of this certificate, to the best of my knowledge, information and belief, the items of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed on this 29th day of March 2018.

"Signed"
Marianne Utiger

Marianne Utiger, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

Francois Chabot, P.Eng
300 3ième Avenue Est,
Val-d'Or, Québec, J9P 0J6
Tel: (819) 874-3100
Email: francois.chabot@qc.eldoradogold.com

This certificate applies to the technical report prepared for Eldorado Gold Corporation – Lamaque ("Eldorado") entitled "Technical Report, for the Lamaque Project, Quebec, Canada'' signed on March 29, 2018 (the "Technical Report") and effective date March 21st, 2018.

I, François Chabot, P. Eng., of Val-d'Or, Québec do hereby certify:

I am an Operations Manager with Eldorado Gold Corporation – Lamaque, with a business address at 300 3ième Avenue Est, Val-d'Or, Québec J9P 0J6.

This certificate applies to the technical report entitled ''NI 43-101 2018 Prefeasibility Study Update for the Lamaque Project'' (the "Technical Report" or the "report"). The effective date of the report is March 21st, 2018, and the signature date is March 29th, 2018.

I am a graduate of Geological Engineering from Université Laval de Québec (1986). I am a member in good standing of the Ordre des Ingénieurs du Québec (OIQ No. 43977). I have 30 years of experience in mining geology, mining, environment, and operation management including several years working on permitting, environmental management for mining property, reclamation and mining social acceptability.

I have read the definition of "qualified person" ("QP") set out in National Instrument 43-101/Regulation 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a QP for the purposes of NI 43-101.

I have working full time on the Lamaque Project, including the Triangle Project, since 2013, first with Integra Gold and now for Eldorado Gold, July 2017.

I am responsible for section 20 and contributed to sections 1, 25 and 26 of the report titled "Technical Report, for the Lamaque Project, Quebec, Canada" (the "Technical Report" or the "report"), prepared for Eldorado Gold. The effective date of the report is March 21, 2018, and the signature date is March 29, 2018.

I am not independent of Eldorado Gold Corp., as defined by section 1.5 of NI 43-101.

I have read NI 43-101 and the items of the Technical Report that I am responsible for have been prepared in compliance with that instrument.

As of the date of this certificate, to the best of my knowledge, information and belief, the items of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed on this 29th day of March 2018

"Signed"
François Chabot

____
François Chabot, P.Eng.