NI 43-101 Technical Report
Feasibility Study for the Lynn Lake Gold Project, Manitoba, Canada

Prepared for

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List of Acronyms

AACE	Association for the Advancement of Cost Engineering International
AAS	Atomic Adsorption Spectroscopy
ABA	Acid Base Accounting
AEP	Annual Exceedance Probability
Ag	Silver
AGI	Alamos Gold Inc. (referred to as Alamos or the Company)
Ai	Abrasion Index
ALS	ALS Global
AMPX	AMPX Corporation
AMSL	Above Mean Sea Level
AP	Acid Potential
APS	Azimuth Pointing System
ARD	Acid Rock Drainage
ASL	Analytical Solutions Ltd.
ATV	All-Terrain Vehicle
Au	Gold
Axb	JK Parameter
BIF	Banded Iron Formation
BT	Burnt Timber
BWI	Ball Mill Work Index
CaO	Calcium Oxide
CAPEX	Capital and Sustaining Capital Expenditure
CCME	Canadian Council of Ministers of the Environment
CCTV	Closed-Circuit Television
CD Regulation	Classes of Development Regulation
CEA	Canadian Environmental Assessment
CEAA	Canadian Environment Assessment Act
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIL	Carbon-In-Leach
CIP	Carbon-In-Pulp
CLPA	Crown Land Property Agency
CMU	Concrete Masonry Unit
CN	Cyanide
CND	Cyanide Destruction

CN_F	Cyanide Final Titration
CN_T	Total Cyanide
CN_{WAD}	Cyanide Weak Acid Dissociable
CRA	Commercial, Recreational, or Aboriginal
CO_3	Carbonate
COSEWIC	Committee on the Status of Endangered Wildlife in Canada
CRM	Certified Reference Materials
CSD	Critical Solids Density
CSS	Close Size Setting
CV	Coefficient of Variation
CWI	Bond Low Energy Impact test
CWQG	Canadian Water Quality Guidelines
D&C	Design and Construct
DCIP	Direct Current Induced Polarization
DDH	Diamond Drill hole
DDP	Delivered Duty Paid
DFO	Fisheries and Oceans Canada
DO	Dissolved Oxygen
DTH	Down the Hole
DWT	Drop Weight Test
EA	Environmental Assessment
ECCC	Environmental and Climate Change Canada
EDA	Exploratory Data Analysis
EDF	Environmental Design Flow
EDGM	Earthquake Design Ground Motion
EDM	Electronic Distance Measure
EGL	Effective Grinding Length
EIS	Environmental Impact Statement
EM	Electromagnetic
EMP	Environmental Management Plan
EPC	Engineering, Procurement, and Construction
EPCM	Engineering Procurement and Construction Management
EPP	Environmental Protection Plan
FAL	Fresh Water Aquatic Life
FAT	Factory Acceptance Test

FCA	Free Carrier
Fe	Iron
FL	Gordon
FEL	Front End Loader
FIFO	Fly-in – Fly-out
FOS	Fine Ore Stockpile
FOT	Free on Truck
FS	Feasibility Study
FTE	Full Time Equivalent
Ga	Gigayears ago
G&A	General and Administrative
GPS	Global Positioning System
GRG	Gravity Recoverable Gold
GT	Grade Thickness
H_2O_2	Hydrogen Peroxide
HART	Highway Addressable Remote Transducer
HAZOP	Hazard and Operability Study
HBED	Hudson Bay Exploration and Development
HCl	Hydrochloric Acid
HDPE	High-Density Polyethylene
HHERA	Human Health and Ecological Risk Assessment
HLEM	Horizontal Loop Electromagnetic
HMI	Human-Machine Interface
HRIA	Heritage Resources Impact Assessment
HSEC	Health Safety Environment and Community
I/O	Input / Output
ICP	Inductively Coupled Plasma
ID3	Inverse Distance Weighting Method
IDF	Inflow Design Flood
IP	Induced Polarization
IR	Information Request
IRR	Internal Rate of Return
ISO	International Standards Organization
JK DWT	JK Drop Weight Test

JSZ	Johnson Shear Zone
KPI	Key Performance Indicator
LC50	Lethal Concentrations to 50% of the Test Organisms
LCS	Local Control Station
LG	Low Grade
LGD	Local Government District
LiDAR	Light Detection and Ranging
LLGB	Lynn Lake Greenstone Belt
LLGP	Lynn Lake Gold Project (referred to as the Project)
LOM	Life of Mine
MB CDC	Manitoba's Conservation Data Centre
MB ESEA	Manitoba's The Endangered Species and Ecosystems Act
MC	Master Composite
MCC	Motor Control Centre
MCL	MacLellan
MCE	Maximum Credible Earthquake
MCFN	Marcel Colomb First Nation
ML	Metal Leaching
MMER	Canadian Metal Mining Effluent Regulation
MMR	Manitoba Mineral Resources Ltd.
MSD	Manitoba Sustainable Development
MSE	Mechanical Stabilized Earth
MSOG	Manitoba Water Quality Standards, Objectives, and Guidelines
NaCN	Sodium Cyanide
NAG	Net Acid Generation
NaOH	Sodium Hydroxide
NN	Nearest Neighbour
NP	Neutralization Potential
Non-PAG	Not-Potentially Acid Generating
NPR	Neutralization Potential Ratios
NPV	Net Present Value
NQ	Drill Core Size (47.6 mm diameter)
NSR	Net Smelter Return
OCS	Operator Control Station
OEM	Original Equipment Manufacturer

OG	Ore Grade
OPEX	Operating Expenditure
OREAS	Ore Research and Exploration Pty Ltd Assay Standards
P&E	P&E Mining Consultants Inc
PAG	Potentially Acid Generating
$PbNO_3$	Lead Nitrate
PCA	Principal Components Analysis
PCS	Process Control System
PEP	Project Execution Plan
PID	Proportional-Integral-Derivative (Controllers)
PGA	Peak Ground Acceleration
PLC	Programmable Logic Controllers
PLUP	Provincial Land Use Policies
PMF	Probable Maximum Flood
PQ	Drill Core Size (85 mm diameter)
PR	Provincial Road
QA	Quality Assurance
QC	Quality Control
QP	Qualified Person
RAS	Required at Site
RC	Reverse Circulation
RCMP	Royal Canadian Mounted Police
RM	Reference Materials
ROM	Run of Mine
RWI	Bond Rod Mill
SAG	Semi-Autogenous Grinding (mills)
SARA	Species at Risk Act
SFE	Shake Flask Extraction
SG	Specific Gravity
SMC	SAG Mill Comminution
SMBS	Sodium metabisulphite
SO_2	Sulphur Dioxide
SOCC	Species of Conservation Concern
TCLP	Toxicity Characteristic Leaching Procedure
TIC	Total Inorganic Carbon

TK/TLRU	Traditional Knowledge/Traditional Land and Resource Use
TMF	Tailings Management Facility
TSL	TSL Laboratories
TSS	Total Suspended Solids
UCS	Uniaxial Compressive Strength
UF	Underflow
UG	Underground
UTM	Universal Transverse Mercator
UV	Ultra-Violet
VC	Valued Component
VFD	Variable Frequency Drive
VLF-EM	Very Low Frequency- Electromagnetic
VPSA	Vacuum Pressure Swing Adsorption
vs.	Versus
WI	Work Index
XRD	X-Ray Diffraction

Units of Measure

°	Degrees
˚C	Degrees Celsius
D_{50}	50th Percentile Particle Size
dBA	Decibel A-Weighting
ea	Each
ft	Feet
g/L	Grams per litre
g/t	Grams per tonne
GB	Gigabyte
h	Hours
ha	Hectare
hp	Horse power
kg	Kilogram
km	Kilometre
kPag	Kilopascal Gauge
kW	Kilowatt
L	Litre
L/s	Litres per second
lb	Pound
lbf	Pound force
L_{eq}	Sound level
m	Metre
M	Million
Ma	Millions of Years Before Present
Mbps	Megabytes per second
mm	Millimetre
MVA	Megavolt amperes
µm	Micrometer
m^2	Square metre
m^3	Cubic metre
min	Minute
mg/L	Milligram per litre
mL	Millilitre
MWh	Megawatt-hours

oz	Troy ounce
Pa	Pascal
pH	Measure of a solution’s acidity
$PM_{2.5}$	Particulate Matter (2.5 microns)
PM_{10}	Particulate Matter (10 microns)
ppb	Parts per billion
ppm	Parts per million
µm	Micrometre (Micron)
%	Percent
t	Tonne
t/d	Tonne per day
t/h	Tonne per hour
oz	Ounce
$\mu g/m^3$	Microgram per cubic metre
V	Volt
y	Year

1 SUMMARY

1.1 Introduction

Ausenco Engineering Canada Inc. (Ausenco) and a group of engineering and environmental consultants were engaged by Alamos Gold Inc. (Alamos) to complete the Project Feasibility Study (FS) and the NI 43-101 Technical Report for the Lynn Lake Gold Project (LLGP or the Project) located close to the Town of Lynn Lake, Manitoba.

The LLGP will be built as two conventional open pit mines with a centralized processing plant facility and a tailings management facility. The LLGP is composed of two properties: MacLellan and Gordon. The processing plant, located at MacLellan, has an expected nominal processing throughput of 7,000 t/d with an estimated 10.4-year production life.

The Technical Report and FS responsibilities of the engineering consultants are as follows:

- Ausenco was commissioned by Alamos to manage and coordinate the work related to the Project FS and the NI 43-101 for the LLGP. Ausenco was engaged also to develop the feasibility level design of the process plant and infrastructure for the Project;
- Q'Pit Inc. (Q'Pit) was commissioned to develop the design of the open pit final limits, the phase development and long-term mine plan, the selection of the equipment type and size, related estimates with respect to the open pit mine operating personnel and the estimates of the mine capital and operating costs for the life of the Project and the pre-production related quantities for the mine development roads. Q'Pit is responsible for the Mineral Reserve estimate;
- Stantec Consulting Ltd. (Stantec) was commissioned to support environmental planning, assessment, licensing, and permitting;
- Golder Associates Ltd. (Golder) was commissioned to complete the feasibility level design of the Tailings Management Facility (TMF) at the MacLellan site, slope design recommendations for the MacLellan and Gordon pits and stockpiles, as well as the layout and sizing of the water management structures for both sites. Golder was also responsible for undertaking various geotechnical and hydrogeological investigations in support of the feasibility level design and to aid in the sourcing of borrow materials for construction;
- RePlan was commissioned to complete the Lynn Lake Workforce Accommodation Study;
- All the consultants were engaged to provide input and contributed to the development of the Operating Cost (OPEX) and Capital and Sustaining Capital Expenditures (CAPEX); and
- Alamos reviewed and developed those elements of the Project relating to the geological setting and mineralization, Mineral Resources, market studies and contracts and economic analysis.

All costs are in Q2 2017 Canadian dollars unless otherwise stated.

1.2 Property Description

Located in Northern Manitoba, the LLGP consists of two primary sites, MacLellan and Gordon. These mine sites and their surrounding properties are within an easterly distance of 7 km and 37 km, respectively, of the Town of Lynn Lake. Presently, the LLGP's land portfolio is comprised of 247 dispositions, including 238 staked mining claims and nine Crown Leases, eight mineral and one surface, all of which the 100% recorded interest is held by AGI's wholly-owned subsidiary, Carlisle Goldfields Limited (Carlisle). As of the effective date of this report, all mining claims and leases are active and in good standing.

1.2.1 MacLellan Property

Formerly operated as an underground gold and silver mine between 1986 and 1989, the MacLellan site and its surrounding property is accessible via a 4.6 km all-weather gravel road traversing from the former mine site to Provincial Road (PR) 391 and into the Town of Lynn Lake. The whole of the MacLellan Property, which includes the historical MacLellan mine site lands, covers an area of 3,248 ha by way of nineteen mining claims, four Crown mineral leases and one Crown surface lease.

The historical MacLellan site has been in a 'care and maintenance' phase since 1989 with very little reclamation having taken place. This site currently consists of an all-weather gravel access road, power transmission line (abandoned pole line), and infrastructure from the former underground mine, such as a head frame, hoist house, shaft, access ramp, maintenance and other storage buildings, core shack and racks, vent raise, and mine water settling ponds.

1.2.2 Gordon Property

The Gordon site, historically referred to as the Farley Lake Open Pit Mine, previously operated as a two-pit open pit gold mine from 1996 to 1999. Accessible by way of a 15 km all-weather gravel road leading from PR 391, the former mine site is located approximately 37 km east of the Town of Lynn Lake. The Gordon Property consists of seventy-one mining claims and four Crown mineral leases totalling 12,915 ha.

After closure of the historical mine (see Section 1.4), the Gordon site underwent a reclamation process. The present day site consists of the above-mentioned gravel access road, a bridge across the Hughes River, two mine rock storage areas and two overburden storage areas that have been capped, and two water-filled open pits. All buildings and infrastructure from the historical operations have been removed.

1.3 Accessibility, Climate, Local Resources, Infrastructure and Physiography

Private all-weather gravel access roads connect both the MacLellan and Gordon sites with PR 391. PR 391 is a provincial road connecting Lynn Lake to Leaf Rapids (105 km east) and Thompson, Manitoba (315 km southeast), under the authority of Manitoba Infrastructure, Region 5.

The Town of Lynn Lake (Town) is a former mining community. The privately-operated Lynn Lake Airport services chartered flights and has a 1,700 m paved runway.

The LLGP is located within a remote, rugged region of the Boreal Shield Ecozone, in a climatic region characterized by short, cool summers and long, cold winters. The terrain consists of mostly hilly, till veneered bedrock, with intervening low areas of organic terrain ranging from level to moderately sloping (0-15%). Surface water features and peat generally occupy the topographic lows. Soils in the region are thin, poorly drained, and acidic, with organic soils typical in bogs and peat plateaus, and discontinuous permafrost is widespread.

The general area of the LLGP overlaps with the Paleoproterozoic Lynn Lake Greenstone Belt (LLGB) within the Churchill Structural Province of the Canadian Shield. The LLGB is comprised of volcanic rocks of the Wasekwan Group, sedimentary rocks of the Sickle Group, and plutonic intrusions. Overburden geology is characterized as glaciolacustrine sediments overlying either bedrock or a discontinuous regional sand till. Organic deposits consist of a thin veneer with thicker accumulations observed in low lying areas. Isolated pockets of glaciofluvial sediments are also present. A series of bedrock valleys near the MacLellan site are present where overburden is greater than 28 m thick.

1.4 History

1.4.1 MacLellan Property

Exploration on the MacLellan Property commenced in 1946 when Noranda Mines Limited (Noranda) carried out magnetometer and geological surveys on a portion of the Property. Numerous companies have explored the MacLellan Property and surrounds between 1946 and 2000. Production at the MacLellan Property under SherrGold Inc. (SherrGold) and LynnGold Resources Inc. (LynnGold) between 1986 and 1989 resulted in 969,680 t grading 5.36 g/t gold with an average mill rate of 900 t/d. Carlisle, as AMPX Corporation (AMPX), acquired the MacLellan Property in 2005 and actively explored the MacLellan Property up to 2014.

1.4.2 Gordon Property

Gold occurrences in the Lynn Lake area were first discovered by prospectors along the Johnson Shear in the late 1930s and early 1940s. Mapping in the area discovered gold in frost heaved boulders at the west end of Farley Lake. The deposit was mined from two small open pits during the period 1996-1999 by Black Hawk Mining Inc. (Black Hawk Mining) and reportedly produced 214,800 oz of gold.

1.5 Geological Setting and Mineralization

The LLGP is located within the Churchill Structural Province of the Canadian Shield. It consists of tholeiitic to calc-alkaline mafic volcanic and volcaniclastic rocks with minor rhyolite, dacite and sediments.

The LLGB, comprised of the younger north and older south belts, is part of a larger litho-structural unit which extends in a northeasterly direction from the La Ronge Greenstone Belt in Saskatchewan.

Both the north and south belts are comprised of steeply north dipping mafic/ultramafic to felsic volcanic rocks, clastic sediments, oxide facies banded iron formation, and mafic to felsic plutonic rocks. The belts are separated and bound to the north and south by large granitic batholiths.

The MacLellan Property is located on the western portion of the north trend within the Churchill Structural Province about 30 km west of the Gordon area. The north trend is thought to be a north facing homocline in the Agassiz Metallotect within the Wasekwan Group. The area is underlain by picritic basalts, biotitic and siliceous siltstones and lesser felsic volcanics.

The MacLellan deposit is stratabound between a sequence of east-west trending clastic and chemical sedimentary rocks interlayered with picritic flows and tuffs. This section is encompassed by volcanoclastic rocks to the north and south and felsic volcanics further to the south. Metamorphic grade is amphibolite. Minor felsic volcanics are located to the south of the mine area and gabbroic intrusions occur locally. The entire LLGP is bounded by granitic intrusions to the north and south.

The Gordon Property is hosted within the Agassiz Metallotect or Rainbow Trend of the north belt. The Rainbow Trend is a tectostratigraphic assemblage of ultramafic flows (picrites), banded oxide-facies iron formation, associated exhalative and clastic sedimentary rocks, and volcanic flows. This Trend represents a relatively narrow, strike-continuous stratigraphic-structural unit that occurs over a 70 km strike length from west of the shear hosted MacLellan deposit through the Gordon Property to southwest of Barrington Lake. This trend hosts all known gold mineralization in the north belt.

The Gordon Property is composed of pillowed basalts, dacitic flows, siliceous sediments (argillites, greywackes, etc.), intermediate tuffaceous sediment and banded iron formation (BIF) which have been extensively folded into an east-west striking and steeply north-dipping sequence.

1.6 Deposit Types

1.6.1 MacLellan

The MacLellan deposit developed through multiple, distinctive stages of hydrothermal fluid infiltration with no evidence that shear zone development resulted in significant dilation and vein formation. Unlike many other Precambrian lode gold deposits, the MacLellan deposit, according to this model, is unrelated to shear zone formation.

1.6.2 Gordon

The Gordon deposit is an epigenetic iron formation-hosted gold deposit. It is thought that oxide facies banded iron formation was selectively replaced by gold and sulphur-bearing fluids along faults and fractures in the iron formation. This is supported by the concentration of gold relative to the base metals in the deposit; sharp change from barren oxide facies iron formation to gold and sulphide-bearing iron formation and the fact that the deposits and the surrounding alteration zone crosscut stratigraphy.

1.7 Exploration

More recent exploration on the properties has been performed by Carlisle, AuRico Gold Inc. (AuRico) and now Alamos whom have merged with AuRico in 2015 and purchased Carlisle in 2016. Exploration programs include mapping and sampling, airborne magnetic survey, IP Surveys and exploration drilling.

1.8 Drilling

1.8.1 MacLellan

Drilling on the property has been conducted in a number of campaigns by prior operators and by the Company. A total of 1,490 surface and underground drill holes comprising 221,991m of drilling have been completed on the property. Of that total, 201 holes comprising 69,281m were drilled by Carlisle and Alamos during the period 2007-2016.

1.8.2 Gordon

Drilling on the Gordon Property has been conducted by Hudson Bay Exploration and Development (HBED) and Manitoba Mineral Resources Ltd (MMR) in 1985-1995 and Carlisle from 2012-2014, and the Company from 2015-2016. A total of 485 drill holes comprising 77,039 surface holes have been completed on the property. Of that total, 139 holes comprising 26,740 m were drilled by Carlisle and Alamos during the period 2012-2016.

1.9 Sample Preparation, Analyses and Security

Since 2011 drill core samples from MacLellan and Gordon were submitted to accredited commercial laboratories and assayed for gold using industry-standard fire assay techniques. The resource database also includes drill core samples for two earlier periods of exploration. From 1945 to 1985, Sherritt Gordon Mines Ltd. (Sherritt) and HBED drilled both properties and it is assumed that samples were assayed for gold by fire assay with a gravimetric finish. Additional drilling by Granduc Mining Corporation (Granduc) and Black Hawk Mining in the 1990s also generated samples where it is assumed fire assay was used for gold determinations. No details are available for sample preparation and assaying of the historical drill core samples.

1.10 Data Verification

Alamos geologic staff conducted a comparison between the resource drill hole database and the original assay certificates from the Sherritt site laboratory for pre-2007 drill holes for MacLellan, and the 654 series of drill holes at Gordon that were drilled by MMR during the period of February 1985 to April 1995.

For MacLellan, a total of 6,719 samples were checked, and 48 errors were identified and corrected. For Gordon, a total of 3,872 samples were checked, with only three errors identified and corrected in the database.

1.11 Metallurgical Test Work

1.11.1 Historical

A simple, whole ore leach process was used to treat MacLellan ore (SherrGold, 1980s) and Gordon ore (Black Hawk Mining, 1990s) in the old Lynn Lake Copper-Nickel concentrator (Sherritt) which was adapted for gold processing.

1.11.2 Test Programs

Alamos have conducted four phases of metallurgical testing since 2015, all at SGS Canada Inc. (SGS) Lakefield, Ontario facilities, using drill core composited into master, global, variability, reserve grade and location-specific samples. Objectives for each of the four stages of metallurgical test work were to:

1) Finalize flow sheet selection, optimize the main process parameters, determine comminution data and assess metallurgical variability;

2) Obtain comminution data (using whole PQ core); metallurgical tests on the products of the comminution tests;

3) Assess the process behaviour with internal dilution; test additional samples from the far eastern end of the MacLellan deposit; and

4) Assess the comminution and process behaviour for samples from the initial three years of mine production for both deposits.

1.11.3 Mineralogy

For MacLellan ore, the gold occurs as a mix of native gold, electrum and kustellite. The gold is fine grained, and predominantly liberated or exposed. Micro probe mineralogy confirmed that some gold is associated with arsenopyrite as micro-encapsulations.

For Gordon ore, the gold was found almost exclusively as native gold and is almost completely found as liberated or exposed attachments to iron oxides.

Although the gold is fine grained, both deposits have some coarse gold (> 100 µm) at an apparent rate of 1 grain per 2 kg sample.

1.11.4 Comminution

The deposits are of average hardness from a grinding perspective, but have a high to very high competency for SAG milling. The work indices from all tests were:

- Rod mill work index, average was 18.6 kWh/t; and
- Ball mill work index, average was 14.8 kWh/t from a range of 10-20 kWh/t.

The JK parameter Axb average was 28, with a range of 20-40.

Although similar in characteristics, the Gordon ore is more competent and harder than the MacLellan ore.

1.11.5 Extraction and Recovery

The optimum grind size determined for plant feed blends of MacLellan and Gordon ores and based on grind sensitivity trends, specific power required per ore type, unit cost of power and cyanide consumption with varying grind size, was a P_{80} of 75 µm. Gordon ore is strongly grind sensitive and MacLellan is not significantly grind sensitive in the range of grind sizes considered.

The overall recovery from whole ore leaching was comparable to that from gravity plus leaching of the gravity concentrate. Overall recovery was not improved using flotation and leaching of reground float concentrate and float tails. The standard laboratory tests used a gravity stage ahead of leaching to improve reconciliation of calculated and assayed heads and to mitigate potential misinterpretation of the test results from spotty or higher than usual residue grades.

The (unweighted) average gold recovery from tests on MacLellan “ore-grade” samples was 89.4% with a range of 80-98%, excluding low-recovery outlier results on two samples with high arsenic grades. The corresponding silver recovery was 52% with a range of 30-80%.

Similarly, the average gold recovery for Gordon “ore-grade” samples was 92.7% with a range of 85-98%. No significant silver grades occur in Gordon ores.

Pre-aeration with oxygen at a nominal pH 10 prior to leaching was essential to improve extraction kinetics and reduce consumption of sodium cyanide in the downstream leaching circuit. Six hours was provided for the pre-aeration duty based on oxygen uptake tests on various ore types including high-sulphur pyrrhotite and pyrite ores.

A total leach and adsorption time of 30 hours was suitable for these ores. Leach extraction is essentially complete ahead of six-stages of counter-current carbon-in-pulp adsorption, as confirmed by a series of simulations using proprietary modelling carried out by SGS Lakefield.

1.11.6 Detoxification and Environmental

Final leach/CIP tailings slurry is amenable to cyanide detoxification with the SO_2/air(O_2)/Cu^{2+} process. The not-to-exceed concentration of 10 mg/L weak acid dissociable cyanide was achieved at normal reagent doses.

Environmental tests comprising toxicity characteristic leaching procedure, shake flask extraction, acid base accounting, humidity cell tests and sub-aqueous column tests were completed on selected composite samples and leach products.

1.12 Mineral Resource Estimates

The Open Pit Mineral Resource estimate is based on data available from 1,945 drill holes drilled from both surface and underground, comprising 287,647 m of non-zero assayed gold intervals.

Separate block models were constructed for both the MacLellan deposit and the Gordon deposit and the Mineral Resource estimate is constrained by mineralized shapes, based on a 0.50 g/t Au cut-off grade. The MacLellan deposit and Gordon deposit block models have been depleted for historic underground and open pit mining, respectively.

Table 1-1 – LLGP Mineral Resource Statement, December 01, 2017

MacLellan					
Category	**kTonnes**	**Au Grade (g/t)**	**Ag Grade (g/t)**	**Au koz**	**Ag koz**
Measured	2,110	1.86	5.34	126	362
Indicated	2,236	1.24	4.24	89	305
Total Measured and Indicated	***4,347***	***1.54***	***4.77***	***215***	***667***
Inferred	750	1.62	2.80	39	67

Gordon					
Category	**kTonnes**	**Au Grade (g/t)**	**Ag Grade (g/t)**	**Au koz**	**Ag koz**
Measured	9	1.72	n/a	0.47	n/a
Indicated	451	1.96	n/a	28	n/a
Total Measured and Indicated	***460***	***1.95***	***n/a***	***29***	***n/a***
Inferred	615	1.30	n/a	26	n/a

MacLellan Underground					
Category	**kTonnes**	**Au Grade (g/t)**	**Ag Grade (g/t)**	**Au koz**	**Ag koz**
Measured	0	0.00	0.00	0	0
Indicated	1,161	4.37	6.23	163	233
Total Measured and Indicated	***1,161***	***4.37***	***6.23***	***163***	***233***
Inferred	116	3.82	3.43	14	13

Total					
Category	**kTonnes**	**Au Grade (g/t)**	**Ag Grade (g/t)**	**Au koz**	**Ag koz**
Measured	2,119	1.86	5.31	127	362
Indicated	3,848	2.27	4.34	281	537
Total Measured and Indicated	***5,967***	***2.12***	***4.69***	***407***	***900***
Inferred	1,481	1.66	1.69	79	80

Notes:

- *The Mineral Resources are reported at an assumed gold price of US $1,400/ounce, and an assumed silver price of US $22.00/ounce;*
- *The Mineral Resource estimate was completed by Mr. Jeffrey Volk, CPG, FAusIMM, Director of Reserves and Resources for Alamos Gold Inc.;*
- *Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resources estimated will be converted into Mineral Reserves;*
- *Open pit Mineral Resources are stated as contained within potentially economically open pit above a 0.42 g/t AuEq cut-off for MacLellan and 0.62 g/t Au for Gordon, and includes external dilution at zero grade outside the 0.50 g/t Au solid;*
- *Mineral Resources for the MacLellan Underground are stated above a 2.0 g/t Au cut-off. MacLellan Underground block grades are undiluted;*
- *Totals may not add due to rounding;*
- *Contained Au and Ag ounces are in-situ and do not include metallurgical recovery losses; and*
- *Mineral Resources are exclusive of Mineral Reserves.*

1.13 Mineral Reserve Estimate

The estimates of the Mineral Reserves were carried out based on the detailed open pit limit designs for the Gordon and MacLellan deposits and using the Measured and Indicated Mineral Resources of the block models of the two deposits. The estimates were carried out using cut-off grades of 0.69 Au g/t for Gordon and 0.47 equivalent Au g/t for MacLellan, calculated on the basis of the design parameters of the Project, which include a gold price of US $1,250/Au oz and an USD/CAD exchange rate of 0.75. The current geological model estimation methodology inherently introduces dilution in the estimate of the block gold grades of 13% and 15%, at zero grade, for MacLellan and Gordon respectively. Q'Pit is of the opinion that no additional dilution and mining recovery factors are applicable.

The Mineral Reserves for the LLGP are listed in Table 1-2 with the Au and Ag grade estimates based on the diluted grades of the block model.

Table 1-2 – LLGP Mineral Reserve Statement, December 01, 2017

	Class	Tonnage (Mt)	Au Grade (g/t)	Ag Grade (g/t)	Au oz Contained (x1000)	Ag oz Contained (x1000)
Gordon	Proven	2.31	2.82	n/a	210	n/a
	Probable	6.41	2.27	n/a	468	n/a
	Total Proven and Probable	8.72	2.42	n/a	678	n/a
MacLellan	Proven	9.55	1.91	5.01	586	1,539
	Probable	8.53	1.32	3.79	361	1,039
	Total Proven and Probable	18.08	1.63	4.43	947	2,578
Lynn Lake Gold Project	Proven	11.86	2.09	4.03	796	1,539
	Probable	14.94	1.73	2.16	829	1,039
Total Proven and Probable		26.80	1.89	2.99	1,625	2,578

Notes:

- *Mineral Reserves reported are in agreement with the CIM Definition Standards for Mineral Resources and Mineral Reserves*
- *The Mineral Reserve is estimated using metal prices of US $1,250/Au oz and US $15.00/Ag oz.*
- *Totals may not add up due to rounding.*
- *The estimates were carried out using cut-off grades of 0.69 Au g/t for Gordon and 0.47 Equivalent Au g/t for MacLellan and a metallurgical Au recovery of 89-94% for Gordon and 91-92% for MacLellan.*
- *The design parameters applicable are detailed in Section 15 of this report.*
- *The estimate of the Mineral Reserves was carried out under the supervision of Efthymios Koniaris, PhD., P.Eng., of Q'Pit Inc.*

1.14 Mining Production Plan

The development of the Gordon and MacLellan deposits is planned as a conventional shovel truck operation as both deposits outcrop and are amenable to mining using open pit methods.

The mine plan production summary is listed in Table 1-3.

Table 1-3 – Summary of Key Quantities for Lynn Lake Gold Project Mine Plan

Year	Y-2	Y-1	1	2	3	4	5	6	7	8	9	10	11	Total
Gordon Tonnes Mined (kt)	0	3,911	11,000	12,954	13,000	10,500	6,045	1,200	0	0	0	0	0	58,608
MacLellan Tonnes Mined (kt)	1,522	3,472	9,525	14,095	14,012	14,070	20,771	24,721	23,129	17,010	13,749	6,006	1,300	163,382
Total Tonnes Mined (kt)	1,522	7,383	20,524	27,049	27,012	24,570	26,816	25,921	23,129	17,010	13,749	6,006	1,300	221,991
Gordon														
Ore Tonnes Mined (kt)	0	120	1,647	1,651	1,805	1,738	1,406	356	0	0	0	0	0	8,723
Au Grade (g/t)	0	1.82	2.79	2.36	2.09	2.19	2.52	3.53	0	0	0	0	0	2.42
Au Ounces (oz x1000)	0	7	148	125	122	122	114	40	0	0	0	0	0	678
Waste Tonnes Mined (kt)	0	3,790	9,352	11,303	11,195	8,762	4,639	844	0	0	0	0	0	49,886
MacLellan														
Ore Tonnes Mined (kt)	0	251	1,060	1,227	906	1,213	2,242	2,168	1,866	1,813	2,507	2,110	715	18,080
Au Grade (g/t)	0	1.64	1.79	1.68	1.36	1.41	1.78	1.81	1.42	1.4	1.43	1.8	2.31	1.63
Au Ounces (oz x1000)	0	13	61	66	40	55	128	126	85	81	115	122	53	947
Ag Grade (g/t)	0	3.47	5.18	5.19	3.89	3.69	3.92	5.46	4.72	3.8	3.86	4.47	5.65	4.44
Ag Ounces (oz x1000)	0	28	177	205	113	144	283	380	283	221	311	303	130	2,578
Waste Tonnes Mined (kt)	1,523	3,221	8,464	12,867	13,106	12,857	18,529	22,553	21,263	15,197	11,241	3,895	585	145,302
Process Summary														
Process Nominal Capacity (t/d)	0	Variable	7,000	7,000	7,000	7,000	7,000	7,000	7,000	7,000	7,000	7,000	7,000	
Annual Mill Feed (kt)	0	322	2,555	2,555	2,555	2,555	2,555	2,555	2,555	2,555	2,555	2,555	930	26,803
Au Grade Contained (g/t)	0	1.59	2.51	2.28	1.91	2.06	2.63	2.04	1.27	1.18	1.42	1.59	1.91	1.89
Au Contained (oz x 1000)	0	16	206	187	157	169	216	167	104	97	116	131	57	1,625
Ag Grade Contained (g/t)	0	2.7	1.97	2.08	1.38	1.42	2.67	4.66	3.53	3.5	3.84	4.17	4.99	2.99
Ag Contained (oz x 1000)	0	28	162	171	113	117	220	383	290	288	315	343	149	2,578
Strip Ratio (Waste/Ore)	n/a	18.9	6.6	8.4	9	7.3	6.4	9.3	11.4	8.4	4.5	1.8	0.8	7.3

Key characteristics of the Life of Mine (LOM) plan are as follows:

- Concurrent operations on Gordon and MacLellan are planned at start-up of operations. The development targets the early depletion of Gordon in Year 6, due to its higher grade and lower stripping ratio. The mine life of MacLellan is 10.4 years, in addition to the two-year pre-production period;
- The phased development of both open pits is planned;
- The LOM calls for peak mining rate of 13 Mt/y at Gordon and 24.7 Mt/y at MacLellan. The mine plan calls for total mining in the range of 20.5 to 27.0 Mt/y for the first seven years of operation. Rate of vertical advance also limits the mine capacity in the final years of mining; and
- The mine equipment from Gordon is planned to be relocated to MacLellan between Year 4 and Year 6, as the mine capacity in Gordon is reduced when the pit nears depletion.

The scale of the operation as well as site specific operating conditions including selectivity requirements, and operating in severe climatic conditions in the winter, the remoteness of the mine location, operating around underground openings in MacLellan influenced the selection of mining equipment type and size.

For the primary production loading equipment, the FS for the LLGP open pits plans for the utilization of two 300 t class hydraulic shovels with a 14 m^3 bucket and two 11.6 m^3 front end loaders (FELs). The loading units are initially split between the mine sites.

Rigid body mine trucks of 144 t capacity were selected as the primary mine truck. The peak requirements are seven units at Gordon and 13 units at MacLellan. The MacLellan peak requirements are covered in part by the relocation of the Gordon haulage fleet.

Drilling will be undertaken by 60,000 lbf hydraulic top head drive drills as the primary production drilling unit, utilizing a Down-the-Hole (DTH) rock hammer configuration with a peak fleet of four units.

For the overburden removal at MacLellan, two 75 t to 90 t class hydraulic excavators will be used, combined with fleet of four 39.5 t articulated trucks.

A fleet of 23 highway trucks of 30 t capacity will be used for the haulage of Gordon ores to the process facility at MacLellan.

1.15 Recovery Process

The unit operations used to achieve plant throughput and metallurgical performance are well-proven in the gold/silver processing industry. The flow sheet incorporates the following major process operations:

- Two-stage crushing and stockpile;
- Semi-autogenous grinding (SAG);
- Ball mill grinding and classification;
- Leaching and carbon-in-pulp (CIP) adsorption;
- Desorption and gold room;
- Tailings detoxification and disposal;

- Fresh and reclaim water supply; and
- Reagent preparation and distribution.

Design throughputs for the following parts of the plant are:

- Crushing plant is 7,000 t/d or 490 t/h at 65% availability; and
- Process plant is 7,000 t/d or 317 t/h at 92% availability.

Life-of-mine feed grade to the plant is 1.89 g/t Au and 2.99 g/t Ag. Average production in each full year of production is 143,000 oz Au/y and 118,000 oz Ag/y.

1.16 Infrastructure

1.16.1 Access

Both sites have existing access roads connecting to PR 391. A new 2.5 km long road will be built to access the selected MacLellan process plant site.

1.16.2 Water Management Plan

The overall water management concept for the MacLellan site is to divert non-contact water to reduce the amount of water to be managed at the site, and to collect the contact water for conveyance to:

- The TMF, where water is stored for recirculation to the mill; or
- A collection pond, where sediment control can be provided prior to the release of water to the environment.

There are three non-contact water diversion ditches at the MacLellan site; located at the open pit, the Mine Rock Storage Area and the TMF.

The overall water management concept for the Gordon site is to divert non-contact water to reduce the amount of water to be managed at the site, and collect the contact water for conveyance to a collection pond where sediment control can be provided prior to its release to the environment.

The existing diversion channel that currently connects Gordon and Farley Lakes will have to be re-aligned to accommodate the expansion of the Gordon open pit, resulting in a new diversion ditch north of the open pit.

Water balance modelling was carried out for operations to estimate the amount of water collected at the sites, the amount of water available in the TMF for reclaim, the discharge volumes to the environment (i.e., site excess water), and the operating volume requirements for TMF and collection ponds sizing. Water balance simulations were conducted for the 100-year dry, mean, and 100-year wet annual precipitation conditions.

1.16.3 Tailings Management Facility

The site is located within the sporadic to discontinuous permafrost zone, where permafrost is typically found in 10% to 50% of the land area and in a relatively low seismic hazard region, known as the “stable central region”. The MacLellan TMF is located approximately 3 km northeast of the planned open pit and plant site areas, and 500 m north of Minton Lake.

The subsurface conditions encountered in the low lying areas of the proposed TMF typically can be characterized as a peat / topsoil, overlying a thin layer of glaciolacustrine clay, overlying a layer of sand to silty sand, transitioning into a silty sand till at depth prior to encountering bedrock.

The tailings are silty fine sand size with a specific gravity of 3.0. They are reported to be potentially acid generating (PAG), though not expected to produce acid rock drainage (ARD) during operations, with a high leaching potential for arsenic and a low leaching potential for metals including copper, iron, chromium, and lead. The Mineral Reserve is 26.8 Mt to be processed over 10.4 years excluding pre-production. The tailings / ore ratio is 1 which will produce about 20 Mm^3 of tailings assuming a deposited void ratio (vol. voids / vol. solids) of 1.0.

The tailings will be disposed of as slurry, pushing the tailings pond upstream against the natural topography. Deposition will initially be from start-up dams constructed on the south and east sides of the facility for the first two years. The tailings will be deposited from the dam crest after raising the dams by the downstream method. The dams are stage raised rockfill embankments with lined upstream slopes. The TMF will also be used as a collection pond for a portion of the site contact water runoff and mine dewatering flows. The facility has an emergency spillway and seepage collection system. Closure will include a vegetated overburden cover over the tailings with a small pond on the northern side of the TMF.

The accumulation of water in the tailings facility has been modelled for the 100-year dry, mean, and 100-year wet annual precipitation conditions. During the mine’s operational phase, water will be pumped from the TMF pond via a reclaim pump station for the operation of the process plant. Water recycled from the tailings pond is approximately 200,000 m^3/month. Additional clean make-up water is required in the process plant from an external source (e.g., from the Keewatin River) during the winter under certain annual precipitation conditions. Excess water accumulated in the facility will be stored in the tailings pond.

1.16.4 Power Supply

Average power demand at MacLellan is 13.8 MW, and will be supplied by Manitoba Hydro Line 6, which requires upgrades to operate at 138 kV. Incoming 138 kV will be stepped down at the main station by two 15/20 MVA transformers and distributed at 6.9 kV. A single VFD will be used to start SAG and ball mill motors to minimize voltage drop. 1 MW of emergency power will be provided by a diesel generator.

Average power demand at Gordon is 200 kW, and will be met by two 300 kW diesel generators in duty/standby configuration. Power distribution will be at 4.16 kV.

1.16.5 Accommodation

The camp will be located within the Town of Lynn Lake and will be established prior to Project construction phase, to be used continuously throughout pre-production and operations. Camp infrastructure will be tied into existing Town facilities.

The camp facility will be comprised of a 300 bed purchased camp plus 100 bed leased camp for the two year pre-production period.

1.17 Environmental Studies, Permitting and Social or Community Impact

Environmental baseline studies were initiated for the LLGP in March 2015 and were used to identify environmental constraints during the development of the layouts and designs for the Project. This includes consideration of siting and layout of Project infrastructure as well as consideration of design alternatives from an environmental management and approvals perspective.

There are several federal and provincial regulatory requirements that may apply to the Project, including an environmental assessment (EA) and other environmental permitting obligations. A Project Description was submitted to the Canadian Environmental Assessment (CEA) Agency on July 04, 2017 to initiate the federal EA process under the *Canadian Environmental Assessment Act* (CEAA) 2012 and inform the provincial EA process under *The Environment Act* of Manitoba. A single Environmental Impact Statement (EIS) document is currently being prepared that will be submitted to satisfy federal and provincial EA requirements.

The Project design, including implementing the identified mitigation measures, is not anticipated to cause significant adverse environmental effects, including effects from accidents and malfunctions, effects of the environment on the Project and cumulative effects. No issues have been identified to date that are expected to materially affect the ability of Alamos Gold to extract minerals from the Project. This will be confirmed through the environmental assessment and permitting phases of the LLGP development and may require additional design modifications or mitigation measures to be implemented. As discussions with agencies are still ongoing, and the environmental assessment for the LLGP has just been initiated, the extent of changes to the Project cannot fully be captured in this report.

1.18 Capital and Operating Costs

1.18.1 Capital Cost

The total LLGP capital cost is shown in Table 1-4.

Table 1-4 – Total Capital Cost

Facilities	Total Cost (C $ in millions)	Total Cost (US $ in millions)
Initial Capital	$450.6	$338.0
Sustaining Capital and Closure	$196.9	$147.6
Total Capital Cost	$647.5	$485.6

The summary level initial Direct and Indirect capital costs by area for are shown in Table 1-5.

Table 1-5 – Summary of Initial Capital Costs (Canadian Dollars, Q2, 2017)

Description	Gordon Mine Total Cost ($'000s)	MacLellan Mine Total Cost ($'000s)	Both Mines Total Cost ($'000s)
Direct Cost:			
Mining Initial Capital Lease Payment (Common)			$23,767
Mine Pre-production and Miscellaneous	$20,295	$37,687	$57,982
Process Plant		$96,424	$96,424
Utilities and Services	$2,424	$20,378	$22,802
Tailings Management		$22,785	$22,785
On-site Infrastructure	$9,048	$46,881	$55,929
Off-site Infrastructure:			
Camp and Infrastructure			$15,694
Manitoba Hydro Line			$17,000
Subtotal Direct Costs	$31,767	$224,155	$312,383
Indirect Cost:			
EPCM and Consulting Services			$28,930
Vendor Representatives			$647
Freight			$4,668
Temporary Construction Facilities and Utilities			$4,882
Pre-production Camp Operations			$21,041
First Fills and Opening Stocks			$2,866
Initial Spares			$7,860
Provincial Sales Tax			$12,629
Subtotal Indirect Costs			$83,523
Subtotal Direct + Indirect			$395,906
Project Contingency			$38,432
Sub Total Directs + Indirects + Contingency			$434,338
Owner Cost			$16,280
TOTAL INITIAL CAPITAL COST			**$450,619**

The cost estimate base date is Q2 2017 and the Scope of Work consists of direct costs, indirect costs, Owner's costs and contingency, as follows:

- Direct costs: Costs of all permanent equipment and bulk materials and the installation costs for all permanent facilities including contractor's supervision and management costs, contractor's travelling costs and contractor's administration and profits;
- Indirect costs: Costs of EPCM services, temporary construction facilities and services, construction equipment, freight, vendor erection supervision, commissioning and start-up, first fills and spares;
- Owner's costs: Costs associated with owner's facilities and services during construction, owner's project management, ramp up and general fees; and
- Contingency: A construction contingency to cover necessary work within the defined scope of the Project which cannot be identified or itemized at this stage of the Project development, but is expected to be incurred.

The major facilities (areas) covered in the capital cost estimate are as follows:

- Mine area;
- Process plant;
- Tailings management;
- On-site infrastructure; and
- Off-site infrastructure.

The estimate conforms to AACE Class 3 guidelines for a Feasibility Study Estimate with a -10% to +15% accuracy.

Owner's costs include the following:

- Land;
- Owner's team including construction, startup and commissioning;
- Recruiting, training and site visits;
- IT and communications; and
- Insurance, finance, legal and Lynn Lake office.

1.18.2 Operating Cost

The overall LOM excluding pre-production operating cost is $1,273.2M (US $954.9M) or $48.08/t (US $36.06/t) of ore milled. Table 1-6 presents the total operating costs for the Project.

Table 1-6 – LOM Operating Cost Summary

Cost Centre	M$	$/t milled	% of Total
Mining	627.3	23.69	49
Highway Haulage (Gordon hauled ore averaged over full mill tonnage)[1]	85.9	3.25	7
Processing	382.8	14.46	30
General and Administration	106.9	4.04	8
Accommodation and Transport	81.2	3.07	6
External Refining	4.8	0.18	0
Subtotal OPEX	1,289.0	48.68	100
Royalties and Ag Credits	(15.8)	(0.60)	-
Total OPEX	1,273.2	48.08	-

Notes:

1. *Actual highway haulage operating cost is $9.93/t applied to milled ore from Gordon only. Indicated $/t is average over total mill feed.*

1.18.2.1 *Mining Operating Cost*

The estimate of the mine operating costs was based on mine plan physicals, information compiled from original equipment manufacturers and Q'Pit's information and experience on projects of similar scope and size. This estimate assumes that all of the equipment is owned and operated by Alamos, with a component exchange and rebuild program with the OEM suppliers as well as other parts suppliers in the vicinity of the operation.

The MacLellan mine costs include the cost of re-handling both from the long-term MacLellan stockpiles as well as the ROM stockpile at the primary crusher. ROM stockpile re-handle includes the re-handle of all the ore from Gordon as well as a fraction of the MacLellan ore.

The LOM mine operating costs were estimated at $3.07/t mined for Gordon and $2.90/t mined for MacLellan. The mine operating costs by function are listed in Table 1-7. These costs exclude the pre-production period.

Table 1-7– Mine Operating Cost by Pit and by Area

Mining Cost by Area	Gordon			MacLellan			Total
	$/t	$M	Percent (%)	$/t	$M	Percent (%)	$M
Drilling	0.41	22.6	13	0.34	54.5	12	77
Blasting	0.57	31.2	19	0.54	85.6	19	116.8
Loading	0.43	23.6	14	0.41	65	14	88.6
Hauling	0.86	46.8	28	0.9	142	31	188.8
Support	0.55	30.2	18	0.41	65.6	14	95.8
Assaying	0.06	3.4	2	0.06	8.8	2	12.3
Voids Management	0	0	0	0.03	5.3	1	5.3
Miscellaneous	0.09	4.7	3	0.09	14.1	3	18.8
Engineering and Management	0.15	8	4	0.14	22.4	5	30.4
Initial Spare Parts Recovery	-0.05	-2.7	-2	-0.02	-3.6	-1	-6.3
Total Mining OPEX	3.07	167.7	100	2.9	459.6	100	627.3

1.18.2.2 *Process Operating Cost*

LOM process operating cost is $382.8M or $14.46/t (US $10.84/t) milled. A breakdown of this OPEX value and its unit costs are presented in Table 1-8.

Table 1-8 – LOM Process Operating Cost

Cost Centre	$M	$/t milled	% of Total
Labour	119.1	4.5	31
Power	52	1.96	14
Operating Consumables:			
Reagents	85.8	3.24	22
Steel Liners and Ball Media	82.3	3.11	22
Utilities	6.2	0.23	2
Maintenance	26.7	1.01	7
Vehicles and Mobile Equipment	5.5	0.21	1
Laboratory and Assays	4.6	0.17	1
Water Treatment	0.6	0.02	0
Total Process Operating Costs	382.8	14.46	100

1.18.2.3 G&A / Accommodation Cost

LOM G&A / accommodation cost is $188.1M or $7.10/t (US $5.33/t) milled. A breakdown of this value and its unit costs are presented in Table 1-9.

Table 1-9 – LOM G&A/Accommodations Operating Cost Summary

Department / Area	$M	$/t milled	% of Total
Accommodation and Transportation	81.2	3.07	43
Personnel	53.9	2.04	29
Contract Services	19.1	0.72	10
Assets Operation	2.1	0.08	1
Human Resources	3.3	0.13	2
Infrastructure Maintenance	8.4	0.32	4
Infrastructure Power	2.5	0.09	1
Site Admin, Maintenance and Security	2.5	0.09	1
IT and Telecommunications	2.5	0.1	1
Health and Safety	0.8	0.03	0
Miscellaneous and Permitting	11.7	0.44	6
Total G&A Costs	188.1	7.1	100

1.19 Economic Analysis

1.19.1 Taxes

The LLGP will be subject to provincial, federal and mining taxes as follows:

- Manitoba Mining Tax: sliding scale with rates between 10% and 17%;
- Manitoba Provincial Income Tax: 12%; and
- Federal Income Tax: 15%.

The rates above are current as of the date of this report, and are subject to change in the future. Based on these rates and the financial assumptions used in this report, the LLGP is expected to have payable income and mining taxes of $174.2M (US $130.6M) over its 10.4-year life.

1.19.2 Royalties

The LLGP is subject to a third-party royalty in the first two years of production from the Gordon pit. Total royalty included in the cash flow model is $10.8M (US $8.1M).

1.19.3 Economic Analysis

The Project is economically viable with an after-tax internal rate of return (IRR) of 12.5% and an after-tax net present value at a 5% discount rate ($NPV^{5\%}$) of $164.5M (US $123.4M). Other economic factors used in the economic analysis include the following:

- US $1,250/oz gold, US $16.00/oz silver and a $0.75 USD/CAD exchange rate were used in the cash flow model;
- Discount rate of 5%;
- Closure cost of $28.1M (US $21.1M);
- No salvage assumed at the end of mine life;
- Working capital outflow of $10.0M (US $7.5M) total in Year -1 and Year 1, offset by $10.0M (US $7.5M) total inflow at the end of the mine life;
- Numbers are presented on a 100% ownership basis and do not include inter-company management fees or non-equipment lease financing costs; and
- Exclusion of all pre-development and sunk costs (i.e. exploration and Mineral Resource definition costs, engineering fieldwork and studies costs, environmental baseline studies costs, etc.). However, pre-development and sunk costs are utilized in the tax calculations.

Table 1-10 - Summary of Economic Results

Category	Unit	Value (C $)	Value (US $)
Net Revenues	$M	2,466.8	1,850.1
Operating Costs[1]	$M	1,273.2	954.9
Cash Flow from Operations[2]	$M	1,019.5	764.6
Capital Costs, Sustaining Capital and Closure Costs	$M	647.5	485.6
Total Cash Cost	$/oz	860	645
Mine Site All-In Sustaining Cost	$/oz	993	745
Net After-Tax Cash Flow	$M	372	279
After-Tax $NPV^{5\%}$	$M	164.5	123.4
After-Tax IRR	%	0.125	0.125
After-Tax Payback	Years	4.6	4.6

Notes:

1. *Operating Costs include mining, processing, G&A, royalties, transport and refining costs and silver credit; and*
2. *Cash Flow from Operations includes changes in working capital.*

A sensitivity analysis was performed to test Project value drivers on the Project's NPV using a 5% discount rate. The results of this analysis are demonstrated in Table 1-11. The Project proved to be most sensitive to changes in metal price followed by foreign exchange, operating costs and capital costs. A sensitivity analysis of the after-tax results was performed using various gold prices. The results of this analysis are demonstrated in Table 1-12.

Table 1-11 – Sensitivity Analysis – After-Tax NPV$^{5\%}$

	After-Tax NPV$^{5\%}$, Millions of CAD				
	-10%	-5%	100%	5%	10%
Gold Price	$48.7	$111.0	$164.5	$222.4	$274.8
CAD/USD F/X	$275.6	$220.2	$164.5	$118.8	$69.4
Capital Cost	$205.9	$185.6	$164.5	$143.3	$130.0
Operating Cost	$225.3	$195.6	$164.5	$132.9	$108.6

	After-Tax NPV$^{5\%}$, Millions of USD				
	-10%	-5%	100%	5%	10%
Gold Price	$36.5	$83.2	$123.4	$166.8	$206.1
CAD/USD F/X	$186.0	$156.9	$123.4	$93.5	$57.2
Capital Cost	$154.4	$139.2	$123.4	$107.5	$97.5
Operating Cost	$169.0	$146.7	$123.4	$99.7	$81.4

Table 1-12 – Gold Price Sensitivity on NPV and IRR

Gold Price	After-Tax NPV (CAD $M)	After-Tax NPV (US $M)	After-Tax IRR (%)
$1,100	$23.3	$17.5	6.1%
$1,200	$123.5	$92.7	10.6%
$1,250	$164.5	$123.4	12.5%
$1,300	$211.1	$158.3	14.5%
$1,400	$297.0	$222.7	18.0%
$1,500	$386.4	$289.8	21.5%

1.20 Interpretations and Conclusions

This report confirms the technical feasibility and the economic viability of the Project. The LLGP is comprised of two properties, MacLellan and Gordon, and is based on conventional open pit mining with a centralized processing plant facility and tailings management facility. The processing plant, located at MacLellan, has an expected nominal processing throughput of 7,000 t/d with an estimated 10.4-year production life.

The CAPEX was developed according to AACE Class 3 guidelines for a Feasibility Study Estimate with a -10% to +15% accuracy. The initial capital cost of the Project, including processing, initial mine equipment lease payments and pre-production activities, infrastructure, spares and other direct and indirect costs is $450.6M (US $338.0M). Sustaining capital cost is $196.9M (US $147.6M), and includes MacLellan and Gordon primary and support equipment lease payments, spare parts, additional TMF lifts, water management and closure costs. Total Project capital cost including initial and sustaining capital is $647.5 M (US $485.6M).

Overall operating cost for mining ($26.94/t milled, $713.3M LOM), processing ($14.46/t milled, $382.8M LOM), G&A/Accommodation ($7.10/t milled, $188.1 M LOM) and external refining ($0.18/t milled, $4.8M LOM) is $48.68/t milled or $1,289.0M LOM. After adjusting for royalties and silver credits, the expected operating cost is $48.08/t milled or $1,273.2M LOM (US $954.9M).

1.21 Recommendations

1.21.1 Geology and Mineral Resource Estimate

Additional drilling is required to infill some gaps in the underground Mineral Resource drilling data with the potential to increase the underground Mineral Resources, and test the depth extensions of the gold mineralization.

It is also recommended to conduct additional geological and mineralogical studies aimed at improving the understanding of the geological and structural setting of the deposit and revised 3D modeling to assist in future mine planning and refinement of the ARD model.

1.21.2 Open Pit Mining

It is recommended that a refinement of the cut-off grade model be undertaken, based on the results of the Feasibility Study.

It is recommended that the MacLellan pit limit be re-examined taking into account the Feasibility Study estimates of the mining costs, as well as the constraints, if any, of the tailings dam and waste storage capacities. The pit limit design of MacLellan was carried out based on the design parameters at the outset of the Feasibility Study, which proved conservative. The Feasibility Study design parameters and estimated mining costs may support a marginally larger pit limit and Mineral Reserve for MacLellan.

1.21.3 Process Metallurgy and Plant

The following recommendations to the process plant shall be considered for the next phase of engineering:

- Review Mill throughput against current mine plan and determine if optimal throughput was selected;
- Assess if pebble crusher is required to minimize risk to throughput;
- Review the use of CIL, instead of CIP, for the purpose of capital savings in both equipment and footprint; and
- Review if a carousel type CIP circuit provides capital and operating cost benefits over the standard CIP design included in the Feasibility Study.

1.21.4 Tailings Management

The following tasks will be considered for the next stage of engineering:

- Further development of the mine rock management plan with consideration of materials to be used for construction of the start-up dams that are non-acid generating and non-metal leaching;
- Further evaluate the potential borrow sources for dam construction in terms of quantity available and suitability;
- Further evaluate the requirement for bedrock grouting beneath the dams;
- Conduct additional geotechnical investigations along the alignment of the perimeter dams to further characterize the geotechnical subsurface conditions, thermal regime, and hydrogeology. The results of the investigation will be used to refine the dam design and seepage cut-off requirements;
- Complete thermal modelling of the TMF to evaluate the potential for water to freeze within the facility then buried by additional tailings and not thaw the following summer. Water tied up as ice will reduce the storage;
- Revisit the site wide water management plan to eliminate pumping of additional water to the TMF from other sources (i.e. open pit dewatering, plant site runoff, waste rock and overburden stockpile runoff). Additional water management ponds should be considered around the plant site to reduce the pumping requirements and the volume of water to be stored in the TMF to reduce its overall risk profile;
- Finalize the tailings dam design; and
- Complete a comprehensive closure plan.

1.21.5 Environment and Community Engagement

The environmental recommendations are as follows:

- Environmental assessment, other environmental permitting and planning, mitigation, management and follow-up monitoring plans, and associated consultation/engagement activities should proceed with the objective of gaining environmental approval for the LLGP in line with the overall Project schedule;
- Geochemical characterization should continue as the processing and mining plans are detailed, with modification to the mine rock and tailings management plans as appropriate; and
- Estimated closure and reclamation costs should be reassessed in the next phase of development as more detailed engineering designs become available.

2 INTRODUCTION

This NI 43-101 Technical Report is a summation of the Feasibility Study prepared to provide Alamos with sufficient information to determine the economic feasibility of developing the LLGP, located close to the Town of Lynn Lake, Manitoba.

In November 2014, AuRico entered into a joint venture agreement with Carlisle, acquiring a 25% interest in the LLGP for an initial cash contribution of $5.0M, with the option to earn up to a 60% interest by funding $20.0M on the Project over a three year period and delivering a Feasibility Study. In January 2016, Alamos consolidated full ownership of LLGP through its acquisition of Carlisle. Throughout this report Carlisle, AuRico, and Alamos may be used interchangeably from the period of 2015 onward, and can be collectively referred to as “the Company”.

Ausenco and a group of engineering and environmental consultants were engaged by Alamos later in 2016 to complete the Feasibility Study and the NI 43-101 Technical Report for the LLGP.

The LLGP will be built as a conventional open pit mine with at site processing plant facilities and tailings management facilities. The LLGP is composed of two properties (MacLellan and Gordon). The processing plant, located at MacLellan, has an expected nominal processing throughput of 7,000 t/d with an estimated 10.4-year production life.

The NI 43-101 Technical Report and Feasibility Study responsibilities of the engineering consultants are as follows:

- Ausenco was commissioned by Alamos to manage and coordinate the work related to the Feasibility Study and the NI 43-101 Technical Report for the LLGP. Ausenco was also engaged to develop the feasibility level design of the process plant and infrastructure for the Project;
- Q’Pit was commissioned to develop the design of the open pit, the phase development and long-term mine plan, the selection of the equipment type and size, related estimates with respect to the open pit mine operating personnel and the estimates of the mine capital and operating costs for the life of the Project and the pre-production related quantities for the mine development roads. Q’Pit also produced the Mineral Reserves;
- Stantec was commissioned to support environmental planning, assessment, licensing, and permitting;
- Golder was commissioned to complete the feasibility level design of the TMF at the MacLellan site, slope design recommendations for the MacLellan and Gordon open pits and stockpiles, as well we as the layout and sizing of the water management structures for both sites. Golder was also responsible for undertaking various geotechnical and hydrogeological investigations in support of the feasibility level design and to aid in the sourcing of borrow materials for construction;
- RePlan was commissioned to complete the accommodation study (Environmental Resource Management (ERM), 2017);
- All the consultants were engaged to provide input and contributed to the development of the OPEX and CAPEX; and
- Alamos reviewed and developed those elements of the Project relating to the geological setting and mineralization, Mineral Resources, market studies and contracts and economic analysis.

2.1 Terms of Reference

All units of measurement are in metric, unless otherwise stated.

The monetary units are in Canadian dollars, unless otherwise stated.

In some instances, the Gordon Property may historically be referred to as the Farley Property. The property name was changed from Farley to Gordon in 2016.

2.2 List of Qualified Persons

Table 2-1 – Section Qualified Persons

Section	Description	Qualified Person	Company
1	Summary	All in part	
2	Introduction	Paul Staples	Ausenco
3	Reliance on Other Experts	Paul Staples	Ausenco
4	Property Description and Location	Paolo Toscano	Alamos
5	Accessibility, Climate, Local Resources, Infrastructure and Physiography	Paolo Toscano	Alamos
6	History	Jeffrey Volk	Alamos
7	Geological Setting and Mineralization	Jeffrey Volk	Alamos
8	Deposit Types	Jeffrey Volk	Alamos
9	Exploration	Jeffrey Volk	Alamos
10	Drilling	Jeffrey Volk	Alamos
11	Sample Preparation, Analyses and Security	Jeffrey Volk	Alamos
12	Data Verification	Jeffrey Volk	Alamos
13	Mineral Processing and Metallurgical Testing	Eddie McLean	Ausenco
14	Mineral Resource Estimates	Jeffrey Volk	Alamos
15	Mineral Reserve Estimates	Efthymios Koniaris	Q'Pit
16	Mining Methods	Efthymios Koniaris 16.2 – Luiz Castro	Q'Pit Golder
17	Recovery Methods	Eddie McLean	Ausenco
18	Project Infrastructure	Paul Staples Parts of 18.3 and Parts of 18.14 – Karen Mathers Parts of 18.3 – Karen Besemann Parts of 18.3 – Adwoa Cobbina Parts of 18.3 and Parts of 18.14 – Rui Couto	Ausenco Stantec Golder Golder Golder
19	Market Studies and Contracts	Paolo Toscano	Alamos
20	Environmental Studies, Permitting and Social or Community Impact	Karen Mathers	Stantec
21	Capital and Operating Costs	Paul Staples Parts of 21.3 and 21.6 – Efthymios Koniaris Parts of 21.3 – Rui Couto	Ausenco Q'Pit Golder
22	Economic Analysis	Paolo Toscano	Alamos
23	Adjacent Properties	Paolo Toscano	Alamos
24	Other Relevant Data and Information	Paul Staples	Ausenco
25	Interpretations and Conclusions	All in part	
26	Recommendations	All in part	
27	References	All in part	

2.3 Site Visits

The following Qualified Persons (QP's) visited the LLGP site as indicated below:

- Paul Staples, P. Eng., Ausenco, visited the site from September 26, 2016 to September 27, 2016;
- Eddie McLean, B.Sc. (Met), FAusIMM, Ausenco, visited the site from September 26, 2016 to September 27, 2016;
- Jeffrey Volk, CPG, FAusIMM, Alamos, has visited the site on numerous occasions with last site visit occurring from May 14, 2016 to May 17, 2016;
- Paolo Toscano, P. Eng., Alamos, has visited the site on numerous occasions with last site visit occurring October 11, 2017 to October 12, 2017;
- Luiz Castro, PhD., P. Eng., Golder, visited the site from June 25, 2015 to June 27, 2015;
- Rui Couto, MASc, P. Eng., Golder, visited the site from September 26, 2016 to September 27, 2016.
- Efthymios Koniaris, PhD., P. Eng., Q'Pit, visited the site from May 31, 2017 to June 01, 2017; and
- Karen Mathers, P. Geo., FGC, Stantec, has visited the site on numerous occasions with last site visit occurring October 11, 2017 to October 12, 2017.

3 RELIANCE OF OTHER EXPERTS

Ausenco has relied on the input from Alamos and qualified independent consulting companies in preparing this report. This report was prepared using the reports and documents noted in Section 27 “References”.

Ausenco’s responsibility was to assure that this NI 43-101 Technical Report met the stipulated guidelines and standards considering that sections of this Report were contributed by Ausenco, Stantec, Golder, Q’Pit, Alamos, or other Alamos consultants.

The information, conclusions, opinions and estimates contained herein are also based on data, reports, and other information supplied by Alamos and the other party sources.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Introduction

The LLGP is situated in Northern Manitoba, approximately 820 km northwest of Manitoba's capital, Winnipeg, and involves the redevelopment of two historical gold mines, MacLellan and Gordon (collectively, the "Properties"). Respectively, the Properties are located approximately 7 km northeast and 37 km east of the Town of Lynn Lake. The distance between the Gordon and MacLellan sites is approximately 30 km (Figure 4-1).

The overall Project lands are presently comprised of a portfolio of 247 dispositions including 238 mining claims, eight Crown mineral leases and one Crown surface lease, all of which the recorded interest are held 100% by Carlisle, a wholly-owned subsidiary of Alamos. All mining claims and leases are active and in good standing as of the effective date of this report.

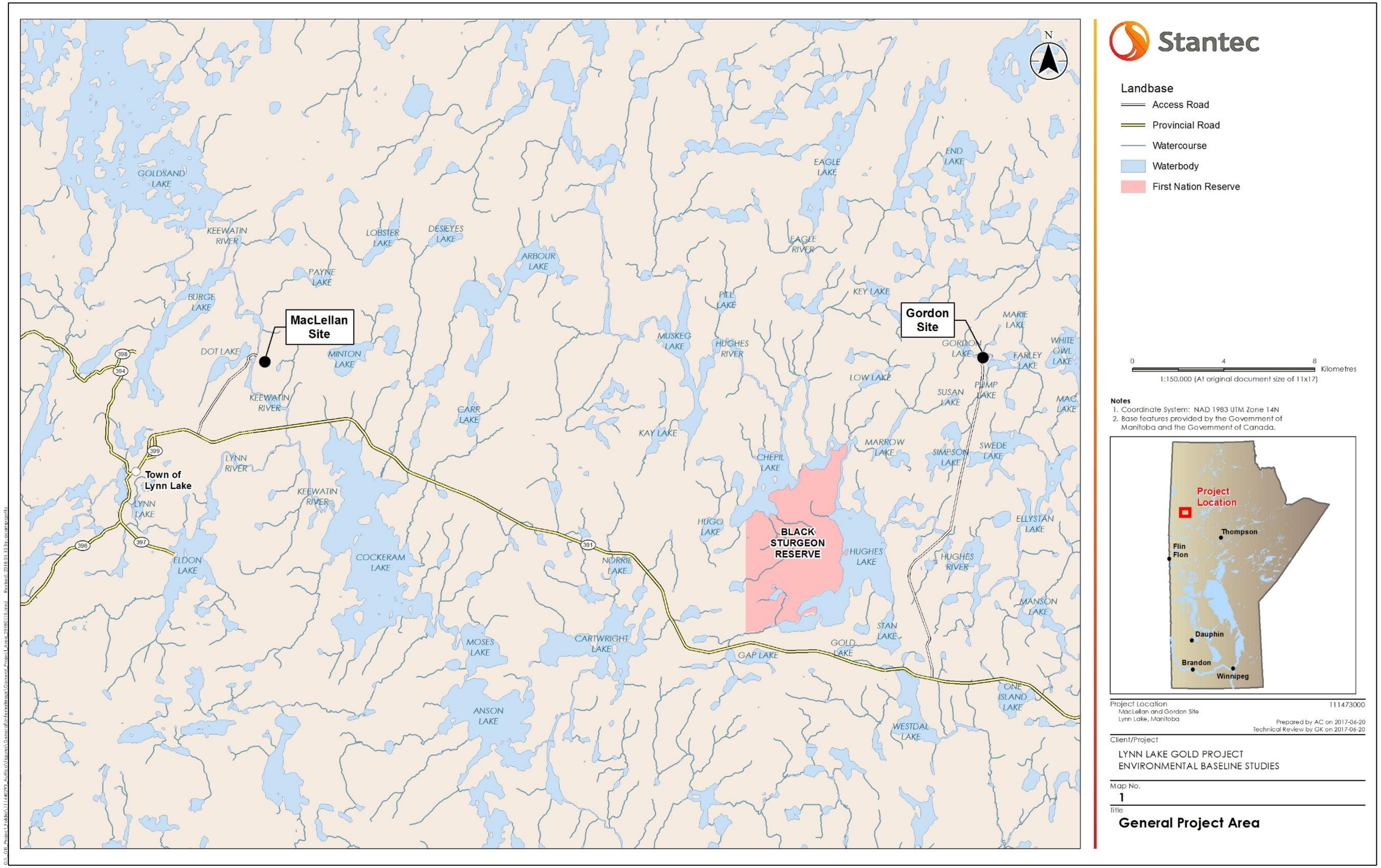


Figure 4-1: General Project Area

Source: Stantec (2017)

4.2 Historical Overview of Property Acquisition

LLGP's property portfolio is comprised of gold and base metal mining claims and leases in the LLGB. Included in the overall land package are the former producing mines of Burnt Timber (which produced from 1993 to 1996), Farley Lake Open Pit Mine (which produced from 1996 to 1999) and the underground MacLellan mine (which produced from 1986 to 1989). The present-day disposition of the property package has been achieved by assembling lands over a period of time through grass-root staking activities, exercises of earn-ins under options and purchase acquisitions.

The Company also holds beneficial ownership interests in numerous mining claims located within LLGP's Nail-Franklin area (78.03%) and the Shoe-Lace area (50.31%). On December 31, 2005, Glencairn Gold Corporation (Glencairn), formerly known as Central Sun Mining Inc., entered into a Property Acquisition Agreement that sold certain mining leases and mining claims to AMPX, a private company. The agreement included properties referred to as the Lynn Lake Properties, being a diverse group of gold exploration claims and leases in the LLGB of Northern Manitoba, covering approximately 20,000 ha, as well as the former producing underground MacLellan and Nisku mines.

AMPX completed an acquisition agreement with Glencairn in December 2006 that required AMPX to complete a minimum financing of $2.0M and complete a 'Going Public Transaction' on or before December 31, 2006. In this regard, AMPX changed its name to Carlisle and obtained a stock exchange listing by having its final prospectus approved and accepted on December 28, 2006. As part of the process, Carlisle raised gross proceeds of $4.3M by way of its initial public offering. Carlisle also met the requirement to incur expenditures on the properties of a minimum of $1.0M on or before February 01, 2007 and thus earned its complete interest in the properties under terms of the agreement. Under the terms of agreement, a Net Smelter Return (NSR) Royalty Agreement dated December 31, 2005 was entered into, granting a 2% net smelter royalty on gold and silver products and for all other products produced from any of the Lynn Properties (the "NSR"). As the successor in interest to Glencairn, B2Gold Inc. (B2Gold) became the holder of this NSR which covered the MacLellan Property (save and except one mineral and surface lease parcel), a portion of the Gordon Property, and other areas of the LLGP land package. Pursuant to a NSR Royalty Purchase Agreement dated June 07, 2017, B2Gold's NSR interest was bought out by Alamos.

In 2013, following the conversion and re-staking of a majority of the then existing Crown mineral leases to mining claims for cost-saving purposes, based upon a clearer understanding of the regional geology, several mining claims were added to the property portfolio resulting in an increased geologic strategic land position in Lynn Lake. Examination of the Farley Lake area geology warranted the staking of 35 additional claims, totalling 7,306 ha, contiguous to the existing claim block adjacent to the former Farley Lake mine area. Similarly, an additional twelve contiguous claims were staked to the west of the Linkwood area, along the Johnson Shear, for an additional 2,476 ha.

On November 20, 2014, AuRico completed a $5.6 million private placement with Carlisle to take a 19.9% interest in the company. In conjunction with the private placement, AuRico entered into a joint venture agreement on November 11, 2014 with respect to Carlisle's Lynn Lake Gold Camp, pursuant to which AuRico acquired a 25% interest in the properties for an initial cash contribution of $5.0 million. In April 2015, AuRico and Alamos announced that they had entered into a definitive agreement to combine their respective companies. AuRico and Alamos merged pursuant to Articles of Arrangement dated July 02, 2015 with the resulting amalgamated company continuing under the name Alamos Gold Inc. On January 07, 2016, Alamos completed the acquisition of Carlisle and its 100% ownership of the LLGP.

4.3 Access Rights and Permits

As the 100% recorded holder of the Crown lands comprising the LLGP, the Company is granted the exclusive right to prospect, explore and develop the Crown minerals underlying the mining claims lands. The Company has secured further access and rights to the Project lands by having issued and maintaining Crown mineral leases and a Crown surface lease. As a Crown mineral lease lessee, the Company has also been granted exclusive rights to the Crown minerals and mineral access rights to work, mine and erect buildings for the efficient mining and production of minerals. Under its' Crown surface lease, the Company's right has been defined to use the surface of one of its Crown mineral lease areas for the efficient and economical performance of its' mining operations.

All mining claims have been located by staking out on the ground as per *The Mines and Minerals Act* (Manitoba). Once recorded, a mining claim is in good standing for two years plus sixty days. Assessment work, in the amount of $12.50/ha for each of the 2nd to 10th years, and $25.00/ha for the 11th year and for each year thereafter, is required to be completed and/or filed annually on each mining claims. A mining claim will be renewed for an indefinite period so long as assessment filings/reportings are completed annually.

All Crown mineral leases held by the Company have been issued for a term of 21 years and are currently considered non-producing. Annual rental for mineral leases not in production is based upon $12.00/ha or fraction thereof, with the annual rent paid being not less than $200.00/y. On the 5th, 10th and 21st anniversaries of the mineral lease, a detailed statement of exploration work performed on the Crown mineral lease parcel must be filed.

The Company's Crown surface lease, which is located within the MacLellan Property, is renewable on an annual basis, for a term that cannot exceed the lease term of the mineral lease to which it relates. Annual rental for the surface lease is $5.00/ha or portion thereof, with a minimum annual rent to be paid of $100.00.

Although authority to enter the mining claim and mineral lease lands is pursuant to *The Mines and Minerals Act* (Manitoba), work permits are required to conduct field work and other activities on Crown lands. Work permits are issued by Manitoba Sustainable Development (MSD). There are currently five work permits issued for the LLGP, four in support of mineral exploration and associated activities, and one in support of activities related to environmental monitoring/surveying to support the environmental assessment, the particulars of which are as follows:

- Work Permit WP 2017-01-15-004 issued under the authority of Section 7(1)(c) of *The Crown Lands Act* (Manitoba), authorizes the carrying out of an operation on Crown Lands located in the LLGP's MacLellan and Gordon (formerly Farley Lake) mine sites and areas surrounding the mine sites for the purposes of environmental monitoring/surveying to support Environmental Assessment. The permit notes that work conducted in accordance with this permit excludes the areas of Crown surface lease SL-163 and the Black Sturgeon Reserve;
- Work Permit WP 2017-01-15-005 authorizes the carrying out of an operation on Crown Lands located in LLGP's Arbor/Muskeg Lake areas, on those mining claims described in the work permit, for the purposes of mineral exploration and related activities;
- Work Permit WP 2017-01-15-006 authorizes the carrying out of an operation on Crown Lands located in LLGP's Gordon Lake area, upon the mining claims and mineral leases described in the issued work permit, for the purposes of mineral exploration and related activities;

- Work Permit WP 2017-01-15-007 authorizes the carrying out of an operation on Crown Lands located in LLGP's MacLellan mine site area, upon the mining claims and leases detailed within the work permit, for the purposes of mineral exploration and related activities; and
- Work Permit WP 2017-01-15-008 authorizes the carrying out of an operation on Crown Lands located in LLGP's McVeigh Lake/Burnt Timber areas, upon the mining claims described within this work permit, for the purposes of mineral exploration and related activities.

All work permits must be renewed annually at the end of April. As of the effective date of this report, all of the above work permits were in good standing.

To access mining claim and lease areas, Crown Land permits for road and/or trail access must be obtained through MSD' Crown Land Property Agency (CLPA). There are currently two Crown Land permits issued in support of the LLGP in respect of 'all weather roads', as more particularly described below. Each permit is effective for a period of one year from January 01 to December 31, and is renewable annually on payment of the prescribed annual permit fee plus an additional $1.00/km of road.

- Crown Land Permit No. GP 70394 authorizes the use of Crown Lands for an all-weather access road to the Burnt Timber mine site and adjoining claim group. The purpose of the Burnt Timber Access Road is to provide access to the mineral dispositions held by the Company. A condition of this issued permit is that the access road remain publicly accessible as a roadway utilized by resource users, and it is to be maintained as a safe, cleared right-of-way for public travel through the land;
- Crown Land Permit No. GP 70395 authorizes the use of Crown Lands for an all-weather road from PR 391 to the MacLellan mine site for a distance of 4.6 km with a 10 m right-of-way. The purpose of the MacLellan Mine Road is to access the MacLellan mine site and surrounding lands, from PR 391. A condition of this issued permit is that it provides access to Grey Owl Outfitters Inc.'s site located north of this access road's northern bend; and should Grey Owl Outfitters Inc. change hands or name, equal access rights must be granted to the new outfitter; and
- Crown Land Permit No. GP 70396 authorizes the use of Crown Lands for an access road from PR 391 to Farley Lake (now Gordon) mine site for a distance of 15.0 km with a 10 m right-of-way. The purpose of the Gordon Lake Access Road is to access the Farley Lake (now Gordon) Open Pit Mine while permitting and Feasibility Study work goes on; and if the Project is feasible, the road will be used as a haul road to transport from this location to the proposed mill at the MacLellan site via PR 391.

As of the effective date of this report, these Crown Land permits were in good standing.

4.4 MacLellan

4.4.1 Location and Property Description

Located 7 km northeast of the Town of Lynn Lake, the MacLellan Property's geographic coordinate for its' mine site (specifically the centroid for the proposed open pit) is (UTM Zone 14N): 380900 easting and 6307500 northing.

The MacLellan Property, which surrounds the historical MacLellan mine site, is situated on Crown lands that are currently comprised of nineteen mining claims, four Crown mineral leases and one Crown surface lease, covering an area of 3,248 ha. See Table 4-1 below. All mining claims and Crown mineral leases comprising the MacLellan Property are 100% held by the

Company. The current disposition of identified lands for future development of the MacLellan site is four mineral leases, one surface lease, and eleven mining claims.

Table 4-1 – MacLellan Property Mining Lease and Claims

Name	Disposition Number	Disposition Tenure	Area (ha)	Date Granted	Annual Expiry Date	Term Expiry Date
Mine Site	ML68	Lease	289	01/04/1992	01/05/2018	01/04/2038
	ML299	Lease	21	01/04/1992	01/05/2018	01/04/2038
	ML304	Lease	17	01/04/1992	01/05/2018	01/04/2038
	ML305	Lease	27	01/04/1992	01/05/2018	01/04/2038
	SL163	Lease	289	12/05/2015	12/05/2018	11/06/2018
Rainbow 101	P5490E	Claim	192	16/09/1985	15/11/2018	N/A
Rainbow 100	P5489E	Claim	192	16/09/1985	15/11/2018	N/A
Rainbow 102	P5484E	Claim	145	10/10/1985	09/12/2018	N/A
Mac 2	P5478E	Claim	28	16/09/1985	15/11/2018	N/A
Shag	CB10340	Claim	31	21/01/1980	22/03/2018	N/A
Mac 1 FR.	P5477E	Claim	9	16/09/1985	15/11/2018	N/A
MAC10844	MB10844	Claim	209	28/03/2013	27/05/2018	N/A
MAC10845	MB10845	Claim	204	28/03/2013	27/05/2018	N/A
MAC10846	MB10846	Claim	254	28/03/2013	27/05/2018	N/A
MAC10847	MB10847	Claim	239	28/03/2013	27/05/2018	N/A
MAC10848	MB10848	Claim	226	28/03/2013	27/05/2018	N/A
MAC10849	MB10849	Claim	210	28/03/2013	27/05/2018	N/A
MAC10851	MB10851	Claim	223	28/03/2013	27/05/2018	N/A
MAC10852	MB10852	Claim	159	28/03/2013	27/05/2018	N/A
MAC10853	MB10853	Claim	148	28/03/2013	27/05/2018	N/A
MAC10854	MB10854	Claim	220	28/03/2013	27/05/2018	N/A
MAPLE	W47799	Claim	160	22/06/1983	21/08/2018	N/A
TOAST	W47808	Claim	25	22/06/1983	21/08/2018	N/A
FRENCH	W47809	Claim	20	22/06/1983	21/08/2018	N/A
Total		4 Leases 19 Claims	3,248			

The MacLellan site was formerly operated as an underground gold and silver mine between 1986 and 1989, and was closed as a result of high operating costs and falling gold prices. This site currently consists of a 4.6 km gravel access road, power transmission line (abandoned pole line), infrastructure from the former underground mine, maintenance and other storage buildings, and former mine water settling ponds. Some of the existing infrastructure will be demolished during the Project construction phase; however, some demolition activities may be phased, depending on the location of the former infrastructure and its overlap with the footprint for the new mine infrastructure.

4.4.2 Environmental Liabilities

Alamos has not assumed any environmental liability with regard to any of the off-site milling and processing operations conducted by prior owners of the MacLellan site. Tailings deposited elsewhere in Lynn Lake by former operators and other materials around the Lynn Mill from prior base metal operations, which have been rehabilitated but remain in place are the responsibility of others.

The environmental exposure assumed by Alamos is limited to the prior mining operations conducted on the acquired property. The mine was closed in 1989 and has been in a 'care and maintenance' phase since, with little reclamation completed. The Manitoba government is aware of the current state of the property and has identified no environmental concerns at this time.

4.4.3 Impact of Previous Mining Activities

The environmental baseline studies completed in support of the LLGP and discussed herein in Section 20 have considered both the natural environmental setting and the historical activities. The historical activities in particular provide field-scale data on potential environmental effects that could result from future mining activities, even though the mining operations proposed are different than those previously undertaken.

4.5 Gordon

4.5.1 Location and Property Description

Located 37 km east of the Town of Lynn Lake the geographic coordinate for the Gordon mine site (specifically the centroid for the proposed open pit) is (UTM Zone 14N): 412400 easting and 6307800 northing.

The Gordon Property, which encompasses the Gordon mine site’s historic footprint, is situated on Crown lands, and consists of seventy-one mining claims and four Crown mineral leases totalling 12,915 ha. All mining claims and Crown mineral leases comprising the Gordon Property are 100% held by the Company. Presently, two mining claims and four Crown mineral leases comprise the current disposition of the lands identified for the future development of the Gordon site.

Table 4-2 – Gordon Property Mining Lease and Claims

Name	Disposition Number	Disposition Tenure	Area (ha)	Date Granted	Annual Expiry Date	Term Expiry Date
KEL11368	MB11368	Claim	256	04/29/2013	06/28/2018	N/A
KEL11384	MB11384	Claim	220	04/29/2013	06/28/2018	N/A
KEL11370	MB11370	Claim	256	04/29/2013	06/28/2018	N/A
KEL11369	MB11369	Claim	256	04/29/2013	06/28/2018	N/A
KEL11375	MB11375	Claim	256	04/29/2013	06/28/2018	N/A
KEL11380	MB11380	Claim	256	04/29/2013	06/28/2018	N/A
KEL11378	MB11378	Claim	240	04/29/2013	06/28/2018	N/A
KEL11383	MB11383	Claim	160	04/29/2013	06/28/2018	N/A
KEL11376	MB11376	Claim	256	04/29/2013	06/28/2018	N/A
KEL11381	MB11381	Claim	256	04/29/2013	06/28/2018	N/A
KEL11372	MB11372	Claim	218	04/29/2013	06/28/2018	N/A
KEL11385	MB11385	Claim	220	04/29/2013	06/28/2018	N/A
KEL11373	MB11373	Claim	200	04/29/2013	06/28/2018	N/A
KEL11366	MB11366	Claim	256	04/29/2013	06/28/2018	N/A
KEL11377	MB11377	Claim	256	04/29/2013	06/28/2018	N/A
KEL11386	MB11386	Claim	110	04/29/2013	06/28/2018	N/A
KEL11365	MB11365	Claim	124	04/29/2013	06/28/2018	N/A
KEL11374	MB11374	Claim	162	04/29/2013	06/28/2018	N/A
KEL11367	MB11367	Claim	256	04/29/2013	06/28/2018	N/A
KEL11379	MB11379	Claim	225	04/29/2013	06/28/2018	N/A
KEL11382	MB11382	Claim	180	04/29/2013	06/28/2018	N/A
KEL11371	MB11371	Claim	256	04/29/2013	06/28/2018	N/A
ROB 10837	MB10837	Claim	130	12/18/2012	02/16/2018	N/A
LESS	CB11416	Claim	65	07/11/1981	09/09/2018	N/A
CART #1	CB10335	Claim	195	02/04/1980	04/05/2018	N/A
	CB6036	Claim	84	07/18/1977	09/16/2018	N/A
	CB6031	Claim	126	07/13/1977	09/11/2018	N/A
GORD 1	P8591E	Claim	8	08/30/1985	10/29/2018	N/A
ROB 10842	MB10842	Claim	145	12/18/2012	02/16/2018	N/A
FAR 7	P8597E	Claim	160	04/25/1985	06/24/2018	N/A
ARBOUR 21	W48175	Claim	16	06/03/1983	08/02/2018	N/A

Name	Disposition Number	Disposition Tenure	Area (ha)	Date Granted	Annual Expiry Date	Term Expiry Date
RAINBOW 2	W45592	Claim	223	02/15/1983	04/16/2018	N/A
ROB 10836	MB10836	Claim	240	12/18/2012	02/16/2018	N/A
FAR 2	P8552E	Claim	75	03/11/1985	05/10/2018	N/A
CAUSES	CB11418	Claim	146	07/11/1981	09/09/2018	N/A
	CB9238	Claim	98	10/16/1978	12/15/2018	N/A
FAR 4	P8594E	Claim	256	04/25/1985	06/24/2018	N/A
ROB 10840	MB10840	Claim	240	12/18/2012	02/16/2018	N/A
FAR 6	P8596E	Claim	256	04/25/1985	06/24/2018	N/A
FAR 1	P8551E	Claim	78	03/11/1985	05/10/2018	N/A
SMOKING	CB11417	Claim	98	07/11/1981	09/09/2018	N/A
ARBOUR 30	W46957	Claim	20	09/26/1983	11/25/2018	N/A
ARBOUR 28	W46952	Claim	254	09/26/1983	11/25/2018	N/A
FAR 5	P8595E	Claim	128	04/25/1985	06/24/2018	N/A
ROB 10843	MB10843	Claim	240	12/18/2012	02/16/2018	N/A
ARBOUR 23	W48177	Claim	96	06/03/1983	08/02/2018	N/A
FIRES	CB11419	Claim	195	07/11/1981	09/09/2018	N/A
FAR 11	P8593E	Claim	128	04/25/1985	06/24/2018	N/A
	CB6035	Claim	195	07/13/1977	09/11/2018	N/A
ROB 10835	MB10835	Claim	256	12/18/2012	02/16/2018	N/A
ROB 10839	MB10839	Claim	125	12/18/2012	02/16/2018	N/A
ARBOUR 26	W48180	Claim	80	07/20/1983	09/18/2018	N/A
FAR 3	P8553E	Claim	35	03/11/1985	05/10/2018	N/A
ROB 10850	MB10850	Claim	16	12/18/2012	02/16/2018	N/A
ROB 10834	MB10834	Claim	256	12/18/2012	02/16/2018	N/A
ROB 10832	MB10832	Claim	256	12/18/2012	02/16/2018	N/A
ROB 10838	MB10838	Claim	106	12/18/2012	02/16/2018	N/A
	CB6037	Claim	146	07/18/1977	09/16/2018	N/A
ARBOUR 29	W46953	Claim	96	09/26/1983	11/25/2018	N/A
FAR 12	P8592E	Claim	192	04/25/1985	06/24/2018	N/A
ROB 10841	MB10841	Claim	165	12/18/2012	02/16/2018	N/A
	CB6034	Claim	65	07/13/1977	09/11/2018	N/A
NICKEL 4	P8656E	Claim	192	05/07/1985	07/06/2018	N/A

Name	Disposition Number	Disposition Tenure	Area (ha)	Date Granted	Annual Expiry Date	Term Expiry Date
CARE	CB11420	Claim	227	07/11/1981	09/09/2018	N/A
ROB 10833	MB10833	Claim	256	12/18/2012	02/16/2018	N/A
FAR 17	P8937E	Claim	120	01/16/1987	03/17/2018	N/A
FAR 16	P8936E	Claim	220	01/16/1987	03/17/2018	N/A
RAINBOW 3	W45593	Claim	160	02/15/1983	04/16/2018	N/A
FAR 10	P8600E	Claim	224	04/25/1985	06/24/2018	N/A
FAR 9	P8599E	Claim	256	04/25/1985	06/24/2018	N/A
NICKEL 1	P8653E	Claim	160	05/07/1985	07/06/2018	N/A
	ML327	Lease	208	03/18/2017	03/09/2018	03/18/2038
	ML325	Lease	93	03/18/2017	03/09/2018	03/18/2038
	ML326	Lease	122	03/18/2017	03/09/2018	03/18/2038
	ML324	Lease	113	03/18/2017	03/09/2018	03/18/2038
Total		71 Claims 4 Leases	12,915			

The Gordon site, historically referred to as the Farley Lake site, was operated as a two-pit open pit gold mine between 1996 and 1999. After closure, this site underwent a reclamation process and currently consists of a 15 km gravel access road, a bridge across the Hughes River, two mine rock storage areas and two overburden storage areas that have been capped, and two water-filled open pits. All buildings and infrastructure have been removed.

4.5.2 Environmental Liabilities

The environmental exposure assumed by Alamos is limited to the prior mining operations conducted on the acquired property. After closure, the site underwent a reclamation process. The Gordon site is currently in compliance with closure plans approved by the Manitoba government.

4.5.3 Impact of Previous Mining Activities

After closure of the historical mine, the Gordon site underwent a reclamation process. The environmental baseline studies completed in support of the LLGP and discussed herein in Section 20 have considered both the natural environmental setting and the historical activities. The historical activities in particular provide field-scale data on potential environmental effects that could result from future mining activities, even though the mining operations proposed are different than those previously undertaken.

4.6 Property Encumbrance

As disclosed by the Province of Manitoba's Mines Branch official Integrated Mining and Quarrying System (IMaQs), the Company's recorded interest in its held mining claims is subject to the following encumbrances:

- Fifty-six mining claims are subject to any rights which have or may be granted to Manitoba Hydro under *The Water Power Act* and *The Water Rights Act*. These statutory rights of Manitoba Hydro, as a Provincial Crown corporation, are quite extensive, and may include flooding and other rights;
- Two mining claims are subject to the right of way of the Canadian National Railway (CN); and
- One mining claim is subject to the right of way of PR 396, as a part of the provincial highway system;

The Company's recorded interest in the LLGP lands may also be subject to those royalty and other agreements, as were publicly disclosed as 'Permitted Encumbrances' in both Schedule E of the Joint Venture Agreement between Carlisle and AuRico, effective as of November 10, 2014 (posted to www.sedar.com on November 19, 2014 and retrievable therefrom) and in Schedule C to the Arrangement Agreement between Alamos and Carlisle, dated as of October 15, 2015 (posted to www.sedar.com on October 23, 2015 and retrievable therefrom).

Despite efforts undertaken through investigations and searches, and disclosure received, it is not possible to exclude the possibility that other undisclosed or unrecorded agreements or instruments may exist.

4.7 Royalty Obligations

The MacLellan Property is not subject to any royalties. The Gordon Property is subject to royalties on a portion of the production in the first two years coming from the Gordon open pit. These royalties stem from agreements entered into by Granduc Mining Corporation in favour of Manitoba Mineral Resources Ltd. (MMR) and Mingold Resources Inc. (Mingold) in connection with their respective former interest percentages. Under the provisions of the January 06, 1995 royalty agreement (the MMR Agreement), a Gold Royalty and a Base Metal Royalty (as each defined) were agreed to be paid to MMR. An advance royalty payment of $1.0M (the Advance Royalty) was also paid to MMR in advance of the MMR Agreement date. The MMR Agreement provided the Gold Royalty payable be based upon:

- For the first 100,000 oz of gold recovered or deemed recovered from the Ore (as defined), 3% of the Gold Returns (as defined) when the average Gold Price (as defined) for the month of production is less than US $400/oz, multiplied by the Consumer Price Index (CPI) Annual Adjustment; 4% of the Gold Returns when the average Gold Price for the month of production is US $400/oz or greater but less than US $500/oz, both multiplied by the CPI Annual Adjustment; and 5% of the Gold Returns when the average Gold Price for the month of production is US $500/oz or greater multiplied by the CPI Annual Adjustment; and

- As to gold recovered or deemed to be recovered from Ore in excess of a cumulative 100,000 oz, 3% of the Gold Returns when the average Gold Price for the month of production is US $425/oz, multiplied by the CPI Annual Adjustment; 4% of the Gold Returns when the average Gold Price for the month of production is US$425/oz or greater but less than US $625/oz, both multiplied by the CPI Annual Adjustment; and 5% of the Gold Returns when the average Gold Price for the month of production is US$625/oz or greater multiplied by the CPI Annual Adjustment.

All payments made are debited against the Advance Royalty. Payment of the Gold Royalty continues until such time as MMR has received an aggregate $7.0M (based upon December 1993 dollars as calculated under the MMR Agreement). Thereafter, a Gold Royalty of 1% of the Gold Returns is paid. For a period of 180 days following the payment of the aggregate $7.0M there exists the right to purchase the Gold Royalty for a purchase price equal to the product obtained when the amount of $1.0M is multiplied by the CPI Monthly Adjustment for the month in which payment of said purchase price is paid.

Previous production at Gordon through 2000 resulted in the payment of $2.8M on the MMR royalty, leaving $4.2M to be paid. Applying the CPI Annual Adjustment since 1994 brings the outstanding balance to $6.2M. Alamos expects to pay this out in the first year of Gordon production.

After adjusting for CPI, the $1.0M MMR buyout option has escalated to $1.4M. Alamos expects to exercise this option in the first year of Gordon production.

Pursuant to the May 08, 1996 agreement (the Mingold Agreement), Mingold was entitled to receive a royalty equal to 0.6725% of the NSR (as defined) from the sale of all Products (as defined) from the property until such time as an amount equal to the total of all Preproduction Capital Costs (as defined) incurred by Granduc and Golden Band Resources Inc. has been recovered. After that time, Mingold was entitled to receive a royalty equal to 1.12% of the NSR from the sale of all Products from the property. Mingold's entitlement to a portion of the Net Smelter Return ceases upon its' receipt of payments totalling $3.0M adjusted to reflect changes in the CPI from May 01, 1992 to the date of each such payment.

Previous production at Gordon through 2000 resulted in the payment of $0.9M on the Mingold royalty, leaving $2.1M to be paid. Applying the CPI Annual Adjustment since 1992 brings the outstanding balance to $3.2M. Alamos expects to pay this out in the first two years of Gordon production.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

The Town of Lynn Lake is a former mining community and is accessible by provincial roads (PR) 391, 394, 396, 397 and 399, which are maintained by Manitoba Infrastructure Region 5. Paved road PR 391 connects the Town of Lynn Lake and the Black Sturgeon Reserve with the Town of Leaf Rapids (105 km east) and the City of Thompson (315 km southeast). Private all-weather gravel access roads connect both the MacLellan and Gordon sites with PR 391.

There is currently no rail service to the Town of Lynn Lake. Although the Sherridon rail line previously connected Lynn Lake to The Pas, passenger and freight service now only run as far north as Pukatawagan (Keewatin Railway Company 2015).

The Lynn Lake Airport has a 1,700 m paved runway and a turf runway measuring 835 m. The airport was serviced by scheduled flights up until 2013, at which time the Town began leasing it to YYL Airport Inc., a locally-owned company. There is currently no commercial flight travel to Lynn Lake; only chartered service.

5.1.1 Accessibility – MacLellan

Access to the MacLellan site is by PR 391 and a 4.6 km gravel road.

5.1.2 Accessibility – Gordon

Access to the Gordon site, is by PR 391 and a 15 km gravel road.

5.2 Proximity to Population Centre, and Transport

Northern Manitoba is a sparsely-populated region with its population concentrated in a few communities, including, most notably, the City of Thompson and the towns of Lynn Lake and Leaf Rapids. Smaller numbers of people reside in reserves, settlements, and other small localities.

Lynn Lake is located approximately 315 km northwest of Thompson and approximately 820 km northwest of Winnipeg. Transportation links to Lynn Lake are limited predominantly to roads or chartered aircraft.

5.3 Climate

The LLGP is located in a climatic region characterized by short, cool summers and long, cold winters. Long-term climate data (1981-2010) from the Lynn Lake Airport monitoring station (Government of Canada 2016a) indicates that the mean annual air temperature is -3.2°C, ranging from an extreme maximum of 35°C (August 11, 1991) to an extreme minimum of -47°C (December 19, 1989). The minimum and maximum monthly mean temperatures measured at the Lynn Lake Airport monitoring station during 1981-2010 were -29°C for January and 22°C for June. There is an annual average of 98 frost-free days. On average, there are 141 days with precipitation per year with an average annual precipitation of 478 mm (318 mm as rain and 160 mm as snow-water-equivalent).

Table 5-1 provides temperature and precipitation details.

Table 5-1 – Representative Climate Values

Parameter[1]	Unit	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Year
Daily Mean Temperature	°C	-24.3	-20.3	-13.0	-3.1	5.6	12.9	16.2	14.7	7.7	-0.6	-12.5	-21.4	-3.2
Average Rainfall	mm	0.2	0.1	1.4	4.5	26.7	60.6	85.3	68.7	57.4	12.2	0.8	0.1	317.9
Average Snowfall	cm	27.6	23.5	24.6	23.9	10.4	1.3	0.1	0.1	3.5	31.3	36.0	26.0	208.1
Average Precipitation	mm	20.3	16.3	19.8	24.1	37.3	61.8	85.4	68.8	61.0	37.6	26.8	18.8	477.9

Notes:

1. *Environment Canada, Lynn Lake Climate Data 1981-2010.*

5.4 Local Resources

With the closing of the Black Hawk operation in 2001, the Town of Lynn Lake experienced a large decrease in employment opportunities and the Town has since been unsuccessful in developing strong secondary industries. The Town has a current population of approximately 650 residents. Business in Lynn Lake is largely tied to the service industry and include three tourist accommodations, two restaurants, two outfitters, two trucking companies, a grocery and dry goods store, a garage, a gas station/salvage business, a mining company (Alamos), and a property management agency.

There are no businesses in the Black Sturgeon Reserve; a community store had previously operated, but is now closed.

Labour for the Project is assumed to come from mostly Manitoba with sources being locally, Thompson, Flin Flon and Winnipeg. Fly-in–Fly-out (FIFO) workers will be housed in an owner supplied accommodation camp managed by a third party.

5.5 Infrastructure

As previously mentioned, infrastructure at the MacLellan site consists of a 4.6 km gravel access road, a bridge across the Keewatin River, power transmission line (abandoned pole line), and infrastructure from the former underground mine, including a head frame, hoist house, shaft, access ramp, maintenance and other storage buildings, core shack and racks, vent raise, and mine water settling ponds.

All buildings and infrastructure from the historical operations at the Gordon site have been removed. The only residual infrastructure includes the 15 km gravel access road, a bridge across the Hughes River, two capped mine rock storage areas, two capped overburden storage areas, and two water-filled open pits.

5.6 Physiography

The LLGP is located within a remote, rugged region of the Boreal Shield Ecozone. The General Project Area (Figure 5-1) supports peat-covered hummocky glacial deposits underlain by an expanse of Precambrian bedrock. The terrain consists of mostly hilly, till veneered bedrock, with intervening low areas of organic terrain ranging from level to moderately sloping (0-15%). Topography slopes from a high of 450 metres above mean sea level (m AMSL) in the west and northwest to a low of 260 m AMSL in the southeast. Steep rocky ridges protrude 30 m to 60 m above lakes and peat-filled depressions. Surface water features and peat generally occupy the topographic lows. Soils in the region are thin, poorly drained, and acidic, with organic soils typical in bogs and peat plateaus, and discontinuous permafrost is widespread.

The General Project Area overlaps with the Paleoproterozoic LLGB within the Churchill Structural Province of the Canadian Shield. The LLGB is comprised of volcanic rocks of the Wasekwan Group, sedimentary rocks of the Sickle Group, and plutonic intrusions. Overburden geology is characterized as glaciolacustrine sediments overlying either bedrock or a discontinuous regional sand diamicton. Organic deposits were observed as a thin veneer with thicker accumulations observed in low lying areas. Isolated pockets of glaciofluvial sediments are also present. A series of bedrock valleys near the MacLellan site are present where overburden is greater than 28 m thick.

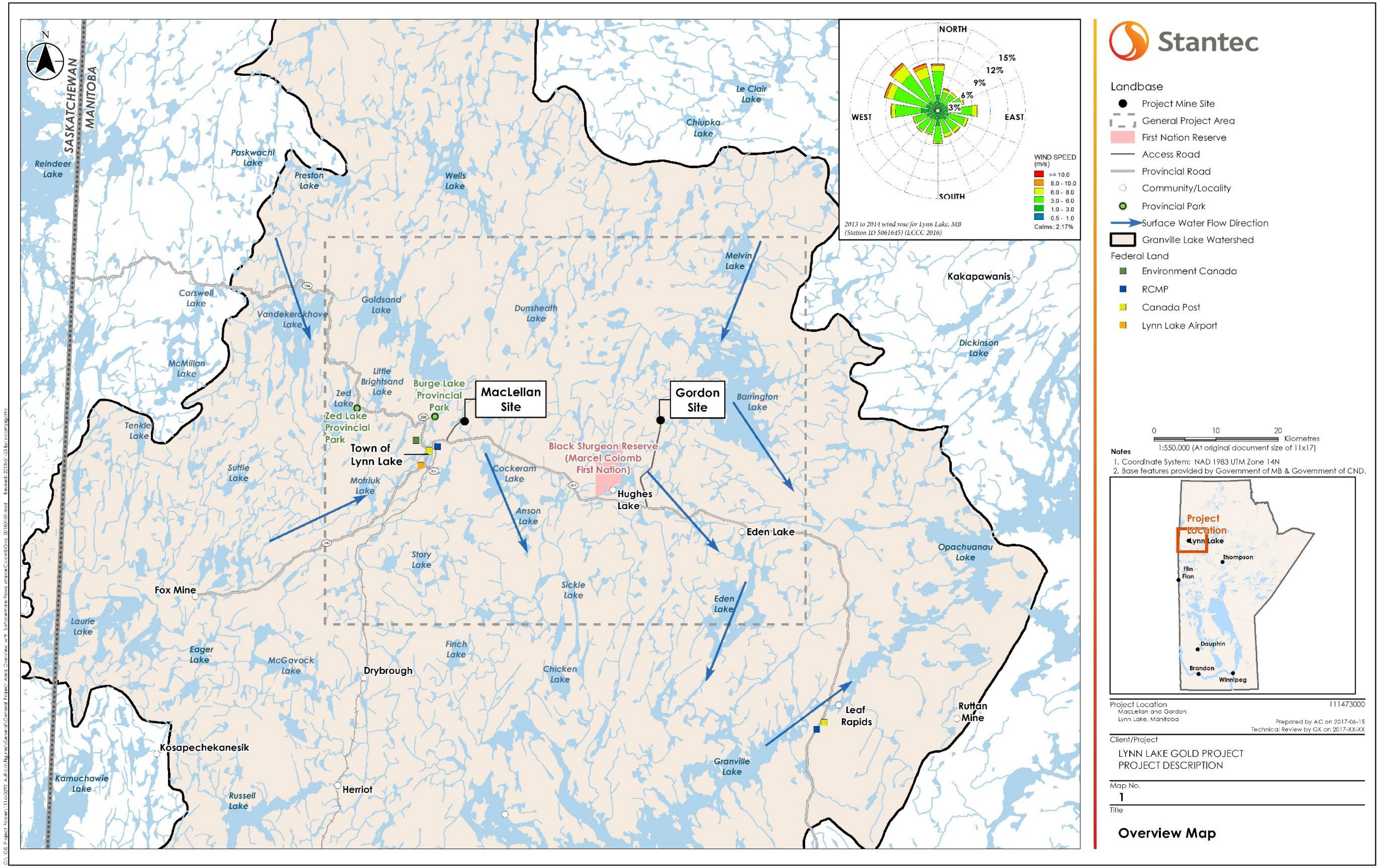


Figure 5-1: General Project Area

Source: Stantec (2017)

6 HISTORY

6.1 MacLellan

6.1.1 Historical Exploration

Exploration on the MacLellan Property commenced in 1946 when Noranda carried out magnetometer and geological surveys on a portion of the property. Numerous companies have explored the MacLellan Property and surrounds between 1946 and 2000. Production at the MacLellan Property under SherrGold and LynnGold between 1986 and 1989 resulted in 969,680 t grading 5.36 g/t Au with an average mill rate of 900 t/d (Chornoby, 1991). Carlisle, as AMPX, acquired the MacLellan Property in 2005 and actively explored the MacLellan Property.

Historical exploration conducted on, and in the vicinity of, the MacLellan Property is summarized in Table 6-1.

Table 6-1 – Historical Exploration on the MacLellan Property

Year	Company	Exploration
1946	Noranda	• First staked in 1946. Magnetometer and geological surveys.
1950	R. Rundle and J.W. Rundle	• Trenching and pits.
1955	Agassiz Mines Limited (Agassiz)	• Drilled 648 m to test magnetic anomalies. Magnetic anomalies equated to magnetite in mafic flows. Diamond Drill Hole (DDH) 3 assayed up to 4.5% Zn, 2.5 g/t Au, and 11 g/t Ag. Electromagnetic (EM) survey.
1956	Aumaque Gold Mines Limited	• Resistivity survey and a 25 drill hole program totalling 3,373 m to test EM anomalies.
1958	Central Manitoba Mines Limited	• 20 drill hole program totalling 2,568 m.
1960	Rayrock Mines Ltd.	• Geophysical survey over a portion of the MacLellan Property.
1965-1967	Agassiz	• Drilled 52 holes totalling 12,446 m.
1969	Agassiz	• A 149 m shaft sunk and 803 m of drifting completed.
1971	Agassiz	• By 1971, 105 holes drilled totalling 21,346 m.
1973	Agassiz	• Shaft dewatered. Drilled 101 underground DDH's.
1974-1975	Bulora Corporation (Bulora) / Agassiz	• Joint venture agreement between Bulora and Agassiz Mines. Drilled 25 underground DDH's, drift and crosscut workings, surface mapping, and thin section work. Underground workings allowed to flood. Four exploration holes drilled.
1976	Bulora	• Agassiz Mines transfers interest to Bulora.
1979	Sherritt	• Acquired an option on the property from a subsidiary company of Agassiz Mines.

Year	Company	Exploration
1980-1985	Sherritt	- Drilled 442 surface and underground DDH's, line cutting, magnetic / horizontal loop electromagnetic (HLEM) surveys, and aerial photomaps. Shaft deepened to 259 m and drifting. Surface and underground mapping. Geochemistry.
1986	SherrGold	- Transfer of property to subsidiary company SherrGold. Drilled 182 underground DDH's. Shaft deepened to 448 m. - Production commences at MacLellan Mine.
1987	SherrGold	- Drilled 32 exploration DDH's and 164 underground DDH's.
1987-1988	SherrGold / LynnGold	- Drilling at MacLellan Mine plus 22 DDH's on exploration targets. Open pit started on crown pillar later abandoned. - Company name change to LynnGold.
1989	LynnGold	- Mine production during 1986-1989 reported at 900,000 tonnes grading 5.4 g/t Au, with average mill rate given as 900 t/d. - Change from cut-and-fill to long-hole open stoping. - Underground drill program tested East Main and Rainbow Zones. - Closure of mine, mine and mill placed on care and maintenance.
1992	Cazador Explorations Limited (Cazador)	- Drilled ten DDH's to test suitability of crown pillar for open pitting. Project abandoned due to risk to mine workings.
1996-2000	Black Hawk / Keystone Gold Inc. (Keystone)	- Review of reserve. Trenching over crown pillar. Geological mapping. Channel sampling. Magnetic survey and prospecting. Drilled 20 DDH's totalling 886 m. Development of crown pillar reserve and pit design. - Mine allowed to flood.
2006	Carlisle	- Carlisle conducted a combined magnetometer, radiometric and very low frequency-electromagnetic (VLF-EM) geophysical (Terraquest 2007 and Karelse et al. 2008) - A compilation and analysis of "Winter-Summer Landsat" and "Digital Topographic" of the Lynn Lake area including the MacLellan Property was initiated.
2007	Carlisle	- Completed a limited surface overburden stripping and subsequent rock channel sampling program over the MacLellan Main zone Collected 395 channel samples from 365 m of channelling to provide continuous coverage from hanging wall to footwall.
2008	Carlisle	- From 2006 to 2008, 62 surface DDH's totalling 21,766 m.

6.2 Gordon

6.2.1 Historical Exploration

Gold occurrences in the Gordon area were first discovered by prospectors along the Johnson Shear in the late 1930's and early 1940's. Mapping in the area discovered gold in frost heaved boulders at the west end of Farley Lake. Note the Gordon Property prior to 2016 was referred to as the Farley Lake Property. A history of exploration on the Gordon Property is summarized in Table 6-2. The deposit was mined from two small open pits during the period 1996-1999 by Black Hawk Mining and reportedly produced 214,800 oz of gold.

Table 6-2 – Historical Exploration on the Gordon Property

Year	Company	Exploration
1945	Sherritt	• Area staked as the Lind claims.
1947	Sherritt	• Drilling carried out on the Lind showing and Pot Hole Lake – Pump Lake area intersecting two gold bearing zones. The claims lapsed in 1976.
1977	HBED	• Area re-staked.
1978-1983	HBED	• Work included line cutting, HLEM surveys and drilling.
1983-1986	HBED and MMR	• Joint venture agreement between two companies. Exploration programs discovered gold mineralization in three zones. Two were thought to be potential open pits.
1986	HBED and MMR	• 15,250 m of drilling completed. Metallurgical testing and feasibility studies were conducted.
1993	MMR and Mingold Resources Ltd.	• Over 20,000 m of drilling completed by 1993. Open pit reserves calculated.
1993	Granduc Mining Corporation (Granduc) and Black Hawk Mining	• The Keystone joint venture acquired the Farley Lake Property as part of its development of the Burnt Timber deposit and reactivation of the Lynn mill.
1995	Granduc and Black Hawk Mining	• Stripping for the open pit began in October. Drilling continued on the Southeast Zone.
1996-1999	Granduc and Black Hawk Mining	• The Farley Lake open pit produced 214,800 oz from 1.7 Mt of ore (Black Hawk Mining 2000). No further work was carried out until the property was acquired by Carlisle.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The LLGB is located within the Churchill Structural Province of the Canadian Shield. The Paleoproterozoic LLGB, a portion of the internal Reindeer zone of the Trans-Hudson orogen, is part of a larger litho-structural unit extending in a northeasterly direction from the La Ronge Greenstone Belt in Saskatchewan (Figure 7-1). To the north, the LLGB is bound by the South Indian Domain, and to the south, by the Kisseynew Domain (Gilbert, Syme, & Zwanzig, 1980); (Beaumont-Smith & Böhm, 2004).

Rocks of the LLGB typically exhibit a middle-amphibolite-facies metamorphic grade. Although, regionally, they have undergone upper-greenschist to upper-amphibolite-facies metamorphism (Gilbert, Syme, & Zwanzig, 1980); (Glendennings, Gagnon, & Polat, 2015).

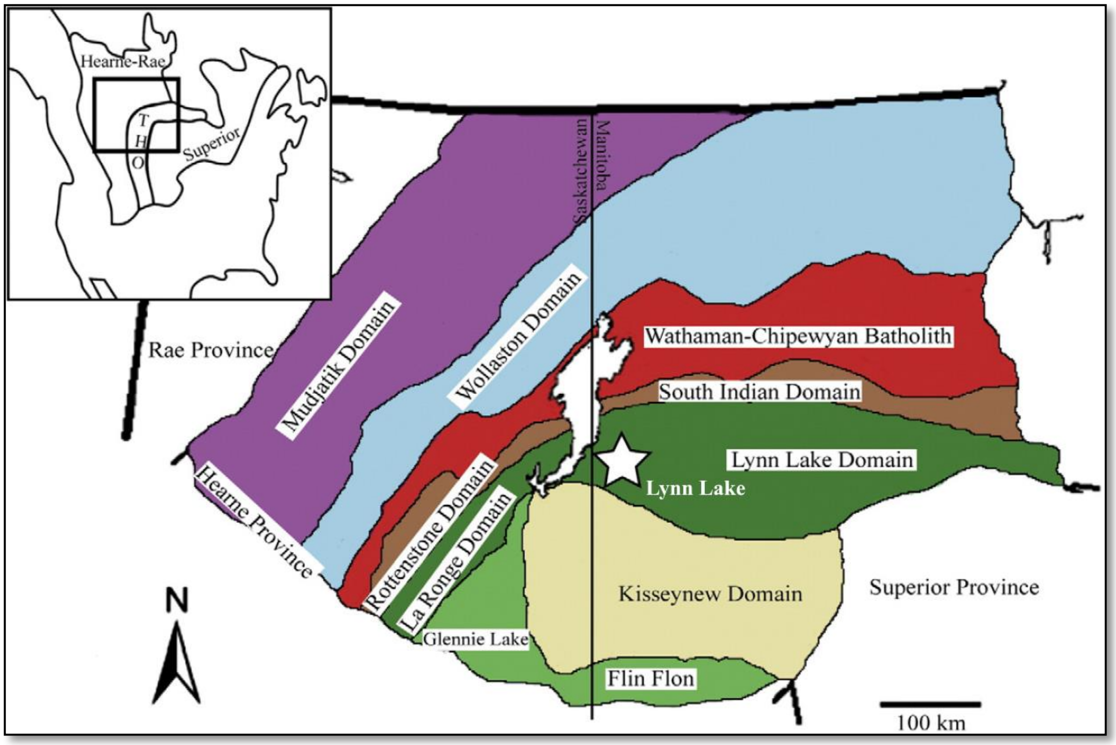


Figure 7-1: Simplified Geologic Map of the Lithotectonic Domains Comprising the Reindeer Zone

Source: Alamos, (2017)

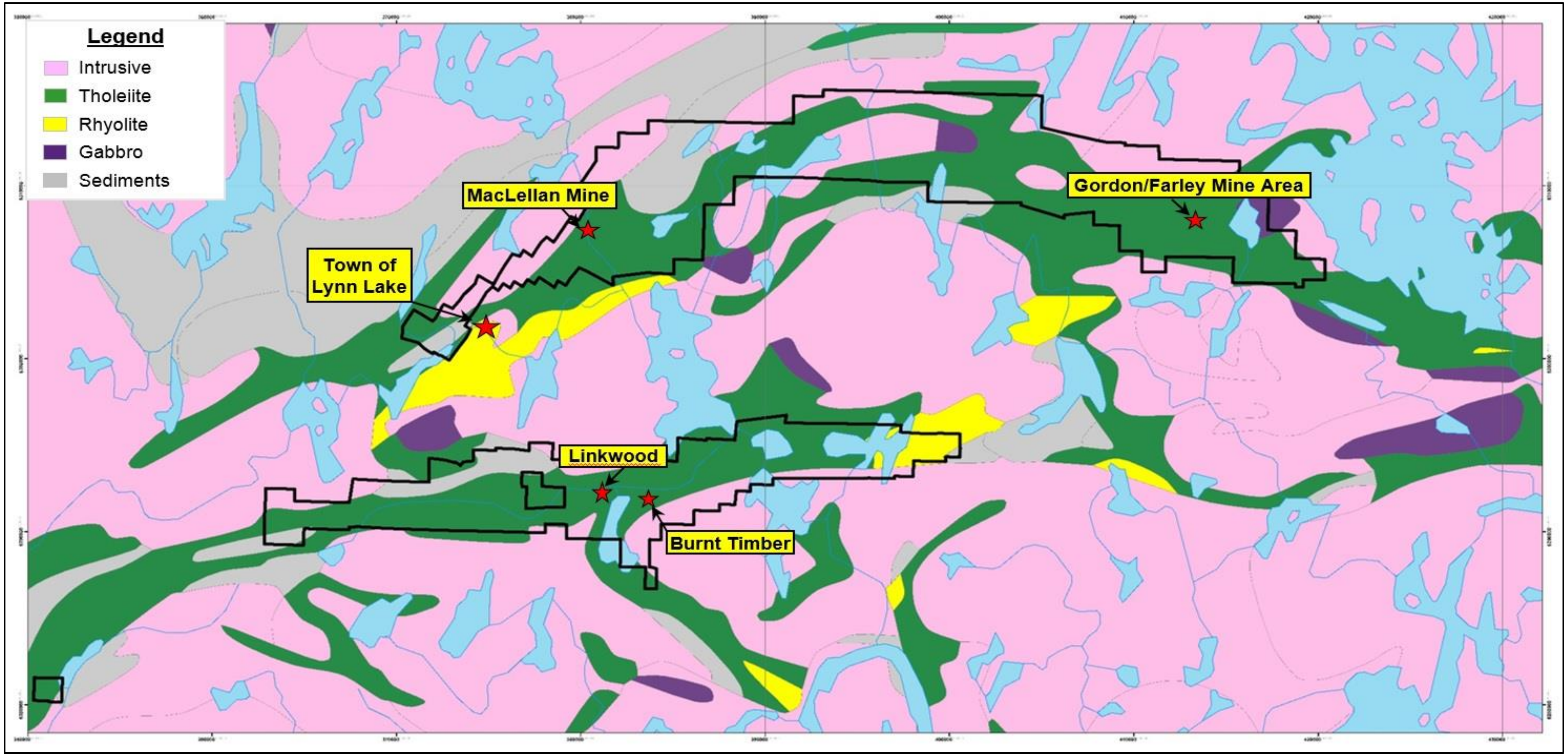


Figure 7-2: Simplified Regional Geology of the LLGB and Alamos Properties

Source: Alamos (2016)

The LLGB is separated into a northern and southern section by granitoid plutons of the Pool Lake intrusive suite (Figure 7-2). The belt consists of contaminated and juvenile volcanic and sedimentary rocks historically assigned to the Wasekwan group, as well as younger Molasse-type sedimentary rocks of the Sickle Group (Bateman, 1945); (Norman, 1933). The temporal relationship to the Sickle Group, dated at ~1,830 Ma (Beaumont-Smith, Machado, & Peck, 2006), separates the Pool Lake intrusives into Pre and Post-Sickle intrusive suites (Gilbert, Syme, & Zwanzig, 1980). An angular unconformity separates the Sickle group from the underlying Wasekwan group and felsic-mafic igneous rocks of the Pool Lake intrusives throughout the middle range and southern belt of the LLGB. In the northern belt, the Ralph Lake conglomerate and Zed Lake greywacke, marine clastic sediments derived from the Wasekwan group, unconformable overly the Wasekwan group (Beaumont-Smith & Böhm, 2004). The relationship of the Ralph Lake sedimentary package and the Sickle group is not yet definite. Both deposits discussed below, MacLellan and Gordon, are situated on the northern belt.

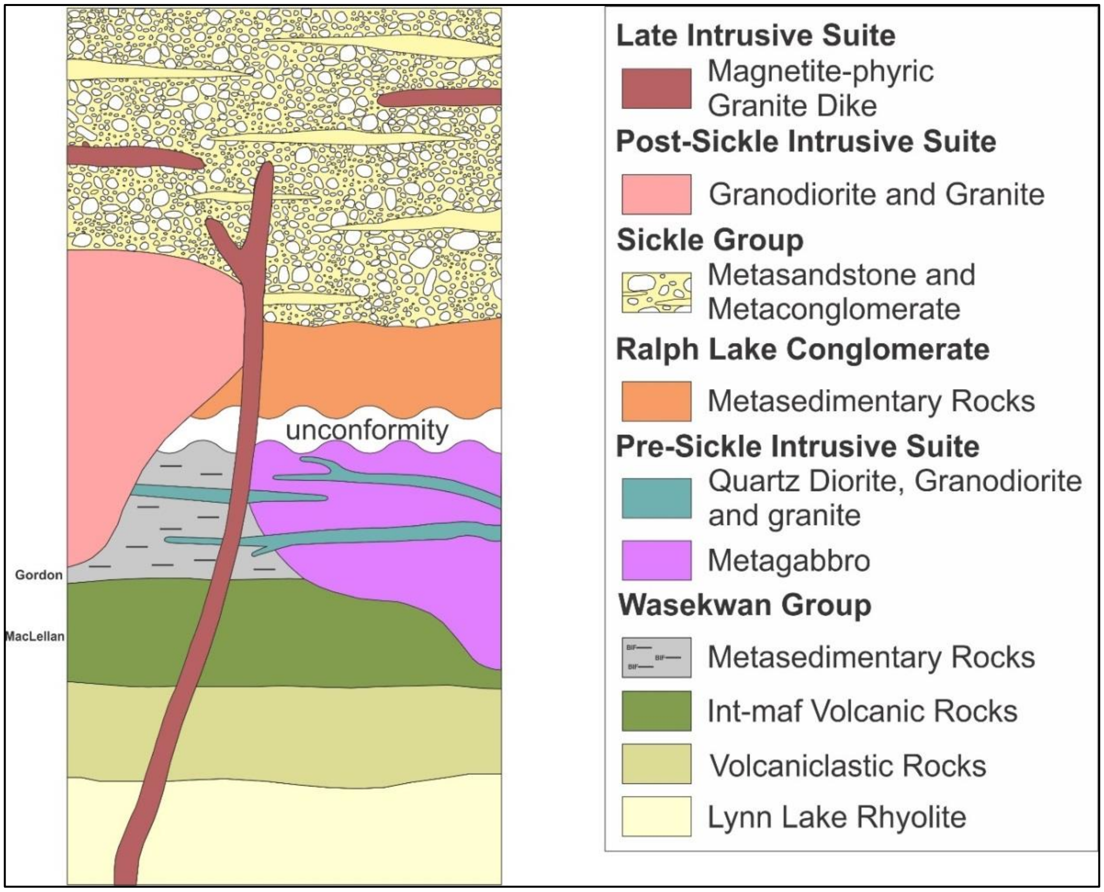


Figure 7-3: Stratigraphic Column of the Northern LLGB

Source: (Baldwin, Syme, Zwanzig, Gordon, Hunt, & Stevens, 1987)

7.2 Property Geology

Mapping of the LLGP properties is not complete. Detailed mapping has only been conducted in the MacLellan and Gordon areas.

Thought to be a north facing antiform, the northern belt is comprised of steeply north dipping, subaqueous sequence of volcanic rocks containing basalt, andesite, dacite and rhyolite overlain by volcaniclastic rocks and epiclastic sediments, intruded by mafic to felsic plutonic rocks (Fedikow & Gale, 1982). Inferring a volcanic arc or back-ark basin depositional environment (Yang & Beaumont-Smith, 2015).

Of importance, is a distinct stratigraphy of ultramafic picrite flows, plagioclase-phyric and aphyric basalt, oxide-facies banded iron formation and related exhalative and epiclastic rocks known as the Agassiz metallotect (Yang & Beaumont-Smith, 2016). This trend represents a relatively narrow, strike-continuous stratigraphic-structural unit that occurs over a 70 km length from west of the shear hosted MacLellan deposit, through the Gordon Property, to southwest of Barrington Lake. This structurally different body is known for hosting gold mineralization and exhibiting intense deformation (Beaumont-Smith & Böhm, 2004), including the MacLellan gold-silver deposit and the Gordon gold deposit.

7.2.1 MacLellan Area

The MacLellan Property is located on the western portion of the northern belt. Hosted within the Agassiz metallotect (historically termed the Rainbow Trend), the previously producing Au-Ag mine outcrops within picritic basalt, near the contact with mafic-intermediate volcaniclastic rocks to the north (Figure 7-4). The deposit is underlain by picritic basalt flows with associated fragmental and minor syn-volcanic pyroxenite, minor biotitic and siliceous siltstones, a thin oxide-facies banded iron formation and lesser felsic volcanics (Figure 7-4).

The MacLellan deposit is stratabound between a sequence of east-west trending clastic and chemical sedimentary rocks interlayered with picritic flows and tuffs. This section is encompassed by volcanoclastic rocks to the north and south and felsic volcanics further to the south. Metamorphic grade in this area is within the middle amphibolite facies. Minor felsic volcanics are located to the south of the mine area and gabbroic intrusions occur locally. The entire LLGB is bounded by granitic intrusions to the north and south.

Typically, rocks around the MacLellan deposit exhibit increased deformation and alteration partly due to the ductility of the host rock. They are seen as: plagioclase-amphibole schist, after mafic-intermediate volcaniclastic and plagioclase-phyric basalts; chlorite-amphibole schist, after ultramafic picrite and mafic flows; biotite-banded chlorite-amphibole schist, after fragmental equivalent of picrite flows; and notably biotite-quartz schist after both the picrite and siltstone, due to intense alteration nearing gneissose structure with recrystallization/coarsening of the quartz bands.

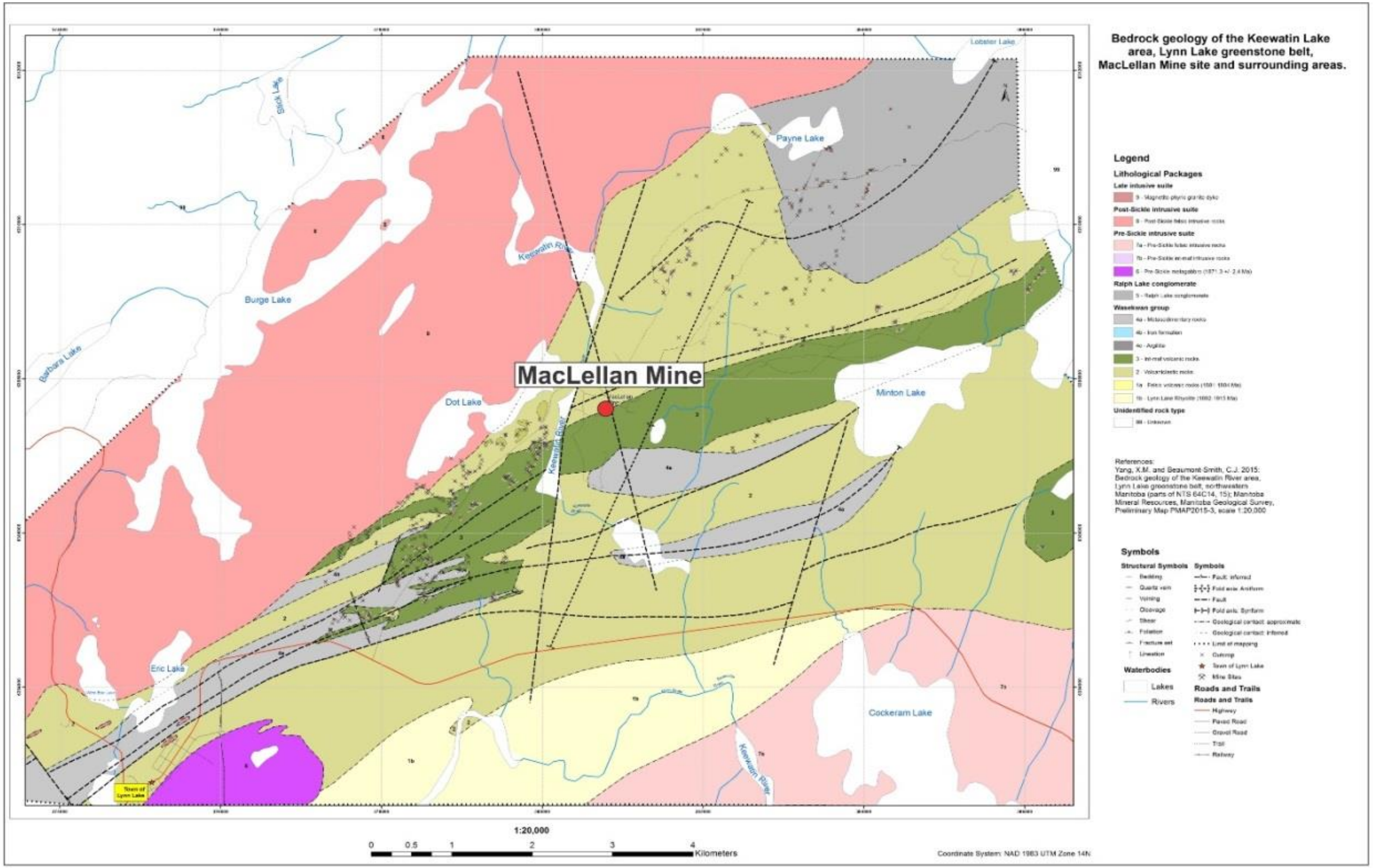


Figure 7-4: Property Geology of the MacLellan Area

Source: (Yang & Beaumont-Smith, 2015)

7.2.2 Gordon Area

The Gordon Property is hosted within the Agassiz Metallotect of the northern belt, which hosts all known gold mineralization in the north belt (Figure 7-5).

The property is composed of pillowed basalts, dacitic flows, siliceous sediments (argillites, greywackes, etc.), intermediate tuffaceous sediment and oxide-facies banded iron formation (BIF) which have been extensively folded into an east-west striking and steeply north-dipping sequence. A small, possibly very late (post-Sickle), quartz-diorite body occurs immediately south of the Gordon Property. Its presence, along with other mineralized diorite plugs elsewhere in the Lynn Lake area, highlights the significance of these intrusions as a possible source of gold-bearing fluids throughout the belt. The southern portion of this property is bounded by extensive massive, medium grained pink granite (Scott, Kuzmich, & Weston, 2015).

Between Gordon Lake and Farley Lake (location of the former open pits), the oxide facies iron formation consists of interbedded laminae of magnetite, chert and layers of chlorite, amphibole, minor silica, and disseminated magnetite. Intermediate dykes (historically termed mafic dykes) ranging from centimetre to metre scale intrude the BIF along stratigraphy. They exhibit biotite altered chill margins, a strong penetrative foliation, and generally poor mineralization. The oxide facies iron formation hosts several zones of gold mineralization (Puritch, Burga, & Wu, 2013).

Rocks in the Gordon area exhibit a metamorphic grade middle amphibolite facies. Pervasive and domain controlled anthophyllite – grunerite alteration is variable within the BIF, and is associated with D2 metamorphism, while randomly oriented porphyroblasts commonly occur within veins, contacts, and fractures and are a result of later hydrothermal alteration.

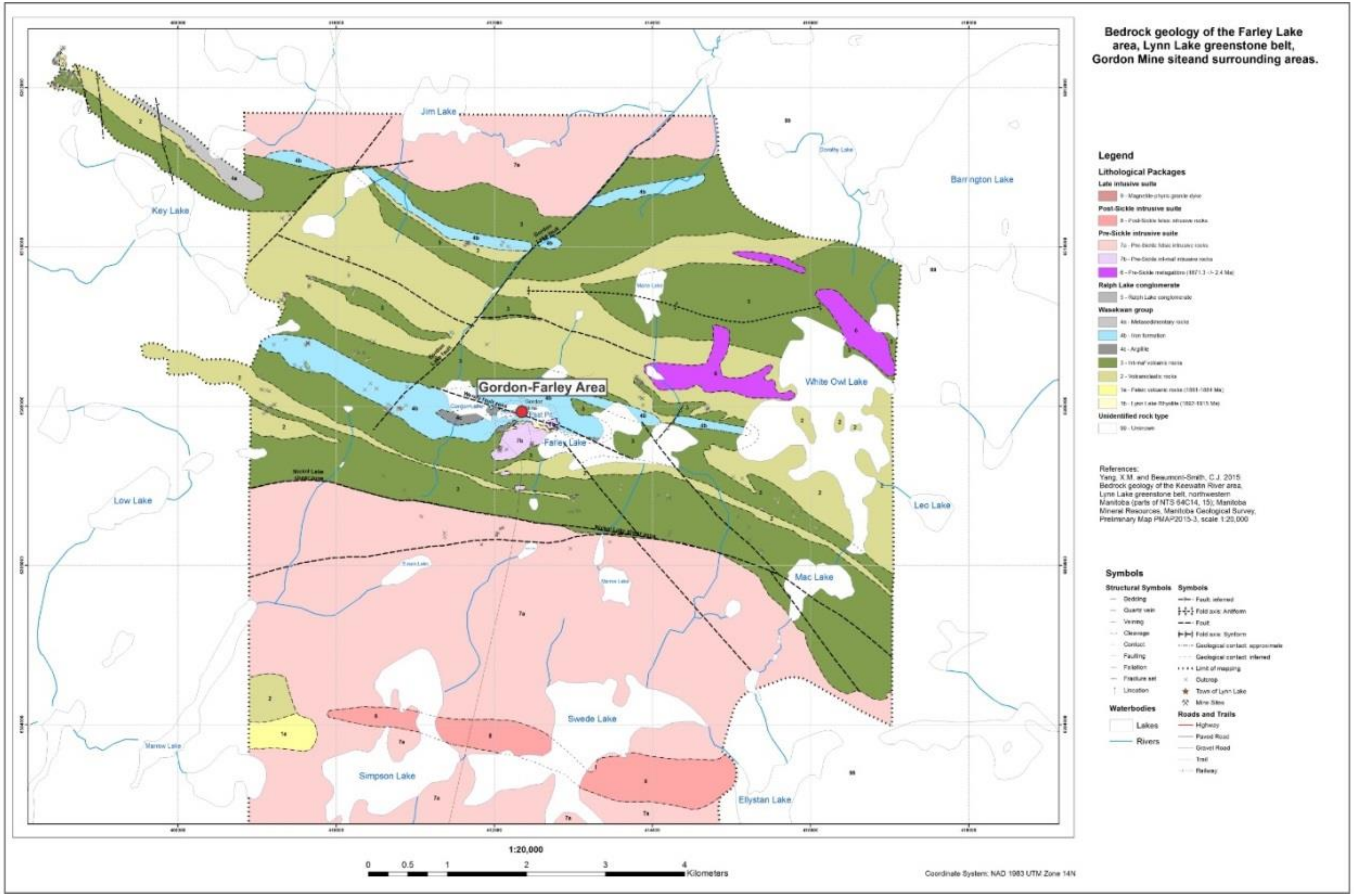


Figure 7-5: Property Geology of the Gordon Area

Source: (Yang & Beaumont-Smith, 2015)

7.3 Structural Geology

Historic and recent mapping of the LLGB, identified a total of 6 deformational events (D_1 – D_6) (Gilbert, Syme, & Zwanzig, 1980); (Beaumont-Smith & Bohm, 2002); (Beaumont-Smith & Böhm, 2004), described below:

- D_1 is generally overprinted or obscured by later fabrics; interpreted as post depositional fabrics;
- D_2 is represented by tight isoclinal folds, a regional penetrative foliation and regional transcurrent shear zones which cut the Sickle group. Associated with formation of the Agassiz metallotect;
- D_3 is represented by northwest trending S-asymmetrical chevron folds and associated crenulation cleavages;
- D_4 is defined by northeast trending Z-asymmetrical chevron folds and associated crenulation cleavages as well as brittle faulting;
- D_5 is evidenced by large, kilometre-scaled open folds; and
- D_6 is recorded by brittle reactivation of D2 shear zones forming narrow pseudotachylite zones.

7.3.1 MacLellan Structures

Around the MacLellan area, ductile D_2 structures dominate with a penetrative foliation and tight isoclinal folds, which increase in intensity within D_2 shear zones, near contacts and in the picrite unit. S_2 is observed as steeply, northwest dipping foliation-schistosity striking northeast (azimuth 045^0). Shear zones and schistosity exhibit dextral shear sense indicators throughout the MacLellan area.

Significant, brittle D_4 structures are present around MacLellan. These are recorded within the picrite (chlorite-amphibole schist), as northeast striking chevron folds and crenulations (azimuth 010^0), and as brittle faults of varying scales outside of the schists. Brittle D_6, pseudotachylite filled fractures and faults are observed on surface commonly cutting sub-parallel to D_2 shears.

7.3.2 Gordon Structures

At Gordon, D_2 deformation is observed as a penetrative foliation and associated tight isoclinal and chevron folding of varying scales; from centimetre-scale, parasitic folds to mesoscopic scale, tight isoclinal-chevron folds with shallow plunges, steepening to subvertical within shear zones. Ductile D_2 east-west striking shear zones are observed along contacts as well as within mafic-intermediate volcanics and cutting gabbroic intrusions. Of significance is the Nickle Lake Shear Zone, which is laterally extensive and cuts between intermediate-mafic volcanic rocks to the north, and post-Sickle intrusives to the south. Several parallel shear zones are observed closer to the Gordon mine, notably immediately south of Farley Lake, along mafic and intermediate volcanics. These shears are manifested by prominent magnetic lineaments.

Observed D_3 structures are rare, observed as close-tight asymmetric S-folds and northwest trending, axial-planar crenulation cleavages. Northwest trending faults cutting through the Gordon deposit and possibly significant to later gold mineralization cut D_2 shears and are thought to be related to the D_3 deformation event (Yang & Beaumont-Smith, 2016).

D_4 structures are commonly observed as steeply plunging F_4 folds, associated with steeply dipping northeast-striking axial planar crenulation cleavages (Yang & Beaumont-Smith, 2016).

D_5 is represented by mesoscopic conjugate folds, kink bands and crenulations. D_6 is observed as brittle-ductile reactivation of D_2 shear zones.

Of importance are shallow iron-carbonate – quartz – sulphide veins and breccia veins that cut stratigraphy at a high angle. They dip to the south at varying angles, ranging from 25^0-45^0, but are observed to shallow out at zones to 15^0. These veins strike east-southeast and cut most structures. Associated veins splay along stratigraphy (S2) commonly between intermediate dykes and oxide-facies BIF. These veins typically exhibit some scale of sulfide replacement, averaging 2.4 m wide zones (Weston, 2014).

7.4 Mineralization

Most volcanic and sedimentary rocks within the north and south belts contain at least some trace disseminated mineralization consisting of pyrrhotite, pyrite, lesser chalcopyrite, sphalerite, galena and arsenopyrite. Gold and silver mineralization can be geochemically anomalous to economic at the MacLellan site, Gordon area as well as the Burnt Timber – Linkwood areas of the southern belt. Gold is usually associated with pyrrhotite-pyrite mineralization and often with minor chalcopyrite, sphalerite, galena and arsenopyrite. Gold mineralization is often associated with narrow quartz and quartz carbonate veins and within semi-massive sulphide lenses 1 to 20 cm thick. Gold is usually very fine grained and is rarely visible and appears to be structurally controlled. The structures, usually quartz veins, are east-west trending.

At MacLellan and Burnt Timber gold-bearing structures dip steeply to the north and are hosted in mafic flows and sediments. Gold seems to be associated with carbonate and silica alteration.

With respect to MacLellan, the high-strain-zone and accompanying hydrothermal alteration localized to the picrite flows and minor intercalated sediments, coupled with its chemical trapping characteristics created favourable conditions for gold mineralization. Hastie (2014) emphasizes the importance of brittle D_4 structures, with supporting evidence for a second hydrothermal fluid event. The first gold-silver mineralizing episode, during ductile D_2 shearing and alteration, is characterized by biotite ± quartz ± sulphide alteration and shows gold-pyrrhotite-pyrite mineralization. The second episode of gold-silver mineralization exhibits fracture-fill Au, arsenopyrite, galena, and sphalerite, associated with D_4 faulting and hydrothermal fluids with Cr-clinochlore + carbonate ± hornblende + quartz alteration. A significant increase in gold-silver mineralization near the intersections of D_2 shear zones and D_4 faults is noted throughout the MacLellan deposit.

At Gordon, gold is often associated with quartz veins cutting folded and contorted Iron Formation, trending east-west and dipping shallowly to the south. Au mineralization is normally trapped in sulfide replacement zones averaging 2.4 m wide, associated with thin, centimetre-scale quartz-sulfide veins and breccia veins. In places, coarse gold flecks are observed within these quartz veins and in iron-carbonate altered fragments within breccia veins. Strong iron-carbonate – chlorite – amphibole alteration is related to some high grade veins.

8 DEPOSIT TYPES

8.1 Orogenic Gold

Both MacLellan and Gordon deposits can be considered as belonging to the class of gold deposits referred to as orogenic. The orogenic gold deposit model (Groves, Goldfarb, Gebre-Mariam, Hagemann, & Robert, 1998) characterizes structurally controlled gold occurrences formed during orogenesis by either metamorphic or deeply sourced magmatic fluids (Figure 8-1), These deposits are thought to have first-order tectonic controls, and are associated with crustal-scale faults which tap sub-crustal source regions, although individual deposits are commonly situated in second-order structures (Groves, Goldfarb, & Santosh, 2016). Any rock type within a greenstone belt, including supracrustal rocks, dykes, or intrusions within or intrusion bounding such belt may host an orogenic gold deposit (Figure 8-2). There is strong structural control of mineralization at a variety of scales but the favoured host is typically the locally most reactive and/or most competent lithological unit. Most of the Precambrian deposits were formed during 2.70–2.60 Ga and 2.10–1.70 Ga and include world class deposits of the Superior Province in Canada, the Yilgarn Craton in Western Australia and the Birimian of West Africa. These epochs appear to be related to rapid crustal growth and accretionary stages of supercontinents.

In greenschist facies settings, mineralisation typically takes place slightly after the metamorphic peak, but in amphibolite facies at the local regional-metamorphic peak. The ore bodies typically form ore shoots. An ore body can be 0.5–50 m wide, 100’s of meters long, and consists typically of a vein network, an en echelon vein swarm, or just of one single large vein. The depth extent of an ore body may well be much larger than its extent along strike.

All deposits are developed by an alteration halo characterized by proximal to distal carbonatization and proximal potassic alteration. Elements enriched typically include As, Au, CO_2, K, Rb, S, Sb, Te, and W; in some cases also Ag, B, Bi, Co, Cu, and Se are enriched. Alteration mineral assemblages, alteration indices based on CO_2 and K, and trace (pathfinder) elements enriched in the deposits can be used in defining exploration targets and vectors to ore in bedrock (Eilu, Ojala, & Sarala, 2011).

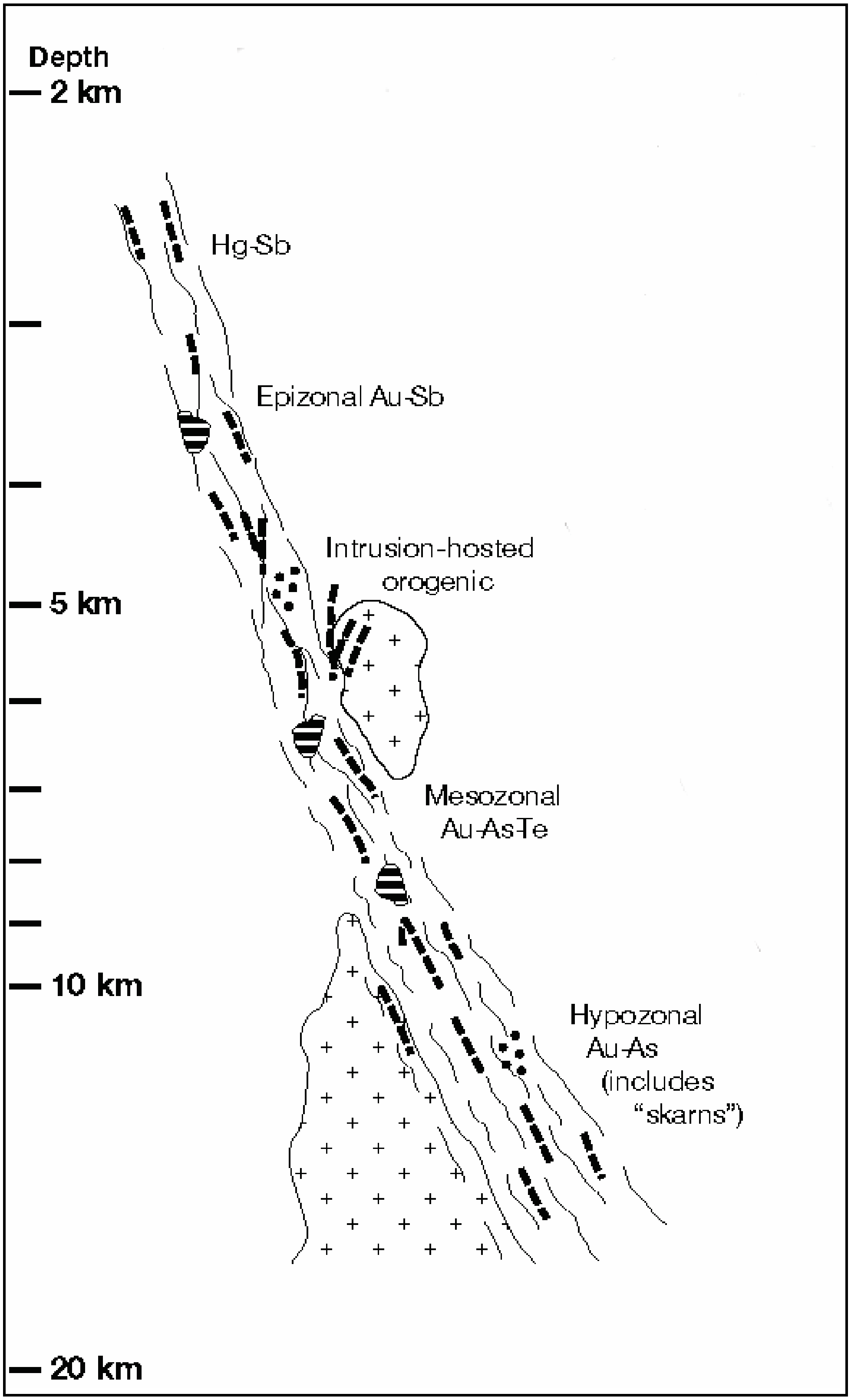


Figure 8-1: Schematic Representation of Crustal Environments of Orogenic Gold Deposits

Source: (Groves, Goldfarb, Robert, & Hart, 2003)

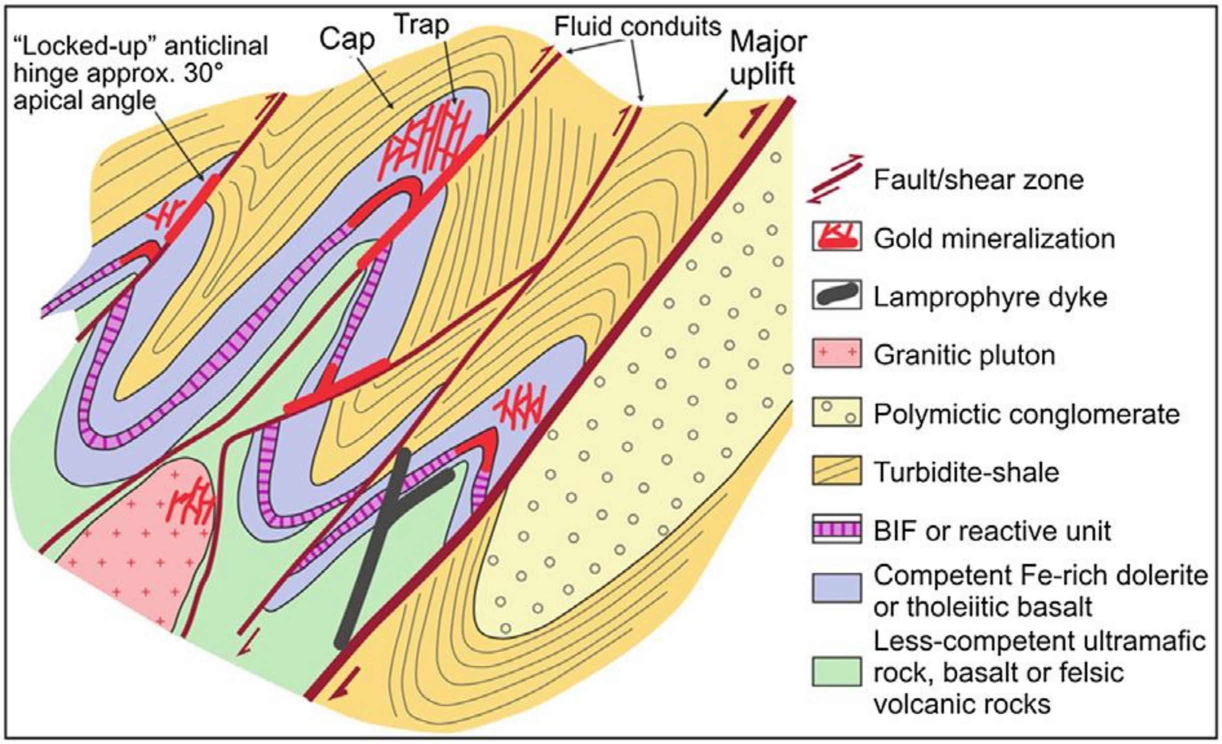


Figure 8-2: Idealized Sketch Illustrating Potential Host Environments for Orogenic Gold Deposits

Source: (Groves, Goldfarb, & Santosh, 2016)

9 EXPLORATION

9.1 Exploration

More recent exploration on the property has been performed by Carlisle and Alamos. Exploration programs include mapping and sampling, airborne magnetic survey, Induced Polarization (IP) Surveys and exploration drilling.

9.1.1 Mapping and Sampling

The Company’s land holdings in the LLGB are low lying with little outcrop, with existing outcropping being well under 5%. Mapping has been completed in the MacLellan area and the results of that mapping program is presented in Section 7.2.1 of this report.

Mapping in the Gordon area is partially complete and will continue through 2017. The geological map of this area is presented in Section 7.2.2.

The area between MacLellan and Gordon has only been reconnaissance mapped to date. Access to this area is poor and mapping continued in 2017. Due to the poor access, helicopter and ATV support is utilized.

9.1.2 Geophysical Surveys

The Company has completed an airborne Total Field Magnetic survey over its entire land package. IP surveys have been completed on select areas including MacLellan southwest and northeast, Gordon and Burnt Timber areas.

9.1.2.1 Airborne Magnetic Survey

The Company had a DIGHEM^{TM} aeromagnetic-electromagnetic survey completed over its entire land package. The helicopter borne survey was carried out at 100 m spaced flight lines with 20 m cells. The results of the +6,000 km survey are presented on Figure 9-1.

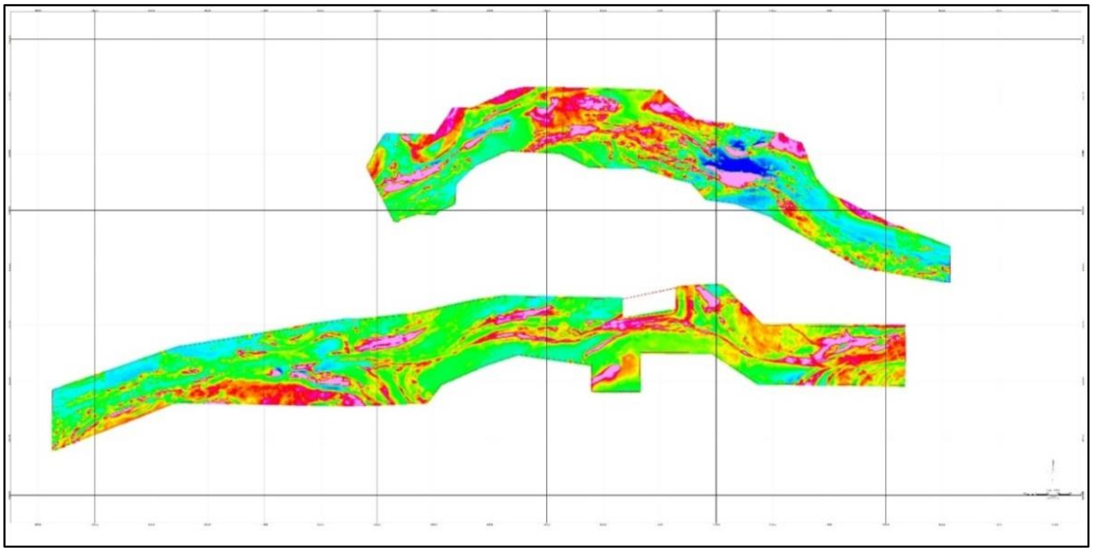


Figure 9-1: Total Field Magnetics

Source: Alamos (2017)

9.1.2.2 *Induced Polarization MacLellan Northeast and Southwest Areas*

In late 2015 and early 2016 Quantec Geophysics (Quantec) completed a DCIP survey northeast of the MacLellan site. Quantec ran a 112 km Pole-Dipole survey on lines at 200 m centers, with a=50 m and N=1 to 8.

The main purpose of the survey was to establish an area of sterile ground suitable for a tailings impoundment.

The chargeability plan of the survey is presented in Figure 9-2 below. The pseudo-sections from this survey are not presented here.

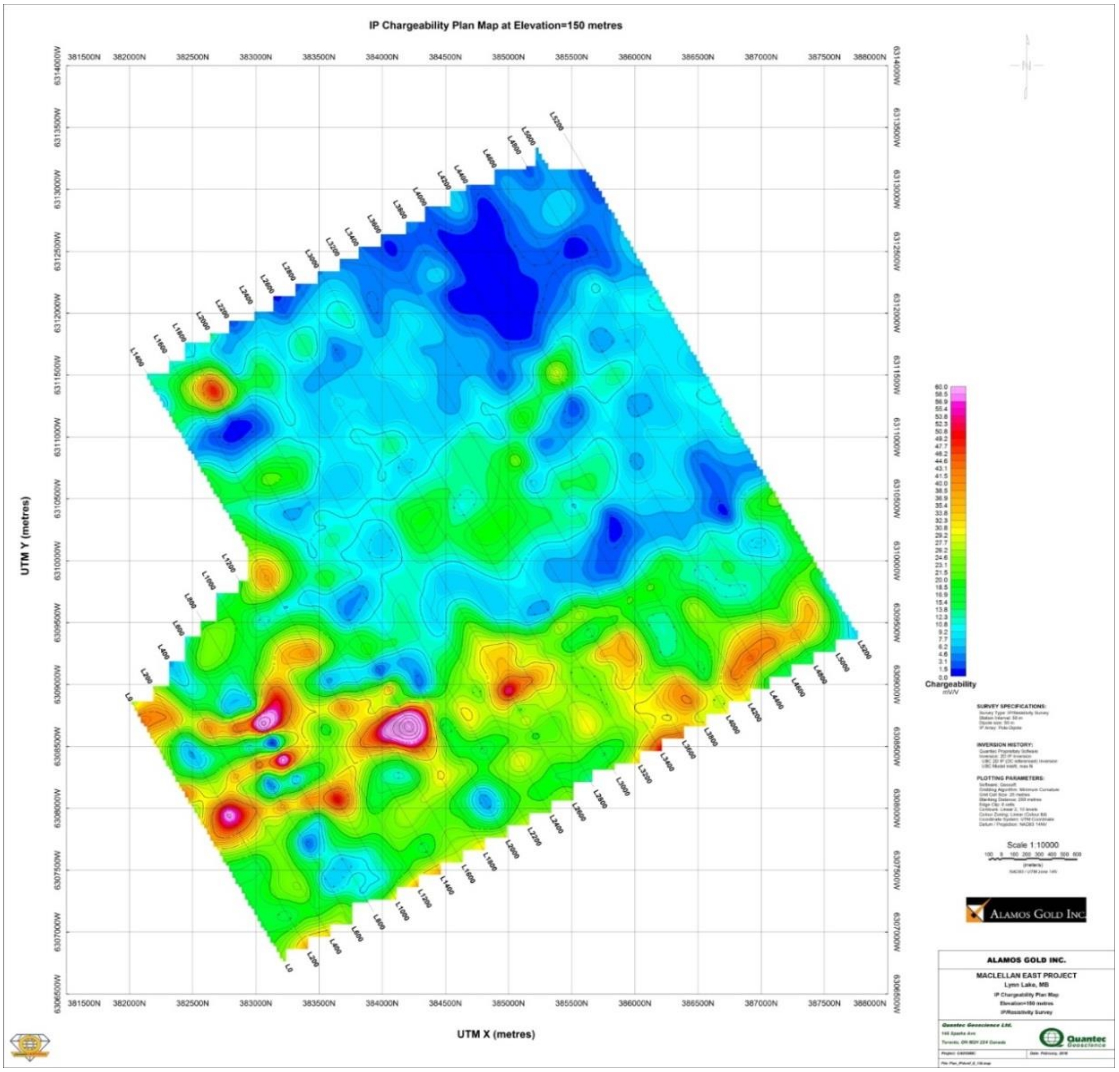


Figure 9-2: MacLellan Northeast IP Survey

Source: Quantec Geoscience (2016)

In 2010 and 2011 Carlisle contracted Quantec to run a Pole-Dipole IP survey over 149 km just to the southwest of the MacLellan site. Line spacing was 100 m with a=50 m and N=1 to 8. The results of the survey are presented below in Figure 9-3. The pseudo-sections from the survey are not presented here.

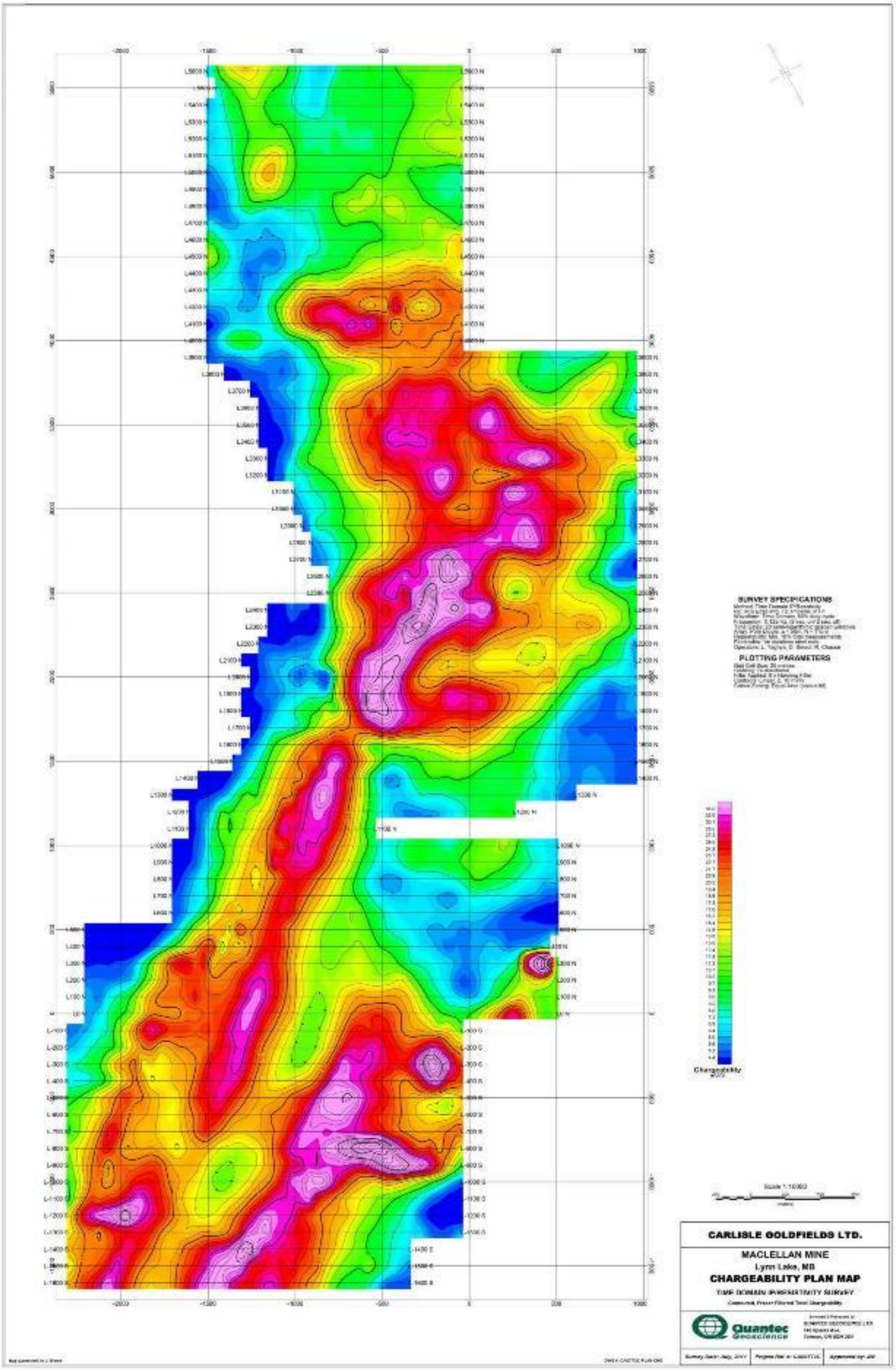


Figure 9-3: MacLellan Southwest IP Survey

Source: Quantec Geoscience (2011)

9.1.2.3 *Induced Polarization Gordon*

In 2012 Quantec was contracted by Carlisle to run a standard pole-dipole Induced Polarization survey on the Gordon area. The survey was 60.7 km long at 200 m centers with n=1 to 8 spacing's at 50 m.

The purpose of the survey was to locate areas that might be mineralized for drill testing on the flanks of the Gordon Mineral Resources.

The chargeability plan of the survey is presented in Figure 9-4. The pseudo-sections from this survey are not presented here.

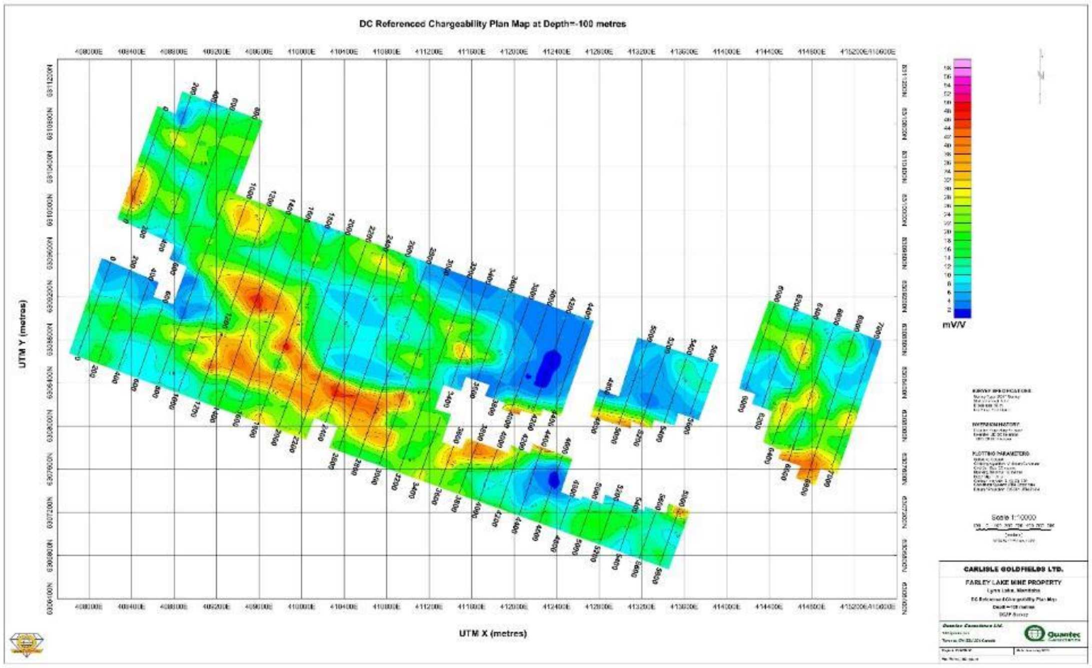


Figure 9-4: Gordon IP Survey

Source: Quantec Geoscience (2012)

9.1.2.4 *Induced Polarization Burnt Timber-Linkwood*

In 2011 Quantec was contracted by Carlisle to run a standard pole-dipole array IP survey in the Burnt Timber-Linkwood area on the southern belt "Johnson Shear" area. The survey was 53 km long at 200 m centers with n=1 to 8 spacings at a=50 m.

The purpose of the survey was to possibly identify the continuation of gold mineralization westwards of Burnt Timber to the Linkwood area. The IP survey did not cover the majority of the Burnt Timber Mineral Resource and the IP Chargeability affect seems washed out moving from west to east towards Burnt Timber. None of the anomalies from this survey have been drill tested.

The chargeability plan of the survey is presented in Figure 9-5 below. The pseudo-sections from this survey are not presented here.

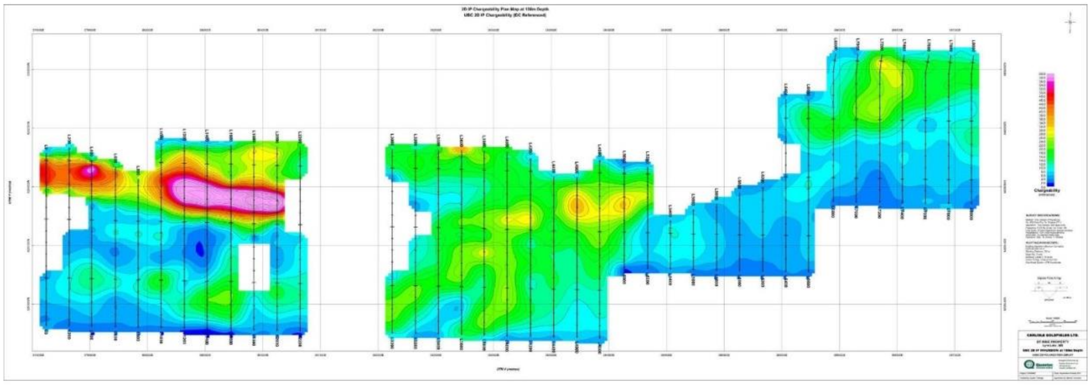


Figure 9-5: Burnt Timber-Linkwood IP Survey

Source: Quantec Geoscience (2011)

9.2 Exploration Diamond Drilling

There are over 3,000 diamond drill holes totalling over 400,000 m completed on the Lynn Lake properties dating back to the mid-1940s. Much of this drilling was concentrated on MacLellan and Gordon Mineral Resource areas. Company drilling in these areas was largely in-fill Mineral Resource and geotechnical drilling and will be discussed in Section 10 of this report.

Exploration drilling in 2016 was largely dictated by accessibility. The fall was very wet and freeze-up was slow to take hold. Due to muddy conditions drilling could only be undertaken in higher dryer areas. Exploration used two skid mounted drills utilizing NQ sized equipment. One drill from Black Hawk Drilling Ltd. (Black Hawk Drilling) of British Columbia drilled exploration holes beneath known resources at MacLellan. The second drill was from Dorado Drilling Ltd. (Dorado Drilling), also from British Columbia, and was used to test IP Anomalies to the southwest of the MacLellan Mineral Resource. Between the two drills 11,718 m were drilled in 21 holes in 2016. The location and lengths of the holes is summarized in Table 9-1.

Table 9-1 – Drilling Summary

Hole ID	Easting	Northing	Elevation (m)	Length (m)	Azimuth	Dip
16MCX001	381016	6308032	335	914.63	160	-66
16MCX001A	381016	6308032	335	210.37	160	-63
16MCX002	380801	6308113	335	1,310.64	165	-70
16MCX003	380668	6307968	335	1,033.27	175	-67
16MCX004	379439	6307009	350	326.00	150	-50
16MCX005	380511	6307836	335	905.25	160	-64.5
16MCX006	379081	6306556	350	333.00	150	-50
16MCX007	378853	6306354	350	374.00	150	-50
16MCX008	378898	6306544	350	348.00	150	-50
16MCX009	380455	6307745	335	906.63	165	-70
16MCX009A	380455	6307745	335	283.46	165	-65
16MCX010	378379	6305953	350	345.00	150	-50
16MCX011	378476	6306099	350	381.00	150	-50
16MCX012	378643	6306136	350	324.00	150	-50
16MCX013	378393	6306507	350	669.00	120	-55
16MCX014	380756	6307967	335	1,122.88	165	-72
16MCX015	379290	6306746	350	393.00	150	-50
16MCX016	380772	6307055	345	450.00	130	-50
16MCX017	380790	6306932	343	328.50	130	-50
16MCX018	380294	6307776	335	434.00	165	-64
16MCX019	380710	6306835	343	325.50	35	-50
			Total	11,718.13		

9.2.1 MacLellan Deeps

Seven holes were drilled by Black Hawk Drilling below known Mineral Resources at MacLellan. All but one reached the target horizon. The location of the holes pierce points and results are presented on Figure 9-6 and in Table 9-2. Further drilling is required on MacLellan Deeps to access its down-dip potential.

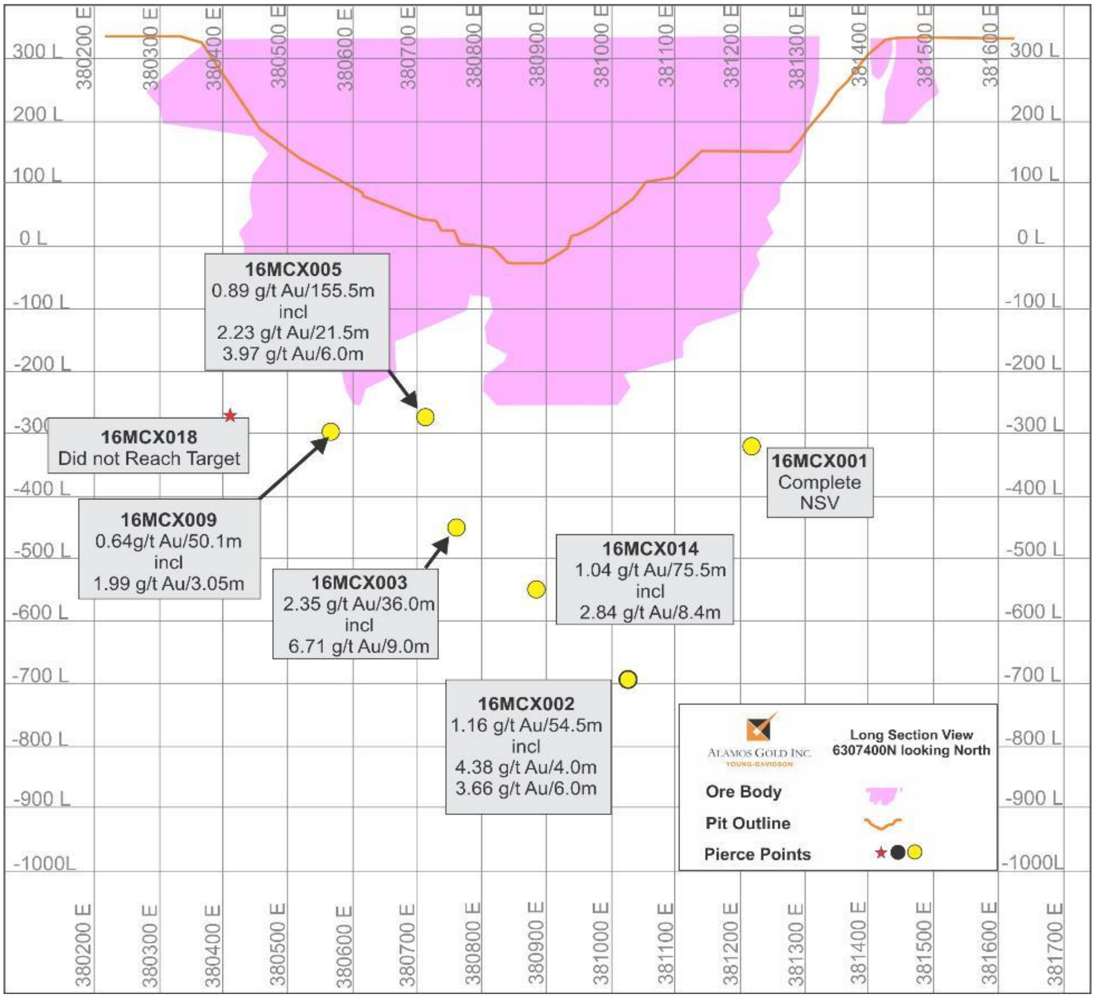


Figure 9-6: Longitudinal Section through the MacLellan Deposit, Showing 2017 Deep Drilling

Source: Alamos (2017)

9.2.2 MacLellan Southwest IP Targets

Dorado Drilling completed 13 holes largely designed to test IP Anomalies immediately southwest of the MacLellan Mineral Resources. The location of the holes is presented on the IP Chargeability Plan on Figure 9-7. The assay results are presented in Table 9-2.

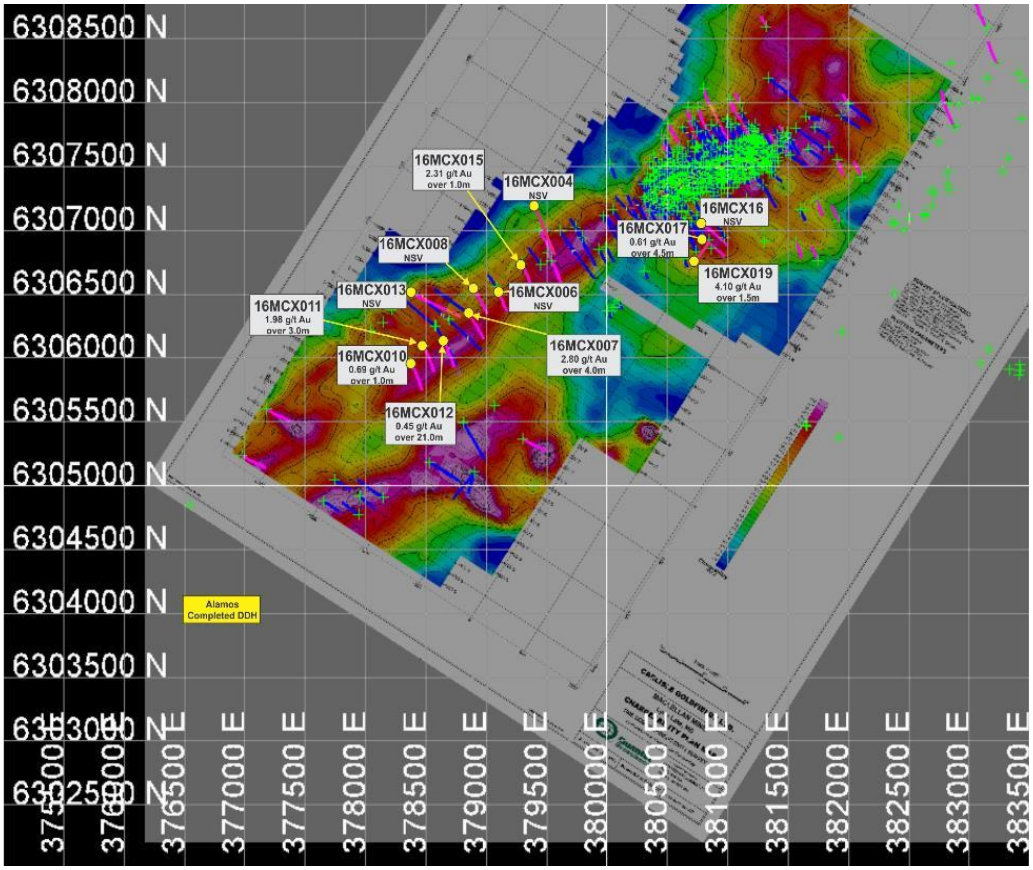


Figure 9-7: MacLellan Southwest Exploration Drilling

Source: Alamos (2017)

Table 9-2 – Exploration Drilling Results

Drill Hole	Area	From (m)	To (m)	Interval (m)	Au (g/t)
16MCX001	MacLellan Deeps				No significant intersections
16MCX002	MacLellan Deeps	1,122.50	1,177.00	54.50	1.16
		Incl.	Incl.	Incl.	
		1,127.00	1,131.00	4.00	4.38
		1,159.00	1,165.00	6.00	3.66
16MCX003	MacLellan Deeps	881.00	917.00	36.00	2.35
		Incl.	Incl.	Incl.	
		891.50	900.50	9.00	6.71
16MCX004	MacLellan SW I.P. Targets				No significant intersections

Drill Hole	Area	From (m)	To (m)	Interval (m)	Au (g/t)
16MCX005	MacLellan Deeps	716.50	872.00	155.50	0.89
		Incl.	Incl.	Incl.	
		716.50	738.00	21.10	2.23
		732.00	738.00	6.00	3.97
		752.50	757.00	4.50	3.58
16MCX006	MacLellan SW I.P. Targets				No significant intersections
16MCX007	MacLellan SW I.P. Targets	269.00	273.00	4.00	2.80
16MCX008	MacLellan SW I.P. Targets				No significant intersections
16MCX009	MacLellan Deeps	690.00	740.10	50.10	0.64
		Incl.	Incl.	Incl.	
		694.50	695.50	1.00	3.01
		713.23	716.28	3.05	1.99
		724.35	727.90	3.55	1.26
		734.00	735.00	1.00	2.17
16MCX009A	MacLellan Deeps Did Not Reach Target				No significant intersections
16MCX010	MacLellan SW I.P. Targets	50.00	51.00	1.00	0.69
16MCX011	MacLellan SW I.P. Targets	157.50	160.50	3.00	1.98
16MCX012	MacLellan SW I.P. Targets	183.60	204.60	21.00	0.45
16MCX013	MacLellan SW I.P. Targets				No significant intersections
16MCX014	MacLellan Deeps	945.00	1020.50	75.50	1.04
		Incl.	Incl.	Incl.	
		1,000.10	1,008.50	8.40	2.84
16MCX015	MacLellan SW I.P. Targets	285.00	286.50	1.50	4.29
		330.00	331.00	1.00	2.31
16MCX016	MacLellan South				No significant intersections

Drill Hole	Area	From (m)	To (m)	Interval (m)	Au (g/t)
16MCX017	MacLellan South	36.00	37.15	1.15	0.80
		43.50	48.00	4.50	0.61
16MCX018	MacLellan Deeps				No Samples
	Abandoned-no samples				
16MCX019	MacLellan South	45.00	46.50	1.50	4.10
	Did not make it to target				
17MCX001	MacLellan Pit NE	119.00	120.50	1.50	0.42
	MG15-49 follow up				
17MCX002	MacLellan Pit NE	23.00	24.50	1.50	1.18
	MG15-49 follow up				
17MCX003	MacLellan Pit NE	59.00	140.00	81.00	1.22
		Incl.	Incl.	Incl.	
		89.00	128.00	39.00	1.78
		95.00	116.00	21.00	2.74
		108.74	116.00	7.26	4.70
17MCX004	MacLellan				No significant intersections
	NE I.P. Target				
17MCX005	MacLellan				No significant intersections
	NE I.P. Target				
17MCX006	MacLellan				No significant intersections
	NE I.P. Target				
17MCX007	MacLellan				No significant intersections
	NE I.P. Target				
17MCX008	MacLellan NE I.P. Target	182.50	184.00	1.50	2.33
		311.50	313.00	1.50	1.94
17MCX009	MacLellan NE I.P. Target				No significant intersections
17MCX010	MacLellan Pit NE	85.60	90.50	4.90	1.34
		Incl.	Incl.	Incl.	
		89.00	90.50	1.50	3.10
17MCX011	MacLellan Pit NE	90.20	91.65	1.45	1.76
		304.95	305.75	0.80	1.13
		315.50	317.00	1.50	5.43
		366.50	368.00	1.50	2.24

Drill Hole	Area	From (m)	To (m)	Interval (m)	Au (g/t)
17MCX012	Follow-up on 17MCX003	8.00	26.50	18.50	3.67
		Incl.	Incl.	Incl.	
		17.00	21.00	4.00	14.25
17MCX013	Follow-up on 17MCX003	27.50	81.00	53.50	1.09
		Incl.	Incl.	Incl.	
		34.00	47.50	13.50	2.58
17MCX014	Follow-up on 17MCX003	71.80	105.50	33.70	2.18
		Incl.	Incl.	Incl.	
		72.70	75.50	2.80	6.08
		90.20	100.20	10.00	3.42
17MCX015	Follow-up on 17MCX003	135.00	144.10	9.10	0.50
		160.00	169.00	9.00	1.34
		208.50	211.50	3.00	2.90
17MCX016	Follow-up on 17MCX003	165.00	176.70	11.70	0.91
17MCX017	Follow-up on 17MCX003	144.50	146.00	1.50	1.25
17MCX018	Follow-up on 17MCX003- MacLellan NE	181.50	183.00	1.50	0.61
		184.50	186.00	1.50	0.54
17MCX019	Follow-up on 17MCX003	148.50	185.00	36.50	3.85 / 3.33 cut
		Incl.	Incl.	Incl.	
		148.50	155.50	7.00	3.57
		172.00	185.00	13.00	8.67 / 7.19 cut
17MCX020	Follow-up on 17MCX003- MacLellan NE				No significant intersections
17MCX021	MacLellan NE Pit area	60.50	61.60	1.10	1.00
17MCX022	MacLellan NE Pit area				No significant intersections
17MCX023	MacLellan NE Pit area	141.50	142.70	1.20	2.93
17MCX024	MacLellan NE Pit area	238.00	246.00	8.00	1.60
17MCX025	MacLellan NE Pit area	81.00	82.50	1.50	2.65
		121.50	123.00	1.50	0.72
		201.33	202.52	1.19	2.90

10 DRILLING

10.1 Historical Drilling

10.1.1 MacLellan

Drilling on the property has been conducted in a number of campaigns by prior operators and by the Company. A summary of these campaigns by year is provided in Table 10-1. A plan map of the deposit showing all drilling through 2016 is provided in Figure 10-1.

Table 10-1 – Summary of Historical Drill Programs

Years	Hole Series	# of Holes	Meters	Type
Unknown	1000	7	2,390	Surface
Unknown	96	2	703	Surface
Unknown	99	16	4,102	Surface
1955-1982	A	180	34,302	Surface
1959	D59	6	1,439	Surface
1959	DOT59	2	524	Surface
1965-1984	AS	109	19,310	Surface
1969-1975	U	181	9,527	Underground
1980	D80	5	740	Surface
1980	D83	6	1,060	Surface
1980-1988	MU	189	16,648	Underground
1983-1986	MS	5	634	Surface
1984	TA	4	671	Surface
1984	TAS	3	605	Surface
1984-1985	AU	222	32,506	Underground
1985-1987	DOT	5	2,230	Surface
1987	AT	51	490	Surface
1987-1988	RBS	22	9,521	Surface
1987-1988	RBX	28	5,256	Surface
1988-1989	NU	142	4,198	Underground
1989	RU	78	5,073	Underground
1992	MACCP92	12	360	Surface
1994	DS94	5	421	Surface
2007-2011	MG	115	48,586	Surface
2015	MG	78	17,535	Surface
2016	MCD	17	3,161	Surface

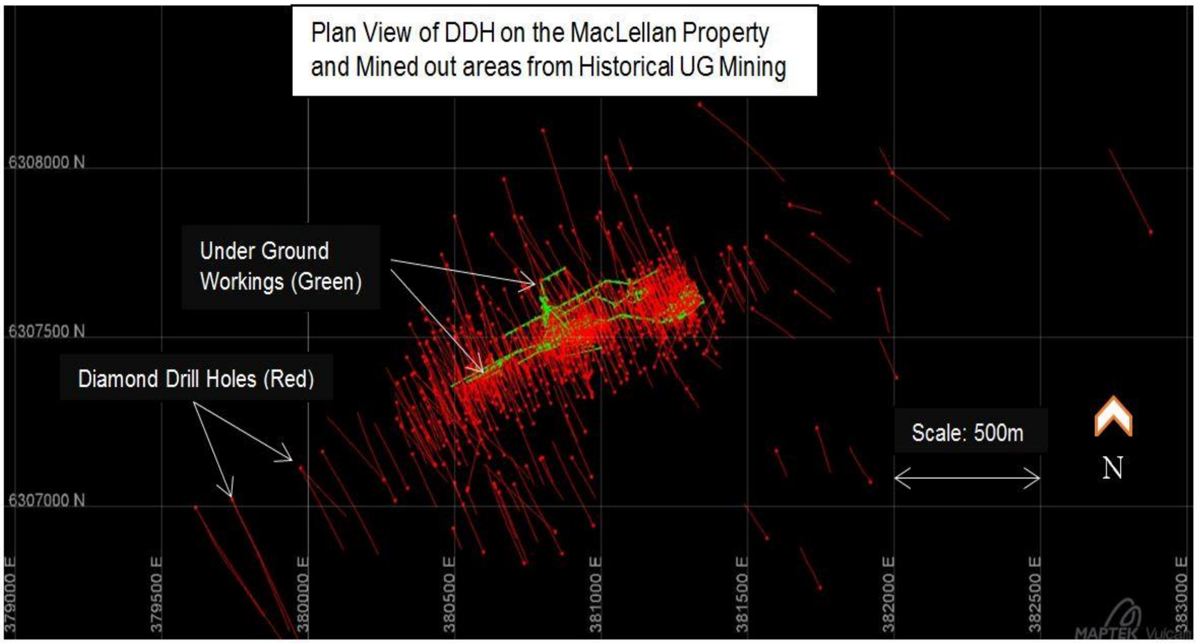


Figure 10-1: Plan View of MacLellan Drill Holes and Historical Underground Workings

Source: Alamos (2017)

10.1.2 Gordon

Drilling on the Gordon Property has been conducted by HBED and MMR. Between 1985 and 1995, by Carlisle from 2012 to 2014, and by Alamos from 2015 to 2016. Table 10-2 summarizes the drilling at Gordon.

Table 10-2 – Summary of Historic Drill Programs

Years	Hole Series	# of Holes	Meters	Type
1985-1995	654	346	50,299	Surface
2012-2014	FL	44	12,134	Surface
2015	FL	84	12,891	Surface
2016	FCD	11	1,715	Surface

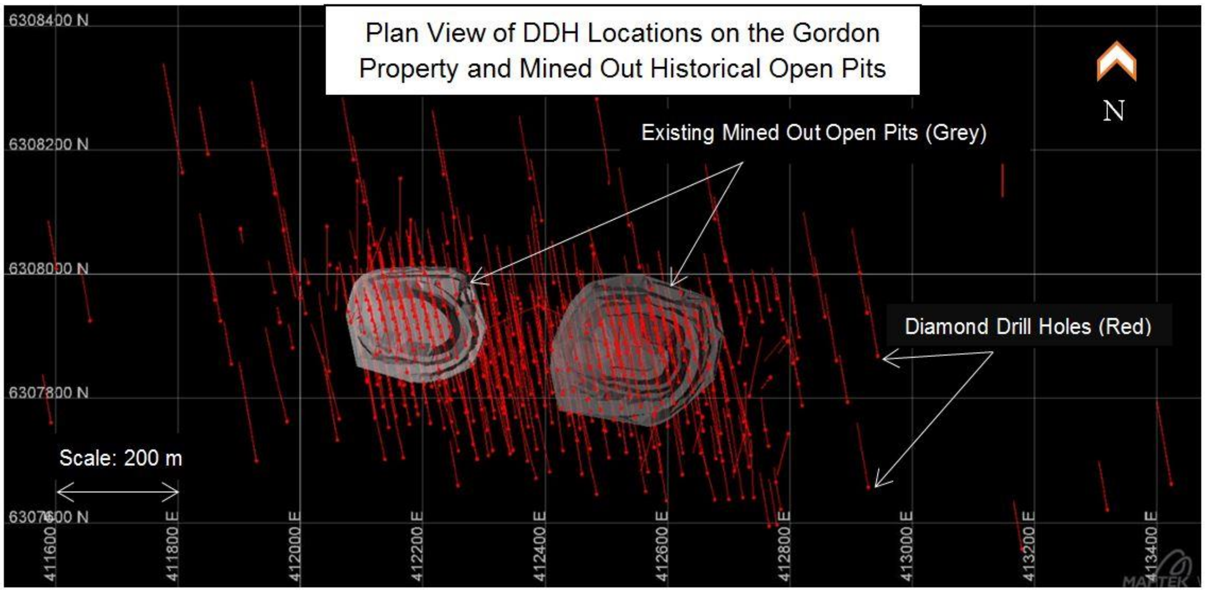


Figure 10-2: Plan View of Gordon Drill Holes and Historical Open Pits

Source: Alamos (2017)

10.2 2015-2016 Drilling Campaigns

In 2015, the Company initiated drill programs at the Gordon and the MacLellan properties. The drill program commenced in March 2015 and ran continuously through August 2015. An additional four holes were drilled during October and November 2015. The Company executed a condemnation drilling program at both sites from July 2016 to October 2016.

10.2.1 Objectives

10.2.1.1 2015 Drilling Program

The overall objectives of the 2015 drilling program were to:

- Provide verification of the historic drill data;
- To test targets outside of the pit designs from the Preliminary Economic Assessment completed by Tetra Tech in 2013; and
- Provide additional drill data density to areas with historical Mineral Resources to upgrade the classification for Mineral Resource estimation purposes.

10.2.1.2 2016 Drilling Program

The objectives of the 2016 drilling program were to:

- Test IP anomalies from the condemnation geophysical IP grid completed in early 2016; and
- Sterilize areas for potential infrastructure development.

The 2015 and 2016 drilling programs were carried out using three diamond drill rigs from three different diamond drill contractors: Element Drilling Ltd. from Gimli, MB; Dorado Drilling from

Vernon, BC; and Black Hawk Drilling from Smithers, BC. All drills were diesel powered hydraulic wireline coring machines. Drill core was recovered in 3 m NQ core tubes and was placed in labelled wood core boxes with wood marking blocks to indicate the meterage of the hole.

10.2.2 MacLellan Property Mineral Resource Drilling

The 2015 drilling program commenced in May of 2015 and concluded in July of 2015. A total of 78 core holes totalling 17,535 m were drilled. Drill holes were oriented at 333° or 153° azimuth, depending on whether the collar location was to the north or the south of the mineralized zone and were collared from -37° to -90° dip.

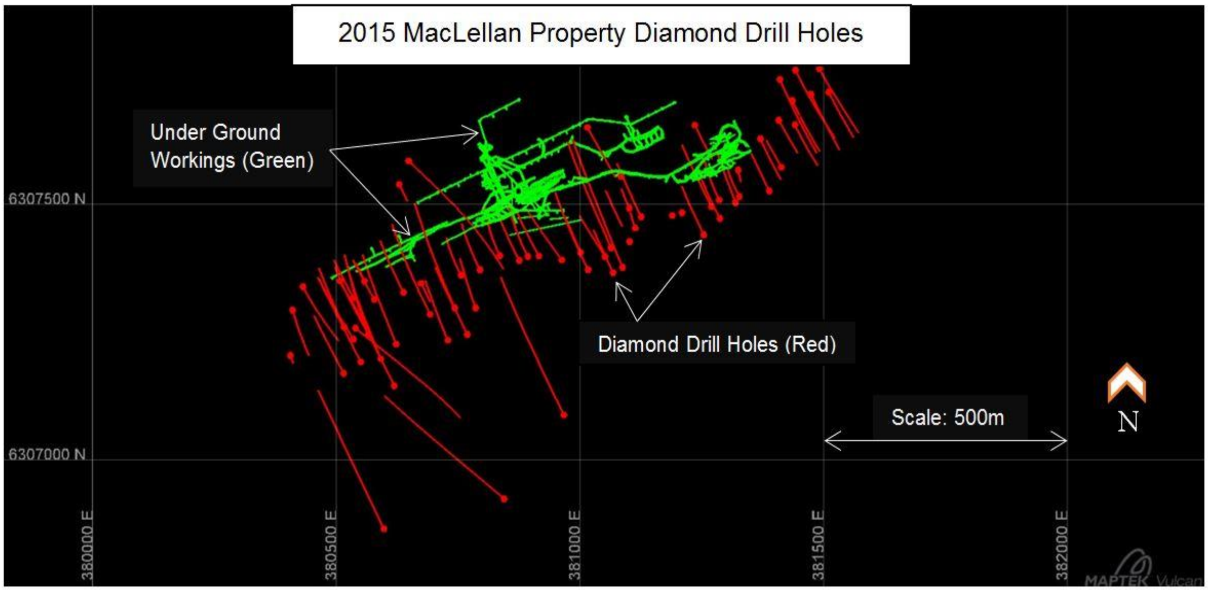


Figure 10-3: 2015 MacLellan Property Diamond Drill Holes and Historical Underground Workings

Source: Alamos (2017)

10.2.3 Gordon Property Mineral Resource Drilling

The 2015 drilling program commenced in March 2015 and concluded in August 2015, and totaled 12,891 m in 84 holes. Drill holes at the Gordon site were oriented at 350° or 170° azimuth and ranged in dip from -35° to -90°. The majority of holes were drilled in the 350° orientation due to the ore body dipping to the south. A total of 24 holes were collared on ice: 16 on the flooded existing open pits; and eight on the northwest corner of Farley Lake.

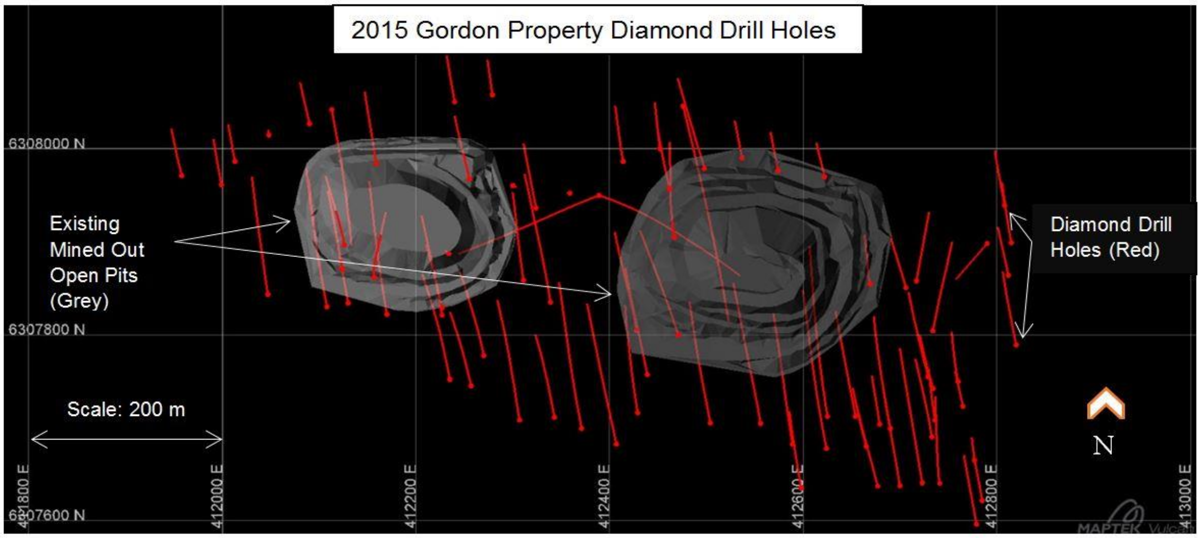


Figure 10-4: 2015 Gordon Property Diamond Drill Holes and Historical Open Pits

Source: Alamos (2017)

10.2.4 MacLellan Property Condemnation Drilling

The 2016 condemnation drilling program commenced in August 2016 and concluded in October 2016. A total of 3,161 m were drilled in 17 holes. Drill holes targeted geophysical IP anomalies within the boundaries of proposed infrastructure sites and were oriented at 333° or 153° azimuth and collared between -45° and -55° dip.

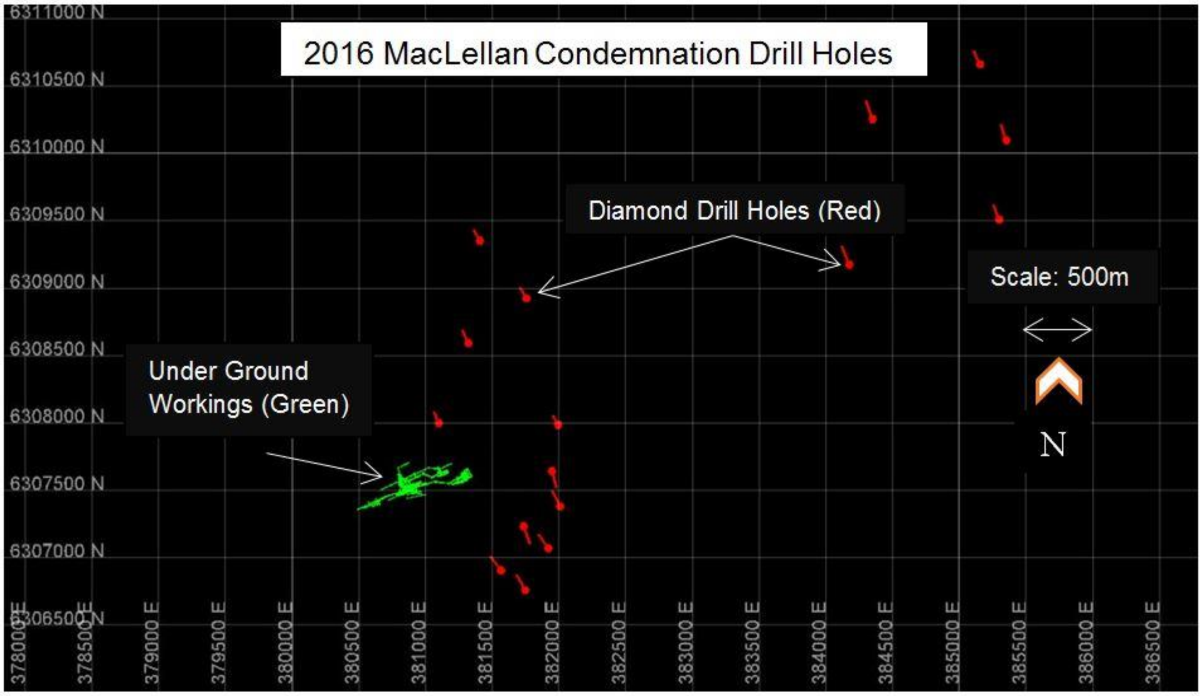


Figure 10-5: 2016 Condemnation Diamond Drill Holes at the MacLellan Property

Source: Alamos (2017)

10.2.5 Gordon Property Condemnation Drilling

The 2016 condemnation drilling program commenced in July 2016 and concluded in August 2016, and totaled 1,715 m in 11 holes. Drill holes targeted IP anomalies within the boundaries of proposed infrastructure sites and were oriented at 350° azimuth and -55° dip.

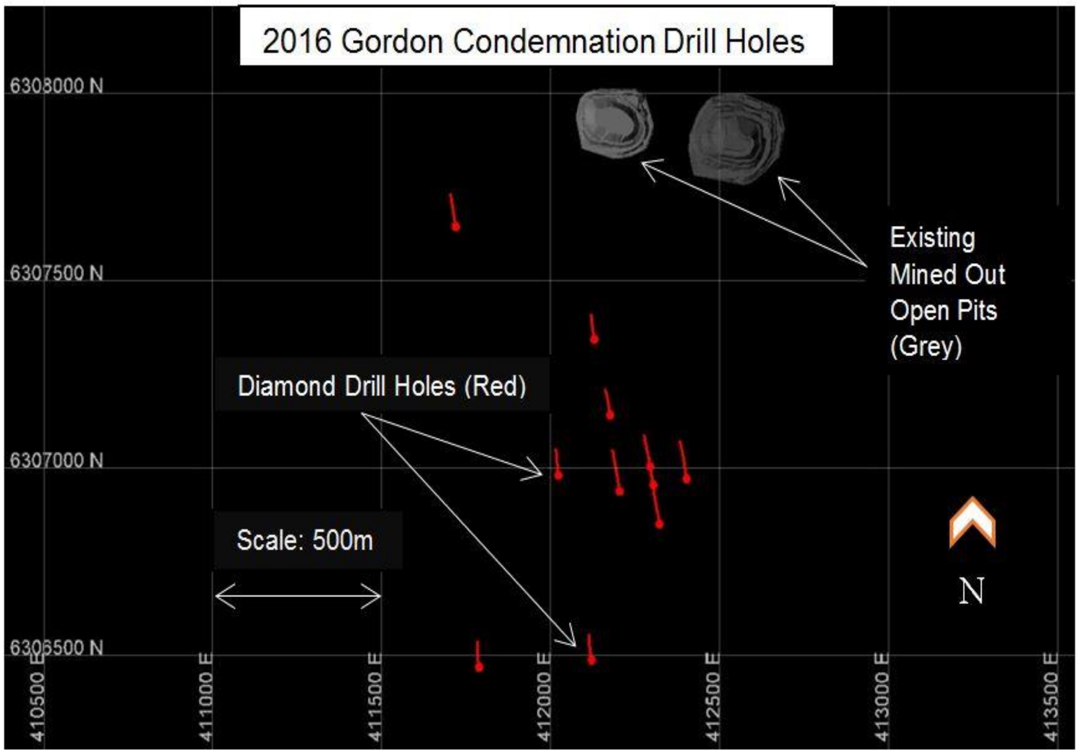


Figure 10-6: 2016 Condemnation Diamond Drill Holes at the Gordon Property

Source: Alamos (2017)

10.3 Drill Logs

The core was delivered by the drill contractor directly to the secured Alamos core shack in Lynn Lake twice per day. Core was then logged, photographed, and sampled by Alamos' geologists. Geologists recorded geotechnical data, lithological data, alteration, structure, and mineralization. Sulphide mineralization was recorded for individual percentages of pyrite, pyrrhotite, chalcopyrite, arsenopyrite, sphalerite, and galena. Core is then split using a diamond-blade saw with 1/2 of the core sent to the assay lab, and 1/2 of the core placed back in the core box to be stored at the core storage area at the MacLellan Property.

10.4 Drill Collar and Down Hole Surveys

Drill hole collars were marked in the field using a handheld Global Positioning System (GPS) and the drill was aligned using a Reflex APS tool. Drill hole collars were marked with a wooden stake and an aluminum tag indicating the hole inside diameter, direction, and dip once the drill has been moved off the drill site. Drill collars were surveyed by an independent surveying contractor prior to finalization of the database for Mineral Resource estimation purposes. Downhole surveying was conducted by project geologists using a Reflex Gyro downhole survey tool at 10 m intervals.

10.5 Core Size

All Mineral Resource drill holes undertaken by Carlisle and Alamos were drilled using NQ (47.6 mm) sized core tubes. PQ (85 mm) sized core was also utilized for metallurgical sampling.

10.6 Core Recovery

Drilling is carried out to meet industry standards with excellent core recovery at both properties. Core recovery at the Gordon Property averaged 95% and 97% at the MacLellan Property.

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Sample Preparation, Analyses and Security

Since 2011, drill core samples for the MacLellan and Gordon properties have been submitted to accredited commercial laboratories and assayed for gold using industry-standard fire assay techniques. The Mineral Resource database also includes drill core samples for two earlier periods of exploration. From 1945 to 1985, Sherritt and HBED drilled both properties and it is assumed that samples were assayed for gold by fire assay with a gravimetric finish. Additional drilling by Granduc and Black Hawk Mining in the 1990’s also generated samples where it is assumed fire assay was used for gold determinations. No details are available for sample preparation and assaying of the historical drill core samples.

All of Carlisle’s samples were submitted to TSL Laboratories (TSL) in Saskatoon, Saskatchewan. TSL is an ISO 9001:2000 and ISO 17025 accredited laboratory.

All of Alamos’ samples were submitted to ALS Global (ALS) for crushing and pulverization in Thunder Bay, Ontario. The prepared sample pulps were sent to the ALS laboratory in Vancouver, British Columbia for analysis.

11.2 Sample Lengths

During the exploration phase samples are generally taken on 1.5 m intervals in potentially mineralized zones, with some samples less than 1.5 m to preserve lithologic intervals. For orebody definition drilling, most sampling is on 1 m intervals broken around lithological changes as required.

11.3 Sample Preparation

Information on the historical methods used for sample preparation and gold assaying are not readily available. It is assumed that conventional crushing, pulverizing, and fire assaying techniques were carried out for core samples.

The sample preparation and assay methods since 2011 are summarized in Table 11-1. Both TSL and ALS crushed the entire sample to at last 90% passing 2 mm. Both laboratories pulverized samples to 95% passing 106 µm, although ALS used a 1000 g split and TSL used a 250 gram split.

No aspect of the sample preparation process was conducted by an employee, officer, director or associate of the Company.

Table 11-1 – Summary of Preparation and Assay Methods

Procedure	TSL Laboratories	ALS Global
Crushing	95% passing 2 mm	90% passing 2 mm
Pulverizing	250 g to 95% passing 106 microns	1000 g to 95% passing 106 microns
Gold Assay	30 g fire assay with inductively coupled plasma (ICP) finish and 0.005 g/t Au detection limit.	50 g fire assay with AAS finish and 0.01 g/t Au detection limit (method AA-26)
Over Limit Gold Assay	30 g fire assay with gravimetric finish for over 10 g/t Au and 0.01 g/t detection limit	Screened metallics for over 10 g/t Au
Screened Metallics	Screening at 106 microns and fire assaying of fine and coarse fractions	Screening at 100 microns and fire assaying of fine and coarse fractions
Multi-element	Silver only by AAS and 3:1 HCl + HNO_3 digest	Aqua regia digest with 37 elements (method ME-ICP41)

11.4 Analysis

Drill core samples used for Mineral Resource estimation have been assayed for gold by standard fire assay methods. A 50 g aliquot was used at ALS and a 30 g aliquot was used at TSL.

The fire assay method is described by ALS as:

> "A prepared sample is fused with a mixture of lead oxide, sodium carbonate, borax, silica and other reagents as required, inquarted with 6 mg of gold-free silver and then cupelled to yield a precious metal bead. The bead is digested in 0.5 mL dilute nitric acid in the microwave oven. 0.5 mL concentrated hydrochloric acid is then added and the bead is further digested in the microwave at a lower power setting. The digested solution is cooled, diluted to a total volume of 10 mL with de-mineralized water, and analyzed by atomic absorption spectroscopy against matrix-matched standards."

Screened metallic assays are requested when initial gold results are greater than 10 g/t.

11.5 Security

Core samples are transported by Gardewine Transport to ALS in Thunder Bay. Individual sample intervals are placed into rice bags with a bar code tag generated by ALS, and 7-10 samples are placed into a plastic bag sealed with a zip tie. A chain of custody procedure was strictly followed during transportation.

11.6 Quality Control and Quality Assurance

No information has been compiled that describes the quality control (QC) and quality assurance (QA) procedures and results for the pre-2011 drilling programs. The main form of QC/QA in the past would have been periodic re-assaying of anomalous samples.

Carlisle relied on the internal quality control procedures at TSL and did not submit QC materials to the laboratory.

Since 2015, Alamos has maintained a QC program for surface drill core that includes:

- Insertion of blanks and RMs with each batch of submitted samples;
- Creation of preparation duplicates; and
- Submission of pulps to a secondary laboratory for check assays.

For the 2015 drilling, a total of 281 core duplicates were collected at MacLellan and 219 core duplicates were collected at Gordon. The summary of the quarter core duplicates can be found in Table 11-2 for MacLellan and Table 11-3 for Gordon.

Table 11-2 – Summary of Core Duplicate Results MacLellan

Analyte	Method	Project	N	N>10 times detection limits	% of Duplicates >10 times detection limit within +/ – 20%
Au	AA26	MacLellan	281	86	19%
Ag	ICP41	MacLellan	281	28	46%
As	ICP41	MacLellan	281	98	58%
Fe	ICP41	MacLellan	281	225	91%
S	ICP41	MacLellan	281	281	99%

Table 11-3 – Summary of Core Duplicate Results Gordon

Analyte	Method	Project	N	N>10 times detection limit	% of Duplicates > times detection limit within +/ – 20%
Au	AA26	Gordon	219	31	42%
Ag	ICP41	Gordon	219	1	0%
As	ICP41	Gordon	219	78	56%
Fe	ICP41	Gordon	219	219	99%
S	ICP41	Gordon	219	112	77%

The gold deportment is different in the MacLellan and Gordon deposits which is reflected in the difference between the gold reproducibility in the core duplicates.

Core duplicates were not continued in 2016 as sufficient data had been collected to represent the uncertainty associated with drill core sampling.

The reference materials inserted with samples are purchased from a third-party supplier, OREAS. Quality control samples were inserted at a rate of approximately 4% which is consistent with industry standards.

The QC data was audited by Lynda Bloom of Analytical Solutions Ltd. (ASL). A summary of the 2015 and 2016 audit results is provided below.

11.7 Accuracy as Determined by Blanks and Reference Materials

11.7.1 Blanks

Barren coarse material (“a blank”) is submitted with samples for crushing and pulverizing to determine if there has been contamination or sample cross-contamination in preparation. Elevated values for blanks may also indicate sources of contamination in the fire assay procedure (contaminated reagents or crucibles) or sample solution carry-over.

A total of 1,084 blanks were inserted with samples. Two different coarse blank samples were inserted during the 2015-2016 drilling programs, LLK-BLK001 and LLK-BLK002. LLK-BLK-001 is OREAS 26a, a basalt blank chip certified for Au less than 1 ppb. LLK-BLK002 is described as a granite crushed to 10 mm, and certified for Au less than 5 ppb.

Blanks are determined to have failed when they assay more than ten times the detection limit of the analyte in question; for Au, any value over 100 ppb.

Using the defined failure criteria for Au, there was one QC failure for LLK-BLK001 and seven QC failures for LLK-BLK002. Six of eight cases have been identified as database errors where the sample that was analysed is a certified reference material and not a blank material. The two other cases may be either sample switches or a result of contamination from a previous sample.

There is a very low failure rate for blanks and no evidence of systematic gold contamination based on the blanks that were inserted with the samples.

11.7.2 Reference Materials

Reference Materials (RM) are submitted with samples for assay to identify:

- If there were assay problems with specific sample batches; and
- Possible long-term biases in the overall dataset.

The definition of a quality control failure is when:

- Assays for a RM are outside ±three standard deviations or 10%; and
- Assays for two consecutive RMs are outside ± two standard deviations.

The LLGP purchased OREAS certified reference material. OREAS RMs are well homogenized, assayed at a minimum of 15 recognized mineral testing laboratories, and are certified in accordance with International Standards Organization (ISO) recommendations. The certified reference materials (CRMs) are provided as pulps packaged individually for easy insertion into the sample stream. The materials selected represent a variety of sulphide bearing gold ores.

Five different certified reference materials were analyzed approximately 1,089 times in regular sequence with the samples. There are 108 cases where the results for gold were classified as a QC failure or as an Outlier.

Fifty-two cases identified as Outliers are cases where the RM was mis-identified in the database, and are not a reflection of laboratory performance. Outliers are excluded to assess the overall laboratory performance for accuracy.

Of the remaining 56 cases, there are 13 cases where two consecutive RMs are both outside ±2 standard deviations, there are 25 cases where the gold result is outside ±3 standard deviations and there are five cases where they are both.

The remaining 13 cases are Outliers, with percent of expected greater than 10%. It is recommended that Outliers or QC failures outside 10% be followed up by requesting that the laboratory repeat analyses for five cases before and after the QC failure.

Charts showing the performance for the three most commonly used RMs are provided in Figure 11-1, Figure 11-2 and Figure 11-3.

Table 11-4 – Summary of Reference Material Performance

RM	N	Outliers Excluded	Failures Excluded	Au ppm		Observed Au ppm		Percent of Accepted
				Accepted	Std. Dev.	Average	Std. Dev.	
OREAS 251	61	-	5	0.504	0.015	0.512	0.016	102%
OREAS 208	34	1	-	9.250	0.440	9.452	0.380	102%
OREAS 206	354	19	11	2.200	0.080	2.207	0.046	100%
OREAS 201	306	20	19	0.514	0.017	0.518	0.015	101%
OREAS 19a	236	8	15	5.490	0.100	5.516	0.103	100%
Total	991		Weighted Average					101%

Note:

1. *These calculations and all subsequent calculations and figures exclude mislabels and Outliers*

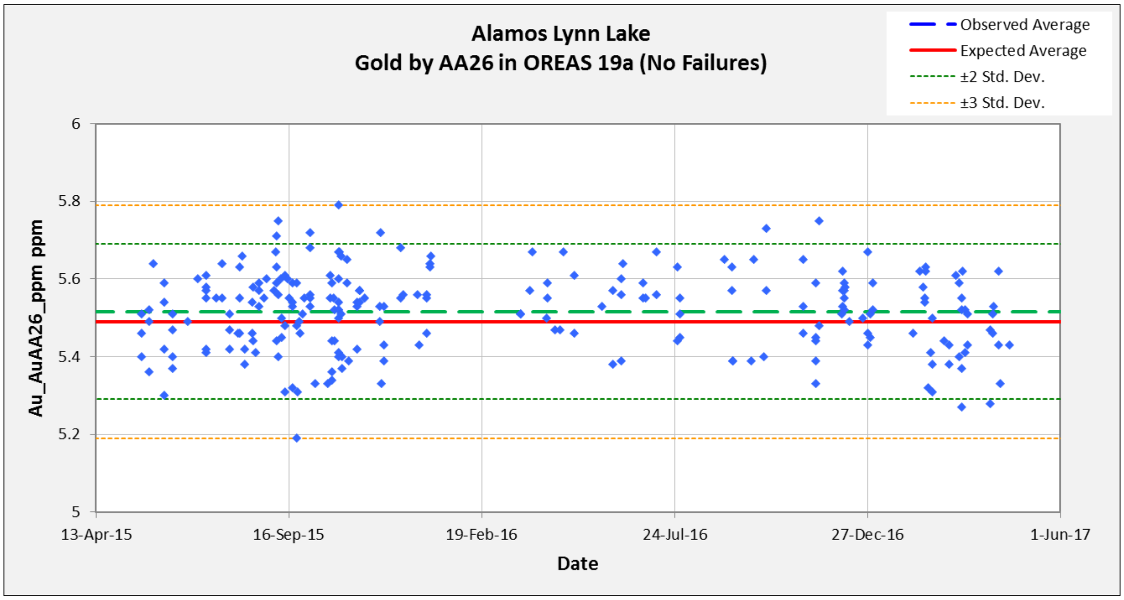


Figure 11-1: Performance Chart for RM OREAS19a

Source: Alamos (2017)

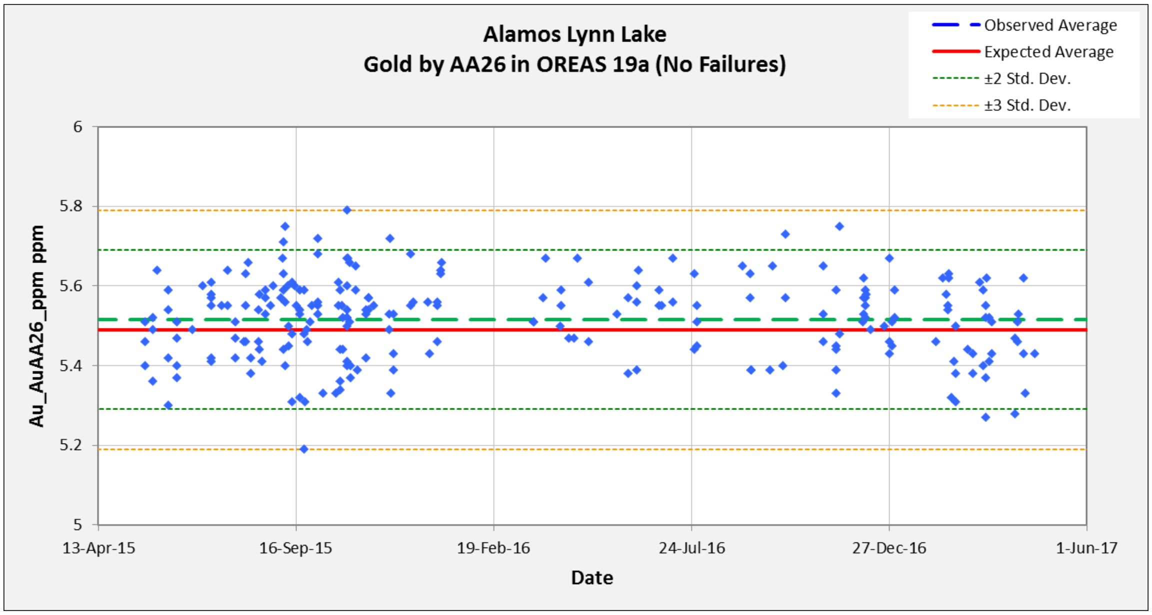


Figure 11-2: Performance Chart for OREAS201

Source: Alamos (2017)

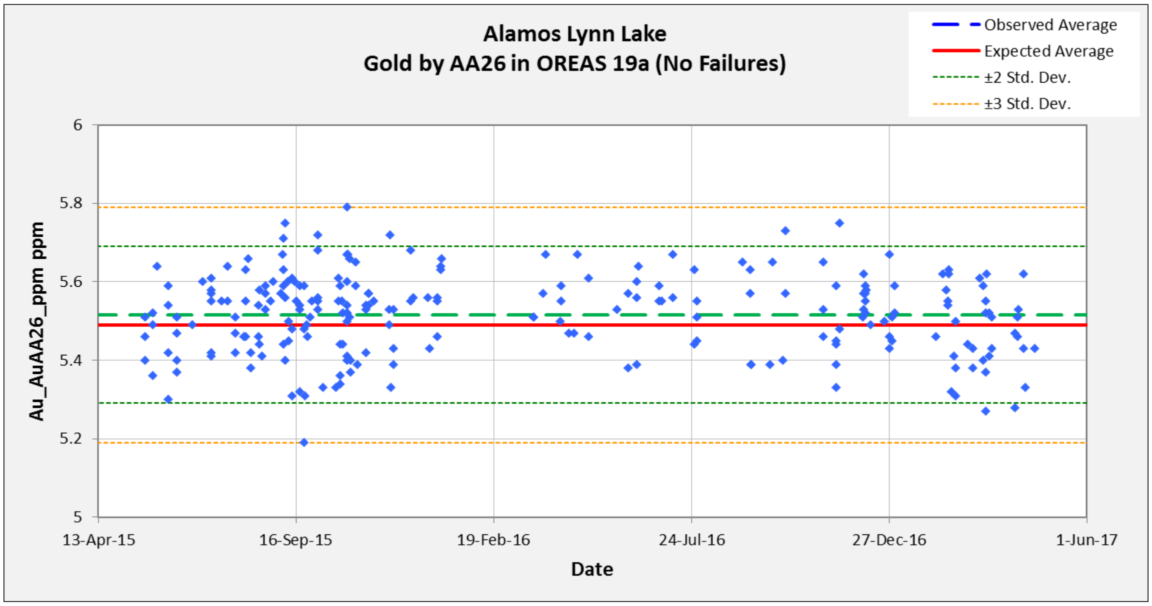


Figure 11-3: Performance Chart for OREAS206

Source: Alamos (2017)

Laboratory performance, based on reference material results, is acceptable and analytical data are considered useable for Mineral Resource estimation.

11.8 Laboratory Pulp Duplicates

Laboratories routinely assay a second aliquot of the sample pulp for their internal quality control monitoring. The assays for pulp duplicates provide an estimate of the reproducibility related to the uncertainties inherent in the analytical method and the homogeneity of the pulps.

The precision or relative percent difference calculated for the pulp duplicates indicates whether pulverizing specifications should be changed and/or whether alternative methods, such as screened metallics for gold, should be considered.

The original and duplicate assays for MacLellan are plotted in Figure 11-4. There were a total of 790 laboratory duplicate gold assays in the database. No lab duplicate analyses were provided for the Gordon Property.

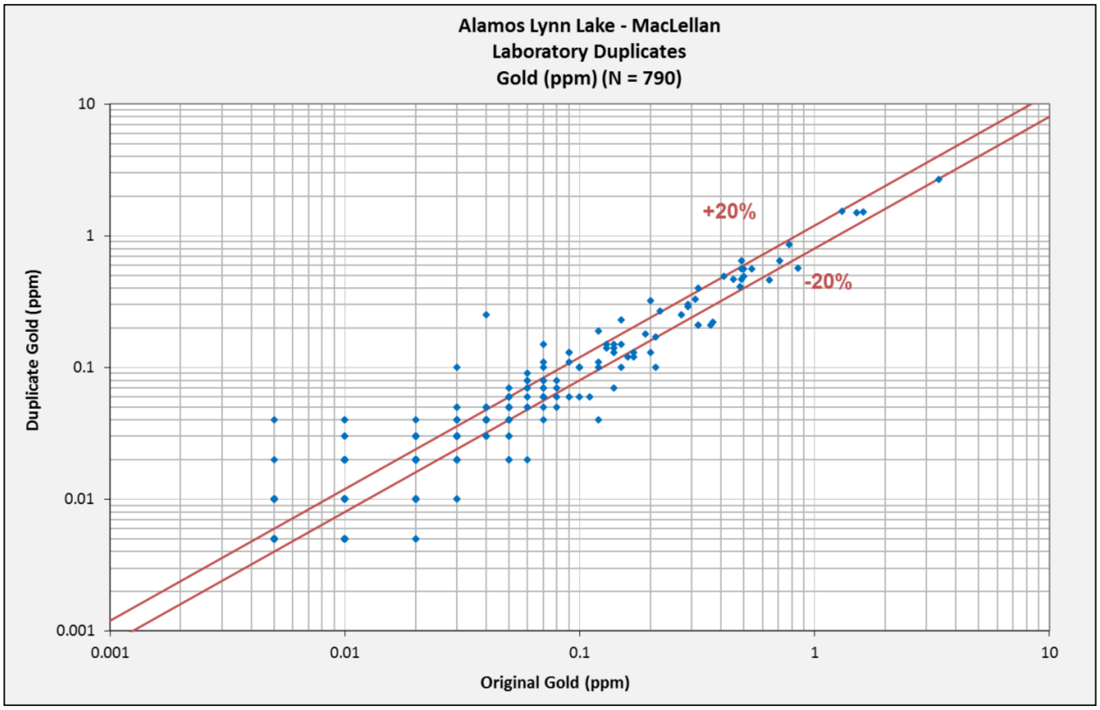


Figure 11-4: Laboratory Pulp Duplicates for Gold at MacLellan

Source: Alamos (2017)

The calculated precision for Au values greater than ten times detection limit (0.1 g/t Au) is 56%. It is suspected that the precision for pulps is relatively poor due to the presence of free gold.

Visible gold is occasionally observed in drill core logging which could explain the assay variability associated with pulp duplicates.

11.9 Screened Metallics Assays

Screen metallics analyses were conducted on 209 samples by ALS on 2015 samples. The gold assay protocol Au-SCR24 employs AAS analyses of two 50 g splits of the fine (<100 micron) fraction and a gravimetric analysis of the entire coarse (>100 micron) fraction.

From previous metallurgical testing, it was determined that gold grains at MacLellan average in size between 20-30 microns and are up to 150 microns. Gravity concentration recovers up to 42% of the gold. At Gordon, 95% of the gold grains are less than 10 microns and gravity concentration results in low gold recovery.

The percentage of gold in the coarse fraction of the screen metallics was calculated and plotted against the total grade (Figure 11-5). The plot shows that 70% of the fire assay screen metallics have greater than 10% of the gold reporting to the coarse fraction +150 mesh (~100 µm).

High grade samples (Au>10ppm) will be difficult to subsample due to the high proportion of gold in the coarse fraction and screened metallics was recommended by ASL.

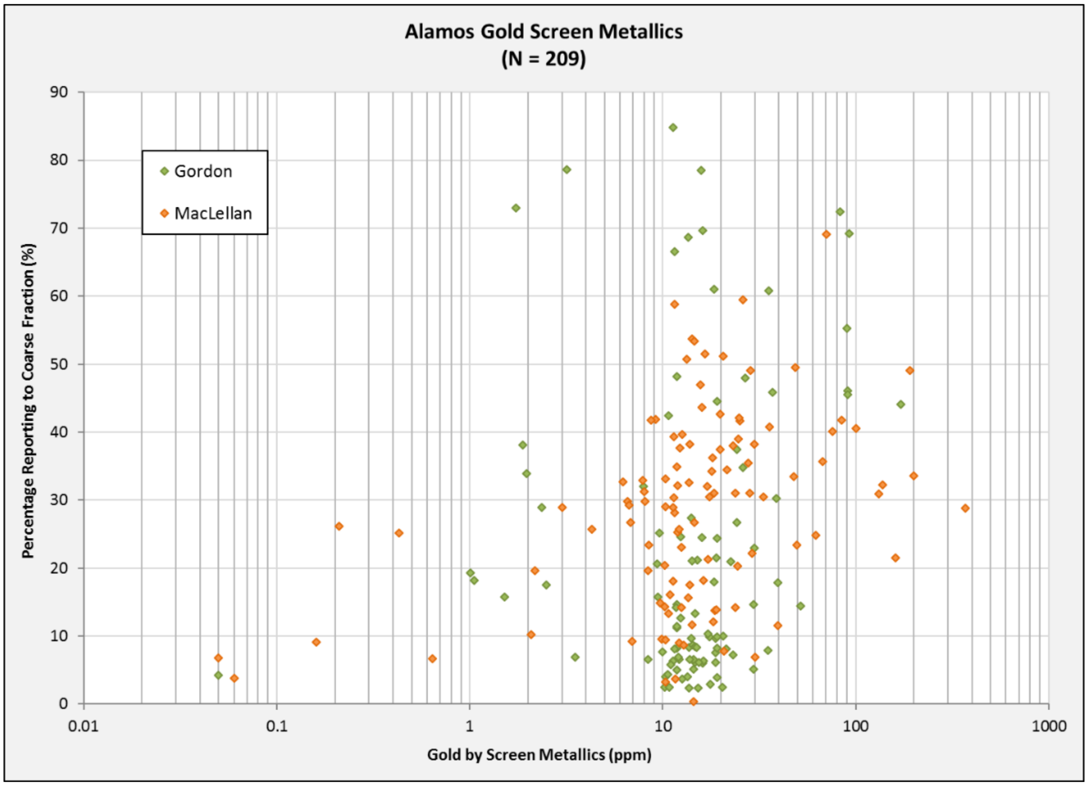


Figure 11-5: Percentage of Gold Reporting to Coarse Fraction

Source: Alamos (2017)

11.10 Check Assays at Secondary Laboratory

Check assays are used to augment the assessment of bias based on the RMs and in-house control samples. The same pulp that was assayed originally is submitted to a different laboratory for the same analytical procedures.

Check assays were completed by the company in 2016. A total of 495 sample pulps were submitted to TSL, Saskatoon for check assays in Q4 2016. TSL is an ISO17025 accredited laboratory.

TSL assayed the samples with fire assay – ICP finish for samples with less than 5 g/t and by fire assay with gravimetric finish for Au greater than 10 g/t. Samples between 5 g/t and 10 g/t were fire assayed with an AAS finish. For comparison, the primary lab, ALS, assayed all samples by fire assay with an AAS finish (AA26). Sample with Au greater than 10 g/t were also run through screen metallics.

The scatterplot in Figure 11-6 shows a reasonable correspondence between the ALS and TSL Au assays.

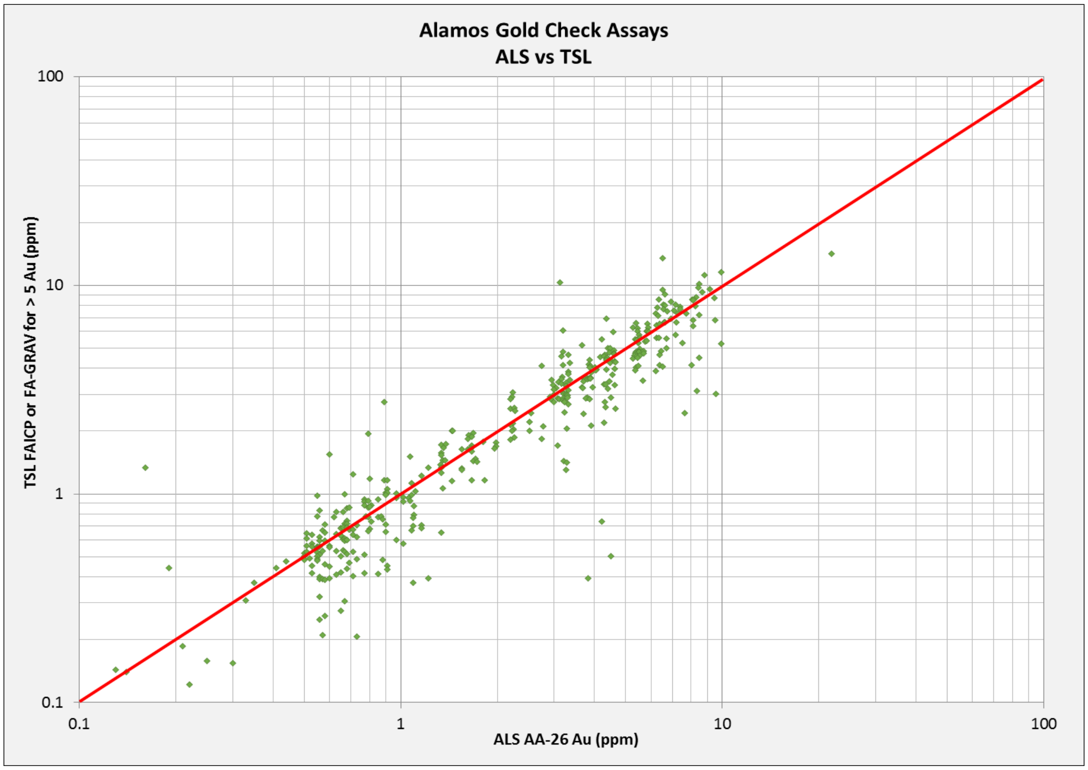


Figure 11-6: Scatterplot of Check Assays for Gold

Source: Alamos (2017)

For 58% of the samples, the ALS and TSL assays agree with ±25%. This is an acceptable range of repeatability given the deportment of gold (i.e. coarse gold particles). Previous studies of the amount of gold reporting to the coarse fraction (based on screen metallics data) showed that most samples have more than 10% of the gold reporting to the coarse fraction of the screen metallics assays.

For assays that do not agree within ±25% there are twice as many cases where ALS reports higher than TSL (N=108), than where ALS reports less that TSL (N=47). It is not clear if this represents a bias towards high Au values reported by ALS or is a symptom of poor reproducibility of the pulps.

11.11 Conclusion

The Company maintains a rigorous assay quality control for the LLGP. Blanks and reference materials are inserted with drill core samples on a routine basis. Results are reviewed when received and non-conformities are discussed with the commercial laboratory. In addition, sample pulps are routinely submitted for check assays to an accredited commercial laboratory.

There was no evidence of assay bias or systematic contamination identified based on the quality control program.

The responsible QP is of the opinion that the Project maintains a QC program that meets or exceeds industry standards. Sample preparation, security, and analytical procedures are all industry-standard and produce analytical results for gold with accuracy and precision that is suitable for Mineral Resource estimation.

12 DATA VERIFICATION

The following italicized sections are excerpted from previous P&E Mining Consultants Inc. (P&E) technical reports.

12.1 P&E Data Verifications – MacLellan

12.1.1 P&E 2006 Data Verification – MacLellan

The MacLellan Property was visited by Mr. Eugene Puritch, P. Eng. on June 19, 2006. A tour of the existing infrastructure, i.e. shaft and hoist buildings, stockpiles, ramp portal, and drill sites was undertaken subsequent to which ten assay verification samples were taken from seven drill holes. Sample intervals were taken from a variety of low, medium and high-grade mineralized material. The chosen intervals were then sampled by taking the remaining half-split core. The samples were then documented, bagged, and sealed with packing tape and taken by Mr. Puritch to the SGS Canada laboratory in Don Mills, Ontario for analysis.

At no time, prior to the time of sampling, were any employees or other associates of Carlisle Goldfields advised as to the location or identification of any of the sample intervals to be collected nor did they at any time have access to the sampled material A comparison table of the verification sampling results is shown below. The P&E results for Au were satisfactory and clearly demonstrate that the tenor of the mineralization is similar to what was originally reported.

It is the author's opinion that the previous sample preparation, security and analytical procedures appear to be satisfactory. (Puritch & Ewert, 2006)

Table 12-1 – MacLellan Check Assay Results from 2006 P&E Technical Report

Sample No.	DDH No.	From (ft)	To (ft)	Original Assay	P&E Assay	Bulk Density
				Au g/t	Au g/t	
B775347	MACCP92-6	24.0	27.0	0.61	0.51	3.03
B775348	MACCP92-6	31.0	34.0	5.44	2.88	3.08
B775349	MACCP92-1	28.0	31.0	0.80	1.73	3.01
B775350	MACCP92-5	41.0	44.0	5.08	1.14	2.85
B774901	MACCP92-4	27.0	30.0	16.01	6.92	2.95
B774902	MACCP92-8	26.0	29.0	0.91	0.40	2.96
B774903	MACCP92-8	16.0	19.0	1.80	0.75	3.00
B774904	MACCP92-7	16.0	19.0	1.57	1.35	2.90
B774905	MACCP92-6	18.0	21.0	0.26	0.07	3.06
B774906	MACCP92-2	44.0	48.0	4.45	0.86	3.03

Reference: P&E (2006)

12.1.2 P&E 2008 Data Verification – MacLellan

The MacLellan Property was visited by Mr. Eugene Puritch, P.Eng., on June 19, 2006, June 09, 2007, and again on June 02, 2008. A tour of the existing infrastructure, i.e. shaft and hoist buildings, stockpiles, ramp portal, and drill sites was undertaken. Eleven assay verification samples were taken from seven drill holes during the 2007 site visit. Sample intervals were *taken from a variety of low, medium and high-grade mineralized material. The chosen intervals were then sampled by taking the remaining half-split core. The samples were documented, bagged, and sealed with packing tape and taken by Mr. Puritch to SGS Mineral Services laboratory in Don Mills, Ontario for analysis.*

At no time, prior to the time of sampling, were any employees or other associates of Carlisle Goldfields advised as to the location or identification of any of the sample intervals to be collected nor did they at any time have access to the sampled material. A comparison table of the verification sampling results is presented below. The P&E results for Au were satisfactory and clearly demonstrate that the tenor of the mineralization is similar to what was originally reported, with the original results at times higher than the P&E results, and the P&E results at times higher than the Carlisle results. (Puritch, Ewert, & Karelse, 2008)

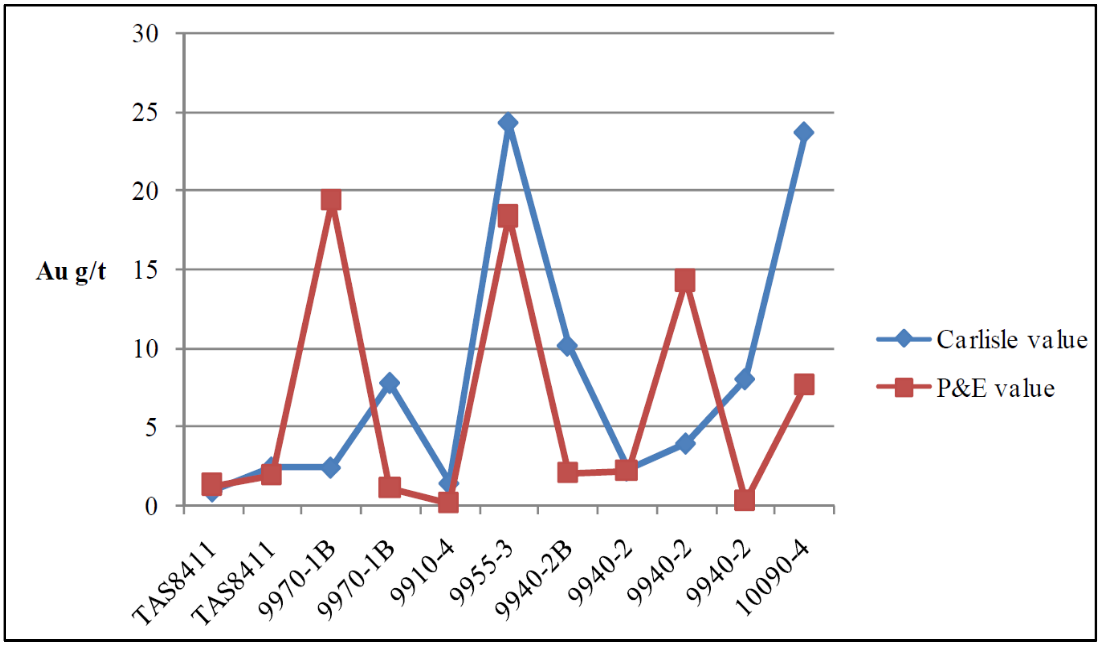


Figure 12-1: MacLellan Check Assay Results from 2008 P&E Technical Report

Reference: (Puritch, Ewert, & Karelse, 2008)

12.2 P&E 2013 Data Verification – Gordon

The Farley Property was visited by Mr. David Burga, P. Geo., from May 5 to 7, 2013 at which time he collected ten samples by quarter sawing the half core remaining in the core box from the 2012 drill program.

Samples were selected through a range of grades from high to low. At no time were any officers or employees of Carlisle advised as to the identification of samples to be select.

During the site visit, samples were tagged with unique sample numbers and bagged. Mr. Burga brought the samples back to P&E's office in Brampton, Ontario, where they were couriered to AGAT Laboratories in Mississauga.

AGAT is accredited by the Standards Council of Canada and conforms to the requirements of CAN-P-1579; Requirements for the Accreditation of Mineral Analysis Testing Laboratories. The latest certificate for proficiency was issued in June 2012.

Gold was analyzed using lead collection fire assay with ICP-OES finish. Graphs of gold values for samples taken during the site visit versus the original sample values can be seen in Figure 12-2.

Considering the site visit samples were quarter core (for the new drilling) and therefore weighed less than the original half core, (i.e. difference in sample volume) and considering the fact that core duplicates can't be expected to have excellent precision due to inherent geologic variability, the comparison between the original results and the P&E results demonstrates that the tenor for the two metals are similar (Puritch, Burga, & Wu, 2013).

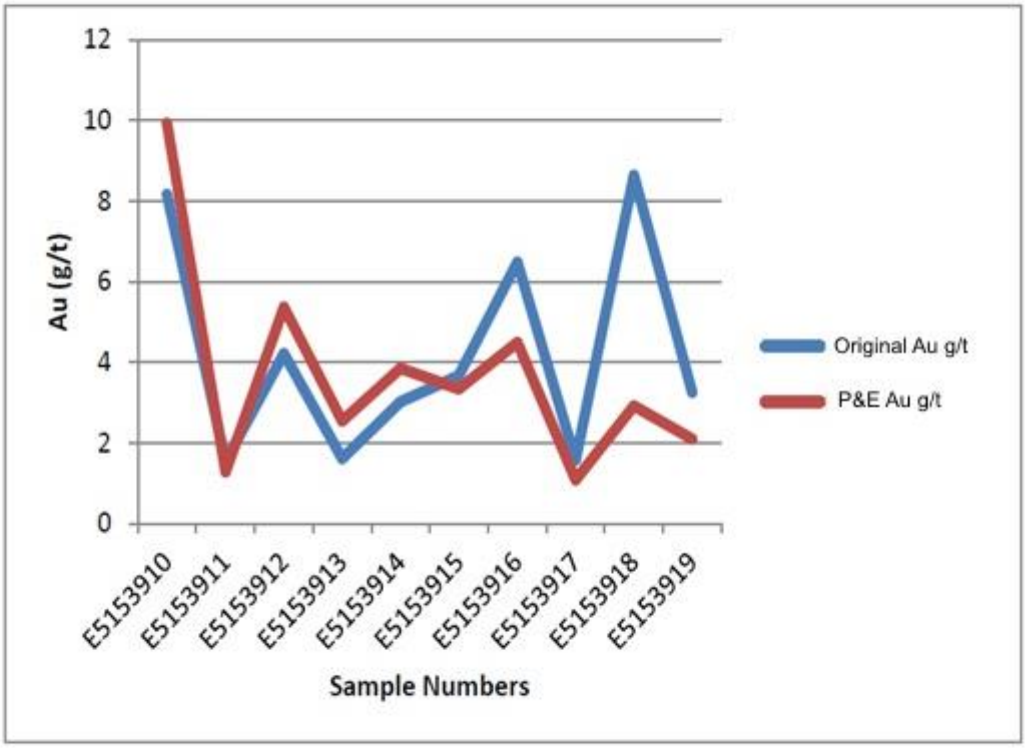


Figure 12-2: Gordon Check Assay Results for Gold from 2013 P&E Technical Report

Reference: (Puritch, Burga, & Wu, 2013)

12.3 Alamos 2016 Database Verification

Alamos geologic staff conducted a comparison between the Mineral Resource drill hole database and the original assay certificates from the Sherritt site laboratory for pre-2007 drill holes for MacLellan, and the 654 series of drill holes that were drilled by MMR during the period of February 1985 to April 1995.

12.3.1 MacLellan

The objective of the validation was to verify the gold assay results for at least 10% of the historic drilling at the MacLellan Property. The historic Mineral Resource drilling consists of several different generations of surface and underground drilling that took place prior to 2007.

The total number of samples validated was 6,767 samples out of a total number of 59,139 samples for the historic drilling on the MacLellan Property for a total of 11.44% of the samples.

In total 6,719 of the 6,767 samples were validated (99.29%) with 48 errors identified. There were also several holes that had data entry or conversion factor issues for the entire hole. A total of 25 holes were identified with errors. These errors were corrected in the current drill hole assay database.

Table 12-2 – Drill Hole Errors Identified – MacLellan Historic Assays

Grade Range (Au)	Total Au Errors	Au Higher Than Data Base	Au Lower Than Data Base
0-0.49 g/t	9	3	6
0.5-0.99 g/t	3	1	2
1.0-2.49 g/t	14	8	6
2.49-4.99 g/t	8	6	2
5.0-9.99 g/t	5	3	2
10+ g/t	9	2	7
Total	48	23	25

From the 99.29% validation rate, it can be concluded that there is a high degree of confidence that the gold results in the Alamos database accurately reflect the original gold assay certificates for the historic drilling at the MacLellan Property, and that these data are suitable for use in Mineral Resource estimation.

12.3.2 Gordon

The objective of the validation was to verify the gold assay results for 10% of the historic drilling at Gordon. The historic Mineral Resource drilling consists of the 654 series of diamond drill holes that were drilled by MMR between February 1985 and April 1995.

The total number of samples validated was 3,875 samples out of a total number of 38,750 (10% total) for the 654 series boreholes. The samples validated represent 5,176 m of sample out of a total of 50,300 m of drilling (10.29% total).

In total, 3,872 of the 3,875 samples were validated (99.9%) with only three errors identified. These errors were corrected in the current drill hole assay database.

From the 99.9% validation rate, it can be concluded that there is a high degree of confidence that the results in the current Alamos database accurately reflect the original assay certificates for the 654 series of diamond drill holes drilled by MMR, and that these data are suitable for use in Mineral Resource estimation.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Test Work Program

13.1.1 Historical Operation and Test Work

The MacLellan deposit was mined in the 1980s by SherrGold, a subsidiary of Sherritt. The Gordon deposit was mined in the 1990's by Black Hawk Mining.

The ore was processed in the Sherritt Lynn Lake Cu-Ni concentrator which was adapted for gold processing. A simple whole ore leach process was used. Historic operations reports and limited metallurgical testing reports were reviewed.

Approximately 1.0 Mt of MacLellan ore grading 5.3 g/t Au and 17 g/t Ag was processed, with average recoveries of 86% and 47% for gold and silver, respectively (SherrGold, 1986 to 1989). Approximately 1.7 Mt of Gordon ore was processed grading 4.2 g/t Au with an average recovery of 92% gold (Black Hawk Mining, 1996 to 1998).

MacLellan metallurgical reports (Lakefield, 1982) (Sherritt, 1985) (Sherritt, 1987) (Sherritt, 1987) (Sherritt, 1988) suggested the gold was closely associated with the sulphides and that some of the gold was either encapsulated or in solid solution with the pyrite, pyrrhotite and arsenopyrite. There was a notable negative relationship between arsenic head grade and gold recovery. The laboratory data also suggested that the ore may have been grind sensitive. Pre-production testing suggested that a recovery of 89% for gold should have been expected.

There were reports (Witteck, 1987) (Carre, 2014) that suggested that a high sulphide (pyrrhotite) zone of Gordon ore processed one summer caused extreme cyanide consumption which left the ore "cyanide starved" and resulted in poor gold recovery. The mill had no pre-aeration, and thus the likely oxidation of pyrrhotite during leaching was the cause of the problem.

Carlisle conducted two metallurgical test programs at SGS Vancouver in 2011 (SGS-Van, 2011) and 2012 (SGS-Van, 2013) on samples from the MacLellan deposit.

The first program was conducted in 2011 on 43 core samples composited together to represent the deposit and assayed 2.0 g/t Au, 3.6 g/t Ag, 2.1% S and 0.02% As. Mineralogical examination found that pyrrhotite was the predominant sulphide and that amphibole was the predominant mineral phase. Grindability tests characterized the sample as medium hard. Cyanidation tests recovered up to 96% of the Au and 52% of the Ag.

The second test program was conducted in 2012. Four samples were prepared: Composite (overall), Nisku, Main and West. Nisku was from the east end of the deposit, and Main from the central portion. The head assays ranged from 1.7 g/t to 4.7 g/t Au, 5.4 g/t to 27.5 g/t Ag and 0.17% to 0.75% As. The three variability samples had a similar response with gravity gold recovery from 40% to 47% and total gold recovery of 86% to 91%. Silver recovery varied from 37% to 67%.

The program also included cyanide destruction (CND) testing using the SO_2/air/Cu^{2+} process. The feed liquor had ~ 400 mg/L CN_{WAD} and ~ 1400 mg/L CN_T. The liquor was heavily fouled with Fe and SCN with minor levels of Cu and Zn. The CND testing was able to reduce CN_{WAD} to < 10 mg/L, but CN_T remained high.

CIP modelling was conducted. The modelling showed the circuit should achieve a gold loading on carbon of approximately 2000 g/t and require a carbon transfer rate of 7 t/d for 5,000 t/d mill throughput. Solution gold losses were nominally 0.01 mg/L Au.

Limited environmental testing was conducted. The acid base accounting (ABA) test found that the Composite sample had 1.8 times more neutralizing potential than sulphide sulphur in the ore; hence, this would be considered unlikely to produce acid.

13.1.2 Test Work for the Feasibility Study, from 2015

Alamos has conducted four phases of testing at SGS Lakefield since 2015 (SGS-LR, 2015) (SGS-LR, 2016) (SGS-LR, 2016) (SGS-LR, 2017).

The initial phase finalized flow sheet selection, optimized the main process parameters, determined comminution data and assessed metallurgical variability.

The second phase conducted on PQ core obtained comminution data and also included metallurgical tests on the samples.

The third phase addressed the process behaviour with internal dilution and tested additional samples from the far eastern end of the MacLellan deposit. Samples were treated similar to the variability samples. The program also repeated some large-scale cyanidation tests from the initial phase for environmental testing. These tests used an extended pre-aeration time which reduced total cyanide consumption. The sample was subjected to CND testing to lower $CN_{(T)}$ limits before undergoing environmental testing. Other gaps in the testing such as Hg levels through the process and final CIP modelling were addressed. Samples were also sent to Base Met Labs in Kamloops, British Columbia, to conduct oxygen uptake tests to determine pre-aeration time and oxygen addition rates for design.

The final phase of work assessed the response of samples from the initial three years of production. Samples in this phase of work were also sent to Jenike & Johanson for material flow properties testing.

13.2 Sample Selection

13.2.1 Guidelines for Sample Selection

Gold deportment and association in minerals and their impact on metallurgical performance were identified from a number of sources: historical operating records, technical and production reports, communications with former operations personnel, communications with geological personnel from previous owners and developers of the properties (Ounpuu, 2016).

The following were identified as key elements in selecting samples for characterisation and variability testing from each of the two deposits:

- At MacLellan:
    - Au versus As relationship and occurrence in arsenopyrite;
    - Au versus S (almost all as sulphide sulphur) relationship and occurrence in pyrite; and
    - Mineralogy and diagnostic leach testing which indicated sub-micron gold and/or solid solution gold in the sulphide minerals.

- At Gordon:
    - Mineralogy examination indicated sub-micron gold locked in silicates was the main source of un-recoverable gold from cyanide leaching.
- At MacLellan and Gordon:
    - High and variable presence of pyrrhotite which caused high to very high cyanide consumption and impacted leach performance (previous operations did not include or practice pre-aeration); and
    - Presence of high graphite in some areas which posed a potential risk to gold recovery due to preg-robbing (tested and threat subsequently discounted as this proved inert to cyanide leaching).

At Gordon, Carlisle geologists reported that the silica veining, dictated by weakness in the rock, allowed mineralized fluids to fill these fractured areas.

Specifically, no characteristic metallurgical response or performance was attributed to, or associated with, a geological feature or ore domain/type (e.g., lithology, alteration, and weathering). It was concluded that mineralisation in the two deposits was attributed to a pervasive silica/sulphide vein system and controlled by structural events at these deposits (Hastie, 2014).

As part of this review, recovery characteristics within domains or trends by domains proved inconclusive. No practical correlation or relationship of recovery with ore types, domains or zones was identified (Ounpuu, 2016).

The sample selection basis by ore grade and by spatial variability is considered appropriate for these deposits (Ausenco, 2016).

13.2.2 Drill Holes

Samples were taken from diamond drill core; no sample preparation or assay rejects were used.

- MacLellan:
    - Six diamond drill holes from Carlisle program (since 2008);
    - Along full strike length, at approximately 200 m intervals and spatially taken to represent roughly equivalent parts of the Mineral Resource;
    - Constraints were that some holes were near the wall of the proposed pit and in some cases contained mineralisation from below the pit wall; and
    - Mostly as ½ core, full core when available, minor amount ¼ core used.
- Gordon:
    - Six diamond drill holes from Carlisle program, in approximate grid pattern;
    - From both Gordon pits: Wendy and East;
    - All as ½ core;
    - Most holes in iron formation and two holes partially through intrusive rocks; and
    - One hole (to the northwest) contained “graphite”.

Drill hole distribution and sample selection guidelines for the deposits were considered appropriate.

13.2.3 Sample Preparation Protocols – Intervals

The 2014/2015 SGS laboratory test program was comprehensive in terms of samples selected and the extent of comminution and metallurgical testing. The interval selection basis for this program is summarized in Table 13-1.

Table 13-1 – Preparation and Interval Selection Summary for 2014/15 Test Program

Description	Units	MacLellan	Gordon
Mineral Resources (PEA, 2014)	Mine Plan (Mt)	18.6	8.4
	Gold Grade (g/t)	1.81	3.06
	Silver Grade (g/t)	3.04	-
Drill holes	DDH (number of)	6	6
Ore Grade (OG)	Total Length (m)	347	218
	Number of Intervals	20	16
	Gold Grade (g/t)	1.90	3.43
	Silver Grade (g/t)	5.0	0.7
Low Grade (LG)	Total Length (m)	140	67
	Number of Intervals	11	5
	Gold Grade (g/t)	0.45	0.53
	Silver Grade (g/t)	2.4	0.2
Benchmarks	ROM (Mt/DDH)	3.1	1.4
	Reserve (oz/interval)	35,000	39,000

The interval selection protocols were reviewed:

- About two-thirds of intervals selected were 15-30 m continuous; the remainder were 5-10 m continuous. This is considered suitable assuming a bench height of 10 m will be used in the mine plan;
- In MacLellan, most of the intervals came from drill core depth 100 to 450 m. Intervals from 25-100 m depth were less than 20% of the total used. This section is not as well-represented, particularly as ore from less than 100 m provides most of the ore to the plant in the initial two years of operation; and
- In Gordon, most of the intervals came from drill core depth 90 to 200 m. Intervals from 20-100 m depth were about 25% of the total used. This section is not as well-represented, particularly as ore from less than 90 m provides most of the ore to the plant in the initial two years of operation.

Although the sample distribution across the Mineral Resource is suitable for metallurgical investigations, ore from the surface to about 100 m is slightly under-represented.

13.2.4 Compositing

Sample compositing protocols were reviewed:

- Grades were consistent with the Mineral Reserve average;

- Total interval meters from MacLellan (487 m) and Gordon (285 m) were considered reasonable for the size of the deposits and nature of the mineralisation;
- Each drill hole represents 3.1 Mt of mined ore from MacLellan and 1.4 Mt from Gordon. This tonnage per hole is reasonable for the range of annual mining rates of ore from these deposits: 0.9-2.6 Mt/y for MacLellan and 1.0-1.6 Mt/y for Gordon which are blended to 2.6 Mt/y feed to the plant; and
- Each interval selected represents 35 000 oz for MacLellan and 39,000 oz for Gordon. These are considered reasonable based on average annual production of about 140,000 oz Au/y.

Sample compositing protocols were considered acceptable.

13.2.5 Classifications

Two grade-based classifications of samples were prepared from the intervals selected: master composites and variability composites for each of the deposits and a Global Master Composite. The number of representative samples in each of these classifications is shown in Table 13-2.

Table 13-2 – Number and Types of Samples for Metallurgical Program

Composite	Classification	MacLellan	Gordon	GLOBAL	Total
Master	Ore Grade (OG)	1	1	1	3
	Low Grade (LG)	1	1	0	2
	Total	2	2	1	5
Variability	Ore Grade (OG)	16	13	0	29
	Low Grade (LG)	8	4	0	12
	Total	24	17	0	41
Benchmark (for variability samples)	Mine Mt/sample	0.8	0.5	-	0.7
	oz/sample	45,000	49,000	-	47,000

The average head grades for each of these classifications are shown in Table 13-3. The ore grade master and variability composites compare well with the reserve average grades.

Table 13-3 – Grades of Samples for Metallurgical Program

Composite	Classification	MacLellan Grades			Gordon Grades		
		Au g/t	Ag g/t	S^{2-} %	Au g/t	Ag g/t	S^{2-} %
Master	Ore Grade (OG)	1.72	5.4	1.53	2.32	0.5	1.93
	Low Grade (LG)	0.36	1.6	1.16	0.47	0.4	0.64
Variability	Ore Grade (OG)	1.72	6.5	1.45	2.91	<0.8	2.15
	Low Grade (LG)	0.34	1.3	1.11	0.5	< 0.5	0.78
Reserves	Life of Mine	1.63	4.4	-	2.42	-	-

13.2.6 Composite Types – Master Composites

Master composites were prepared using an equivalent weight per unit length based on the overall length of the intervals selected for the sub-composites. These were used in the main metallurgical test program for flow sheet development, process optimisation, process criteria selection and design parameter identification for the comminution and metallurgical unit processes, cyanide detoxification, environmental, tailings geotechnical, slurry rheology and settling.

Four master composites by grade and one Mineral Resource-average Global Master Composite were considered reasonable for this deposit since there has been no evidence of dominant ore type or lithology-driven characteristic metallurgical responses. Grades are consistent with those of the Mineral Reserve grade.

13.2.7 Variability Composites

Variability samples were prepared from the intervals selected for the master composites into two main classes: ore grade (OG) and low grade (LG). Forty-one samples were prepared: 24 from MacLellan and 17 from Gordon. The variability samples were considered suitable as these represented 0.8 Mt and 0.5 Mt of mined ore, and 45,000 oz Au and 56,000 oz Au per sample, respectively. The average grades of the variability samples were consistent with the Mineral Reserve grades.

13.2.8 PQ Core Samples

Whole core PQ holes, four from MacLellan and two from Gordon, were used for comminution testing. These drill holes intersected mineralisation closer to the surface, through the heart of the deposits and along strike. Fifteen continuous intervals were prepared as composites for a suite of metallurgical tests including JK Drop Weight testing to provide calibration for the SMC testing.

Comminution competency and hardness characteristics were relatively consistent for the master and variability composites in both ore grade and low grade classifications.

13.2.9 Additional Variability Composites

As part of the third phase of the Feasibility Study test work, various gaps from the previous metallurgical programs were addressed. Samples of internal dilution between well-mineralized intervals were selected for testing. The intervals came from some of the existing holes previously selected for the initial phase of testing. Nine samples were selected from MacLellan and Gordon deposits. The geological team suggested that a sample from the Nisku end of the MacLellan deposit also be tested as this mineralization was considered different. DDH MG15-38 was selected and six samples between 20 to 300 m depth in this hole were prepared. Test work followed the same protocols for the variability samples.

The final phase of testing had four composites for each deposit, each which represented a sample from the initial three years of production and pre-production. A continuous interval from three holes was used for each yearly composite. The intervals were selected to cover spatial considerations for each yearly composite. A total of 16 drill holes provided samples for this phase of test work. Test work followed the same protocols for the variability samples.

Figure 13-1 and Figure 13-2 show the locations for the drill holes used in the test program for the MacLellan and Gordon deposits.

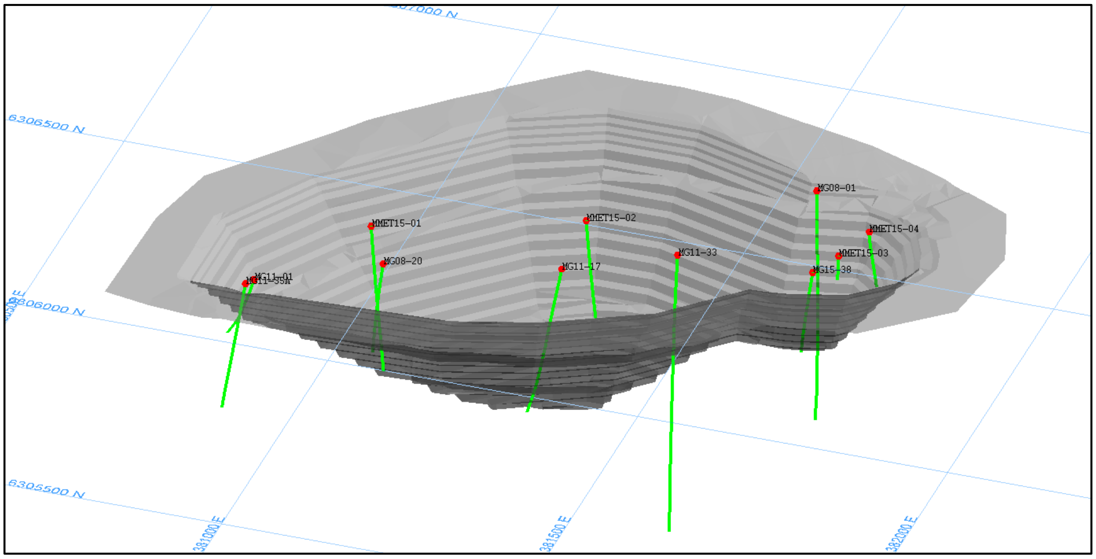


Figure 13-1: MacLellan Drill Hole Locations for Metallurgy Samples

Source: Alamos (2017)

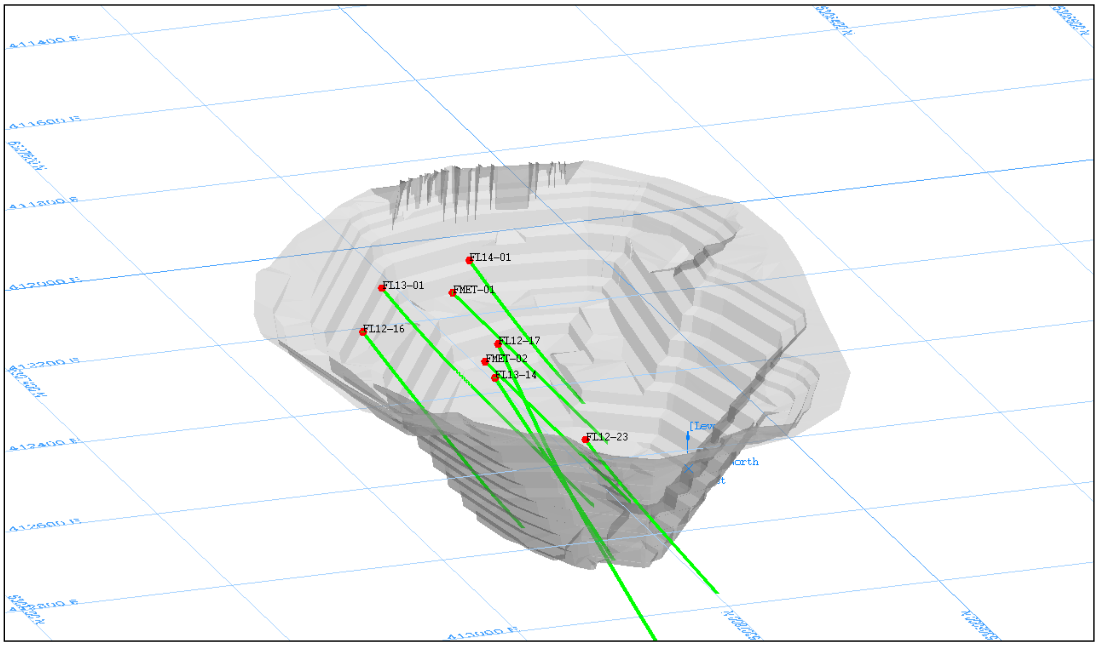


Figure 13-2: Gordon Drill Hole Locations for Metallurgy Samples

Source: Alamos (2017)

13.3 Mineralogy

13.3.1 Historic Mineralogy

Previous programs characterized the main gold deportment for the two deposits. At MacLellan, the gold was found in three metallic phases: native gold (< 20% Au), electrum (~ 35% Au) and kustellite (> 50% Au). Diagnostic leaching conducted at Sherritt in the 1980’s identified a refractory component in the ore. Analysis of test data suggested that gold was intimately associated with both arsenopyrite and pyrite, and possibly to some extent, in pyrrhotite (Ounpuu, 2014). The gold was generally fine grained with > 90% of the observed grains finer than 20 µm (SGS-Van, 2013). At Gordon, the gold was almost exclusively as native gold. The gold had an average particle size of 4 µm with < 10% of the observed particles greater than 10 µm (Witteck, 1987). These examinations focused on panned/upgraded gravity concentrates with limited examination of the non-gravity gold.

13.3.2 Mineralogy on Metallurgical Products

Two mineralogical examinations were conducted in the latest metallurgical test program; both supported the metallurgical test work. The first examination was conducted on flotation concentrates from both deposits then, expanded to include gravity concentrates. Flotation concentrates were examined since 93% of the gold was recovered to this product.

For MacLellan, 64% of the gold was as electrum and 36% as native gold. 87% of the gold was liberated or exposed. The gold grains had a maximum grain size of 12 µm (Mineralogical, 2015).

For Gordon, the gold was found almost exclusively as native gold. 98% of the gold was, almost equally, either liberated or as attachments to iron oxides (magnetite). There was minimal association with the sulphides. Almost all the gold was < 20 µm in equivalent diameter.

An examination made on an upgraded gravity concentrate from a MacLellan 10 kg test charge showed similar results. For Gordon, observations were similar.

Overall, the gold size distribution in the deposits was generally fine, with gold mostly < 20 µm. Examination of the total gravity concentrate from the 10 kg test found coarse gold (> 100 µm) in the deposits at a rate of 1 grain per 2 kg ore. These few coarser grains could account for 50% of the gravity gold recovered.

13.4 Head Assays

A total of 79 samples and five composites have been tested. All samples were submitted to characterize the sample by a full suite of assays which included:

- Au by screen metallic;
- Ag, Cu, As, Sb, Hg by direct assay;
- Carbon (C_{Total}, $C_{Graphitic}$, $C_{Organic}$, CO_2, CO_3)
- Sulphur speciation (S_{Total}, Sulphide Sulfur (S=), SO_4, Acid Insoluble SO_4, So)
- Pyrite and pyrrhotite assay; and
- ICP scan for a further 28 elemental analysis.

Key assays for the composites tested are shown in Table 13-4 (SGS-LR, 2015).

Observations from the head assay results:

- The samples tested had a range of gold assays from <0.1 g/t to > 25 g/t Au. Most of the samples were in the 0.5 g/t Au 3 g/t Au range;
- Ag:Au ratio in MacLellan is variable but generally approximately 4:1;

- Minimal Ag occurs in the Gordon deposit;
- Pyrrhotite is the predominant sulphide mineral for both deposits, with lesser pyrite and notable arsenopyrite in MacLellan;
- Almost all of the sulphur occurs as sulphide sulphur;
- Both deposits contain carbonate minerals, predominantly as calcite, but also as ankerite. MacLellan generally has more CO_3 than Gordon (~ 3% CO_3 versus 1% CO_3);
- Low levels of graphite and organic carbon were assayed for the Gordon deposit but test work did not show any signs of preg-robbing due to carbonaceous material. This confirms the early observation by Witteck on the Gordon deposit;
- Low levels of Cu, Zn and Ni in the deposits were assayed. These can contribute to the cyanide consumption; and
- Mercury levels are generally very low. A few samples from MacLellan had detectable levels of Hg. Further analysis of Hg levels on carbon and in leach liquors showed very low levels of cyanide-soluble mercury (SGS-LR, 2016). Correlation of elevated Hg with elevated Zn in samples indicated that Hg is likely to be associated with sphalerite.

Table 13-4 – Summary of Head Assays

Element	Unit	MacLellan Ore Grade	MacLellan Low Grade	Gordon Ore Grade	Gordon Low Grade	Global Master
		Master Composite	Master Composite	Master Composite	Master Composite	Composite[1]
Au (S.M.)	g/t	1.57	0.23	2.48	0.5	2.34
Ag	g/t	5.4	1.6	0.5	0.4	3
Cu	%	0.016	0.017	0.01	0.005	0.014
Zn	g/t	1100	314	63	87	617
Pb	g/t	485	52	< 20	5.3	257
Ni	g/t	417	515	< 30	47	227
As	%	0.16	0.013	< 0.001	0.002	0.093
Sb	%	0.002	-	<0.002	-	-
Hg	g/t	< 0.3	<0.3	<0.3	<0.3	<0.3
C_G	%	< 0.01	<0.01	0.11	0.23	0.02
C_{ORG}	%	0.14	0.09	0.09	0.3	0.08
CO_3	%	3.18	5.27	1.62	1.55	3.56
S_T	%	1.53	1.24	1.94	0.66	1.7
$S^=$	%	1.53	1.16	1.93	0.64	1.61
Py (FeS_2)	%	0.71	0.6	1.65	0.7	1.38
Po ($Fe_{12}S_{13}$)	%	2.97	2.4	2.73	0.75	2.46

Notes:

1. *Global Master Composite is from MCL-OG-MC ~ 58% and FL-OG-MC ~ 42%, by weight.*

13.5 Comminution

The objective of the comminution testing was to characterize the competency (coarser sizes) and hardness/grindability (fine sizes) of ore types from both deposits.

The testing for all samples (79) comprised the Bond Ball Mill Work Index test (BWI), the JK Drop Weight Test (DWT) and the Morrell SMC test. Samples from the PQ core series were subjected to a broader suite of tests including the Abrasion Index (AI), full DWT, BWI and the Bond Low Energy Impact Test (CWI). All tests were conducted by SGS, primarily at the Lakefield location. Table 13-5 summarizes the results for the comminution tests for the various phases of testing. Figure 13-3 and Figure 13-4 present the cumulative frequency plots of the JK Axb and BWI data.

Table 13-5 – Summary of Comminution Test Results by Phase of Testing

Sample Name		Relative Density			JK Parameters			Work Indices (kWh/t)			AI
		DWT	SMC	CWI	A x b[1]	t_a[1,2]	SCSE	CWI	RWI	BWI	(g)
Count		5	68	6	71	71	71	6	15	71	15
MCL-OG Average		-	2.96	-	28.2	0.25	12.7	-	-	14.9	-
MCL-LG Average		-	2.99	-	27.4	0.24	12.8	-	-	13.0	-
Gordon-OG Average		-	3.05	-	25.2	0.21	13.6	-	-	15.1	-
Gordon-LG Average		-	3.01	-	24.9	0.22	13.5	-	-	15.9	-
MCL PQ Average		2.95	2.93	2.97	29.4	0.26	12.4	13.2	16.6	15.3	0.191
Gordon PQ Average		3.18	3.04	3.06	32.9	0.28	11.9	14.4	20.0	16.4	0.359
MCL Internal Dil Average		-	2.90	-	30.0	0.27	12.2	-	-	13.0	-
Gordon Internal Dil Average		-	2.95	-	24.5	0.22	13.4	-	-	16.4	-
Nisko (MCL) Average		-	2.99	-	29.6	0.26	12.4	-	-	13.5	-
MCL Ave.	Early Production	-	2.95	-	28.3	0.25	12.5	-	-	13.8	-
Gordon Ave.	Early Production	-	3.08	-	25.3	0.21	13.7	-	-	14.5	-
Overall Average		3.04	3.00	3.00	28.0	0.25	12.9	13.6	18.6	14.8	0.292

Notes:

1. *A x b and t_a from DWT when available, otherwise from SMC*
2. *t_a value from SMC test is an estimate*

Test conditions and methodology were as follows:

- JK test results were interpreted and reported by JK Tech (JK Tech, 2016) (JK Tech, 2016) (JK Tech, 2017);
- SMC tests were conducted on the 19 mm to 22 mm size fraction as NQ core was used. Minimal re-calibration of the SMC results was required once the JK DWT results were available; and

- Bond tests were conducted at a closing size of 200 mesh (75 µm) which typically results in a P_{80} of about 63 µm. This screen size was selected because historical information for both the MacLellan and Gordon deposit suggested a grind size P_{80} less than 75 µm. As the selected grind size following the test work was P_{80} of 75 µm, the BWI data is considered suitable (slightly conservative) for design.

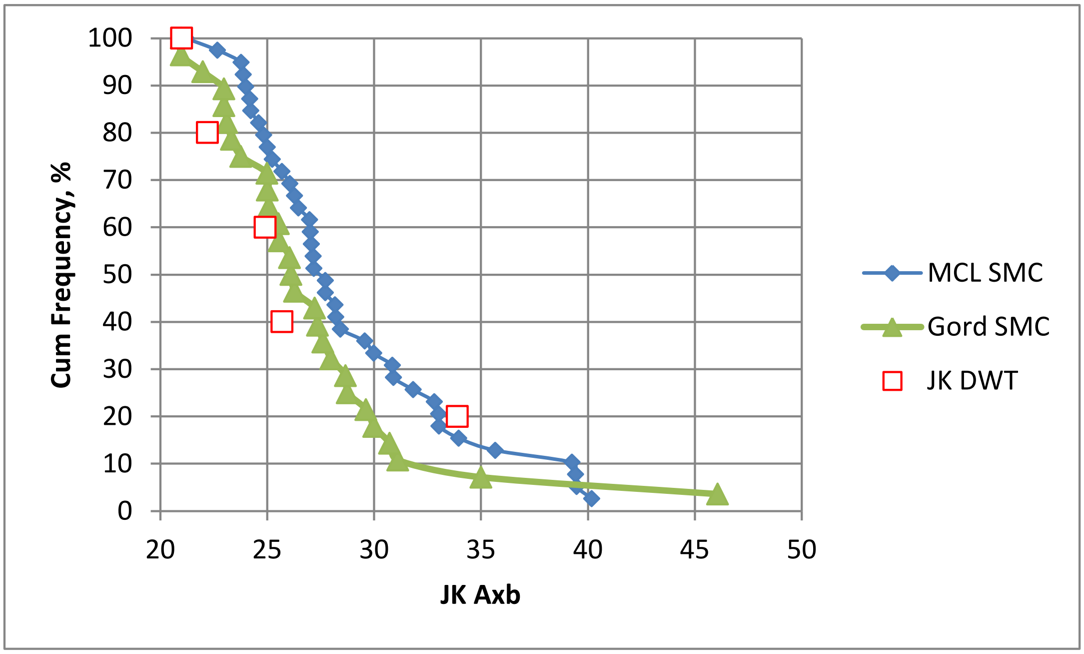


Figure 13-3: Cumulative Frequency Plot of JK Axb Data

Source: JK Tech (2016)

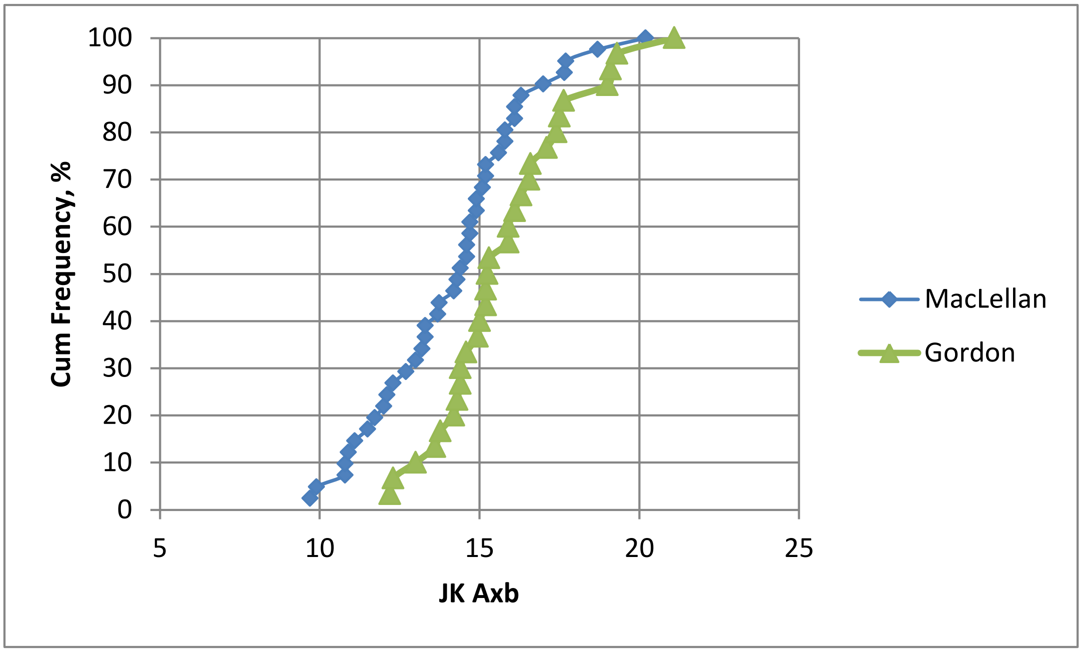


Figure 13-4: Cumulative Frequency Plot of BWI Data

Source: JK Tech (2016)

The results show a good level of consistency by phase of testing. Observations from the comminution testing are:

- The Gordon deposit has a lower average JK Axb parameter (more competent) and a higher average BWI (harder) than the MacLellan deposit;
- The deposits are of average hardness from a BWI perspective, but have a high to very high competency from an Axb perspective;
- The JK Axb data fell in a relatively tight distribution from 20 to 40. The BWI data had a broad range from 10 to 20 which needs to be taken into consideration for plant design;
- There is little difference in the results between ore grade, low grade and dilution samples from each deposit. These three classifications of ore have similar comminution characteristics for the Axb parameter and the BWI; and
- The Gordon deposit has two lithologies: intrusive rocks from the south end of the pit and iron formation throughout the rest of the pit. The intrusive rocks have a more competent Axb (22) compared to iron formation (26) and a higher (harder) BWI (16.4) compared to iron formation (15.1).

13.6 Gravity

13.6.1 Gravity Test Procedure

Most tests included gravity concentration as part of the process and flow sheet development. The procedure used these steps: grind the ore to the target grind size; single pass through a Knelson MD-3 laboratory concentrator; then upgrade to a low–mass, gravity concentrate on a Mozley C-800 Laboratory Mineral Separator. The mass recovery to the Mozley concentrate was targeted for ~ 0.1% by weight. No GRG type tests were conducted. Tests ranged from 1 kg test charges to 100 kg bulk tests for downstream metallurgical test work.

All tests showed some level of gravity recovery. Flakes greater than 100 µm were commonly observed. Because of the impact of this “nugget” effect, a gravity stage was included in the standard test procedure to reduce the impact and influence of nuggets due to variable leach performance with grain size, difficulty in reconciliation of assayed and calculated heads and the risk that test results would be masked due to a disproportionate increase in residue assay from nugget grains. In flotation testing, these coarser flake gold particles are unlikely to be amenable to efficient flotation. The nugget impact in an operating plant is significantly reduced due to the use of cyclones which essentially prevent any nuggets from passing to the leach circuit.

13.6.2 Gravity Testing

Results from the gravity test work are summarized in Table 13-6.

Table 13-6 – Summary of Gravity Gold Recovery Tests

Deposit	Sample	Test Size, wt.	Au Gravity Recovery, %
MacLellan	Initial, Flowsheet Dev.	~1 kg	35.2
	Initial, OG Variability	1 kg	32.3
	Initial, LG Variability	1 kg	39.4
Gordon	Initial, Flowsheet Dev.	~1 kg	24.1
	Initial, OG Variability	1 kg	37.3
	Initial, LG Variability	1 kg	30.9
MacLellan	OG Bulk	100 kg	52.7
	OG Bulk, repeat	80 kg	51.4
	LG Bulk	100 kg	49.9
Gordon	OG Bulk	100 kg	48.5
	OG Bulk, repeat	80 kg	53.0
	LG Bulk	60 kg	35.5
Global	OG Bulk	90 kg	49.9
MacLellan	PQ	10 kg	47.4
Gordon	PQ	10 kg	36.0
MacLellan	Internal Dilution	2 x 10 kg	44.5
Gordon	Internal Dilution	2 x 10 kg	27.6
MacLellan	Nisku	10 kg	67.7
MacLellan	Overall Test Average		46.7
Gordon	Overall Test Average		36.6
Overall	Overall Test Average		42.4

Observations from the gravity testing were:

- The average gravity gold recovery from the test work was 42%. This was not weighted by number of tests, test mass, head grade;
- MacLellan had a higher gravity recovery of 47% compared to Gordon at 37%; and
- The LG and Internal Dilution samples show a comparable level of gravity recoverable gold.

13.6.3 Mineralogy

A mineralogical examination conducted on a gravity concentrate (Mineralogical, 2015) from both deposits showed that most of the gold by occurrence was as relatively fine native gold or electrum particles (typically less than 50 µm). Even one or two larger gold grains account for a significant amount of the gold present in a gravity concentrate. A statistics assessment indicates that there may only be one larger grain of gold mineral in every 2 kg of ore.

13.7 Flotation

Limited flotation testing was conducted. The primary purpose of the testing was to assess if a grind, float, regrind and leach of the float concentrate and leach of the flotation tailings could achieve higher gold recovery than for whole ore cyanidation.

The results for these tests, compared with a (gravity + cyanidation) test, are given in Table 13-7.

Table 13-7 – Comparison of Flotation + Cyanidation with Gravity + Cyanidation

Sample	Feed		CN	Feed Size	Gravity Recovery,	Flotation Recovery,	CN Recovery,	Overall Recovery,	
			Test #	Micron	%	%	%	%	
MCL-OG-MC	ore		2	59	66.9		78.9		93
	F-3	Conc	7	26		93.1	91.9	85.6	91
		Tail	8	59		6.9	78.8	5.4	
FL-OG-MC	ore		5	55	24.1		95.2		96.4
	F-4	Conc	9	16		91.7	98.2	90	97.1
		Tail	10	55		8.3	85	7.1	

The results show little difference in overall recovery. The MacLellan sample shows 2% lower overall Au recovery while the Gordon sample had less than 1% increase in overall recovery when flotation was used. Since the flotation flow sheet showed minimal benefit in overall recovery, it was discontinued from further testing.

13.8 Leaching

13.8.1 Process Options with Leaching

The initial cyanidation test determined optimum process conditions and the standard flow sheet for the two deposits. The ore grade Master Composite for both deposits was the feed for these tests. The baseline conditions were:

- Grind P_{80} ~ 60 µm;
- 16 hour pre-aeration;
- 45% solids by weight;
- pH 10.5 with lime;
- 1 g/L of NaCN maintained; and
- 72 hour leach time with timed subsamples.

The grind size was adapted from the historical operations (80% to 85% passing 200 mesh). Pre-aeration was added to mitigate high cyanide consumption. The extended time, which allowed overnight pre-aeration prior to adding cyanide, was considered more than adequate. The other test conditions were considered consistent with industry standards for successful leaching.

The initial tests compared the following processes:

- Whole ore leaching;
- Gravity and CIL;
- Gravity and leaching; and
- Flotation, regrind of the concentrate, leaching of both concentrate and tailings.

The results from these tests are summarized in Table 13-8.

Table 13-8 – Summary of Initial Flowsheet Selection Test Work

				Recovery, %					
Ore Zone	**Sample**	**Test No's**	**Flowsheet**	**Gravity**		**Flotation**		**Overall**	
				Au	**Ag**	**Au**	**Ag**	**Au**	**Ag**
MacLellan	MCL-OG-MC	CN-1	Whole Ore Leach	…	…	…	…	89.9	66.0
		G-1 / CN-3	Gravity + CIL Leach	66.9	16.7	…	…	93.2	70.6
		G-1 / CN-2	Gravity + Leach	66.9	16.7	…	…	93.0	65.8
		F-3 / CN-7&8	Flotation + Leach (Reground Conc and Tail)	…	…	93.1	88.8	91.0	48.9
Gordon	FL-OG-MC	CN-4	Whole Ore Leach	…		…	…	97.4	43.1
		G-2 / CN-6	Gravity + CIL Leach	24.1	45.3	…	…	96.3	79.7
		G-2 / CN-5	Gravity + Leach	24.1	45.3	…	…	96.4	66.1
		F-4 / CN-9&10	Flotation + Leach (Reground Conc and Tail)	…	…	91.7	71.7	97.1	65.3

Observations from the test work were:

- No significant difference was observed between any of these processes;
- Although the flotation-based flow sheet is the most complex, it shows no significant increase in recovery compared to the other processes;
- CIL offers no benefit in recovery compared to whole ore leaching;
- Gravity shows little if any significant benefit even though the gravity recovery in separate program has been high at nominally 40% and represents 67% recovery in this test; and
- Both deposits show a similar response to different flow sheets. The MacLellan sample achieve an average of about 92% Au and 65% Ag recovery, whilst the Gordon sample achieved an average of 96% Au recovery.

13.8.2 Leaching and Grind Size

The next series of tests evaluated the effect of grind size. The two samples were subjected to different grinds to determine the effect on recovery and residue grade. The tests were a subset of a larger gravity test so that all leaching tests had a similar feed grade. The initial leach sequence tests were also included to assess consistency in response. The results are shown in Figure 13-5.

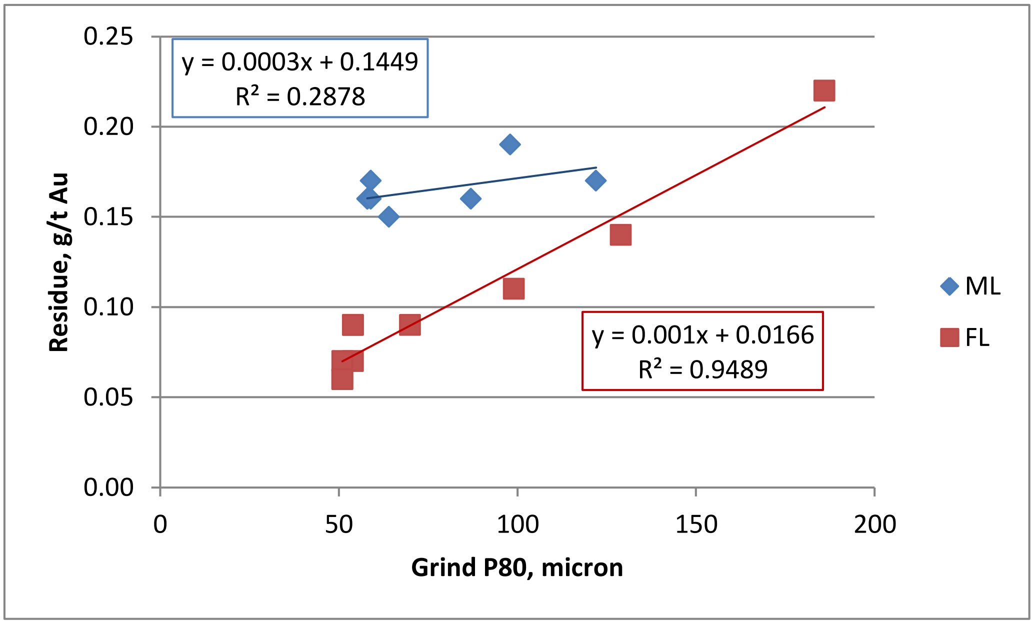


Figure 13-5: Effect of Grind on the Leach Residue Gold Assay

Source: Ounpuu (2015)

This shows that both deposits demonstrate leach effectiveness with grind size, and to varying extents:

- The MacLellan (MCL) grind-leach residue response was relatively minor with a linear slope of 0.0003 g/t Au for each one micron change in grind size; that is, corresponded to 0.01 g/t Au increase for 30 µm increase in grind size; and
- The Gordon (FL) sample has a more significant response to grind size. The linear slope was 0.001 g/t Au for a one micron change in grind; that is, 0.01 g/t Au change for 6 µm increase in grind size.

Sodium cyanide (NaCN) consumption was also influenced by grind, as shown in Figure 13-6. Finer grinds contributed to higher NaCN consumption.

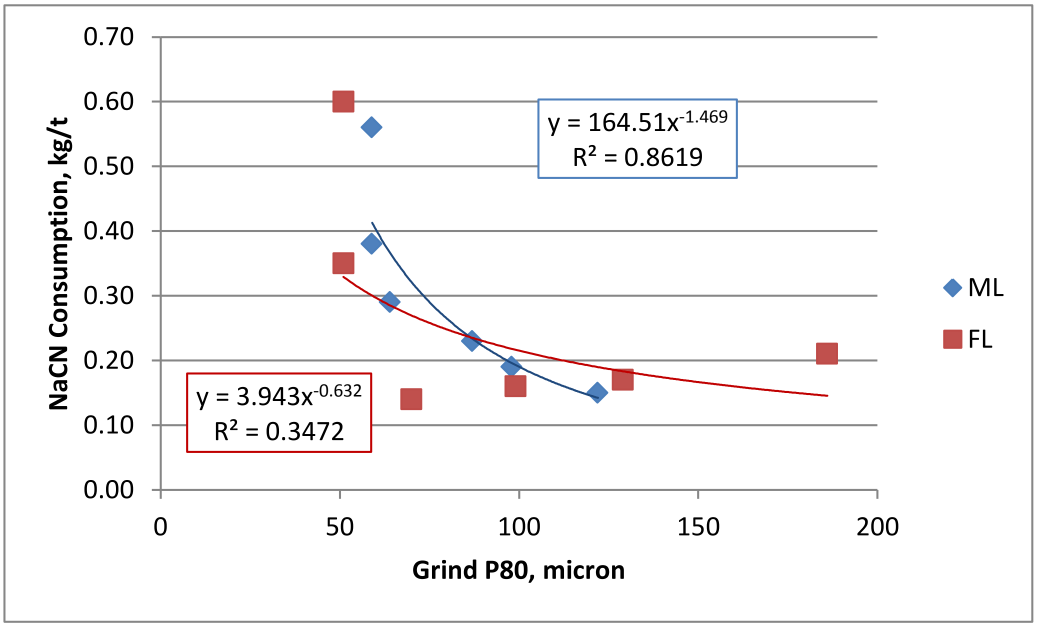


Figure 13-6: Effect of Grind on NaCN Consumption

Source: Ounpuu (2015)

13.8.3 Grind Size

To determine the optimum grind for these ores, a simple operating cost trade-off assessment was conducted in which the incremental cost of grinding finer was compared to the revenue gain from a lower residue assay (Ounpuu, 2015). The following assumptions were used:

- BWI of 15 kWh/t;
- Power cost of $0.025/kWh;
- Grinding media cost of $0.05/kWh;
- NaCN cost of $3.00/kg;
- Trend for impact of grind on residue assay in Figure 13-5;
- Trend for impact of grind on NaCN consumption in Figure 13-6; and
- Gold price = US $1,200/oz.

The results are shown in Figure 13-7 and are presented as the net change ($/t) from a base grind of P_{80} 150 µm.

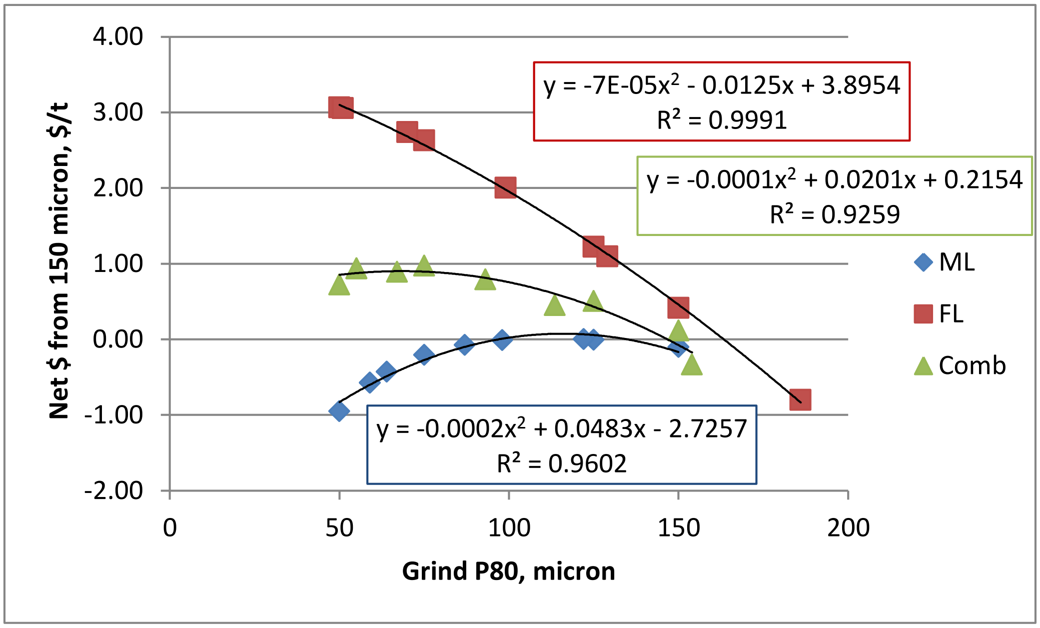


Figure 13-7: Net $/t Change From 150 Micron Grind

Source: Ounpuu (2015)

This shows that the Gordon ore will benefit from a fine grind, and possibly even finer than 50 µm. The MacLellan ore shows little impact with grind size between 75 µm and 150 µm; finer than 75 µm shows a negative benefit. For the target grind selection, a calculated blend in the expected weighted ratio of ores from the two pits was carried out to assess the best common grind size for the two deposits.

The trend for the combined ore blend is shown in Figure 13-7. This indicated that the optimum grind is about P_{80} 70 µm and that there is little net difference for a grind between 50 µm and 75 µm. Grinding coarser than this range would have a negative benefit. Figure 13-8 shows the impact of power cost on the net gain from a 150 µm grind for the blended ore sample.

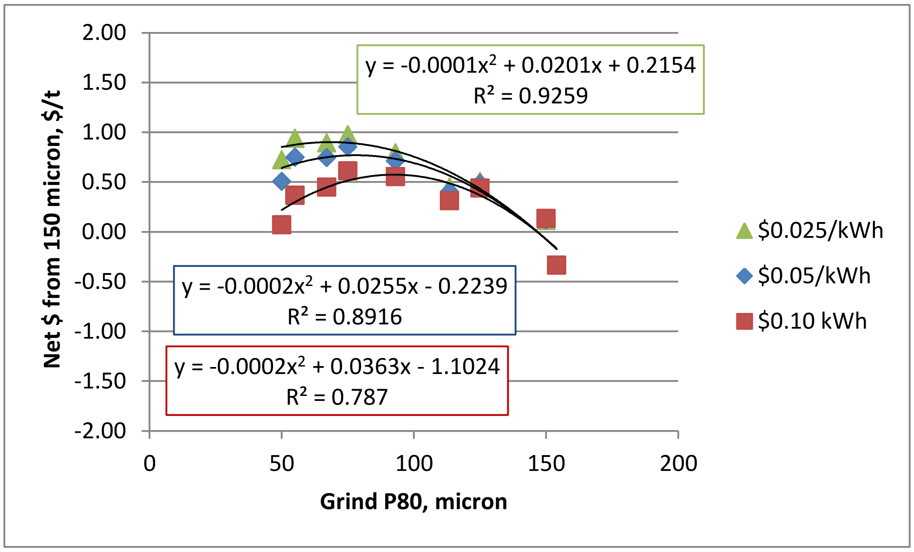


Figure 13-8: Effect of Power Cost on Optimum Grind

Source: Ounpuu (2015)

The impact from higher unit power costs was relatively small for coarser sizes and indicated that the gain from finer grinding the Gordon ore is the primary driver of the economics. At a power cost of $0.10/kWh, optimum grind is about 100 µm with generally little difference in net increment between 75 µm and 125 µm. At a power cost of $0.05/kWh, optimum grind is about 80 µm with little difference between 60 µm and 100 µm. A grind size of 75 µm was selected for the remainder of the test program.

13.8.4 Pre-Leach Treatment

The next series of tests assessed pre-aeration, addition of lead nitrate ($PbNO_3$), use of oxygen and effect of NaCN dose. These test results are summarized in Table 13-9.

Table 13-9 – Summary of Cyanidation Tests Reviewing Pre-aeration and Additives

Test	Gravity	Pre-Aeration h	NaCN g/L	Conditions	Grind P_{80} µm	NaCN kg/t	Overall Au Recovery %	Residue g/t Au
MacLellan								
CN-1		16	1.0		59	0.37	89.9	0.20
CN-2	yes	16	1.0		59	0.38	93.0	0.17
CN-13	yes	16	0.5		87	0.23	91.6	0.16
CN-19		0	0.5	$PbNO_3$	58	0.76	92.1	0.16
CN-20		0	0.5	no pre-aeration	58	1.15	90.2	0.19
CN-21		0	0.5	O_2 12-16 ppm	58	1.18	93.6	0.16
CN-25	yes	1	0.5		75	0.39	91.7	0.15
CN-26	yes	1	0.5, decay		75	0.29	92.4	0.15
Average							**91.8**	
Gordon								
CN-4		16	1.0		51	0.34	97.4	0.06
CN-5	yes	16	1.0		51	0.35	96.4	0.07
CN-18	yes	16	0.5		70	0.14	94.1	0.09
CN-22		0	0.5	$PbNO_3$	54	1.32	97.4	0.07
CN-23		0	0.5	no pre-aer`n	54	1.23	96.2	0.09
CN-24		0	0.5	O_2 12-16 ppm	54	1.05	96.5	0.09
CN-27	yes	1	0.5		75	0.27	95.8	0.10
CN-28	yes	1	0.5, decay		75	0.14	96.0	0.10
Average							**96.2**	

Observations from these tests showed:

- The tests achieved a similar level of gold extraction: on average MacLellan was 92% and Gordon was 96%;
- The Gordon tests showed that the coarser grind tests had higher residues and lower extractions, as expected; and
- Pre-aeration is beneficial. The two tests for each sample without any pre-aeration, but using $PbNO_3$ or oxygen during the leach, achieved a similar gold extraction as the tests with pre-aeration, but at considerably higher NaCN consumption.

The effect of pre-aeration on leach kinetics for ore from each deposit is shown in Figure 13-9.

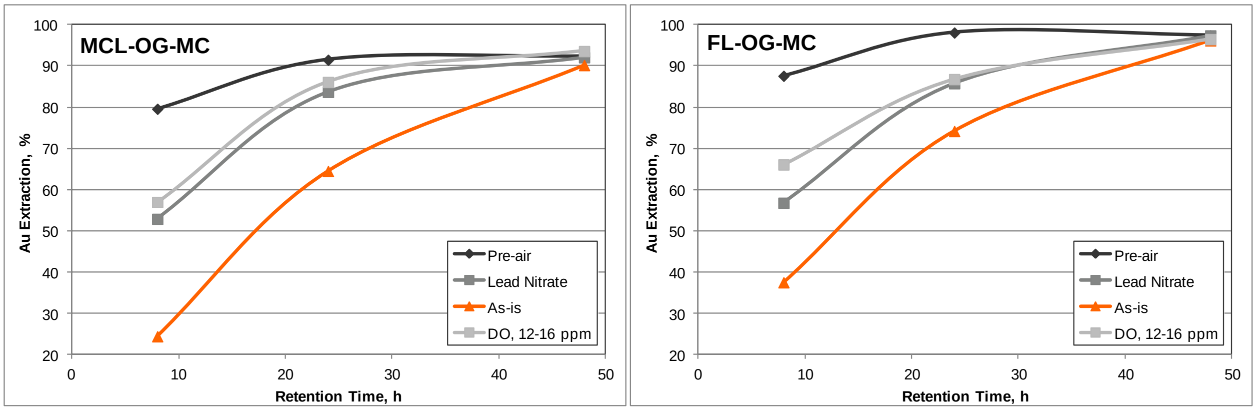


Figure 13-9: Gold Leach Kinetics, With and Without Pre-aeration

Source: SGS (2015)

The test with pre-aeration has the best kinetics with leaching completed within 24 hours while other tests with no pre-aeration required close to 48 hours to complete the leaching. The use of $PbNO_3$ or O_2 improved the kinetics over the base case (as-is) with no pre-aeration.

13.8.5 Oxygen Uptake

Oxygen uptake tests were conducted at Base Met Labs in Kamloops British Columbia (Base Met Labs, 2016). The basic procedure was to sparge O_2 into the pulp to a target of 15 ppm dissolved oxygen (DO) and periodically stop the sparging to measure the oxygen uptake by the pulp. The testing was conducted on the Ore Grade (OG) Master Composites (MC) from both MacLellan and Gordon, as well as a worst-case sample from each deposit (highest %S, pyrrhotite rich) and a high-pyrite, low-pyrrhotite sample from Gordon (FL-OG-11). Table 13-10 summarizes the samples tested. Figure 13-10 presents the results from these tests.

Table 13-10 – Characteristics of Samples Tested at BaseMet Labs

	% S^{2-}	Py % (FeS_2)	Po % ($Fe_{12}S_{13}$)	P_{80} µm
FL-OG-MC	1.93	1.65	2.73	65
FL-Comp 15	2.9	0.55	7.89	123
FL-OG-11	2.55	4.68	0.39	56
MCL-OG-MC	1.53	0.71	2.97	64
MCL-OG-5	2.65	0.64	5.95	74

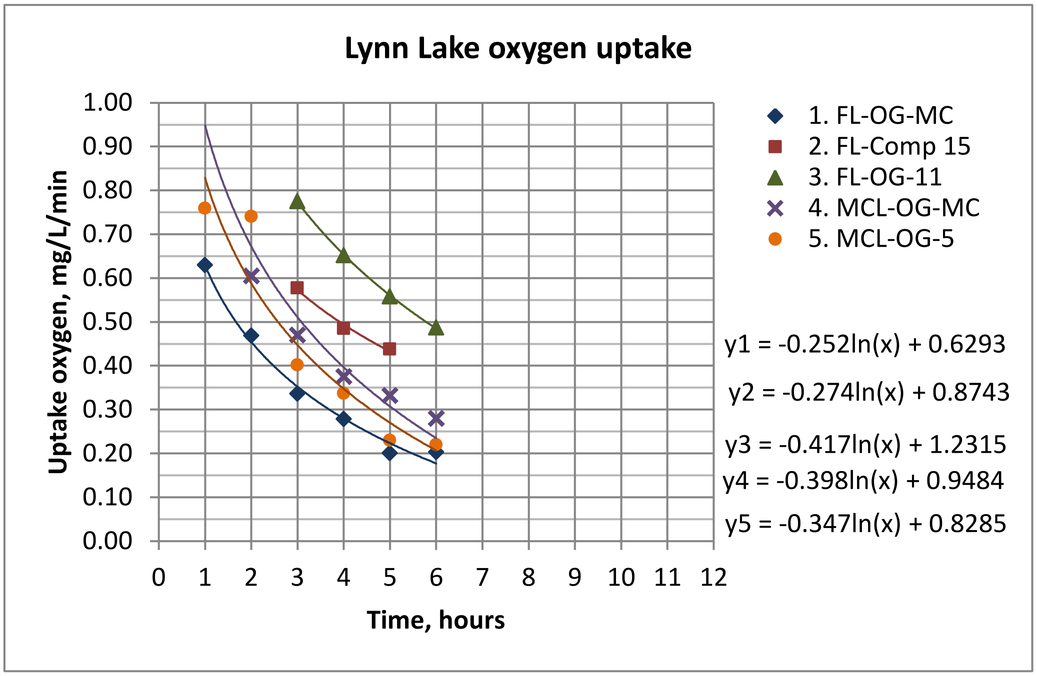


Figure 13-10: Oxygen Uptake Results from BaseMet Labs

Source: Base Met (2016)

The tests show the characteristic negative logarithmic decay in the oxygen uptake with time. Initial uptake values were high at >1 mg/L/min. The uptake stabilized at about six hours and the final 24 hour uptake was 0.14 mg/L/min.

13.8.6 Leaching of Variability Samples

All of the variability samples (79) were submitted for a standard batch test of gravity plus cyanidation of the gravity tailings. The base conditions for these tests were:

- 75 µm grind P_{80};
- Gravity concentrate upgraded to < 0.1% by weight;
- One hour pre-aeration at 1 L/min air for 1 kg test charge;
- 0.5 g/L NaCN through an initial eight hours, then decay;
- pH 10.5 with lime; and
- 48 hour leach time with timed samples, at 45% solids by weight.

The MacLellan ore grade variability composites gravity gold recovery averaged 32.4% and the gold extractions ranged from 44.2% to 97.0%, with an average overall gold recovery of 89.4% (16 composites). The average cyanide and lime consumptions were 0.32 kg/t NaCN and 0.48 kg/t calcium oxide (CaO), respectively. The MacLellan low grade variability composites gravity gold recoveries averaged 39.4% and gold extractions ranged from 64.8% to 90.8% (eight composites). Overall gold recovery averaged 86.0%. The average cyanide and lime consumptions were similar to the OG composites, averaging 0.35 kg/t NaCN and 0.50 kg/t CaO, respectively. Overall silver recoveries averaged 49.6% and 40.0% for the MCL-OG and MCL-LG composites, respectively.

The Gordon ore grade composites average gravity gold recovery was 37.3% and average gold extraction was 88.6%. The overall gold recoveries ranged from 84.9% to 97.6%, averaging 92.7% (13 composites). The average cyanide and lime consumptions were 0.33 kg/t NaCN and 0.70 kg/t CaO, respectively. The four Gordon low grade composites had an average gravity gold recovery of 30.9%, an average gold extraction of 81.9%, and an average overall gold recovery 88.0%. Average cyanide and lime consumptions were lower than the OG composites, 0.13 kg/t NaCN and 0.54 kg/t CaO, respectively.

The FL-LG-01 sample was flagged as from the northwest area of the Gordon pit with a lithology labelled as Graphitic Argillite and with an elevated graphite assay (0.72% C_g). The results were inferior for this sample and the residue higher than for the other Gordon-LG samples. It is not clear if this sample shows any genuine preg-robbing issues.

The results from the PQ Core samples, the internal dilution samples, the Nisku samples and the early production samples essentially confirmed the results from the initial variability results. Some issues were encountered with the PQ core testing. These tests were 10 kg tests and the pre-aeration was insufficient for some of the samples (low DO, high NaCN consumption, low residual CN_{free}). These tests were repeated with longer pre-aeration time which improved the results on two of the four samples. This problem was more prevalent for the Gordon samples. Two tests from the Gordon early production years were repeated due to coarse grind. Normal results were achieved at the correct grind size which further demonstrates that the Gordon deposit is grind sensitive.

Figure 13-11 and Figure 13-12 present the gold recovery as a function of the head assay for the MacLellan and Gordon samples. Both show a typical logarithmic trend of higher recoveries from higher head grades, with the recovery flattening out at average to high head grades. However, five samples for MacLellan and four samples for Gordon did not respond as well as would be expected based on these recovery-head grade relationships.

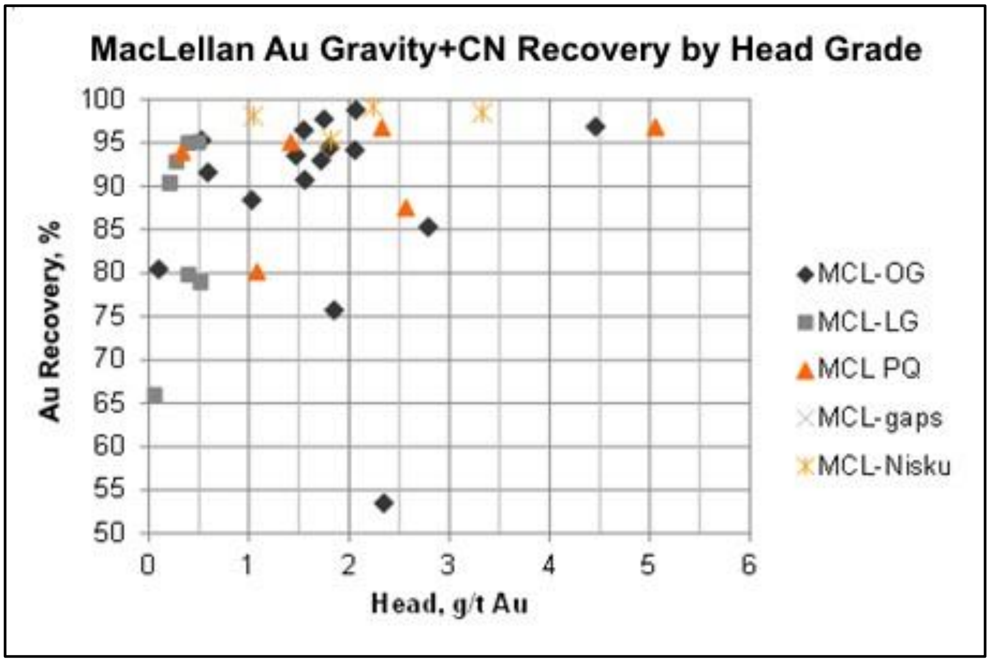


Figure 13-11: Overall Gold Recovery versus Calculated Head Grade – MacLellan

Source: SGS (2015)

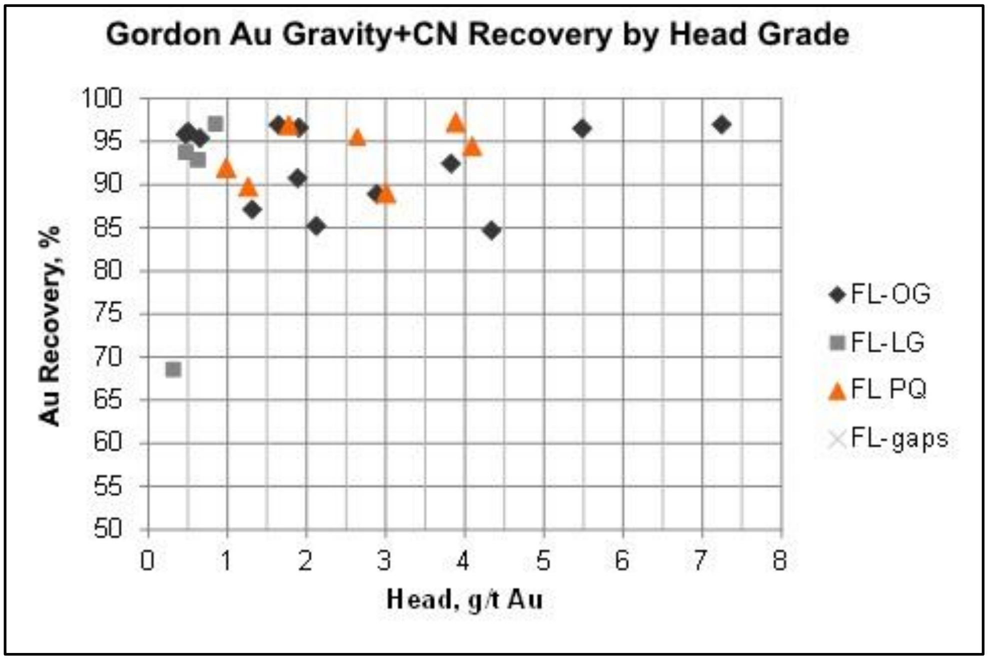


Figure 13-12: Overall Gold Recovery versus Calculated Head Grade – Gordon

Source: SGS (2015)

13.8.7 Mineralogy

One high gold residue sample from each of the Gordon and MacLellan testing was submitted for mineralogy to find out why the residue was higher than expected. The residue from the Gordon deposit was observed to have small (2 µm) native gold grains locked within larger silicate particles (Mineralogical, 2015). This finding supports the previous observation that the Gordon deposit is grind sensitive for gold recovery. The residue from the MacLellan deposit did not find very much gold, yet the residue was about 1 g/t Au. The examination was extended to laser ablation micro probe testing. This found that the arsenopyrite contained gold (Mineralogical, 2015). The ablation profile was in-homogenous which suggests that the gold in arsenopyrite is not as a solid solution but most likely as micro-encapsulation of gold grains in arsenopyrite. There was little gold associated with the pyrite and pyrrhotite. Most of the gold losses in this sample were due to the gold in the arsenopyrite.

A second sample from MacLellan was also submitted as a reference point. The sample had a similar arsenic head grade to the poor responding sample, but with a low residue gold assay. This sample was also found to have some gold associated with arsenopyrite, but at a much lower level. This indicates that gold is associated with the arsenopyrite to a variable degree.

13.9 CIP Modelling

CIP modelling was conducted by SGS to assist with the design of the CIP circuit and to confirm the plant will perform as expected. SGS has adopted the semi-empirical models developed by Mintek in the 1980s (Nicol and Fleming, 1980, Feb 1984, Mar 1984). For CIP modeling in the laboratory, the ore is first leached to completion, and the leached pulp is then treated in a batch reactor with activated carbon to extract the gold cyanide. The rate of extraction is determined by taking samples at timed intervals and analyzing them for gold. Values for the constants k (kinetic constant) and K (equilibrium constant) are then determined by the use of best fit parameters derived from a non-linear, least squares fit of the batch kinetic data.

A total of 18 simulations were made. With normal CIP conditions, target-final solution assay < 0.02 mg/L Au will be achieved with a six stage CIP circuit and carbon loading of 2000 g/t Au.

Final modelling was conducted in late 2016 when the plant throughput and base design was known (SGS-LR, 2016). The results showed acceptable circuit performance under the specified parameters. Final solution losses were 0.013 mg/L Au for the maximum Au and Ag grades scenario. The carbon loaded to 3260 g/t Au with one strip per day and 5.7 tonnes of carbon per day eluted. This required that 46% of the carbon per stage was advanced daily from the specified 25 g/L carbon density in the CIP tanks.

13.10 Cyanide Detoxification

CND design test work was conducted on Global Master Composite leached pulp (CN-77). The objective of the test work was to obtain optimized CND circuit conditions and produce treated pulp containing < 10 mg/L residual weak acid dissociable cyanide (CN_{WAD}) using the SO_2/air/Cu^{2+} detoxification process. The barren leach solution from test CN-77 was assayed prior to CND test work and the analysis is presented in Table 13-11 (SGS-LR, 2015).

Table 13-11 – CND Design Feed Analysis, Global Master Composite CN-77

Analysis, mg/L										
Au	Ag	Cu	Fe	CN_T	CN_{WAD}	CN_F[1]	CN_O	CN_S	CN_T (calc.)[2]	CN_{WAD} (calc.)[3]
< 0.01	< 0.08	12	36	161	61	30	7	250	162	42

Notes:

1. *CN_F (Calculated from CN final titration)*
2. *$CN_T = CN_{WAD} + Fe(2.8)$*
3. *$CN_{WAD} = CN_F = Cu$*

The Lynn Lake pulp responded well to the treatment using the SO_2/air/Cu^{2+} process. A series of batch tests were conducted to optimize the retention time, SO_2 addition rate, along with the hydrated lime and copper sulphate addition. The final test, CND1-4, was determined to be the optimized design and the reagent additions were 3.60 g equivalent SO_2, 2.45 g hydrated lime and 1.41 g Cu per gram CN_{WAD} in the feed to achieve the < 10 mg/L CN_{WAD} target.

The design test conditions were used as a starting point for the bulk CND environmental test work that was completed using each master composite sample. Continuous CND tests were conducted on the five Master Composite samples and a sixth test was added onto the Global Master Composite to produce a higher CN_{WAD} target of 30 mg/L for environmental ageing tests. A one hour residence time was used. The results from these tests are given in Table 13-12. All tests achieved their respective target CN_{WAD} level with reagent additions that matched the expected levels. The first three tests had elevated levels of CN_T (> 50 mg/L) and were subjected to a polishing stage whereby $CuSO_4$ is added to remove Fe and CN_T from solution. This was the action that was taken with the CND-4 test, as there was essentially no CN_{WAD} to remove. The polishing stage was successful in reducing both the Fe and CN_T levels.

It was noted that the bulk cyanidation tests which prepared the feed for the above tests did not get the appropriate degree of pre-aeration which resulted in elevated levels of CNS, Fe and hence CN_T in solution and low levels of CN_{free} at the end of the test. The MCL-OG and FL-OG samples were repeated through the cyanidation and CND testing to produce a nominal concentration of < 10 mg/L CN_T and < 1 mg/L CN_{WAD}. The cyanidation pre-aeration step used three times the aeration rate and twice the time used in the initial tests which was successful in reducing the Fe and CNS in solution and hence the overall cyanide consumption. The results for these CND tests are given in Table 13-13 (SGS-LR, 2016). The one hour residence time was adequate to achieve the target with the adjusted reagent additions. The product solutions were notably better than from the initial series of tests. The second set of tests required about double the SO_2 and lime addition per tonne of ore than the initial tests. The $CuSO_4$ addition was similar. The final additions used were ~1 kg/t SO_2, 0.45 kg/t lime, and 0.10 kg/t Cu added as $CuSO_4.5H_2O$.

Table 13-12 – Summary of Initial Continuous CND Tests

Sample	Test		Solution Analysis, mg/L			Reagent Addition, g/g CN_{WAD}			
			CN_T	CN_{WAD}	Cu	Fe	SO_2	Lime	Cu
Global Master Composite	CND-2	Feed	182	28	5.4	56			
		Product	119	0.02	10.8	57	3.25	0.07	5.2
MCL-OG-MC	CND-3	Feed	157	42	9.4	44			
		Product	51	5.6	20	35	3.6	0.54	2.8
FL-OG-MC	CND-4[1]	Feed	195	2	0.3	110			
		Product	0.2	<1	15.6	0.2	0	0	210
MCL-LG-MC	CND-5	Feed	189	49	10	36			
		Product	17	5	21.5	8	6.45	3.2	3.5
FL-LG-MC	CND-6	Feed	245	140	5	30			
		Product	8	4.3	14.6	4.3	3.0	1.3	0.65
Global Master Composite	CND-7	Feed	124	119	12	14			
		Product	36	15	37	15	2.4	0.94	0.09

Notes:

1. *CND-4 did not need CN WAD removal, but was treated with Cu to remove Fe and* CN_T

Table 13-13 – Summary of the Second Set of Continuous CND Test Results

Sample	Test		Solution Analysis, mg/L			Reagent Addition, g/g CN_{WAD}			
			CN_T	CN_{WAD}	Cu	Fe	SO_2 equiv.	Lime	Cu
MCL-OG-MC	CND-1	Feed	150	125	24	8			
		Product	0.36	0.13	0.8	< 0.1	7.6	3.9	0.31
FL-OG-MC	CND-2	Feed	162	94	6	16			
		Product	0.24	< 0.1	0.5	0.2	8.3	2.4	0.88

13.11 Materials Handling

13.11.1 Ore Flow Characteristics

Samples from both the MacLellan and Gordon deposits were sent to Jenike and Johanson for ore flow characterization (Jenike & Johanson, 2017). The samples were rejects from the PQ core shipped to SGS. The testing found:

- Both samples were cohesive and will need to be handled in mass flow;
- The samples were both sensitive to impact pressure and it is recommended that a low drop height be used;
- The angle of repose was 35º to 45º;
- The draw down angle was 60º to 65º ;
- The Comp 4 MacLellan sample, fines with 8% moisture content was the worst case for flowability. The higher moisture content (10%) showed less friction (i.e. better flow), so there are no concerns with rain/snow melt causing flowability issues; and
- The results are considered relatively consistent with other gold ores.

13.11.2 Settling and Thickening

The Global Master Composite was submitted for a complete solid/liquid separation and rheology test program. The sample was ground to a P_{80} of 80 µm and pH adjusted to 10.6 with lime prior to testing, to represent a pre-leach thickener sample.

The results of the flocculant scoping and static settling tests indicated that the Global Master Composite pre-leach sample responded well to BASF Magnafloc 333 flocculant, a very high molecular weight non-ionic polyacrylamide flocculant.

The optimized dynamic thickening conditions of the Global Master Composite pre-leach sample are summarized in Table 13-14 (SGS-LR, 2015). These results were produced at 10% w/w feedwell density at a dosage of 25 g/t BASF Magnafloc 333 flocculant and 1.14 hour residence time (dry equivalent volume of feed solids versus underflow volume).

The overflow total suspended solids (TSS) was 49 mg/L and the underflow solids content averaged 60.5% w/w solids under these conditions. A thirty-minute period of extended underflow thickening, without feed, resulted in increased underflow solids density to 66.5% w/w solids. This corresponded to an increased underflow yield stress of 103 Pa versus the pre-extended thickening yield stress of 21 Pa. The results are considered typical for a gold ore.

Table 13-14 – Dynamic Thickening Results Summary – Optimum Conditions

Description[1,2]	Units	Value
Dosage Flocculant	g/t	25
Undiluted Feed[3]	% wt.	44
Diluted Feed[4]	% wt.	10
U/F[5]	% wt.	60.4
UF[6] Extend	% wt.	66.5
TUFUA[7]	m²/t/day	0.1
THUA[8]	m²/t/day	0.007
Net Rise Rate	m³/m²/day	88.2
Solids Loading	t/m²/h	0.42
Net Hydraulic Loading	m³/m²/day	3.67
Res. Time Solid vs. UF	h	1.14
Overflow Visual		Hazy
TSS[9]	mg/L	49

Notes:

1. *All values were calculated without a safety factor*
2. *Common conditions: Flocculant: BASF Maganfloc 333 flocculant. Internal feed dilution using recycled overflow, liquor density: ~1000 g/L, underflow raking, ambient temperature.*
3. *Sample Density Prior to Thickener Feed Dilution (note: this density may not represent the actual discharge density of the preceding process stream).*
4. *Autodiluted Thickener Feed Density*
5. *Underflow (UF) Density*
6. *Underflow (UF) Density after 30 minutes of extended thickening (raked, no feed).*
7. *Thickener Underflow Unit Area*
8. *Thickener Hydraulic Unit Area*
9. *Total Suspended Solids (TSS) contained in the overflow.*

13.11.3 Underflow Rheology

The UF from the above testing was submitted for rheological characterization.

The UF sample displayed insignificant inter-particle interactions meaning that the dry solids specific gravity was comparable to their densities in the slurry phase. The rheology test measurement data allowed for Bingham modelling and subsequent interpretation, particularly with respect to the solids density rheological profile. The critical solids density (CSD) was ~65.5% w/w solids, displaying a yield stress of 25 Pa under unsheared flow condition and 7 Pa under sheared conditions.

A certain degree of thixotropic tendency was displayed by the sample at, or above, a density of 61.1% w/w solids. Thixotropic response is a “flow-friendly” behaviour whereby the resistance to flow decreases due to constant shearing. These results are considered typical for a hard rock gold ore. The expected leach feed of 55 %w/w solids should present no issues for settling or rheology based on the above data.

13.12 Environmental Testing

An environmental testing program was carried out using samples from the metallurgical test program to assess the geochemical, acid rock drainage (ARD), and contaminant release potential associated with the samples over time. The acute effects of aged process water on freshwater aquatic species were also investigated.

The environmental testing and the outcomes from this work are discussed in relevant parts of Section 18 and Section 20. The work on the metallurgical samples included:

- Elemental analyses;
- Toxicity Characteristic Leaching Procedure (TCLP);
- Shake Flask Extraction (SFE);
- Analyses and ageing tests of decant solutions;
- Modified acid base accounting and net acid generation tests;
- Humidity cell tests; and
- Sub-aqueous column tests.

Environmental tests were conducted on a total of 52 samples from the metallurgical test program conducted at SGS facilities in Lakefield, Ontario: five ore samples (master composites), 41 cyanide (CN) residues (variability samples), and six CND residues. Selected samples of CND residues were subjected to five humidity cell tests and two sub-aqueous column tests.

At the time of report preparation, all static test work was complete. Kinetic testing (humidity cell and sub-aqueous column) of the CND residues is ongoing (SGS-LR, 2016).

13.13 Conclusions

The main outcomes from the metallurgical test programs are described in the following sub-sections.

13.13.1 Grade Classifications for Ore Types

Two grade-based classifications of samples prepared for metallurgical and physical testing on master composites and variability composites for each of the deposits were OG and LG. The grades of the representative ore grade MC samples for each of the deposits compare well with the corresponding LOM average grades:

- MacLellan:
    - MC head grade, 1.57 g/t Au, 5.4 g/t Ag; and
    - LOM average, 1.63 g/t Au, 4.4 g/t Ag.
- Gordon:
    - MC head grade, 2.48 g/t Au, 0.5 g/t Ag; and
    - LOM average, 2.42 g/t Au, negligible Ag.

13.13.2 Comminution

The average results from tests carried out for the following JK parameter (Axb), work index (WI) and abrasion index (Ai) tests are summarised in Table 13-15. The ball mill work index tests (BWI) were carried out with a closing screen size of 200 mesh (75 µm).

Table 13-15 – Summary MacLellan and Gordon Comminution Tests

	JK	Work Indices (kWh/t)			Abrasion
Sample Name	A x b	CWI	RWI	BWI	Ai (g)
MCL-OG, average	28	-	-	14.9	-
Gordon-OG, average	25	-	-	15.1	-
MCL PQ, average	29	13.2	16.6	15.3	0.19
Gordon PQ, average	33	14.4	20.0	16.4	0.36
Average, all tests; includes low grade, early production, Nisko (MCL), dilution	28	13.6	18.6	14.8	0.29

Typically:

- The Gordon deposit has a lower average JK Axb parameter (more competent) and a higher average BWI (harder) than the MacLellan deposit;
- The deposits are of average hardness from a BWI perspective. The range of values from BWI testing was 10 to 20 kWh/t; and
- The deposits have a high to very high competency from an Axb perspective. The JK parameters occurred in a relatively tight distribution from 20 to 40.

13.13.3 Gravity

Gravity concentration tests comprised grinding to the target grind size, a single pass through a Knelson MD-3 laboratory concentrator followed by upgrading to a low–mass (target ~ 0.1% by weight) on a Mozley C-800 Laboratory Mineral Separator. Low grade and internal dilution tests showed similar gravity response to the ore grade samples. The average (unweighted) gravity gold recovery from each deposit was:

- MacLellan: 47% ; and
- Gordon: 37%.

Although flakes of gold greater than 100 µm were observed in tests, mineralogical examination showed that, in both deposits, most of the gold by occurrence was as relatively fine native gold or electrum particles (typically less than 50 µm).

A gravity stage was included in the standard test procedure to reduce the impact and influence of coarser gold particles in leach performance, to improve reconciliation of assayed and calculated heads and to reduce anomalous test results due to increases in residue assay from the coarser gold particles.

13.13.4 Flotation

The overall recovery from leaching the reground float concentrate and leaching the flotation tailings was compared to the standard gravity/leaching test on each of the Master Composites.

As the overall gold recovery from the flotation-based circuit compared to the standard leaching circuit was 2% lower for MacLellan and negligible difference for Gordon, and as this was accompanied by a more complex and capital-intensive circuit, flotation showed no benefit to the Project and was discontinued from further testing.

13.13.5 Pre-Leach Treatment

A series of pre-leach tests assessed the effect of pre-aeration, addition of lead nitrate ($PbNO_3$), use of oxygen and sodium cyanide concentration on leach efficiency for both ore types.

Although each of the pre-treatment options produced similar leach extractions and gold grades in leach residue after 48 h, tests with pre-aeration using oxygen proved the most beneficial:

- Considerably lower cyanide consumption at 0.2-0.4 kg/t NaCN compared to 1.0-1.3 kg/t for tests without pre-aeration; and
- Best kinetics, with leaching completed within 24 hours while tests with no pre-aeration required close to 48 hours to achieve similar extraction.

Reactive iron sulphide minerals, notably pyrrhotite, responsible for the high cyanide consumption and the high oxygen demand were effectively passivated ahead of leaching by sufficient pre-aeration.

13.13.6 Oxygen Demand

Standard oxygen uptake tests in which oxygen was periodically sparged into the pulp to a target 15 mg/L dissolved oxygen (DO) and the decay rate measured, were carried out on Ore Grade Master Composite samples from both MacLellan and Gordon as well as high-sulphur pyrrhotite-rich and pyrite-rich samples from both deposits.

The tests showed characteristic negative logarithmic decay in the oxygen uptake with time, with a high initial uptake of 0.5-1.0 mg/L/min in the initial two hours, stabilizing after about six hours, and less than 0.2 mg/L/min after 24 hour.

13.13.7 Grind Size

A series of leach tests were carried out over a range of grind sizes to assess extraction, residue grade and cyanide consumption for ores in each of the deposits. These showed that for leach extraction in the range P_{80} 50 µm to 125 µm:

- Gordon ore is grind sensitive;
- MacLellan ore has minor grind sensitivity; and
- Finer grinds contributed to higher cyanide (NaCN) consumption in both samples, particularly finer than P_{80} 100 µm.

An economic analysis of the incremental cost of grinding at finer grind sizes, accounting for power consumption, unit power cost, media wear and cyanide consumption, showed that:

- Gordon ore benefited from finer grinds to nominally P_{80} 50 µm; and

- MacLellan ore showed no marginal benefit with grind size between P_{80} 75 µm and 150 µm, but had a negative benefit for grinds finer than 75 µm.

Using the expected weighted blend of ores from the two pits, the analysis concluded that little net difference in economic benefit occurred for the following grind sizes: between P_{80} 50 µm and 75 µm at the base case power cost of $0.025/kWh; and between P_{80} 60 µm and 10 µm if the base case power cost doubled. A grind size of P_{80} 75 µm was selected for the test program.

13.13.8 Flowsheet Options

Test results for four flowsheets, carried out using the same baseline conditions, on the Master Composite sample for both deposits are summarised in Table 13-16:

- Whole ore leaching;
- Gravity and carbon-in-leach (CIL);
- Gravity and leaching; and
- Flotation, regrind of the concentrate, leaching of both concentrate and tailings.

Table 13-16 – Summary Flowsheet Options

Deposit	Flowsheet	Recovery, % Au	Recovery, % Ag
MacLellan	Whole ore leach	90[1]	66
	Gravity + CIL	93	71
	Gravity + leach	93	66
	Flotation + leach	91	49
Gordon	Whole ore leach	97	43
	Gravity + CIL	96	80
	Gravity + leach	96	66
	Flotation + leach	97	65

Notes:

1. *Higher grade residue than normal; assay not checked/repeated. Recovery is possibly 2-3% higher.*

Both deposits achieved similar gold recoveries for the four flowsheet options: MacLellan about 92% Au and Gordon, 96% Au. Silver recovery from the three leach-based flowsheets was similar and higher than that from the float/leach flowsheet for MacLellan. No significant silver grades are present in the Gordon deposit.

The ores are not preg-robbing as there is no difference in gold recovery between CIL and whole ore leaching or gravity/leach flowsheets.

Although significant gravity recoverable gold was produced in these tests, the gold recovery from whole ore leaching is comparable to that with gravity included.

13.13.9 Leaching

Variability samples from both deposits were submitted for standard batch tests (79 samples) of gravity plus cyanidation of the gravity tailings. The average recoveries and reagent consumptions are summarized in Table 13-17.

Table 13-17 – Summary Variability Samples, Gravity/Leach Standard Test

Sample	Recovery, % Au	Recovery, % Ag	NaCN, kg/t	CaO, kg/t
MacLellan, ore grade	89.4	49.6	0.32	0.48
MacLellan, low grade	86.0	40.0	0.35	0.50
Gordon, ore grade	92.7	-	0.33	0.70
Gordon, low grade	88.0	-	0.13	0.54

Results from standard tests on PQ core samples, the internal dilution samples, the Nisku samples and the early production samples were essentially consistent with the results from the initial variability tests.

13.13.10 Adsorption Modelling

CIP modelling conducted by SGS, using semi-empirical models developed by Mintek and applying k (kinetic constant) and K (equilibrium constant) best-fit parameters from test work to describe a range of operating conditions, concluded that the target final solution assay of < 0.02 mg/L Au would be achieved with a six-stage CIP circuit.

Final modelling, when updates for plant throughput, mine plan and design parameters were available, showed acceptable circuit performance for the specified parameters and for the range of gold and silver grades. The simulations estimated that with loaded carbon of 3,260 g/t Au, one strip per day with a 5.7 t batch of carbon would be required.

13.13.11 Detoxification

Cyanide destruction (CND) continuous test work conducted on leached pulp samples demonstrated that residual weak acid dissociable cyanide (CN_{WAD}) of < 10 mg/L using the SO_2/air/Cu^{2+} detoxification process was achieved. Reagent additions on a weight/weight basis to CN_{WAD} were 3.6 equivalent SO_2, 2.5 hydrated lime and 1.4 Cu in the feed to achieve the target concentration.

Continuous CND tests conducted on five Master Composite samples and a Global Composite sample for CND environmental work all achieved respective target CN_{WAD} levels with reagent additions that matched the expected levels from the previous base-line and optimization test work. Where elevated levels of CN_T were present, a polishing stage in which $CuSO_4$ was added successfully removed the Fe-cyanide species from solution.

13.13.12 Thickening

A comprehensive solid/liquid separation, dynamic thickening and rheology test program was carried out on the Global Master Composite sample at the grind size, pulp density and pH conditions representing the pre-leach thickener. This sample responded well to BASF Magnafloc 333 flocculant.

At a diluted 10% w/w feed well density and a flocculant dosage of 25 g/t, the underflow achieved 60% w/w solids with less than 50 mg/L total suspended solids (TSS) in the overflow. The solids loading unit rate under these conditions was 0.42 t/m²/h.

13.14 Recovery Modelling

13.14.1 Prior Work

Previous test data (Sherritt, ~ 1980's) identified the influence of arsenopyrite and pyrite on recovery of gold. Arsenopyrite had the greater impact as shown by recovery trends with increasing arsenic plotted in Figure 13-13.

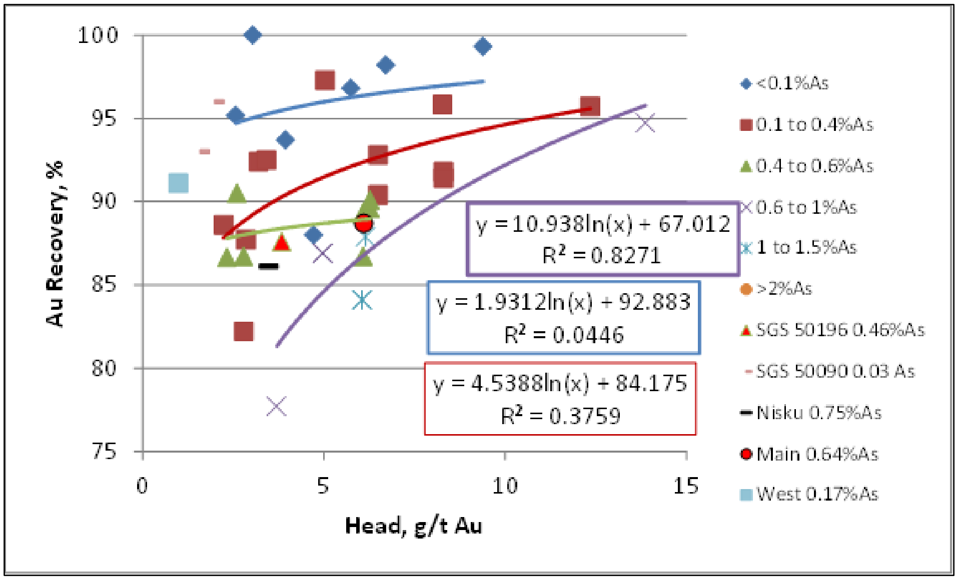


Figure 13-13: Recovery versus Head Grade with Varying Arsenic (after Sherritt Gordon)

Source: Ounpuu (2014)

No prior work to the current series of programs had been done on the Gordon deposit.

13.14.2 Basis

From metallurgical test work since 2015 on 79 samples from the MacLellan and Gordon deposits and five composites prepared from these, a subset of representative tests were assessed for modelling of gold and silver recovery characteristics. This subset comprised 33 tests from MacLellan and 24 tests from Gordon.

The main variables assessed were:

- Au and Ag head grades in the range expected from the mine;
- Sulphide sulphur grades (indicator of pyrite); and
- As grades (indicator of arsenopyrite).

No recovery characteristics or trends were evident from assessment of the following: lithology; location by depth or along strike in the deposits; with occurrence and relative abundance of pyrite, pyrrhotite and arsenopyrite sulphide minerals.

13.14.3 MacLellan

Preliminary assessment of recovery characteristics with feed gold, sulphur and arsenic variables were carried out and discounted because:

- Extractions using algorithms based on Au/S, Au/As and Au/S/As, although reasonable, understated the recoveries compared to test work results;
- The resource model or mine plan does not have S or As grades for plant feed; and
- S and As grades estimated from trends observed in the test work introduced another level of uncertainty. Extractions based on these calculated grades were overstated.

The test data used to assess metallurgical modelling for the MacLellan deposit, after allowing for the following, is shown in Table 13-18.

- By inspection, two outliers with poor leach extraction and with high arsenic (0.3% As compared to the average from tests of 0.1% As) were removed. Mineralogical examination/probe work on samples with elevated levels of arsenic in the feed showed that although gold was present, the gold content was variable and occurred in notably different levels within the arsenopyrite. This variability could not be modelled with any confidence;
- Tests with low arsenic feed grades (< 0.005% As), and which were mostly associated with low gold feed grades (less than 0.5 g/t Au) produced very low residue grades (average 0.03 g/t Au). These were removed from the model analysis as these essentially all produced a fixed residue grade; and
- Tests with very high gold grades (> 20 g/t Au) and very low feed grades (< 0.2 g/t Au) were removed as these were well outside the range of grades to be treated in the plant.

Table 13-18 – MacLellan Test Data Summary Head, Residue and Extraction Results

Sample		Head			Residue	Extr'n
Description	Code	g/t Au	%As	%Stot	g/t Au	% Au
MacLellan Ore Grade Composites MCL-OG	MCL-OG-01	1.72	0.30	1.50	0.12	93.0
	MCL-OG-02	0.59	0.09	1.02	0.05	91.5
	MCL-OG-03	1.81	0.16	1.11	0.10	94.5
	MCL-OG-04	2.07	0.02	1.95	0.04	98.1
	MCL-OG-05	2.06	0.19	2.65	0.12	94.2
	MCL-OG-06	2.79	0.19	2.04	0.23	91.8
	MCL-OG-07	1.75	0.08	0.87	0.05	97.1
	MCL-OG-08	1.03	0.41	2.17	0.12	88.3
	MCL-OG-11	0.53	0.001	0.81	0.03	95.3
	MCL-OG-12	1.55	0.12	1.29	0.06	96.5
	MCL-OG-14	1.56	0.12	1.84	0.15	90.7
	MCL-OG-15	4.46	0.21	2.04	0.14	96.9
	MCL-OG-16	1.47	0.05	1.23	0.10	93.5
Master Comp OG	MCL-MC-OG	1.57	0.16	1.53	0.17	89.2
MacLellan Low Grade Composites MCL-LG	MCL-LG-01	0.21	0.005	0.84	0.02	90.5
	MCL-LG-02	0.50	0.010	1.75	0.03	95.0
	MCL-LG-03	0.28	0.012	0.90	0.02	92.9
	MCL-LG-04	0.21	0.001	0.93	0.02	90.5
	MCL-LG-05	0.40	0.004	1.15	0.02	95.0
	MCL-LG-06	0.40	0.001	1.54	0.08	80.0
	MCL-LG-08	0.52	0.120	1.04	0.11	78.8
Master Comp LG	MCL-MC-LG	0.23	0.01	1.16	0.05	78.3
PQ Core Samples (2016 Data)	Comp 1	1.42	0.15	0.64	0.07	95.1
	Comp 2	2.33	0.07	0.32	0.08	96.6
	Comp 3	5.06	0.04	1.65	0.16	96.8
	Comp 4	0.33	0.001	1.53	0.02	93.9
	Comp 5	1.08	0.10	1.91	0.22	79.6
	Comp 6	2.57	0.00	2.46	0.32	87.5
Additional Samples MCL-MG	Comp 26	1.89	0.01	0.82	0.02	98.9
	Comp 27	0.58	0.02	0.35	0.02	96.6
	Comp 28	2.34	0.02	1.14	0.05	97.9
	Comp 29	1.42	0.24	1.85	0.09	93.7
	Comp 30	2.76	0.51	3.45	0.41	85.1

The plot of gold in residue versus gold in feed is shown in Figure 13-14. Coordinates for the OG and LG master composites are shown for reference. An improved correlation was obtained by assessing the trend for feed gold grades expected in the mine plan, typically in the range 0.5 to 3.0 g/t Au. High grade gold test results (> 4 g/t Au) were not included as this produced poorer correlation. It was assumed that the minimum gold grade in the residue was 0.03 g/t (from the low arsenic observation) and this was set as the intercept for the equation to best fit the test results.

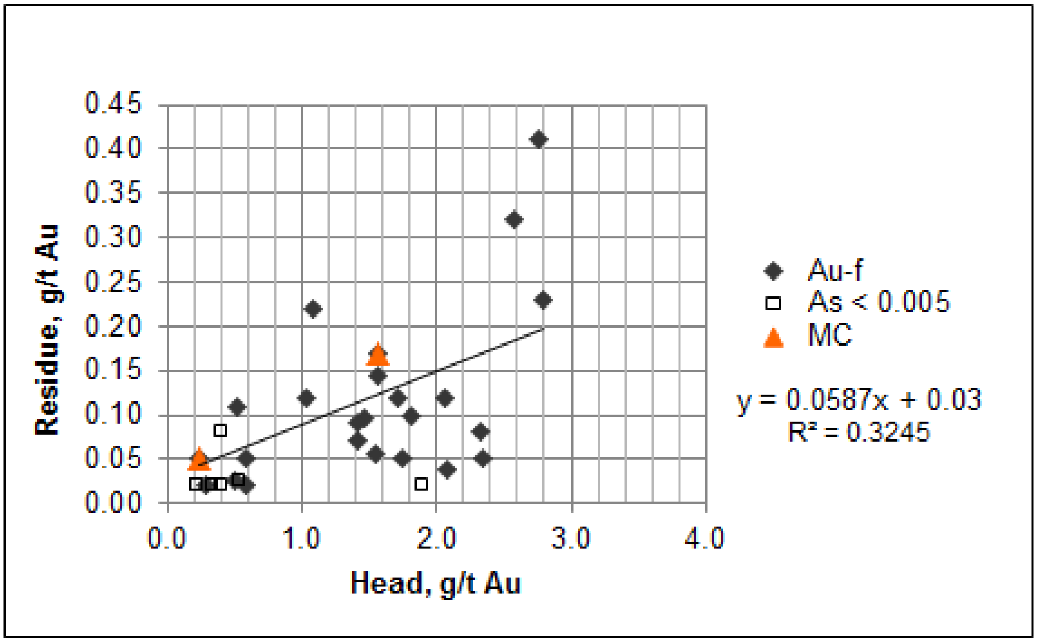


Figure 13-14: MacLellan Relationship Au in Residue Versus Gold in Feed

Source: SGS (2015)

The equation best suited to estimate Au residue grade and then determine extraction, was:

Au-residue, g/t = 0.03 + 0.0587 x Au-feed

Extraction Au, % = (1 – Au-residue / Au-feed) x 100

Recovery of Au from the operation is extracted Au less soluble losses and other minor losses in plant operation, assumed to be constant for recovery forecast purposes:

Recovery Au, % = Extraction (%) – 0.9%

The calculated extractions using the residue model estimate from the feed grade are shown in Figure 13-15. The logarithmic regression curve from the calculated extractions is the best-fit curve from the considerable scatter of data evident.

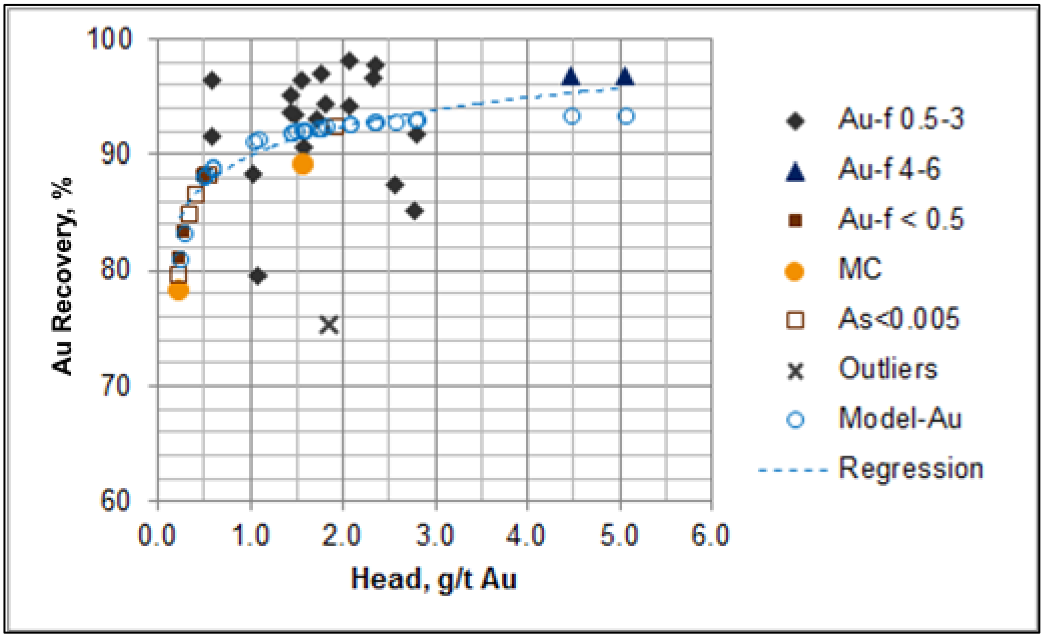


Figure 13-15: MacLellan Gold Extractions from Test Work and by Residue Model

Source: SGS (2015)

The leach extraction of silver with head grade is plotted in Figure 13-16.

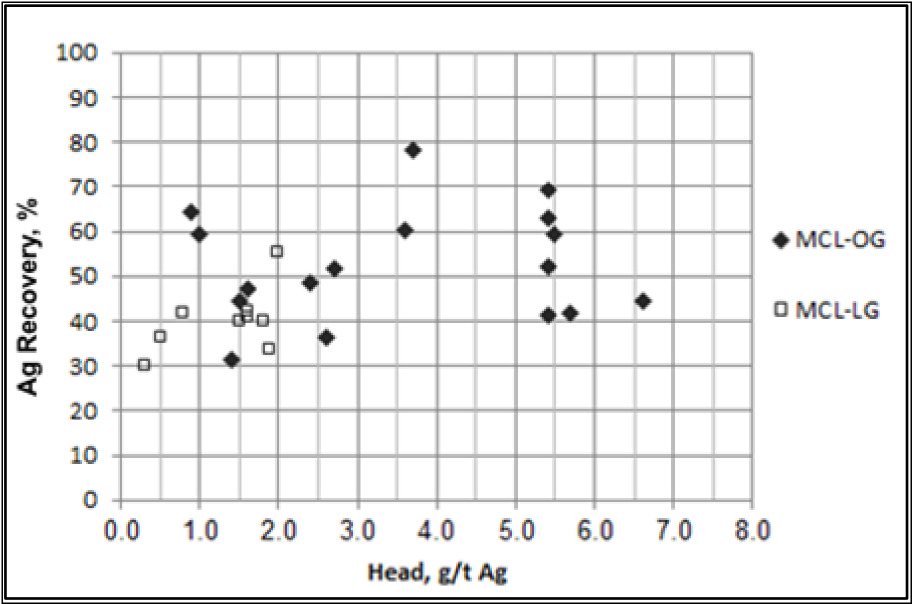


Figure 13-16: MacLellan Leach Extraction Silver with Head Grade

Source: SGS (since 2015)

There is a large scatter of Ag extraction results across all tests (30% to 80%) with high variability in each of the ore categories. As the correlation of data was poor and as no strong trend with feed grade was evident, the average extraction of all tests, 51.9%, was adopted as a constant silver extraction with silver head grade for MacLellan ores.

Recovery of silver is constant extracted silver less soluble losses and other minor losses in plant operation, assumed to be constant for recovery forecast purposes:

Recovery Ag, % = 51.9 – 2.9 = 49.0.

13.14.4 Gordon

The data used to assess metallurgical modelling for the Gordon deposit is shown in Table 13-19.

Table 13-19 – Gordon Test Data Summary Head, Residue and Extraction Results

Sample code	Lithology	Spatial	Pyrite	Pyrrhotite	Auf g/t	Sf %	Aur g/t	Extn % Au
FL-OG-01	Arg IF	West-N	3.05	0.57	1.91	1.65	0.07	96.6
FL-OG-02	Arg IF	West-N	0.98	7.45	4.34	3.27	0.65	85.0
FL-OG-03	Arg IF	West-N	0.27	2.73	0.48	1.15	0.02	95.8
FL-OG-04	Andesite	West-S	0.19	2.00	0.51	0.85	0.02	96.1
FL-OG-05	Sil IF	West-S	0.35	6.44	3.83	2.53	0.29	92.4
FL-OG-06	Arg IF	MidWest-N	5.44	2.71	7.25	3.71	0.22	97.0
FL-OG-07	Arg IF	MidWest-N	0.51	4.34	2.13	1.83	0.28	86.9
FL-OG-08	Arg IF	MidWest-N	0.54	3.24	5.49	1.49	0.19	96.5
FL-OG-09	intrusive	MidWest-S	5.00	0.63	1.32	2.72	0.17	87.1
FL-OG-10	intrusive	MidWest-S	0.28	2.34	0.66	1.06	0.03	95.5
FL-OG-11	Sil IF	MidEast	4.68	0.39	1.65	2.55	0.04	97.6
FL-OG-12	BIF	MidEast	2.40	6.96	2.89	3.17	0.34	88.1
FL-OG-13	Arg IF	East	1.85	5.04	1.89	1.95	0.18	90.7
FL-LG-01	Graph. Argillite	West-N	0.21	3.74	0.32	0.33	0.10	68.8
FL-LG-02	Arg IF	MidWest-N	3.30	3.03	0.85	1.74	0.03	97.1
FL-LG-03	intrusive	MidWest-S	0.41	1.41	0.48	0.76	0.03	93.8
FL-LG-04	BIF	MidEast	0.06	0.71	0.63	0.28	0.05	92.9
Comp 7	BIF	West-N ?	4.16	0.73	1.78	2.07	0.06	96.6
Comp 8	BIF	West-N ?	0.25	1.79	0.99	0.71	0.08	91.9
Comp 9	ARG-BIF	West-N ?	0.81	1.95	3.01	1.04	0.33	89.0
Comp 12	GAB	S ?	3.18	3.73	3.89	2.90	0.11	97.2
Comp 13	BIF	S ?	0.30	1.99	1.27	0.79	0.13	89.8
Comp 14	BIF	S ?	1.28	6.80	4.10	2.74	0.23	94.4
Comp 15	BIF	S ?	0.55	7.89	2.64	2.90	0.12	95.5

Regression analyses of trends and correlations of gold in the residue with gold, silver and arsenic or with minerals in the feed which may influence recovery indicated that:

- Gold in the feed provided the best fit and that the natural logarithm of feed gold was a better predictor than gold feed grade;
- Sulphide sulphur (average 1.7%, range 0.3-3.2%) had little influence and in some cases modelling predicted a negative recovery;
- Pyrite (average 2% and up to 5%) had no influence;
- Pyrrhotite (average 3% and up to 8%) had no influence;
- The ratio of pyrite to pyrrhotite had no influence;
- Arsenic (arsenopyrite) was not significant as this occurs typically less than 10 ppm As; and
- Silver was not used as this occurs as very low and variable grades.

The equation which best predicted the gold residue grade was a logarithmic function of gold in the feed. This was then used to determine extraction as follows:

Au-residue = 0.0978 + 0.0835 x Ln (Au-feed)

Extraction Au, % = (1 – Au-residue / Au-feed) x 100

Recovery of gold from the operation is extracted gold less soluble losses and other minor losses in plant operation, assumed to be constant for recovery forecast purposes:

Recovery Au, % = Extraction (%) – 0.9%

The test data base and calculated values from the residue model are plotted in Figure 13-17.

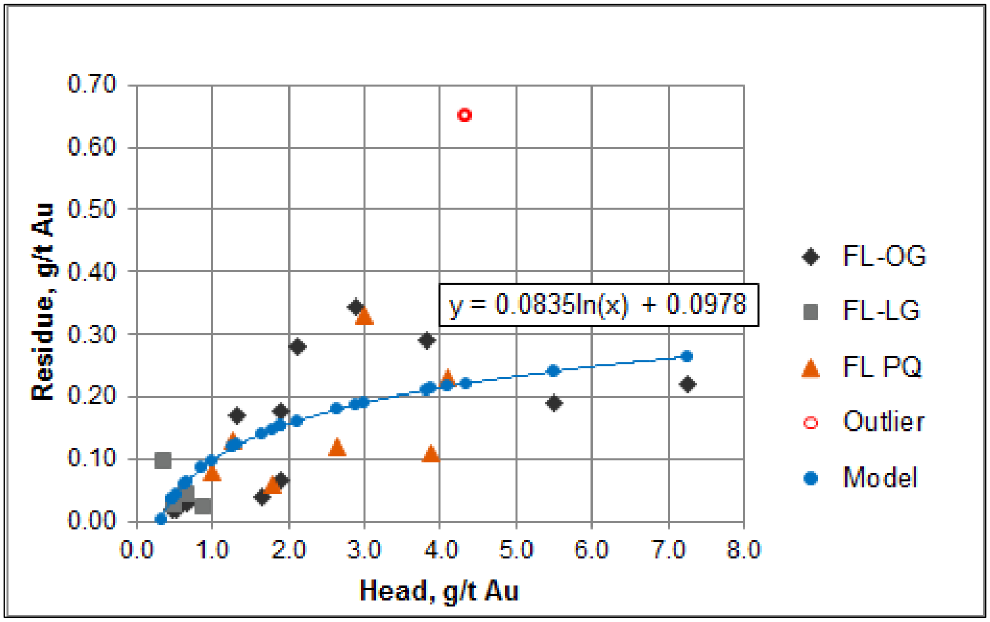


Figure 13-17: Gordon Gold Residue versus Feed Grade and Model Residue

Source: SGS (2015)

The estimated gold recovery by head grade based on the modelled residue grades is plotted with the test results in Figure 13-18.

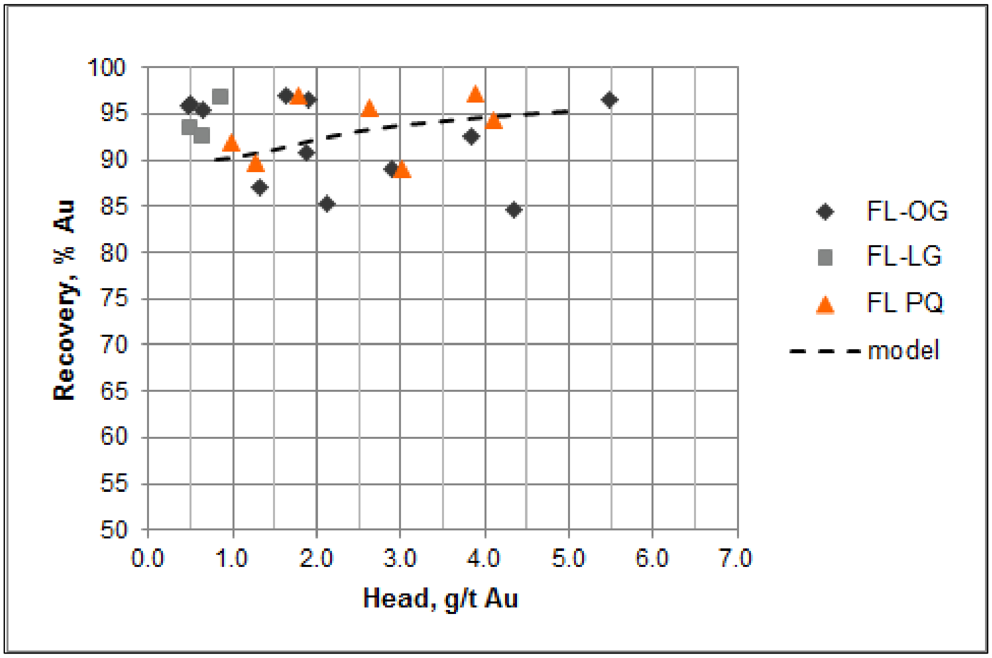


Figure 13-18: Gordon Estimated Gold Recovery by Head Grade

Source: SGS (2015)

Silver is not modeled in Gordon.

14 MINERAL RESOURCE ESTIMATES

14.1 MacLellan

14.1.1 Drill Hole Database

Alamos conducted a Mineral Resource estimate of the MacLellan deposit, which incorporates all drilling data from Alamos and predecessor drilling programs conducted through December 31, 2016. A database was compiled using data from 1,472 drill holes, with collar, survey, geological and assay information, containing a total of 218,434 m of assayed gold intervals. Of these data, 150,938 m (678 holes) are surface drilling, and 67,496 m (794 holes) were drilled from underground. Historic drilling conducted by historic operators was included where data could be verified.

14.1.2 Topography

Existing topography is based on a 2014 LiDAR survey conducted by LiDAR Services International Inc. (LSI), and was used to code the Mineral Resource model. Alamos has reviewed the topography surface in cross sections comparing elevation between the drillhole collars and the topography surface and found close agreement between the two.

14.1.3 Coordinate System

All data is located in NAD 83 Zone 17 geodetic Datum. The local mine coordinate system utilized a translated and rotated version of the above coordinate system during historic mining operations. A number of pre-2015 drill hole collars (89) were located by Barnes and Duncan in a December 2014 survey and were found to be accurately located in the digital drill hole collar database.

14.1.4 Lithology and Grade Modelling

A 0.50 g/t Au grade solid was constructed using Leapfrog modeling software and manually edited to eliminate small volume solids and outlier tonnages. A northwest-southeast cross section showing the resultant solid is provided in Figure 14-1.

A generalized lithologic model was also constructed using Leapfrog modeling software and was utilized to assign specific gravity (SG) to the model blocks. A total of eight generalized lithologies were modeled:

1) Argillite;
2) Banded Iron Formation (BIF);
3) Chert/Quartzite;
4) Felsic Intermediate Schist;
5) Intrusive;
6) Mafic/Ultramafic;
7) Mafic Schist; and
8) Quartz-Carbonate Veins.

SG assignments based on these modeled lithologies are described in a subsequent section of this report.

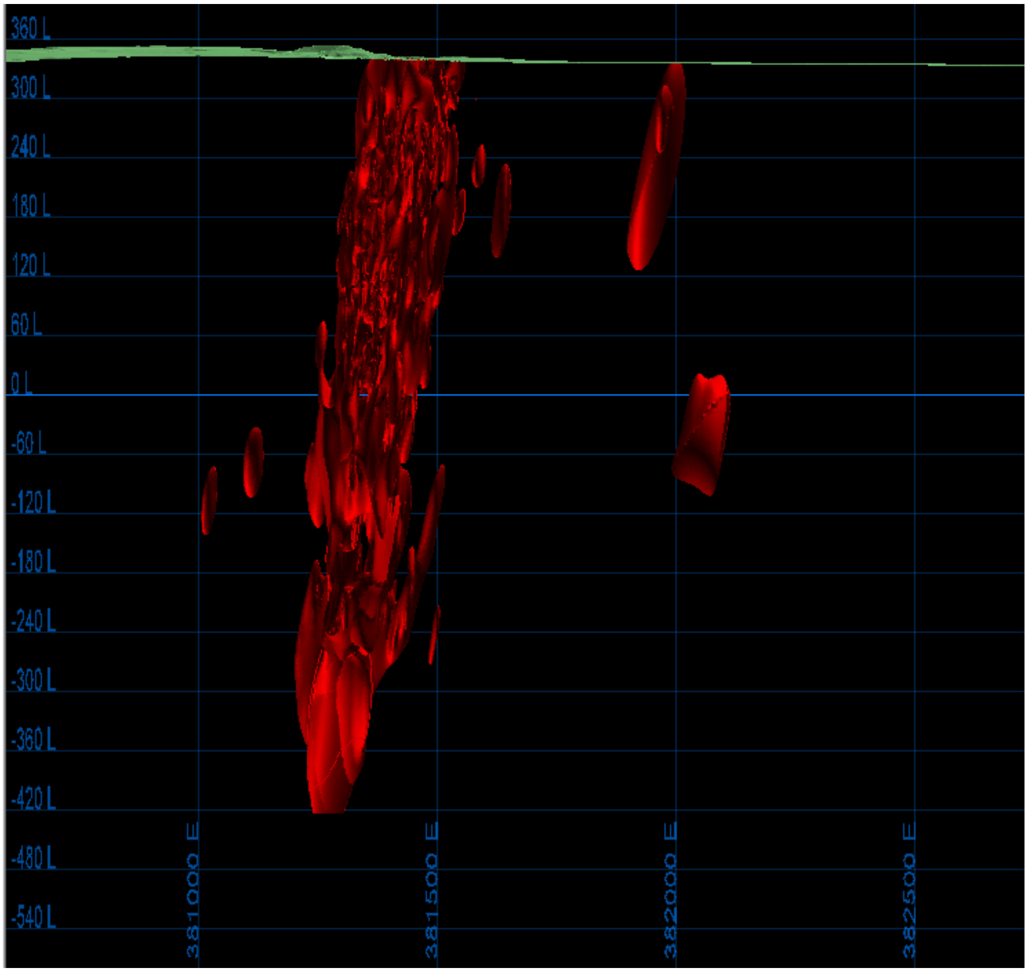


Figure 14-1: WSW - ENE Cross Section Viewed to the North Showing 0.50 g/t Au Solid

Source: Alamos (2017)

14.1.5 Exploratory Data Analysis

Exploratory data analysis (EDA) was conducted using the raw gold assays from the MacLellan Property. These data were analyzed above both global and incremental cut-offs to assess the distribution and grade ranges of the assays internal and external to the 0.50 g/t Au solid. Basic statistics for the raw gold assays are provided in Table 14-1. It can be observed that on a grade-thickness basis, a minor amount of metal above potentially economic cut-off resides external to the 0.50 g/t solid. Most of the intervals external to the grade solid are not of minable width, however this demonstrates that gold occurs external to the main lode system. Statistically, the 0.50 g/t Au solid envelops ~80% of all of the raw assay data in the model area above a 0.50 g/t Au cut-off.

Table 14-1 – Summary Statistics – Raw Gold Grades by Domain

Domain	Au Cut-off (g/t)	Total Meters	Incremental Pct	Statistics above Cut-off				
				Max Grade (Au g/t)	Mean Grade (Au g/t)	Grade Thickness (g/t*m)	Standard Deviation	Coeff. of Variation
All Data	0.01	110,658	76.69%	649.72	0.90	99,097	5.19	5.80
	0.50	25,795	17.27%		3.49	90,061	10.33	2.96
	3.00	6,687	2.46%		10.08	67,371	18.76	1.86
	5.00	3,967	3.58%		14.37	57,003	23.40	1.63
Internal to 0.50 g/t Au Solid	0.01	39,246	47.64%	649.72	2.06	80,742	7.82	3.80
	0.50	20,549	37.42%		3.72	76,526	10.53	2.83
	3.00	5,862	6.05%		9.97	58,450	18.24	1.83
	5.00	3,486	8.88%		14.16	49,381	22.72	1.60
External to 0.50 g/t Au Solid	0.01	71,411	92.65%	498.48	0.26	18,356	2.65	10.30
	0.50	5,246	6.19%		2.58	13,536	9.45	3.66
	3.00	824	0.48%		10.82	8,921	22.06	2.04
	5.00	480	0.67%		15.86	7,622	27.81	1.75

14.1.6 Specific Gravity Analysis

Company geology personnel conducted 2,531 specific gravity tests, using the water immersion method, on diamond drill core from the property during the 2015 drilling program. In addition, analyses were conducted by TSL during the 2007-2012 Carlisle drilling campaigns. A summary of SG determinations by rock type is provided in Table 14-2.

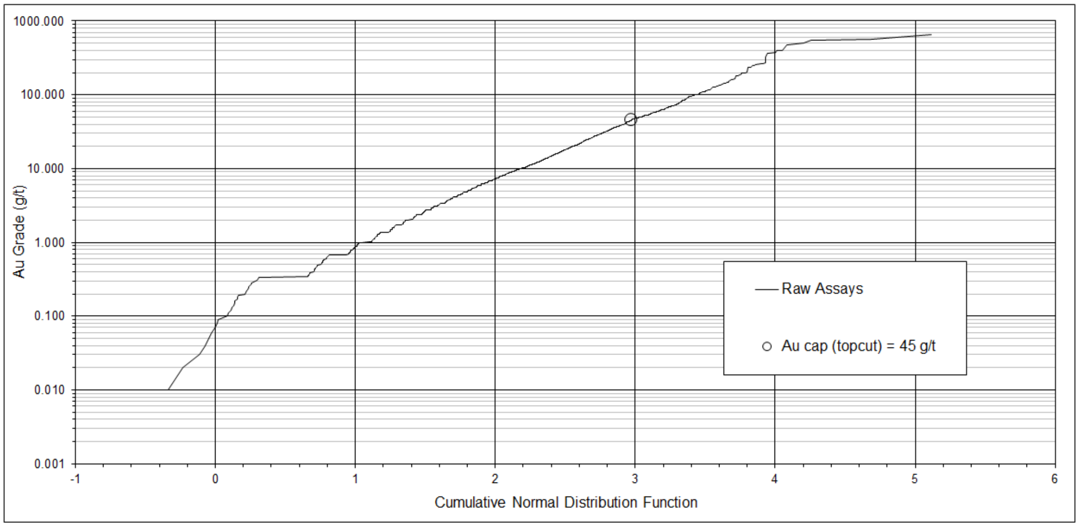
A default SG value of 2.94 was assigned to modeled lithologies that did not receive any SG determinations.

Table 14-2 – SG Results for MacLellan 2015 Samples

Rock Type	Lith Code	Number of Samples	Min SG	Max SG	Avg SG
Argillite	1	0			2.94
BIF	2	5	2.79	3.18	2.94
Chert/Quartzite	3	0			2.70
Felsic Intermediate Schist	4	379	2.65	3.81	2.91
Intrusive	5	5	2.78	3.10	2.90
Mafic/Ultramafic	6	994	2.41	4.26	3.00
Mafic Schist	7	42	2.69	3.12	2.97
Quartz-Carbonate Veins	8	1	2.70	2.70	2.70
All Data		2,531	2.41	4.26	2.96

14.1.7 Evaluation of Outlier Data

The raw drill hole gold assay dataset was inspected globally using log cumulative probability plots to assess for the presence of high-grade outlier values that could adversely impact grade estimation. For this analysis, the datasets both external and internal to the 0.5 g/t Au grade solid were combined, as they are one contiguous deposit. After review of log probability plots, all raw gold assays were capped at 45.0 g/t Au prior to compositing (Figure 14-2). This assay cap (topcut) affects 166 meters of sample, and results in a reduction of 7.23% of gold metal on a grade-thickness (GT) basis. The capped data shows a reduction in the coefficient of variation (CV) to 3.39 as compared to 5.80 for the uncapped data.



	Untransformed Au Statistics								Log Normal Approximation Model		
	Au Cutoff = 0.01 g/t		Au Cutoff = 0.50 g/t		Au Cutoff = 3.00 g/t		Au Cutoff = 45.00 g/t				
	Meters	Au (g/t)	Meters	Au (g/t)	Meters	Au (g/t)	Meters	Au (g/t)	Mean	Standard Deviation	Third Parameter
Raw Assays	111,215	0.892	25,845	3.488	6,693	10.071	166	87.754	-1.12	0.90	0.00
incr. % and grade	76.8%	0.106	17.2%	1.187	5.9%	8.094	0.1%	87.754			
low cut	0.10		45 g/mt percentile		GT lost by capping		percent of GT >= 650 g/mt				
			99.70%		7.23%		0.15%				
Au cap (topcut)	45.00		percent of GT >= 45 g/mt		CV uncapped		CV capped				
			14.83%		5.80		3.39				

Figure 14-2: Cumulative Probability Plot: Raw Gold Assays

Source: Alamos (2017)

14.1.8 Compositing

All capped raw data were composited into 5 m downhole intervals. Composites were back-flagged by the model blocks to assign percentage of composite internal to the 0.5 g/t Au wireframe. Only composites with a wireframe percentage ≥10% were retrieved for use in grade estimation. All composites were length weighted during the grade estimation process.

14.1.9 Variogram Analysis

Pairwise relative variograms were constructed for gold in Sage2001© software using the 5 m capped composite data. The resulting variograms exhibited high nuggets and relatively short ranges overall. The resultant fitted variograms for the major, semi-major and minor axes are provided in Figure 14-3, Figure 14-4 and Figure 14-5. A summary of variogram parameters is provided in Table 14-3. These variogams generally reflect geologic field observations, and due to the extremely high nugget observed in the sample variograms, search orientations were ultimately defined based on directions of geologic continuity and observed ore controls.

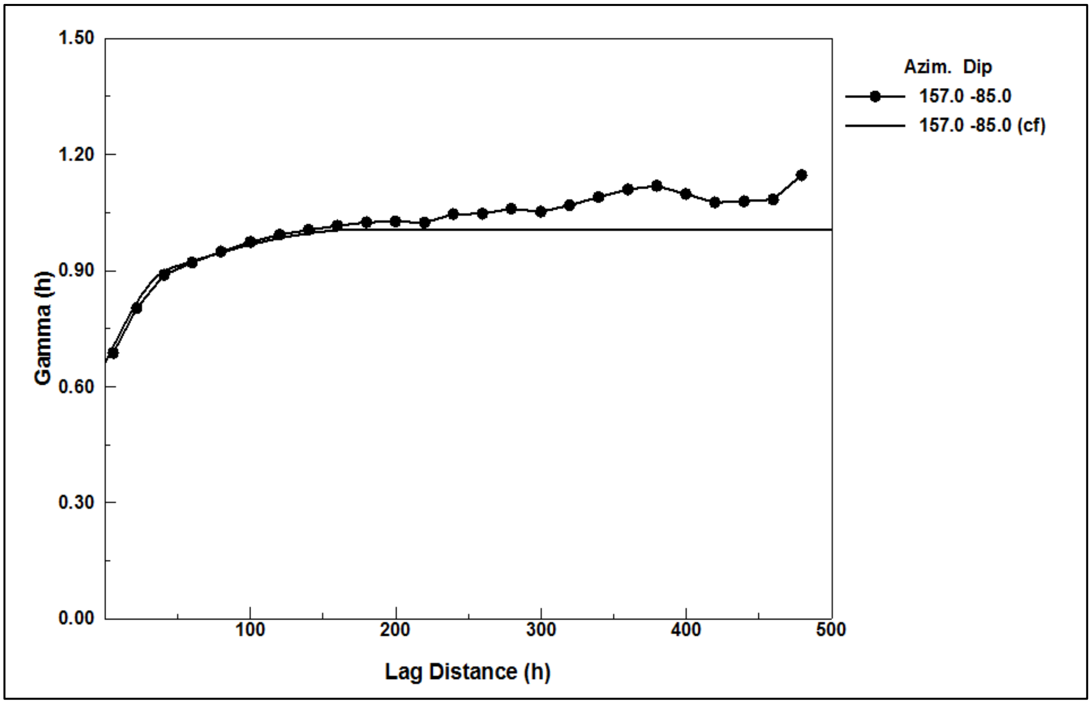


Figure 14-3: Pairwise Relative Variogram and Model: Principle Axis

Source: Alamos (2017)

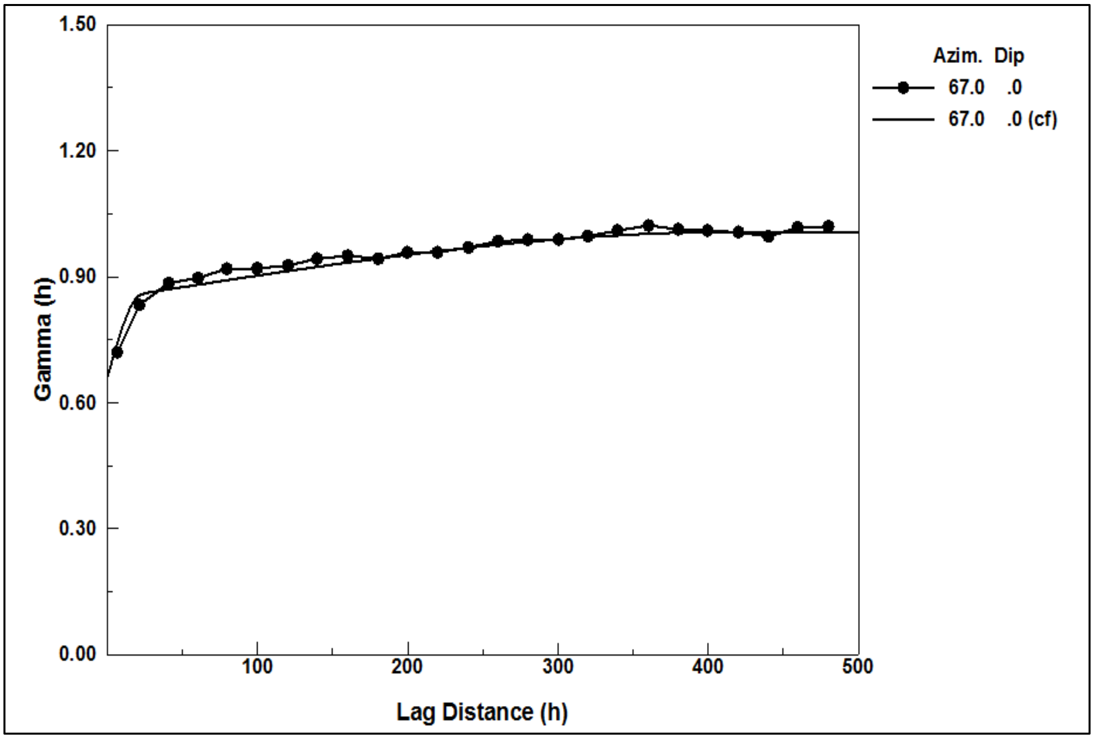


Figure 14-4: Pairwise Relative Variogram and Model: Semi-major Axis

Source: Alamos (2017)

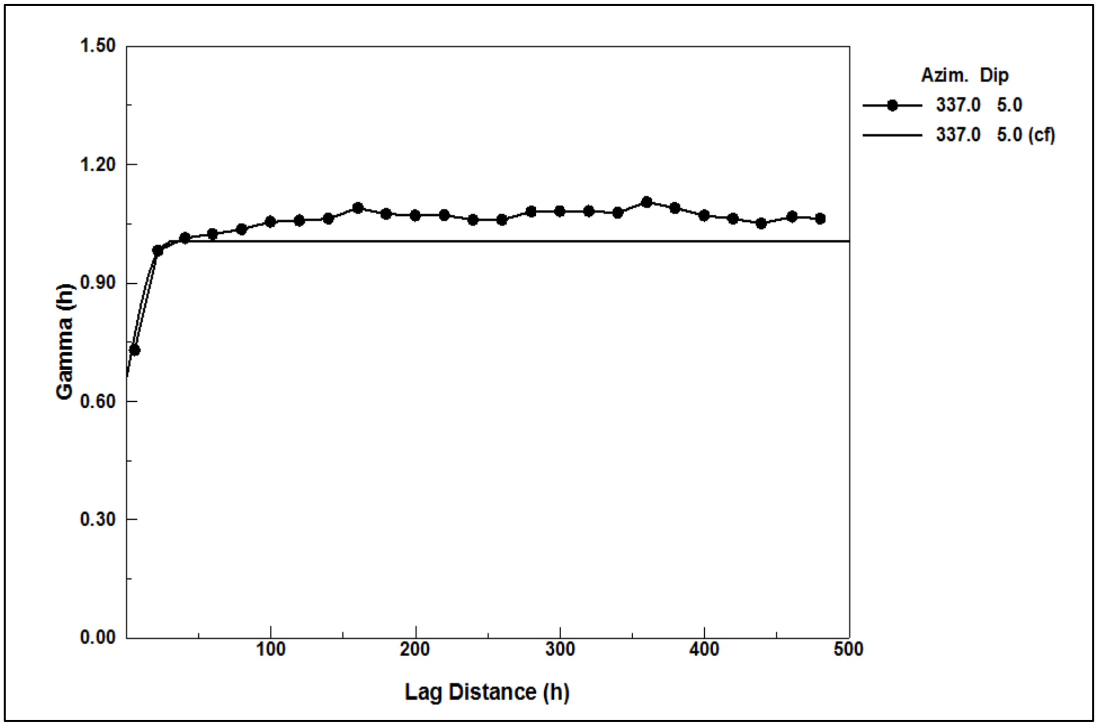


Figure 14-5: Pairwise Relative Variogram and Model: Minor Axis

Source: Alamos (2017)

Table 14-3 – Pairwise Relative Variogram Model Parameters for Au: 5m Composites

MacLellan: Variogram Results (Spherical Model)			
Parameters (First Structure)	**Principal**	**Minor**	**Semi-Major**
Azimuth (deg)	216	152	62
Dip (deg)	89	-1	1
Parameters (Second Structure)	**Principal**	**Minor**	**Semi-Major**
Azimuth (deg)	100	10	68
Dip (deg)	-1	-1	89
Nugget Effect C0	0.719		
1st Structure C1	0.138		
2nd Structure C2	0.151		
1st Range A1 (m)	58	19	51
2nd Range A2 (m)	436	34	314

14.1.10 Block Model Construction

A block model was constructed in Vulcan™ for the MacLellan deposit using the model limits and extents provided in Table 14-4.

Block model construction utilized a constant block size of 5.0 x 5.0 x 5.0 m.

Table 14-4 – MacLellan Model Limits and Extents

Mine Grid Origin[1]	Min (m)	Max (m)	Block Size (X)	Block Size (Y)	Block Size (Z)	No. of Blocks
East	380,290	382,295	5.0	5.0	5.0	401
North	6,306,440	6,307,940	5.0	5.0	5.0	300
Elevation	-500	505	5.0	5.0	5.0	201

Notes:

1. *All blocks are rotated 65° about model origin*

14.1.11 Block Model Grade Estimation

The Mineral Resource estimate was undertaken using Maptek Vulcan™ software employing the inverse distance weighting method (ID3) and nearest neighbour (NN) method for use in model validation. In addition to gold, silver grades were estimated. Model blocks were assigned a percentage of the block internal to the 0.50 g/t Au wireframe and only blocks whose volumes were ≥10% internal to the wireframe were available for grade estimation. Model blocks were also “mined” using the historic underground mining solids and both mined tonnage and grade were zeroed out for the final Mineral Resource tabulation. A summary of the search parameters utilized is presented in Table 14-5. Model block grades were diluted to account for all volume external to the 0.50 g/t Au wireframe at a zero grade. Additional estimates were performed for neutralization potential (NP) and acid potential (AP) to assist in the overall ABA studies related to mine planning. The methodology used for these latter estimates are described in more detail in subsequent sections of this report.

Table 14-5 – Search Parameters for the MacLellan Model – Gold and Silver

Pass Number	Search Orientation[1] (degrees)			Search Distance (m)			Min Comps	Max Comps	Max per DDH
	Bearing (Z)	Plunge (Y)	Dip (X)	Major Axis	Semi-Major Axis	Minor Axis			
1	67	0	-85	20	20	5	3	8	1
2	67	0	-85	45	45	10	2	8	1
3	67	0	-85	100	100	20	1	8	1

Notes:

1. *Vulcan rotations*

14.1.12 Block Model Validation

Various measures have been utilized to validate the resultant Mineral Resource block model. These measures include the following:

- Comparison of drill hole composites with Mineral Resource block grade estimates by zone visually, both in plan and section;
- Statistical comparisons between block and composite data using histogram and cumulative frequency distribution analysis;
- Generation of comparative NN model; and
- Swath plot analysis (drift analysis) comparing the ID3 model with the NN model.

14.1.12.1 Visual Inspection

Visual comparisons between the block grades and the underlying composite grades in plan and section show close agreement, which would be expected considering the estimation methodology employed. An example northwest-southeast cross section and example longitudinal section are provided in Figure 14-6 and Figure 14-7, respectively, displaying block and composite gold grades, 0.50 g/t Au grade solid outlines, US $1,400 Mineral Resource pit (magenta) and existing underground development (brown).

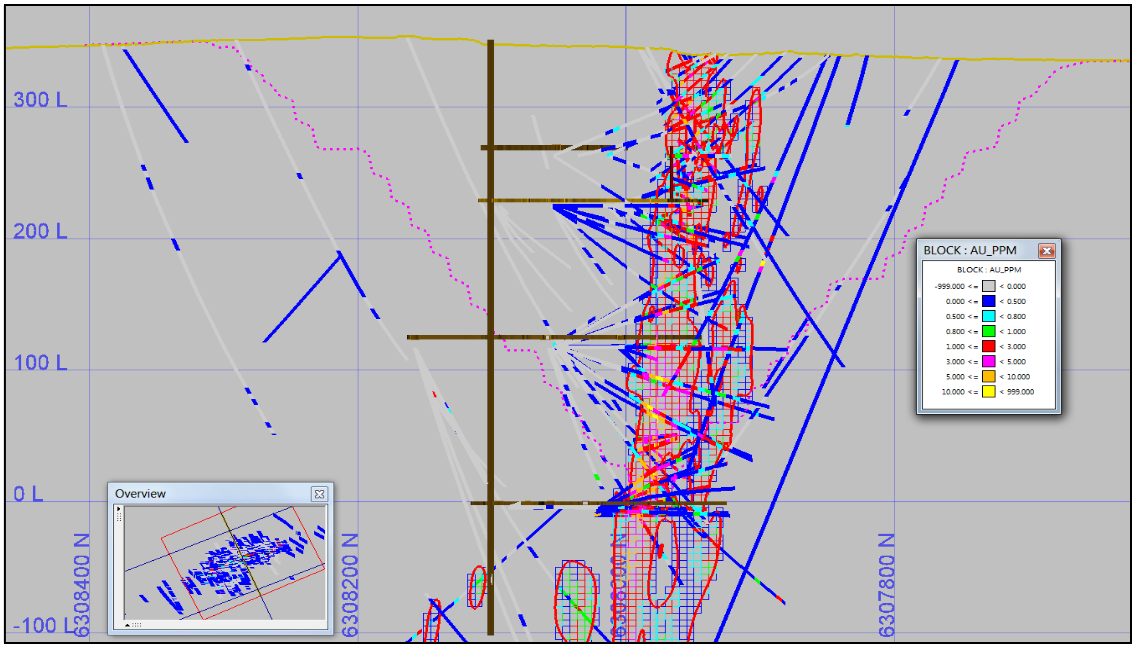


Figure 14-6: Example Northwest-Southeast Cross Section Viewed to East

Source: Alamos (2017)

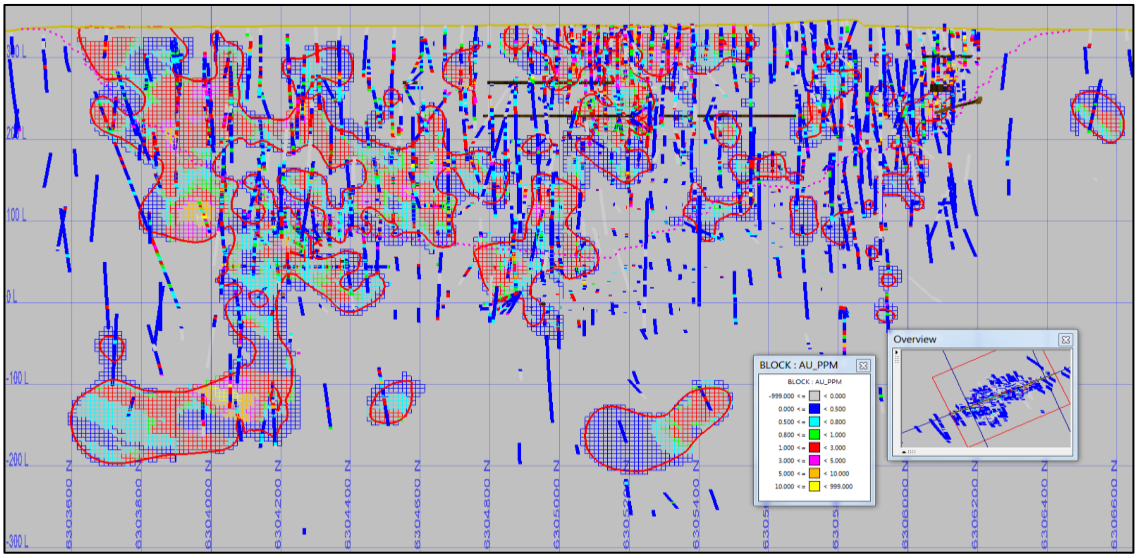


Figure 14-7: Example West-southwest - East-northeast Long Section Showing Model Block Grades

Source: Alamos (2017)

An example level plan displaying block and composite gold grades, 0.50 g/t Au grade solid outlines, US $1,400 Mineral Resource pit (magenta) and existing underground development (brown) are provided in Figure 14-8.

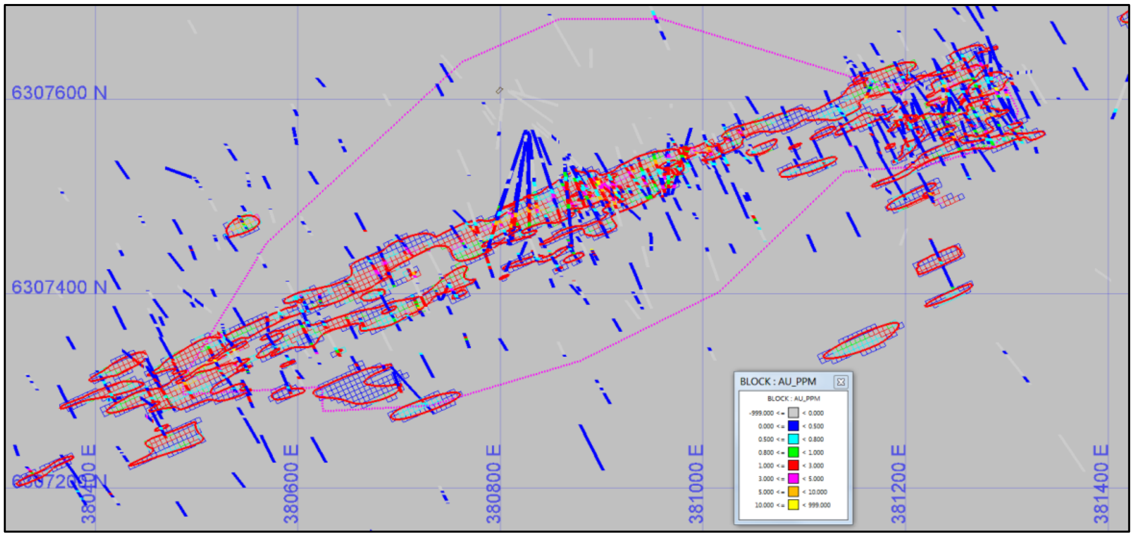
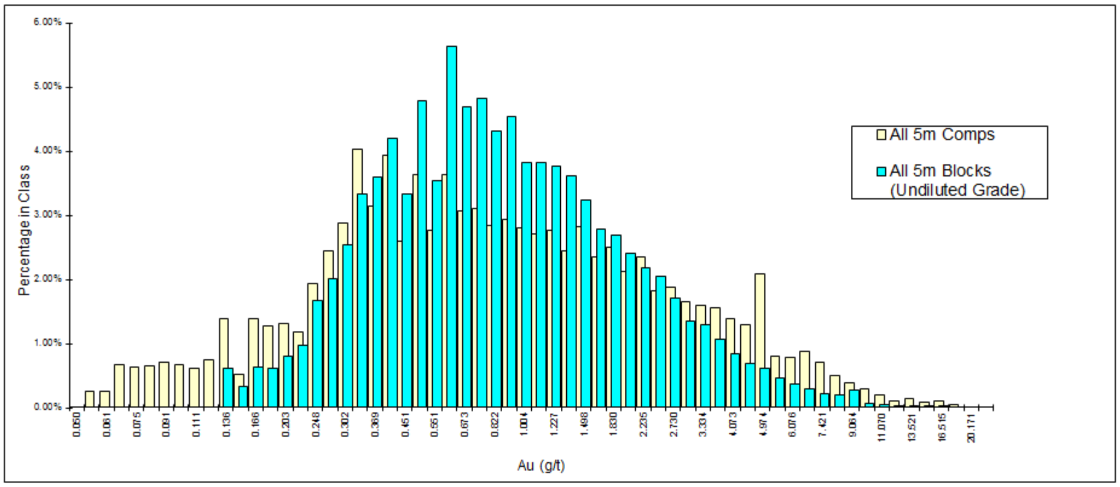


Figure 14-8: Example Level Plan 200 m Elevation Showing Model Block Grades

Source: Alamos (2017)

14.1.12.2 Block-Composite Histogram Comparison

Alamos also conducted statistical comparisons between the grades of the Measured, Indicated and Inferred ID3 blocks contained within the 0.50 g/t Au solid and the underlying gold composite grades. A histogram comparison between block and composite gold grades at the MacLellan is provided in Figure 14-9.



Untransformed Au Statistics

	Au Cutoff = 0.010 g/t		Au Cutoff = 0.500 g/t		Au Cutoff = 1.50 g/t		Au Cutoff = 5.00 g/t	
	meters/tonnes	Au (g/t)	meters/tonnes	Au (g/t)	meters/tonnes	Au (g/t)	meters/tonnes	Au (g/t)
All 5m Comps	53,149	1.557	33,653	2.307	16,270	3.830	3,819	7.567
incr. % and grade	36.7%	0.262	32.7%	0.882	23.4%	2.684	7.2%	7.567
All 5m Blocks (Undiluted Grade)	79,365,271	1.260	57,481,982	1.619	19,885,077	3.011	2,128,887	7.363
incr. % and grade	27.6%	0.319	47.4%	0.883	22.4%	2.489	2.7%	7.363

Figure 14-9: Histogram Comparison between Block and Composite Grades: Gold

Source: Alamos (2017)

Overall, this comparison shows that the model grade distribution for gold is appropriately smoothed when compared with the underlying composite distribution, and that the comparison of average grades and percentages above geologic absolute and incremental cut-offs show good agreement.

14.1.12.3 Comparison of Interpolation Methods

For comparative purposes, additional grades were estimated using NN interpolation methods. The results of the NN model are compared to the ID3 model at a 0 g/t Au cut-off grade for the measured and indicated blocks in Table 14-6 and in Table 14-7 for all Inferred blocks. These comparisons confirm the conservation of metal at a zero cut-off, and shows close agreement on both a tonnage and grade basis within the deposit area.

Table 14-6 – ID3 vs. NN Tonnage – All Measured and Indicated Model Blocks

Model	Tonnes	Au Grade (g/t)	Au oz
ID3	45,568,643	1.448	2,120,816
NN	45,568,643	1.480	2,168,350
Pct Diff (ID3 to NN)	0.00%	-2.24%	-2.24%

Table 14-7 – ID3 vs. NN – All Inferred Model Blocks

Model	Tonnes	Au Grade (g/t)	Au oz
ID^3	16,799,277	1.071	578,375
NN	16,799,277	1.097	592,507
Pct Diff (ID^3 to NN)	0.00%	-2.44%	-2.44%

14.1.12.4 Swath Plots (Drift Analysis)

A swath plot is a graphical display of the grade distribution derived from a series of bands, or swaths, generated in several directions through the deposit. Grade variations from the ID3 model are then compared (using the swath plot) to the distribution derived from the NN grade model.

On a local scale, the NN model does not provide reliable estimations of grade, but on a larger scale it represents an unbiased estimation of the grade distribution based on the underlying data. Therefore, if the ID3 model is unbiased, the grade trends may show local fluctuations on a swath plot, but the overall trend should be similar to the NN distribution of grade.

Swath plots have been generated in three orthogonal directions for the distribution of gold in the model area. Swath plots for gold along the east-west, north-south and vertical directions are shown in Figure 14-10, Figure 14-11 and Figure 14-12.

There is good correspondence between all models in all orthogonal directions. The degree of smoothing in the ID3 model is evident in the peaks and valleys shown in the swath plots, however, this comparison shows close agreement between the ID3 and NN models in terms of overall grade distribution as a function of X, Y and Z location.

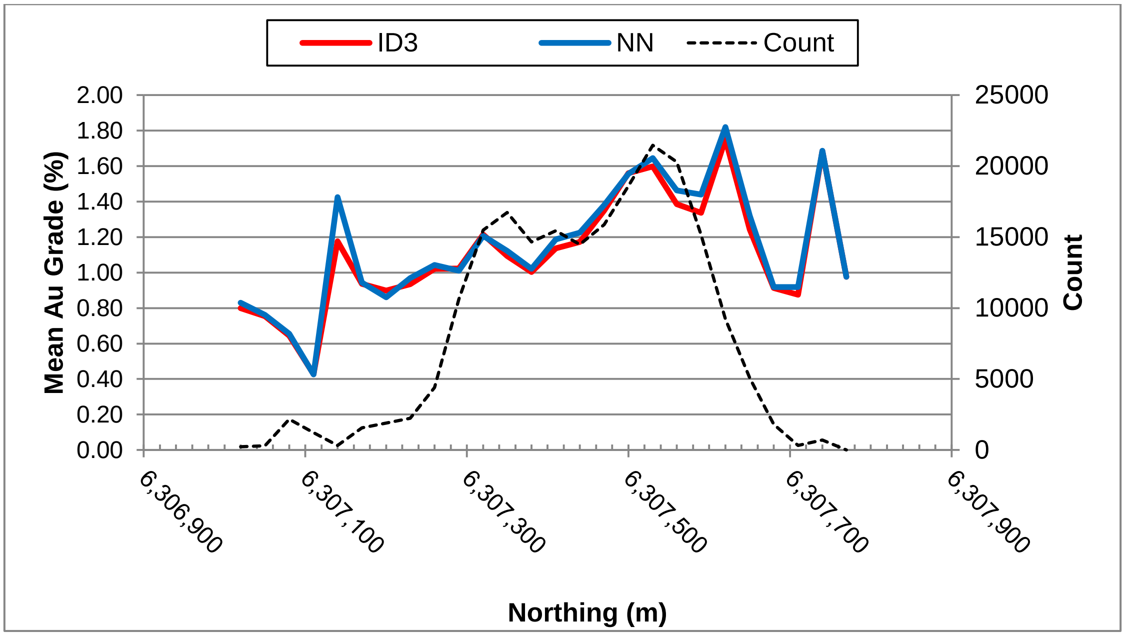


Figure 14-10: E-W Swath Plot, Comparing IDW and NN Model Gold Grades

Source: Alamos (2017)

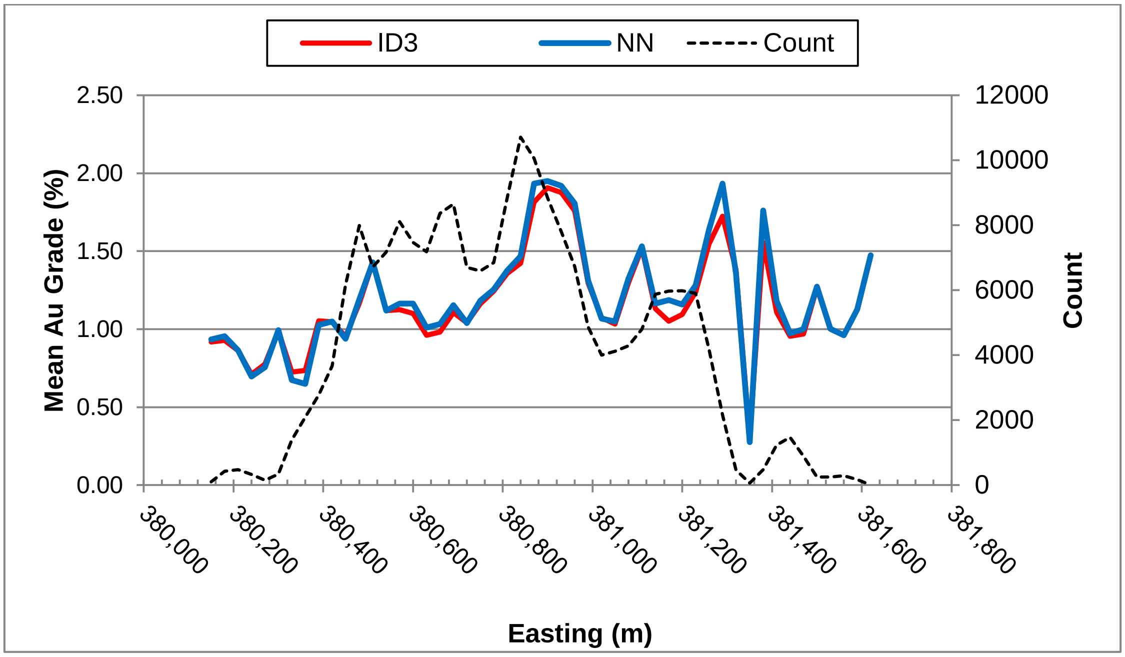


Figure 14-11: N-S Swath Plot, Comparing IDW and NN Model Gold Grades

Source: Alamos (2017)

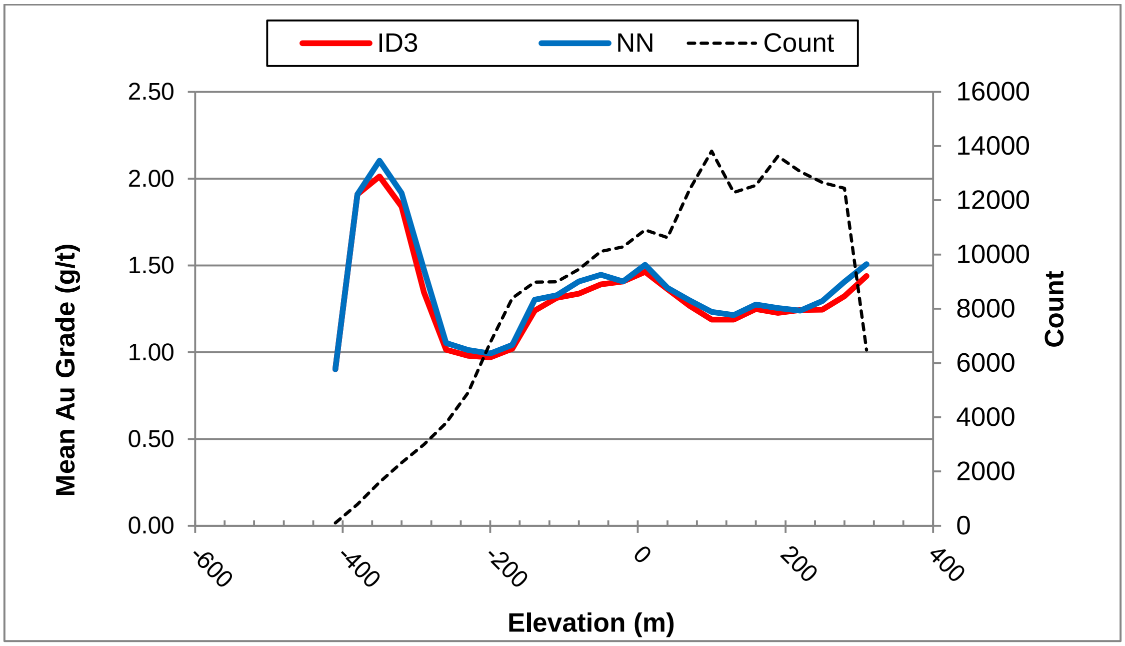


Figure 14-12: Vertical Swath Plot, Comparing IDW and NN Model Gold Grade

Source: Alamos (2017)

14.1.13 Mineral Resource Sensitivity

To assess the sensitivity of the Mineral Resource to changes in gold cut-off grade, Alamos has summarized diluted tonnage and grade above cut-off for all estimated blocks, at a series of increasing gold cut-offs by Mineral Resource category. The cut-off grade sensitivity analysis for all Measured and Indicated blocks (depleted for past mining and inclusive of Mineral Reserves) within the MacLellan deposit are provided in Table 14-8. The cut-off grade sensitivity analysis for inferred blocks within the MacLellan deposit are provided in Table 14-9. It can be observed that the Mineral Resource is reasonably insensitive to cut-off grades in the increment between 0.40 g/t and 0.60 g/t Au, which is likely the grade range of the ultimate open pit cut-off grade. Note that these summaries are not constrained by any pit.

Table 14-8 – Cut-off Gold Grade Sensitivity – All Measured and Indicated Blocks

Au Cut-off	k-Tonnes	Au Grade (g/t)	Ag Grade (g/t)	Au koz	Ag koz
0.20	55,219	1.15	3.02	2,048	5,360
0.30	52,362	1.21	3.13	2,035	5,269
0.40	47,210	1.31	3.31	1,995	5,021
0.50	42,249	1.43	3.49	1,942	4,741
0.60	37,730	1.55	3.68	1,882	4,463
0.70	33,802	1.67	3.85	1,818	4,184
0.80	30,326	1.80	4.03	1,752	3,930
0.90	27,307	1.92	4.20	1,686	3,690
1.00	24,693	2.04	4.38	1,621	3,473

Table 14-9 – Cut-off Gold Grade Sensitivity – All Inferred Blocks

Au Cut-off	k-Tonnes	Au Grade (g/t)	Ag Grade (g/t)	Au koz	Ag koz
0.20	19,769	0.90	1.52	575	969
0.30	18,694	0.95	1.56	569	938
0.40	16,352	1.04	1.62	550	853
0.50	14,645	1.13	1.68	530	790
0.60	13,100	1.21	1.73	509	729
0.70	10,970	1.34	1.81	473	638
0.80	9,280	1.48	1.88	441	562
0.90	7,932	1.61	1.93	411	491
1.00	6,919	1.74	1.96	386	437

14.1.14 Mineral Resource Classification

The Mineral Resources for the MacLellan deposit are classified under the categories of Measured, Indicated and Inferred according to the guidelines as defined by the “CIM Definition Standards for Mineral Resources and Mineral Reserves”, prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council on May 10, 2014.

Classification of the Mineral Resources reflects the relative confidence of the grade estimates. This is based on several factors including; sample spacing relative to geological and geostatistical observations regarding the continuity of mineralization, mining history, specific gravity determinations, accuracy of drill collar locations, quality of the assay data and other factors which can influence the confidence of the mineral estimation.

The classification parameters are defined in relation to the number of drill holes used to estimate the block grades and the block-composite separation distance. These classification criteria are intended to encompass zones of reasonably continuous mineralization.

The following classification parameters were applied to the MacLellan block model:

Measured Mineral Resources

Blocks in the model which are within the 0.50 g/t Au solid that were informed by a minimum of three drill holes on the first estimation search pass. (20 m x 20 m x 5 m)

Indicated Mineral Resources

Blocks in the model which are within the 0.50 g/t Au solid that were informed by a minimum of two drill holes on the second estimation search pass (45 m x 45 m x 10 m).

Inferred Mineral Resources

Blocks in the model that do not meet the criteria for Measured or Indicated Resources and have been informed by a minimum of one drill hole on the third estimation search pass (100 m x 100 m x 20 m).

14.1.15 MacLellan Mineral Resource Statement

The Mineral Resources for the MacLellan deposit have been estimated by Alamos at 4,347 kt grading an average of 1.62 g/t Au classified as Measured and Indicated Mineral Resources; with an additional 750 kt grading an average of 1.62 g/t Au classified as Inferred Mineral Resources. The Mineral Resources are stated above a 0.42 g/t Au equivalent cut-off and are contained within a potentially economically mineable open pit. Mineral Resources are exclusive of Mineral Reserves.

The Mineral Resources are reported in accordance with NI 43-101 and have been classified in conformity with generally accepted CIM “Estimation of Mineral Resource and Mineral Reserves Best Practices” guidelines. The Mineral Resource estimate was completed by Mr. Jeffrey Volk, CPG, FAusIMM, Director of Reserves and Resources for Alamos. Mr. Volk is a Qualified Person within the meaning of Canadian Securities Administrator’s National Instrument 43-101 (NI 43-101)

The effective date of this Mineral Resource estimate is December 01, 2017 and is based on drilling data finalized in December, 2016. The Mineral Resource statement for the MacLellan Property is presented in Table 14-10.

Table 14-10 – MacLellan Mineral Resource Statement, December 01, 2017

	MacLellan				
Category	**kTonnes**	**Au Grade (g/t)**	**Ag Grade (g/t)**	**Au koz**	**Ag koz**
Measured	2,110	1.86	5.34	126	362
Indicated	2,236	1.24	4.24	89	305
Total Measured and Indicated	***4,347***	***1.54***	***4.77***	***215***	***667***
Inferred	750	1.62	2.80	39	67

Notes:

- *The Mineral Resources are reported at an assumed gold price of US $1,400/ounce, and an assumed silver price of US $22.00/ounce;*
- *The Mineral Resource estimate was completed by Mr. Jeffrey Volk, CPG, FAusIMM, Director of Reserves and Resources for Alamos Gold Inc.;*
- *Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resources estimated will be converted into Mineral Reserves;*
- *Open pit Mineral Resources are stated as contained within potentially economically open pit above a 0.42 g/t AuEq cut-off, and includes external dilution at zero grade outside the 0.50 g/t Au solid;*
- *Totals may not add due to rounding;*
- *Contained Au and Ag ounces are in-situ and do not include metallurgical recovery losses; and*
- *Mineral Resources are exclusive of Mineral Reserves.*

14.2 Gordon

14.2.1 Drill Hole Database

Alamos conducted a Mineral Resource estimate of the Gordon deposit, which incorporates all drilling data from the Alamos and predecessor drilling programs conducted through December 31, 2016. A database was compiled using data from 473 drill holes, with collar, survey, geological and assay information, containing a total of 69,213 m of assayed gold intervals. Historic drilling conducted by previous operators was included where data could be verified.

14.2.2 Topography

Existing topography is based on a 2014 LiDAR survey conducted by LSI, and was used to code the Mineral Resource model. Alamos has reviewed the topography surface in cross sections comparing elevation between the drill hole collars and the topography surface and found close agreement between the two.

14.2.3 Coordinate System

All data is located in NAD 83 Zone 17 geodetic Datum. The local mine coordinate system utilized a translated and rotated version of the above coordinate system during historic mining operations. A number of pre-2015 drill hole collars (28) were located by Barnes and Duncan in a December 2014 survey and were found to be accurately located in the digital drill hole collar database.

14.2.4 Lithology and Grade Modelling

A 0.50 g/t Au grade solid was constructed using Leapfrog modeling software and manually edited to eliminate small volume solids and outlier tonnages. A northwest-southeast cross section showing the resultant solid is provided in Figure 14-13.

A generalized lithologic model was also constructed using Leapfrog modeling software and utilized to assign SG to the model blocks. A total of six generalized lithologies were modeled:

1) Argillite;
2) Mafic Dike;
3) Volcanic;
4) Intrusive;
5) Sedimentary; and
6) BIF.

SG assignments based on these modeled lithologies are described in a subsequent section of this report.

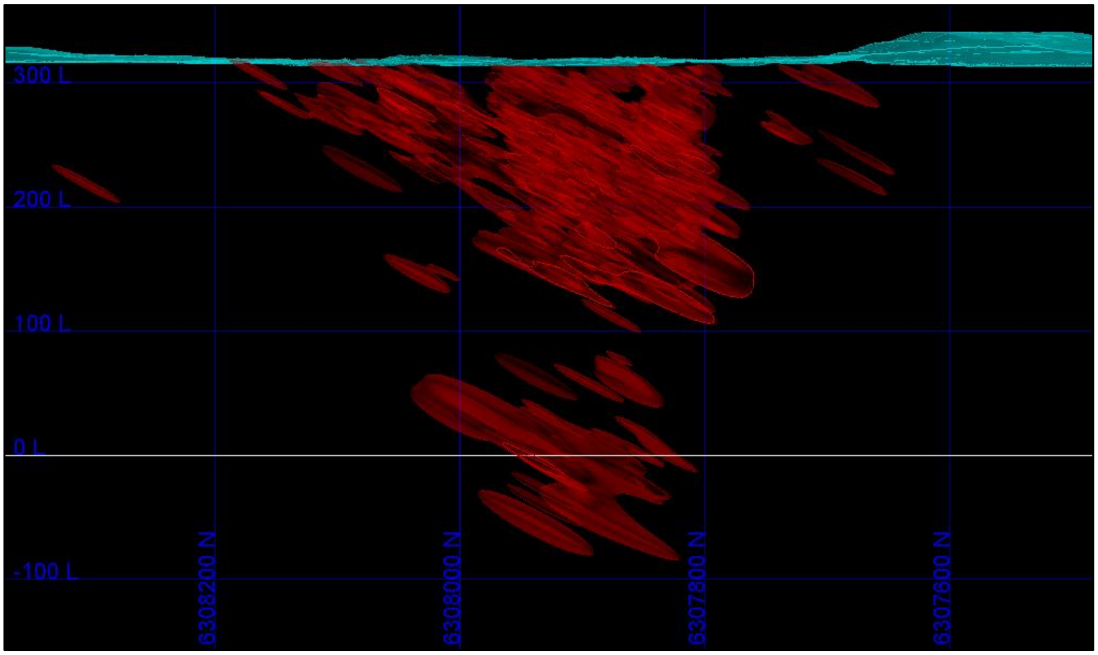


Figure 14-13: Northwest-SouthEast Cross Section Viewed to the North Showing 0.50 g/t Au Solid

Source: Alamos (2017)

14.2.5 Exploratory Data Analysis

EDA was conducted using the raw gold assays from the Gordon Property. These data were analyzed above both global and incremental cut-offs to assess the distribution and grade ranges of the assays internal and external to the 0.50 g/t Au solid. Basic statistics for the raw gold assays are provided in Table 14-11. It can be observed that on a grade-thickness basis, a minor amount of metal above potentially economic cut-off resides external to the 0.50 g/t solid. Most of the intervals external to the grade solid are not of minable width, however this demonstrates that gold occurs external to the main lode system. Statistically, the 0.50 g/t Au solid envelops ~84% of all of the raw assay data in the model area above a 0.50 g/t Au cut-off.

Table 14-11 – Summary Statistics – Raw Gold Grades by Domain

Domain	Au Cut-off (g/t)	Statistics above Cut-off							
		Total Meters	Percent	Max Grade (Au g/t)	Mean Grade (Au g/t)	Grade Thickness (g/t*m)	Incremental Percent	Standard Deviation	Coeff. of Variation
All Data	0.01	36,432	78.03%	348.45	1.71	62,256	45.70%	4.71	2.76
	0.50	8,004	13.62%		4.22	33,816	8.50%	9.13	2.16
	3.00	3,041	2.33%		9.37	28,504	2.20%	12.82	1.37
	5.00	2,193	6.02%		12.38	27,153	43.60%	14.26	1.15
Internal to 0.5 g/t Au Solid	0.01	14,522	52.47%	348.45	2.72	39,472	19.30%	7.03	2.59
	0.50	6,903	27.48%		4.61	31,841	10.90%	9.5	2.06
	3.00	2,912	5.39%		9.46	27,543	3.20%	12.73	1.35
	5.00	2,130	14.66%		12.34	26,285	66.60%	14.08	1.14
External to 0.5 g/t Au Solid	0.01	21,911	94.98%	92.80	1.10	24,163	86.20%	1.35	1.22
	0.50	1,101	4.43%		3.04	3,341	5.10%	5.67	1.87
	3.00	129	0.30%		16.40	2,119	1.30%	14.58	0.89
	5.00	63	0.29%		28.55	1,796	7.40%	19.22	0.67

14.2.6 Specific Gravity Analysis

Company geology personnel conducted 680 specific gravity tests using the water immersion method, on diamond drill core from the property during the 2015 drilling program, as well as analyses conducted by TSL during the 2007-2012 Carlisle drilling campaigns. A summary of specific gravity (SG) determinations by rock type is provided in Table 14-12.

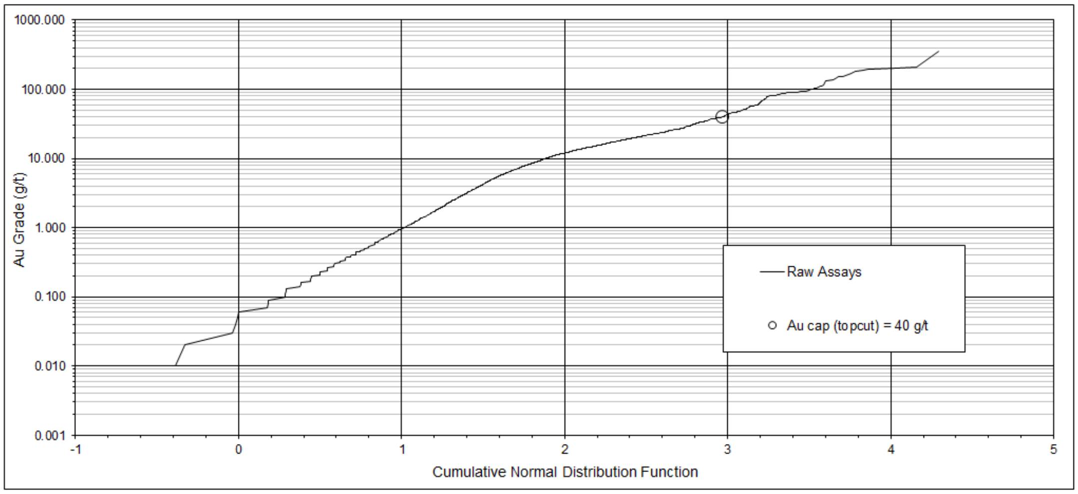
A default SG value of 2.90 was assigned to modeled lithologies that did not receive any SG determinations.

Table 14-12 – SG Results for Gordon 2015 Samples

Lithology	Number	Min SG	Max SG	Avg SG
Argillite	0			2.90
Mafic Dike	0			2.90
Volcanic	61	2.71	3.81	2.90
Intrusive	289	1.85	3.45	2.81
Sediment	10	2.78	2.87	2.81
BIF	312	2.08	3.66	3.07
Other	8	2.75	3.46	2.97
Total	680	1.85	3.81	2.94

14.2.7 Evaluation of Outlier Data

The raw drill hole gold assay dataset was inspected globally using log cumulative probability plots to assess for the presence of high-grade outlier values that could adversely impact grade estimation. For this analysis, the datasets both external and internal to the 0.50 g/t Au grade solid were combined, as they are one contiguous deposit. After review of log probability plots, all raw gold assays were capped at 40.0 g/t Au prior to compositing (Figure 14-14). This assay cap (topcut) affects 55 meters of sample, and results in a reduction of 5.14% of gold metal on a grade-thickness (GT) basis. The capped data shows a reduction in the CV to 3.12 as compared to 4.21 for the uncapped data.



	Untransformed Au Statistics								Log Normal Approximation Model		
	Au Cutoff = 0.01 g/t		Au Cutoff = 0.50 g/t		Au Cutoff = 3.00 g/t		Au Cutoff = 40.00 g/t			Standard	Third
	Meters	Au (g/t)	Meters	Au (g/t)	Meters	Au (g/t)	Meters	Au (g/t)	Mean	Deviation	Parameter
Raw Assays	36,432	1.119	8,004	4.818	3,041	10.629	55	77.415	-1.12	0.90	0.00
incr. % and grade	78.0%	0.078	13.6%	1.256	8.2%	9.390	0.1%	77.415			
low cut	0.10		40 g/mt percentile		GT lost by capping		percent of GT >= 348 g/mt				
			99.64%		5.14%		0.52%				
Au cap (topcut)	40.00		percent of GT >= 40 g/mt		CV uncapped		CV capped				
			10.64%		4.21		3.12				

Figure 14-14: Cumulative Probability Plot: Raw Gold Assays

Source: Alamos (2017)

14.2.8 Compositing

All capped raw data were composited into 5 m downhole intervals. Composites were back-flagged by the model blocks to assign percentage of composite internal to the 0.5 g/t Au wireframe. Only composites with a wireframe percentage ≥10% were retrieved for use in grade estimation. All composites were length weighted during the grade estimation process.

14.2.9 Variogram Analysis

Directional correlograms grams were constructed for gold in Sage2001© software using the 5 m capped composite data. The resulting variograms exhibited moderately high nuggets and moderate ranges overall. The resultant fitted variograms for the major, semi-major and minor axes are provided in Figure 14-15, Figure 14-16 and Figure 14-17, respectively. A summary of variogram parameters is provided in Table 14-13.

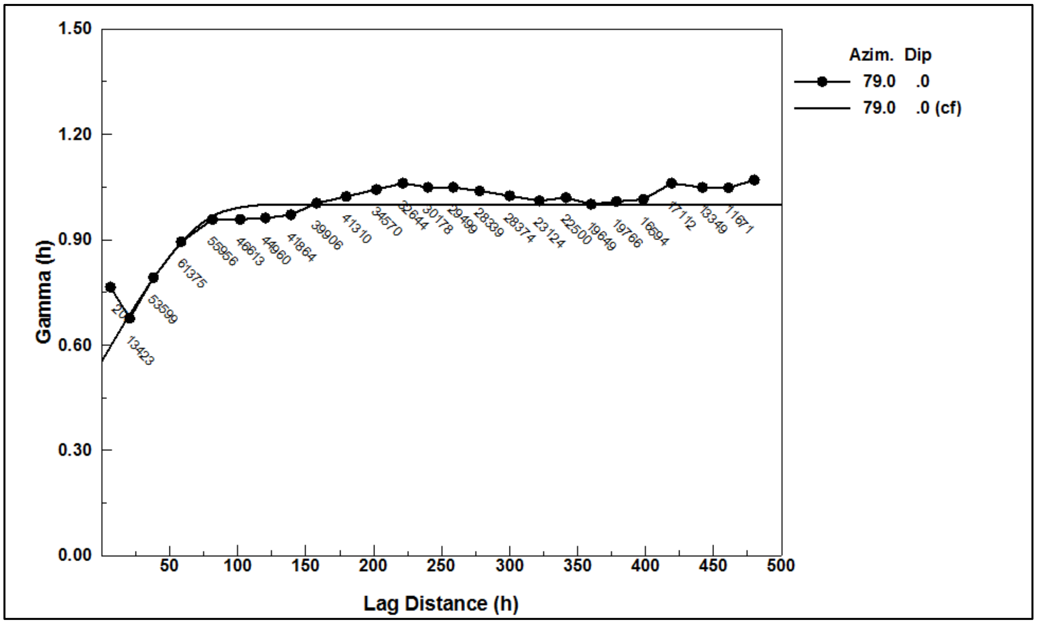


Figure 14-15: Gordon Correlogram and Model – 5 m Composites: Principle Axis

Source: Alamos (2017)

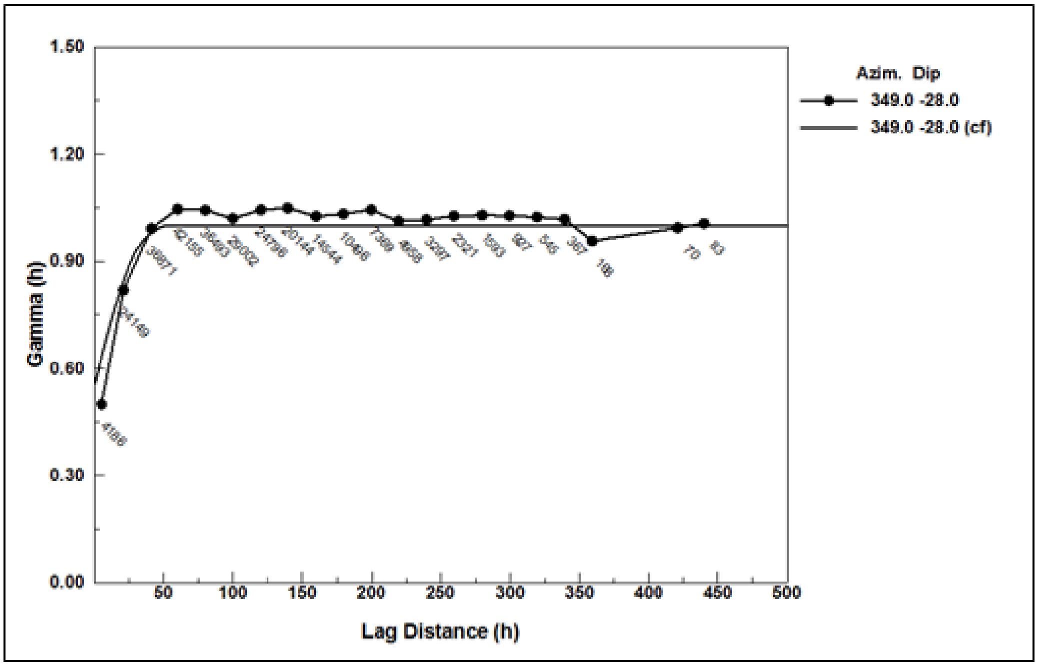


Figure 14-16: Gordon Correlogram and Model – 5 m Composites: Semi-Major Axis

Source: Alamos (2017)

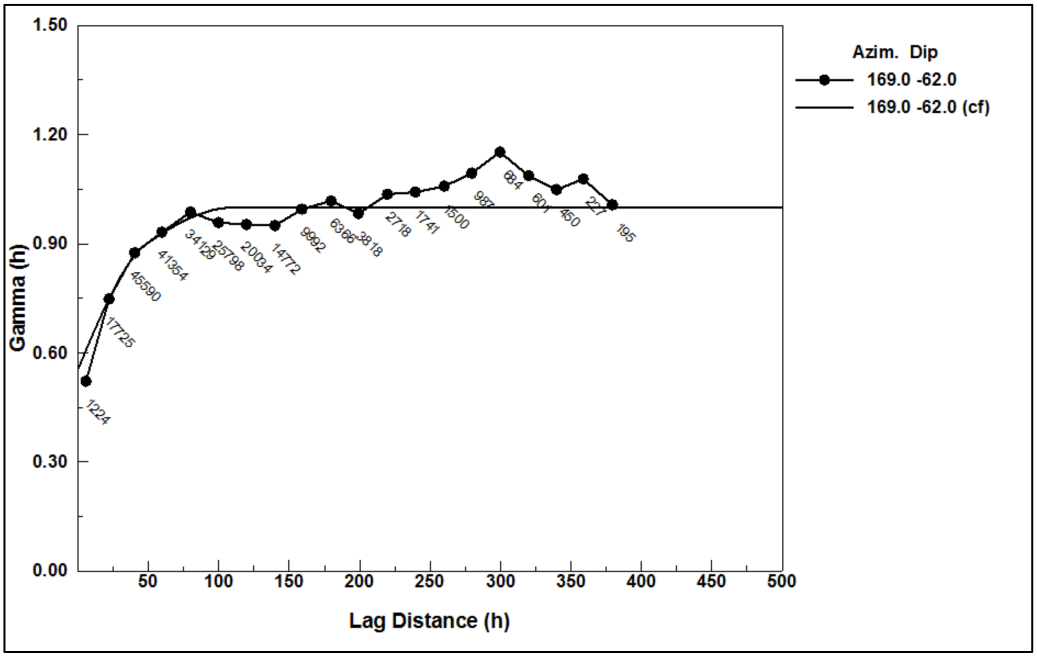


Figure 14-17: Gordon Correlogram and Model – 5 m Composites: Minor Axis

Source: Alamos (2017)

Table 14-13 – Correlogram Model Parameters for Au: 5 m Composites

Gordon: Variogram Results (Spherical Model)			
Parameters (First Structure)	**Principal**	**Minor**	**Semi-Major**
Azimuth (deg)	346	199	90
Dip (deg)	38	47	17
Parameters (Second Structure)	**Principal**	**Minor**	**Semi-Major**
Azimuth (deg)	90	1	328
Dip (deg)	6	-10	78
Nugget Effect C_0	0.555		
1st Structure C_1	0.258		
2nd Structure C_2	0.187		
1st Range A_1 (m)	209	46	118
2nd Range A_2 (m)	170	36	53

14.2.10 Block Model Construction

A block model was constructed in Vulcan™ for the Gordon deposit using the model limits and extents provided in Table 14-14.

Block model construction utilized a constant block size of 5.0 m x 5.0 m x 5.0 m.

Table 14-14 – Gordon Model Limits and Extents

Mine Grid Origin[1] (m)	Min (m)	Max (m)	Block Size (X)	Block Size (Y)	Block Size (Z)	No. of Blocks
East	412,014	413,034	5.0	5.0	5.0	204
North	6,307,344	6,308,179	5.0	5.0	5.0	167
Elevation	-174	426	5.0	5.0	5.0	120

Notes:

1. *All blocks are rotated 79.4° about model origin*

14.2.11 Block Model Grade Estimation

The Mineral Resource estimate was undertaken using Maptek Vulcan™ software employing the inverse ID3 and NN method for use in model validation. Model blocks were assigned a percentage of the block internal to the 0.50 g/t Au wireframe and only blocks whose volumes were ≥10% internal to the wireframe were available for grade estimation. A summary of the search parameters utilized is presented in Table 14-15. Model blocks grades were diluted to account for all volume external to the 0.50 g/t Au wireframe at a zero grade. Additional estimates were performed for NP and AP to assist in the overall ABA studies related to mine planning. The methodology used for these latter estimates are described in more detail in subsequent sections of this report.

Table 14-15 – Search Parameters for the Gordon Model – Gold

Pass Number	Search Orientation[1] (degrees)			Search Distance (m)			Min Comps	Max Comps	Max per DDH
	Bearing (Z)	Plunge (Y)	Dip (X)	Major Axis	Semi-Major Axis	Minor Axis			
1	79.4	0	-28	20	20	5	3	8	1
2	79.4	0	-28	45	45	10	2	8	1
3	79.4	0	-28	100	100	20	1	8	1

Notes:

1. *Vulcan rotations*

14.2.12 Block Model Validation

Various measures have been utilized to validate the resultant Mineral Resource block model. These measures include the following:

- Comparison of drillhole composites with Mineral Resource block grade estimates by zone visually, both in plan and section;
- Statistical comparisons between block and composite data using histogram and cumulative frequency distribution analysis;
- Generation of comparative NN model; and
- Swath plot analysis (drift analysis) comparing the ID3 model with the NN model.

14.2.12.1 Visual Inspection

Visual comparisons between the block grades and the underlying composite grades in plan and section show close agreement, which would be expected considering the estimation methodology employed.

Representative north-northwest – south-southeast cross sections and level plans displaying block and composite gold grade, 0.50 g/t Au grade solid outlines, US $1,400 Mineral Resource pit (magenta) and historic pits (blue) are provided in Figure 14-18, Figure 14-19 and Figure 14-20.

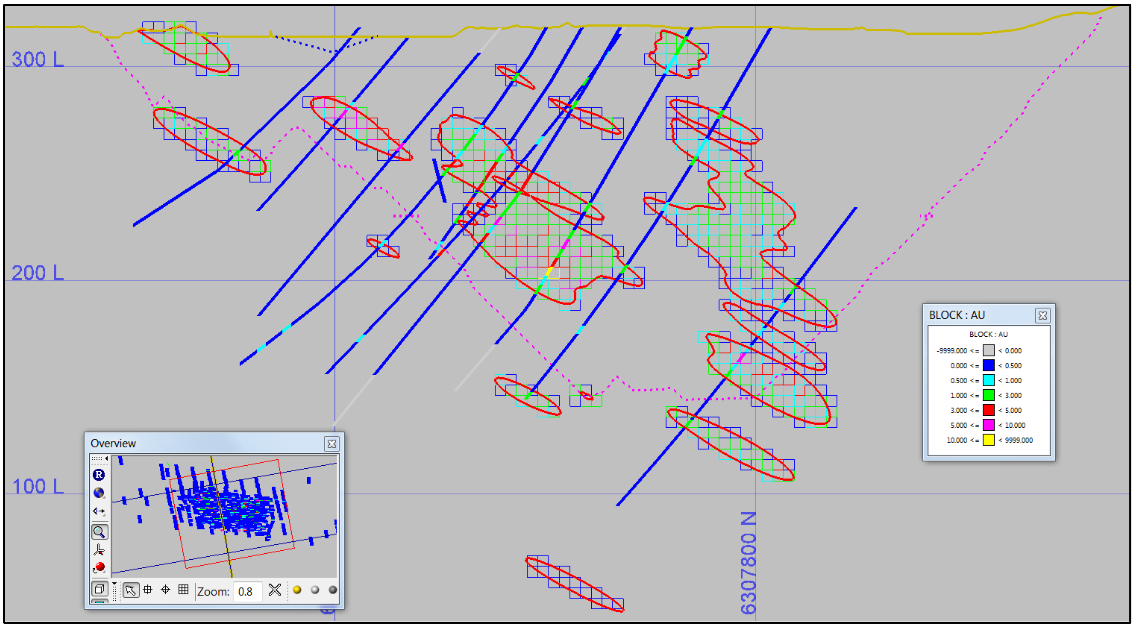


Figure 14-18: Example North-northwest - South-southeast Cross Section 1 Viewed to East

Source: Alamos (2017)

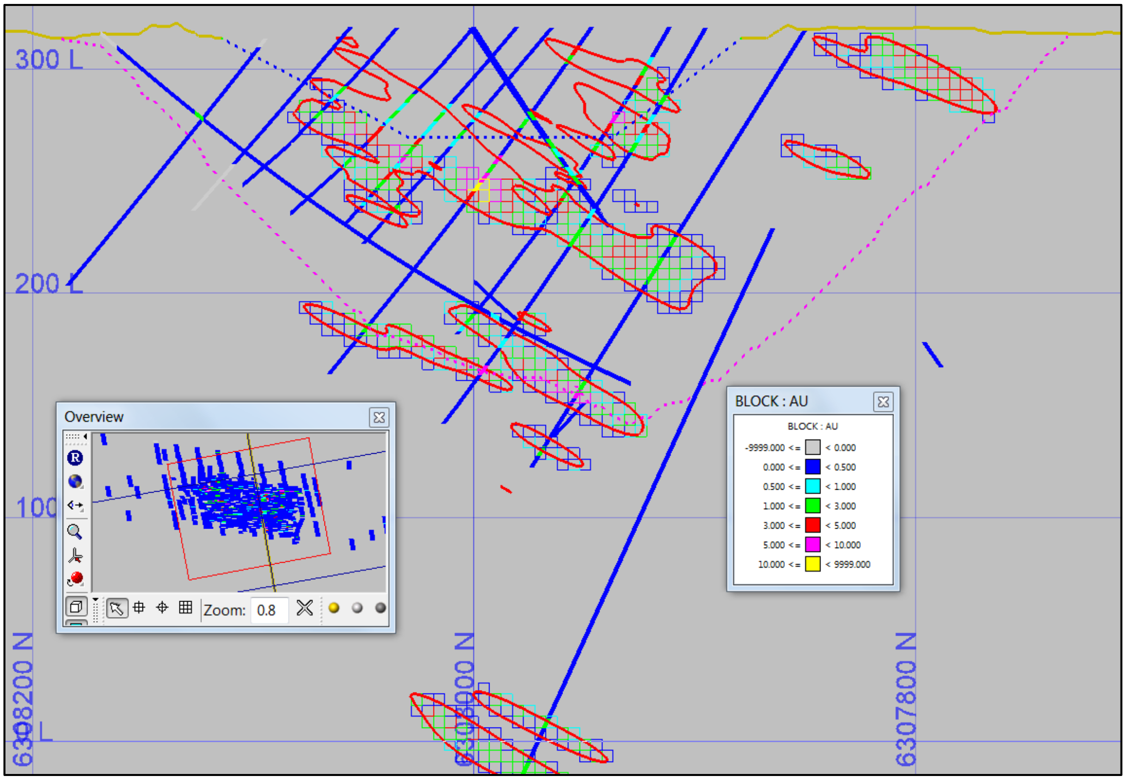


Figure 14-19: Example North-northwest - South-southeast Cross Section 2 Viewed to East

Source: Alamos (2017)

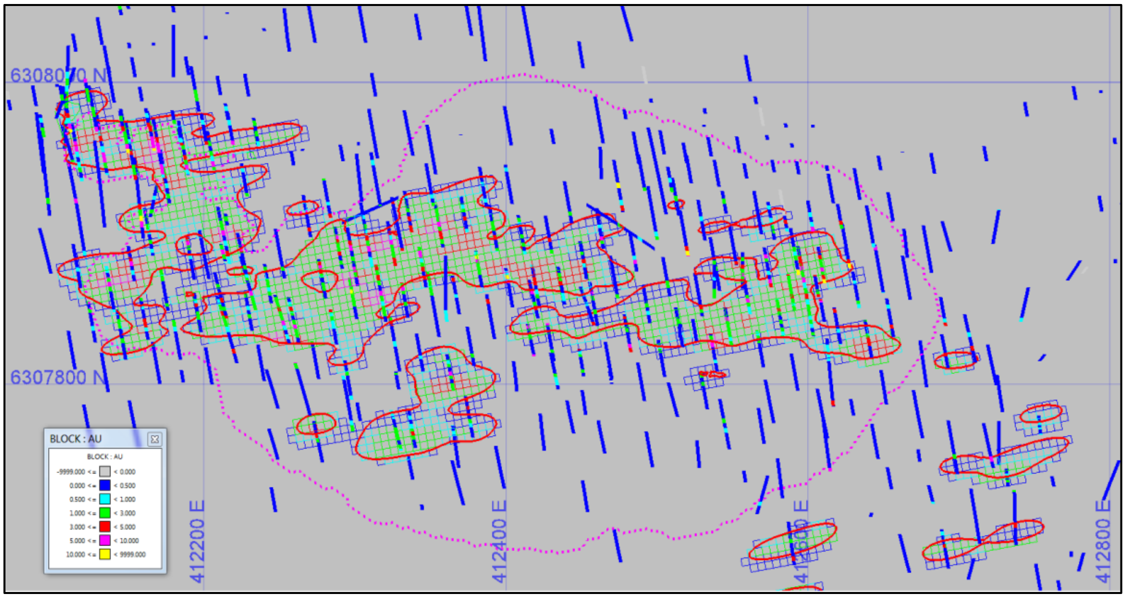
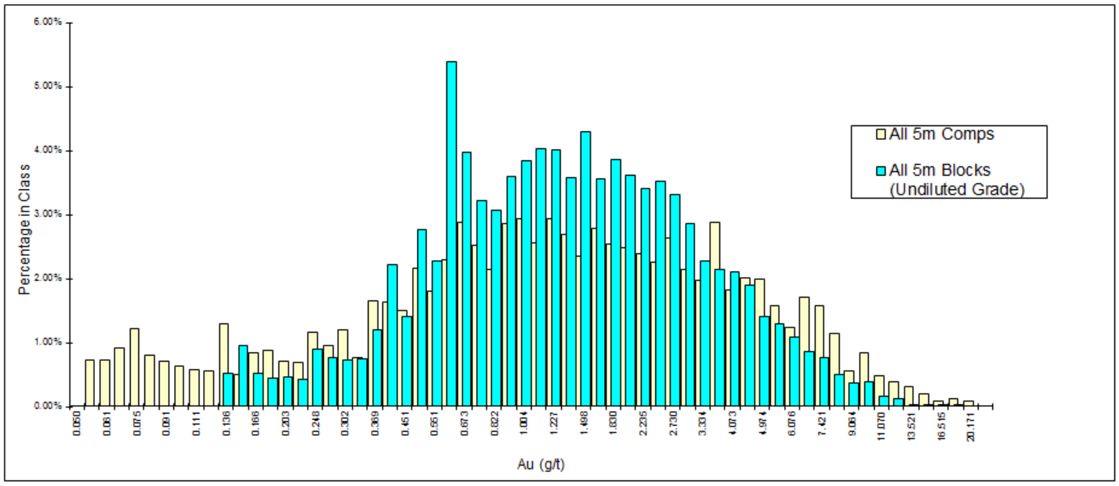


Figure 14-20: Example Level Plan at 230 m

Source: Alamos (2017)

14.2.12.2 Block-Composite Histogram Comparison

Alamos also conducted statistical comparisons between the grades of the Measured, Indicated and Inferred ID3 blocks contained within the 0.50 g/t Au solid and the underlying 5m gold composite grades. A histogram comparison between block and composite gold grades at the Gordon Property is provided in Figure 14-21.



Untransformed Au Statistics

	Au Cutoff = 0.010 g/t		Au Cutoff = 0.500 g/t		Au Cutoff = 1.50 g/t		Au Cutoff = 5.00 g/t	
	meters/tonnes	Au (g/t)	meters/tonnes	Au (g/t)	meters/tonnes	Au (g/t)	meters/tonnes	Au (g/t)
All 5m Comps	16,640	2.124	11,408	3.024	6,779	4.452	2,044	8.187
incr. % and grade	31.4%	0.161	27.8%	0.933	28.5%	2.840	12.3%	8.187
All 5m Blocks (Undiluted Grade)	19,966,979	1.901	16,734,274	2.221	8,782,481	3.397	1,408,374	7.237
incr. % and grade	16.2%	0.246	39.8%	0.922	36.9%	2.664	7.1%	7.237

Figure 14-21: Histogram Comparison between Block and Composite Grades: Gold

Source: Alamos (2017)

Overall, this comparison shows that the model grade distribution for gold is appropriately smoothed when compared with the underlying composite distribution, and that the comparison of average grades and percentages above geologic absolute and incremental cut-offs show reasonably close agreement.

14.2.12.3 Comparison of Interpolation Methods

For comparative purposes, additional grades were estimated using NN interpolation methods. The results of the NN model are compared to the ID3 model at a 0 g/t Au cut-off grade for the Measured and Indicated blocks in Table 14-16, and for all Inferred blocks in Table 14-17. These comparisons confirm the conservation of metal at a zero cut-off, and shows close agreement on both a tonnage and grade basis within the deposit area.

Table 14-16 – ID3 vs. NN Tonnage – All Measured and Indicated Model Blocks

Model	Tonnes	Au Grade (g/t)	Au oz
ID3	15,066,745	2.065	1,000,178
NN	15,066,745	2.092	1,013,595
Pct Diff (ID3 to NN)	0.00%	-1.34%	-1.34%

Table 14-17 – ID3 vs. NN – All Inferred Model Blocks

Model	Tonnes	Au Grade (g/t)	Au oz
ID^3	4,945,875	1.405	223,486
NN	4,945,875	1.443	229,501
Pct Diff (ID^3 to NN)	0.00%	-2.69%	-2.69%

14.2.12.4 Swath Plots (Drift Analysis)

A swath plot is a graphical display of the grade distribution derived from a series of bands, or swaths, generated in several directions through the deposit. Grade variations from the ID3 model are then compared (using the swath plot) to the distribution derived from the NN grade model.

On a local scale, the NN model does not provide reliable estimations of grade, but on a larger scale it represents an unbiased estimation of the grade distribution based on the underlying data. Therefore, if the ID3 model is unbiased, the grade trends may show local fluctuations on a swath plot, but the overall trend should be similar to the NN distribution of grade.

Swath plots have been generated in three orthogonal directions for the distribution of gold in the model area. Swath plots for gold along the east-west, north-south and vertical directions are shown in Figure 14-22, Figure 14-23 and Figure 14-24.

There is good correspondence between all models in all orthogonal directions. The degree of smoothing in the ID3 model is evident in the peaks and valleys shown in the swath plots, however, this comparison shows close agreement between the ID3 and NN models in terms of overall grade distribution as a function of X, Y and Z location.

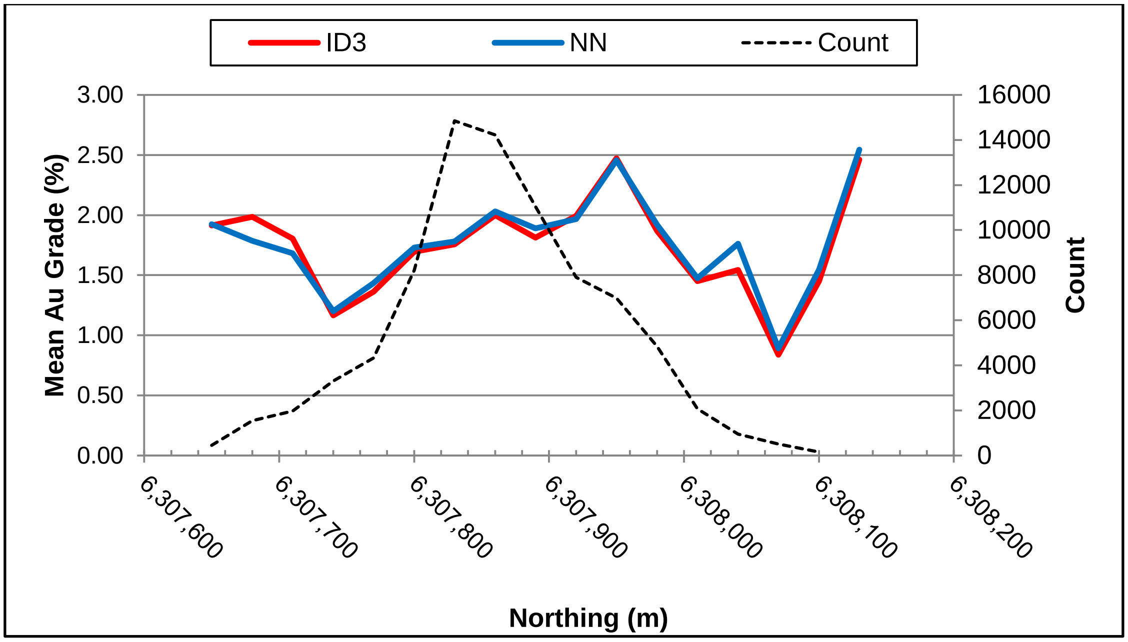


Figure 14-22: E-W Swath Plot, Comparing IDW and NN Model Gold Grades

Source: Alamos (2017)

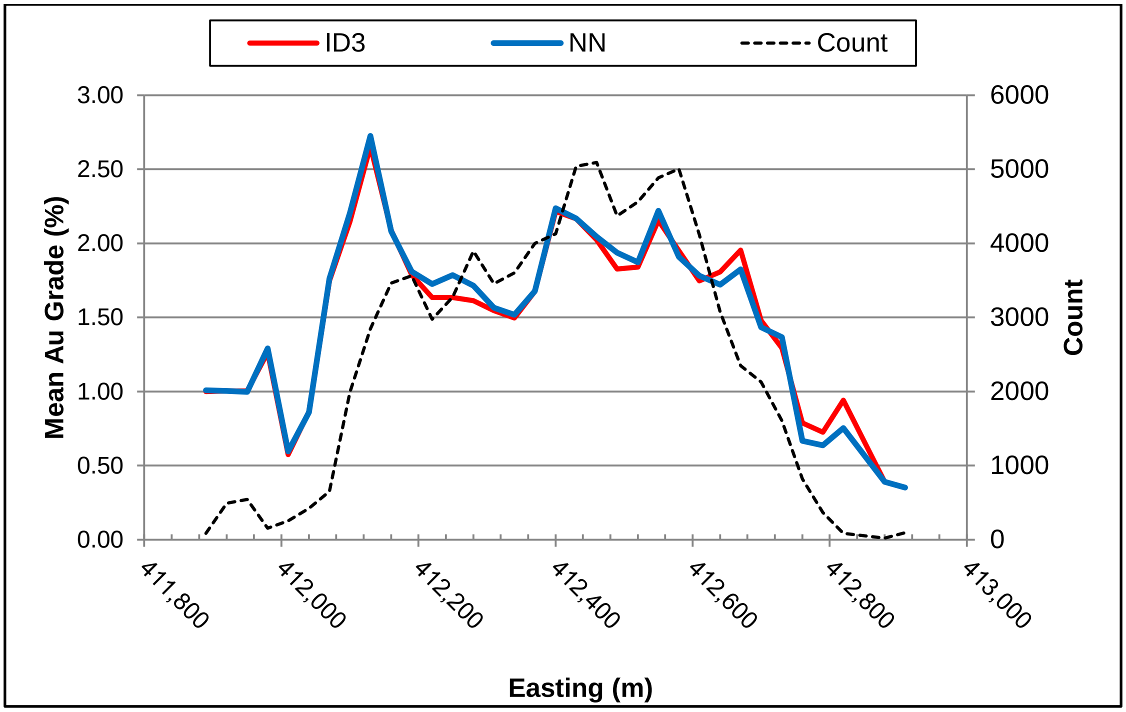


Figure 14-23: N-S Swath Plot, Comparing IDW and NN Model Gold Grades

Source: Alamos (2017)

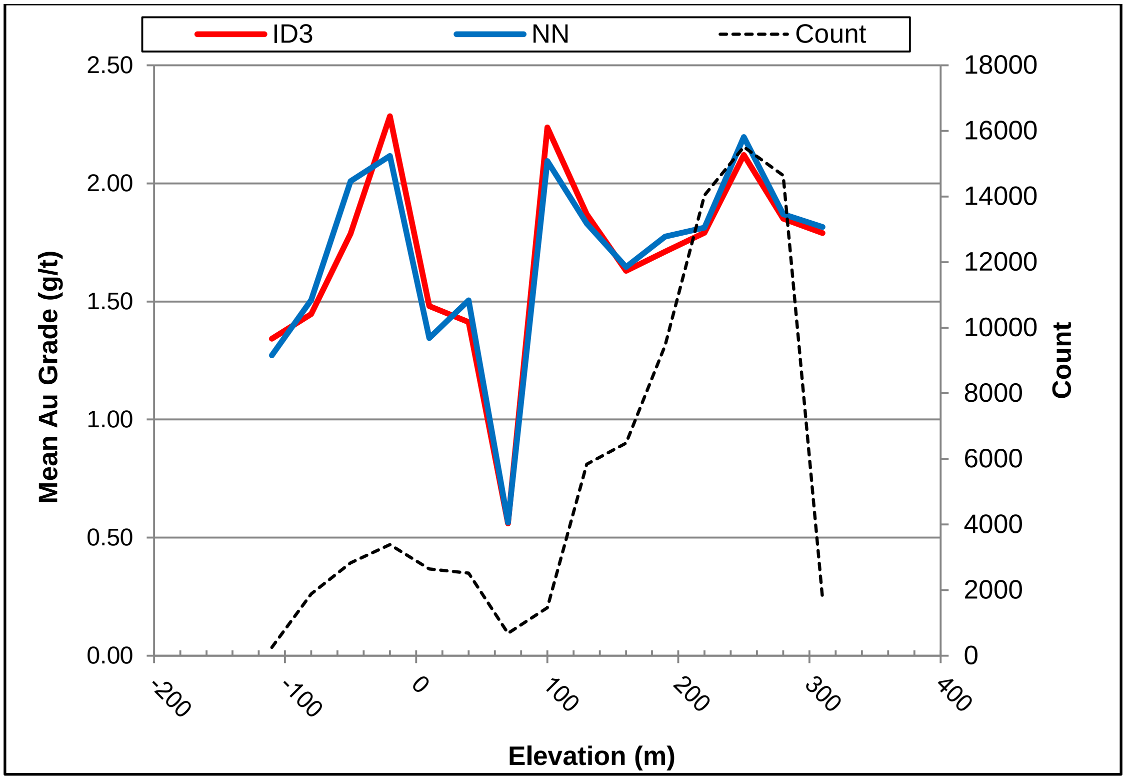


Figure 14-24: Vertical Swath Plot, Comparing IDW and NN Model Gold Grade

Source: Alamos (2017)

14.2.13 Mineral Resource Sensitivity

To assess the sensitivity of the Mineral Resource to changes in gold cut-off grade, Alamos has summarized tonnage and grade above cut-off for the fully diluted blocks, at a series of increasing gold cut-offs by Mineral Resource category. The cut-off grade sensitivity analysis for all Measured and Indicated blocks within the Gordon deposit are provided in Table 14-8. The cut-off grade sensitivity analysis for Inferred blocks within the Gordon deposit are provided in Table 14-19. It can be observed that the Mineral Resource is reasonably insensitive to cut-off grades in the increment between 0.60 g/t and 0.80 g/t Au, which is likely the grade range of the ultimate open pit cut-off grade. Note that these summaries are not constrained by any pit.

Table 14-18 – Cut-off Gold Grade Sensitivity – All Measured and Indicated Blocks

Cut-off	k-Tonnes	Au Grade (g/t)	Au koz
0.20	17,286	1.76	979
0.30	15,961	1.89	968
0.40	14,722	2.02	954
0.50	13,627	2.14	939
0.60	12,597	2.27	920
0.70	11,639	2.41	900
0.80	10,774	2.54	880
0.90	10,024	2.67	859
1.00	9,323	2.80	838

Table 14-19 – Cut-off Gold Grade Sensitivity – All Inferred Blocks

Cut-off	k-Tonnes	Au Grade (g/t)	Au koz
0.20	5,285	1.28	218
0.30	4,781	1.39	214
0.40	4,328	1.50	209
0.50	3,827	1.64	202
0.60	3,343	1.80	193
0.70	2,791	2.03	182
0.80	2,523	2.16	176
0.90	2,335	2.27	170
1.00	2,152	2.38	165

14.2.14 Mineral Resource Classification

The Mineral Resources for Gordon are classified under the categories of Measured, Indicated and Inferred according to the guidelines as defined by the “CIM Definition Standards for Mineral Resources and Mineral Reserves”, prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council on May 10, 2014.

Classification of the Mineral Resources reflects the relative confidence of the grade estimates. This is based on several factors including; sample spacing relative to geological and geo-statistical observations regarding the continuity of mineralization, mining history, specific gravity determinations, accuracy of drill collar locations, quality of the assay data and other factors which can influence the confidence of the mineral estimation.

The classification parameters are defined in relation to the number of drill holes used to estimate the block grades and the block-composite separation distance. These classification criteria are intended to encompass zones of reasonably continuous mineralization.

The following classification parameters were applied to the Gordon block model:

Measured Mineral Resources

Blocks in the model which are within the 0.50 g/t Au solid that were informed by a minimum of three drill holes on the first estimation search pass. (20 m x 20 m x 5 m).

Indicated Mineral Resources

Blocks in the model which are within the 0.50 g/t Au solid that were informed by a minimum of two drill holes on the second estimation search pass (45 m x 45 m x 10 m).

Inferred Mineral Resources

Blocks in the model that do not meet the criteria for Measured or Indicated Resources and have been informed by a minimum of one drill hole on the third estimation search pass (100 m x 100 m x 20 m).

14.2.15 Gordon Mineral Resource Statement

The Mineral Resources for the Gordon deposit have been estimated by Alamos at 460 kt grading an average of 1.96 g/t Au classified as Measured and Indicated Mineral Resources; with an additional 620 kt grading an average of 1.30 g/t Au classified as Inferred Mineral Resources. The Mineral Resources are stated above a 0.62 g/t Au cut-off and are contained within a potentially economically mineable open pit. Mineral Resources are exclusive of Mineral Reserves.

The Mineral Resources are reported in accordance with NI 43-101 and have been classified in conformity with generally accepted CIM “Estimation of Mineral Resource and Mineral Reserves Best Practices” guidelines. The Mineral Resource estimate was completed by Mr. Jeffrey Volk, CPG, FAusIMM, Director of Reserves and Resources for Alamos. Mr. Volk is a Qualified Person within the meaning of Canadian Securities Administrator's National Instrument 43-101 (NI 43-101).

The effective date of this Mineral Resource estimate is 01 December 01 2017 and is based on drilling data finalized in December 2016. The Mineral Resource statement for the Gordon Property is presented in Table 14-20.

Table 14-20 – Gordon Mineral Resource Statement, December 01, 2017

	Gordon				
Category	kTonnes	Au Grade (g/t)	Ag Grade (g/t)	Au koz	Ag koz
Measured	9	1.72	n/a	0.47	n/a
Indicated	451	1.96	n/a	28	n/a
Total Measured and Indicated	***460***	***1.95***	***n/a***	***29***	***n/a***
Inferred	615	1.30	n/a	26	n/a

Notes:

- *The Mineral Resources are reported at an assumed gold price of US $1,400/ounce, and an assumed silver price of US $22.00/ounce;*
- *The Mineral Resource estimate was completed by Mr. Jeffrey Volk, CPG, FAusIMM, Director of Reserves and Resources for Alamos Gold Inc.;*
- *Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resources estimated will be converted into Mineral Reserves;*
- *Open pit Mineral Resources are stated as contained within potentially economically open pit above a 0.62 g/t Au, and includes external dilution at zero grade outside the 0.50 g/t Au solid;*
- *Totals may not add due to rounding;*
- *Contained Au are in-situ and do not include metallurgical recovery losses; and*
- *Mineral Resources are exclusive of Mineral Reserves.*

14.3 MacLellan Underground

Although not incorporated into the Feasibility Study in terms of project economics, the MacLellan underground resource model was updated from the previous estimate (Tetra Tech, 2014). The data utilized for this estimate were included in the statistical/ geostatistical analyses for the MacLellan open pit (Section 14.1) and are not repeated in this section of the report. The updated underground resource model is based on a nominal 1.0 g/t Au solid and modified block model construction parameters, as detailed in following sections.

14.3.1 Lithology and Grade Modelling

Lithologies were assigned as described in previous sections of this report. A nominal 1.0 g/t Au grade wireframe was constructed based on manually constructed cross sectional polygons constructed on 20 m spacing. These polygons were constructed only considering the main central zone of the deposit underlying the resource open pit limit, and are considered to be potentially geologically minable shapes. The resulting solid was clipped using the US $1,400 resource pit shell and assumed no crown pillar and is depicted in Figure 14-25.

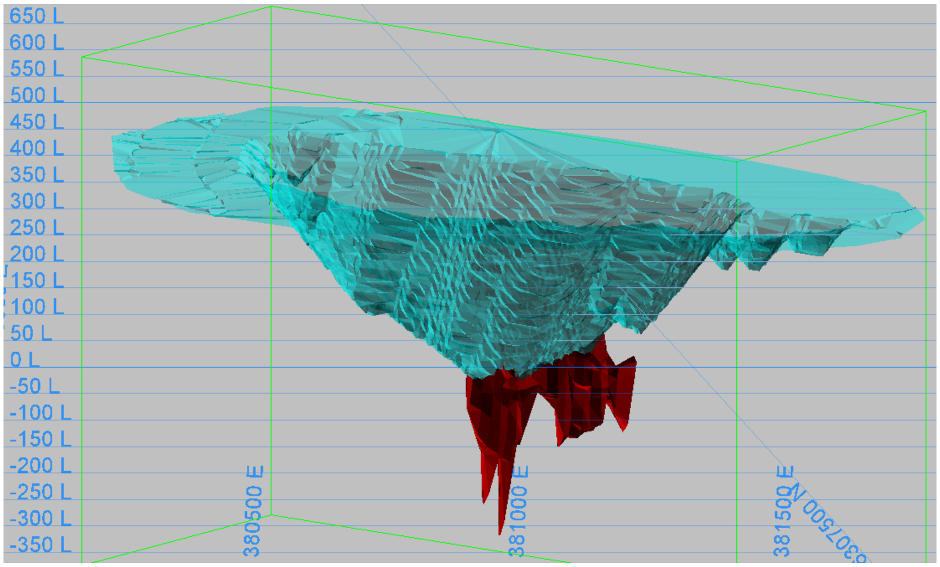


Figure 14-25: 1.0 g/t Gold Underground Solid and $1,400 Resource Pit Viewed to the North

Source: Alamos (2017)

14.3.2 Compositing

All raw gold and silver assays were capped using the same grade thresholds as for the open pit modelling. All assays were composited into 3 m downhole lengths, with composites broken on the 1.0 g/t Au wireframe boundaries.

14.3.3 Block Model Construction

A block model was constructed in Vulcan™ for the MacLellan underground deposit using the model limits and extents provided in Table 14-21.

Block model construction utilized a parent block size of 3.0 m x 3.0 m x 3.0 m, with sub-celling dimensions of 1.5 m x 1.5 m x 1.5 m.

Table 14-21 - MacLellan Underground Model Limits and Extents

Mine Grid Origin[1] (m)	Min (m)	Max (m)	Parent Block Size (X)	Parent Block Size (Y)	Parent Block Size (Z)	Subcelled Block Size (X)	Subcelled Block Size (Y)	Subcelled Block Size (Z)	No. of Blocks
East	380,290	381,892	3.0	3.0	3.0	1.5	1.5	1.5	1,068
North	6,306,720	6,307,320	3.0	3.0	3.0	1.5	1.5	1.5	400
Elevation	-500	505	3.0	3.0	3.0	1.5	1.5	1.5	670

Notes:

1. *All blocks are rotated 65° about model origin.*

14.3.4 Block Model Estimation

Block grades for gold and silver were estimated using identical search parameters to those utilized for the open pit block model. All grade estimates were length weighted to account for composites less that 3 m adjacent to the 1.0 g/t Au wireframe boundaries. All subcells were assigned parent cell grade values.

14.3.5 Block Model Validation

As conducted for the open pit model, the following model validation procedures were utilized to ensure that the grade estimates were valid, and include:

- Comparison of drill hole composites with Mineral Resource block grade estimates by zone visually, both in plan and section;
- Statistical comparisons between block and composite data using histogram and cumulative frequency distribution analysis;
- Generation of comparative NN model; and
- Swath plot analysis (drift analysis) comparing the ID3 model with the NN model.

14.3.6 Visual Inspection

Visual comparisons between the block grades and the underlying composite grades in plan and section show close agreement, which would be expected considering the estimation methodology employed. An example northwest-southeast cross and example level plan on are provided in Figure 14-26 and Figure 14-27, respectively, displaying block and composite gold grades, 1.00 g/t Au grade solid outlines and the US $1,400 Mineral Resource pit (magenta).

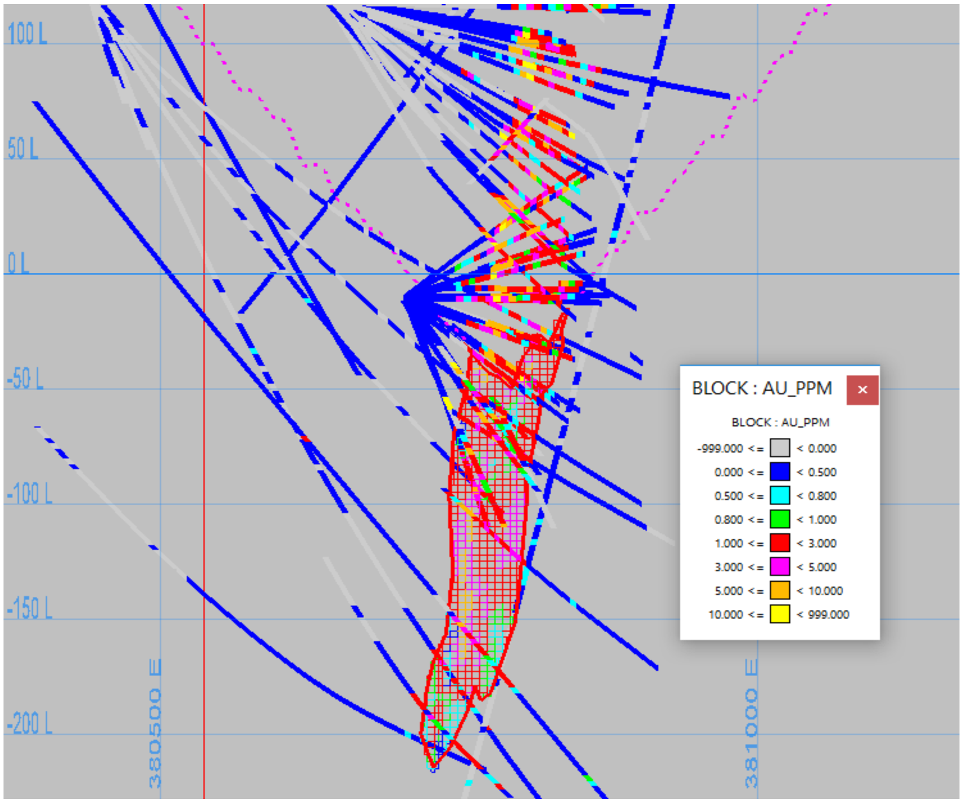


Figure 14-26: Example Northwest-Southeast Cross Section Viewed to East

Source: Alamos (2017)

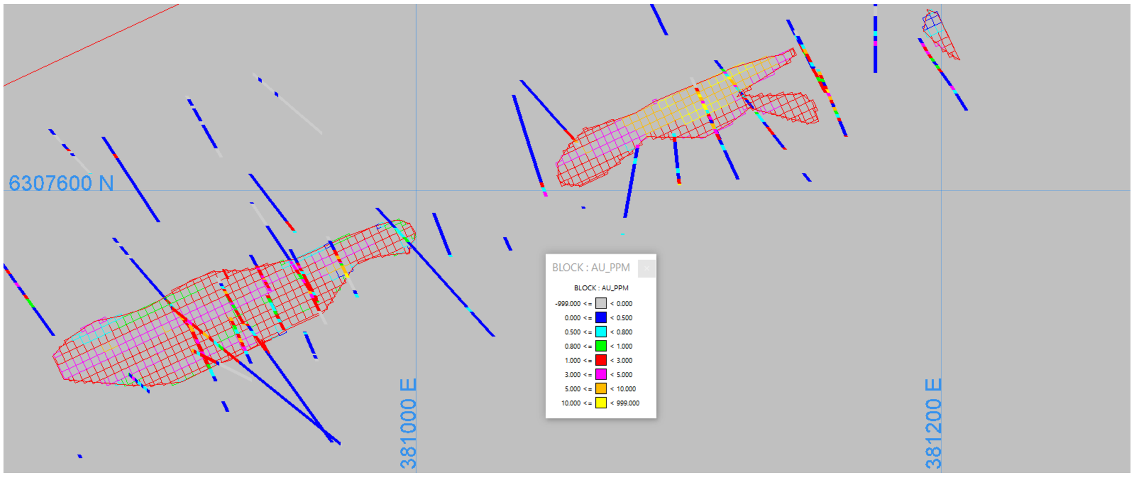
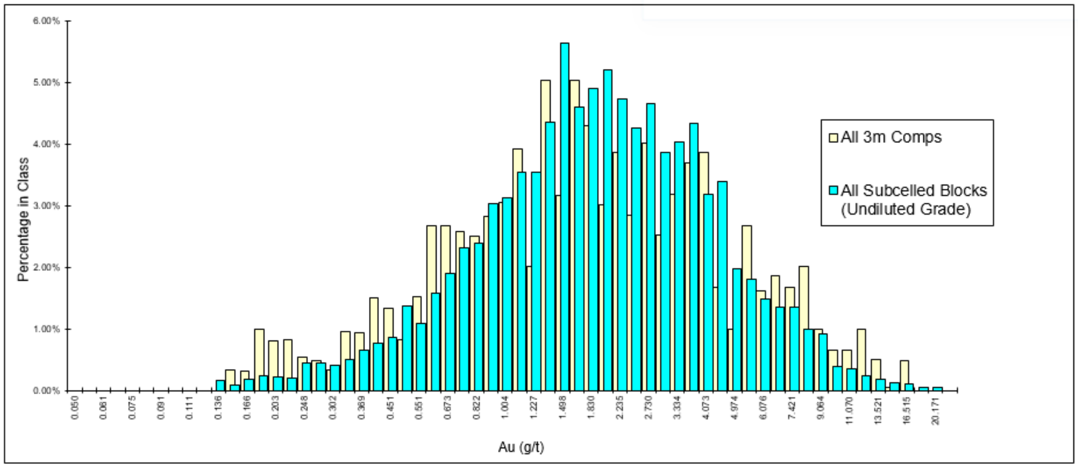


Figure 14-27: Example Level Plan at -86m Elevation Showing Model Block Grades

Source: Alamos (2017)

14.3.6.1 Block-Composite Histogram Comparison

Alamos also conducted statistical comparisons between the grades of the Indicated and Inferred ID3 blocks contained within the 1.0 g/t Au solid and the underlying 5m gold composite grades. A histogram comparison between block and composite gold grades at the MacLellan underground deposit is provided in Figure 14-28. Overall, this comparison shows that the model grade distribution for gold is appropriately smoothed when compared with the underlying composite distribution, and that the comparison of average grade and percentages above geologic absolute and incremental cut-offs show reasonably close agreement.



Untransformed Au Statistics

	Au Cutoff = 0.010 g/t		Au Cutoff = 0.500 g/t		Au Cutoff = 2.00 g/t		Au Cutoff = 5.00 g/t	
	meters/tonnes	Au (g/t)	meters/tonnes	Au (g/t)	meters/tonnes	Au (g/t)	meters/tonnes	Au (g/t)
All 3m Comps	1,784	2.846	1,544	3.250	799	5.154	277	8.804
incr. % and grade	13.4%	0.244	41.8%	1.209	29.3%	3.219	15.5%	8.804
All Subcelled Blocks (Undiluted Grade)	2,528,845	2.758	2,357,749	2.938	1,272,996	4.372	298,157	8.344
incr. % and grade	6.8%	0.279	42.9%	1.255	38.5%	3.157	11.8%	8.344

Figure 14-28: Histogram Comparison between Block and Composite Grades: Gold

Source: Alamos (2017)

14.3.6.2 Comparison of Interpolation Methods

For comparative purposes, additional grades were estimated using NN interpolation methods. The results of the NN model are compared to the ID3 model at a 0 g/t Au cut-off grade for indicated blocks in Table 14-22 and Table 14-23 for all Inferred blocks, internal to the 1.0 g/t gold grade solid. These comparisons confirm the conservation of metal at a zero cut-off, and shows close agreement on both a tonnage and grade basis within the deposit area.

Table 14-22 - ID3 vs. NN Tonnage – All Measured and Indicated Model Blocks

Model	Tonnes	Au Grade (g/t)	Au Oz
ID3	2,292,873	2.78	205,210
NN	2,292,873	2.80	206,665
Pct Diff (ID3 to NN)	0.00%	-0.71%	-0.71%

Table 14-23 - ID3 vs. NN Tonnage – All Inferred Model Blocks

Model	Tonnes	Au Grade (g/t)	Au Oz
ID3	236,428	2.55	19,421
NN	236,428	2.68	20,388
Pct Diff (ID3 to NN)	0.00%	-4.98%	-4.98%

14.3.6.3 Swath Plots (Drift Analysis)

Swath plots have been generated in three orthogonal directions for distribution gold in the model area. Swath plots for gold along the east-west, north-south and vertical directions are shown in Figure 14-29, Figure 14-30 and Figure 14-31.

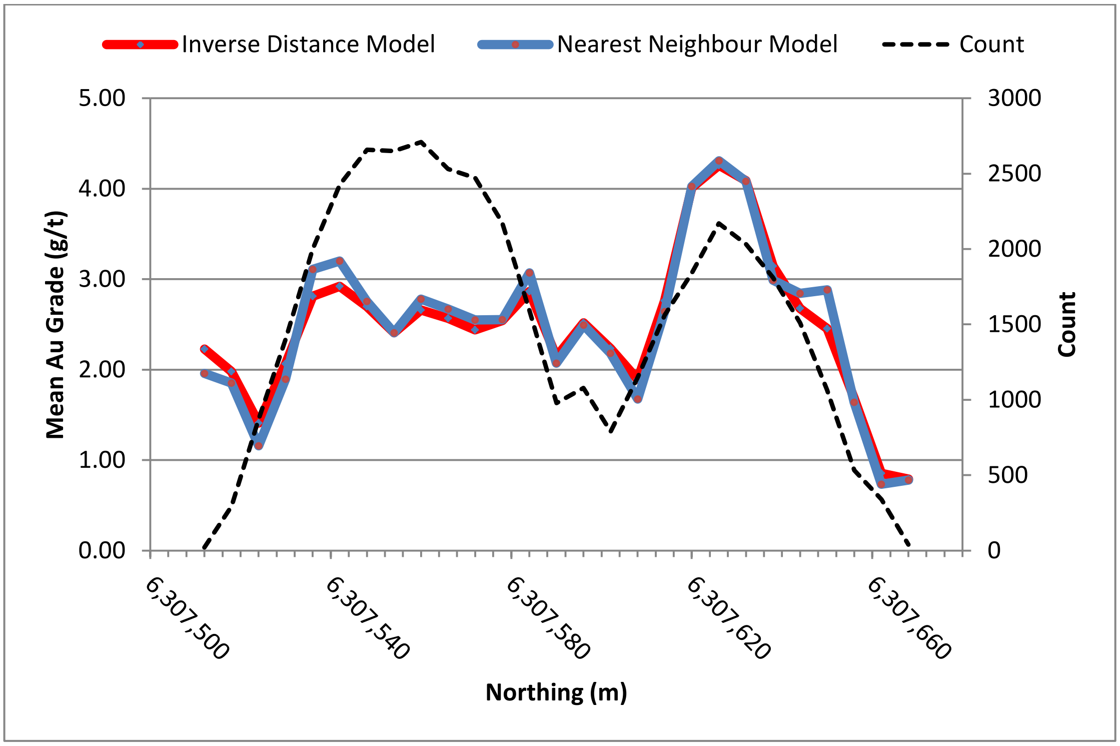


Figure 14-29: E-W Swath Plot, Comparing IDW and NN Model Gold Grades

Source: Alamos (2017)

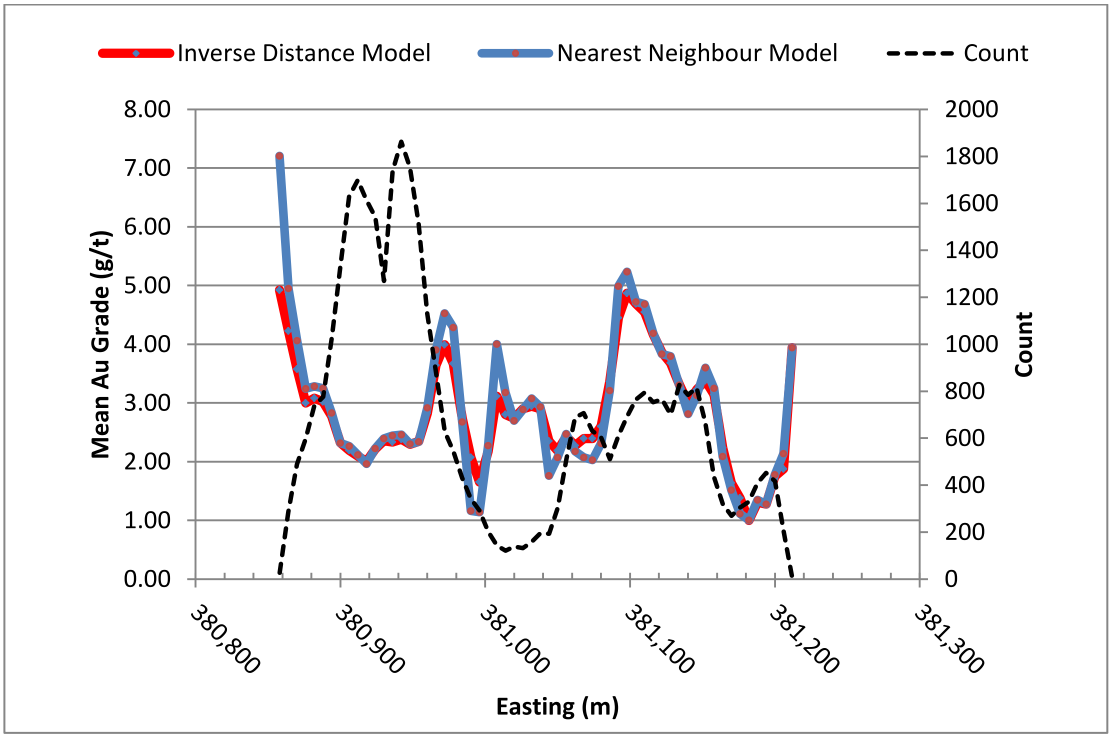


Figure 14-30: N-S Swath Plot, Comparing IDW and NN Model Gold Grades

Source: Alamos (2017)

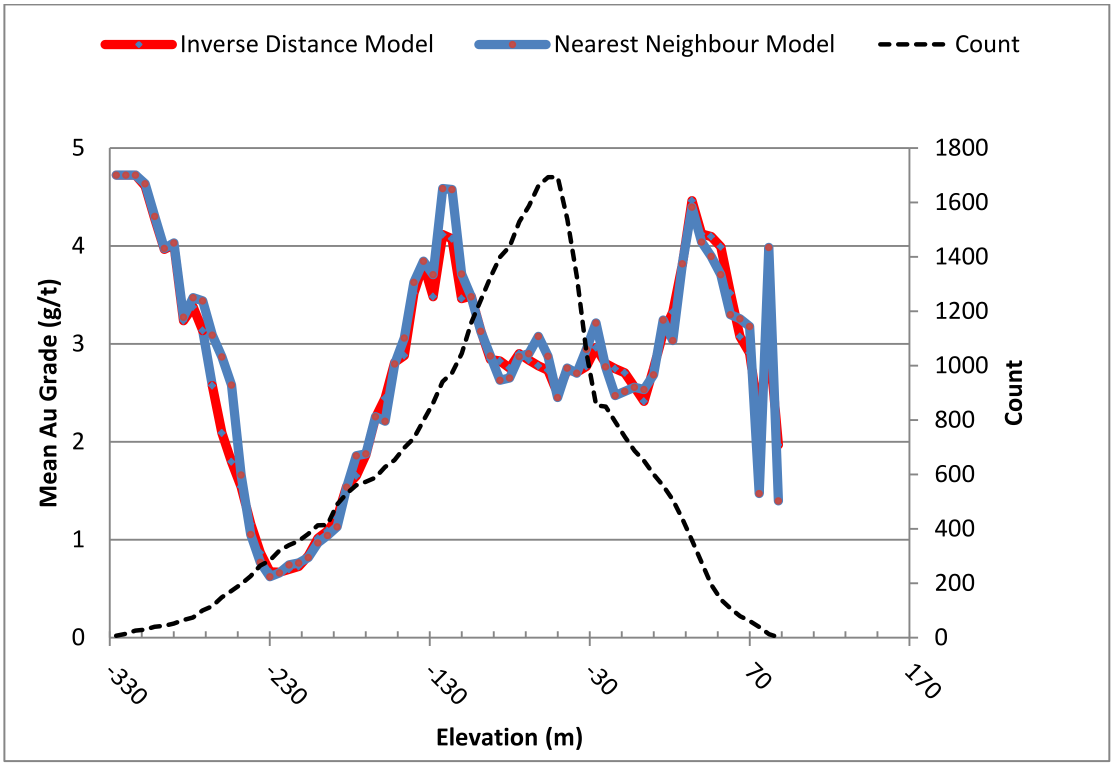


Figure 14-31: Vertical Swath Plot, Comparing IDW and NN Model Gold Grades

Source: Alamos (2017)

There is good correspondence between all models in all orthogonal directions. The degree of smoothing in the ID3 model is evident in the peaks and valleys shown in the swath plots, however, this comparison shows close agreement between ID3 and NN models in terms of overall grade distribution as a function of X, Y, and Z location.

14.3.7 Mineral Resource Sensitivity

In order to assess the sensitivity of the Mineral Resource to changes in gold cut-off grade, Alamos has summarized diluted tonnage and grade above cut-off for all estimated blocks, at a series of increasing gold cut-offs by Mineral Resource category. The cut-off grade sensitivity analysis for all Indicated blocks (depleted for past mining and exclusive of Mineral Reserves) within the MacLellan underground deposit are provided in Table 14-24. The cut-off grade sensitivity analysis for Inferred blocks within the MacLellan deposit are provided in Table 14-25.

It can be observed that the Mineral Resource is reasonably insensitive to cut-off grades in the increment between 1.50 and 2.50 g/t Au equivalent cut-off, which is likely the grade range of the potentially minable underground cut-off grade.

Table 14-24 - Cut-off Gold Grade Sensitivity – All Indicated Blocks

Cut-off	k-Tonnes	Au Grade (g/t)	Au (koz)
0.50	2,089	2.97	199
1.00	1,799	3.33	193
1.50	1,482	3.78	180
2.00	1,161	4.37	163
2.50	912	4.96	146

Table 14-25 - Cut-off Gold Grade Sensitivity – All Inferred Blocks

Cut-off	k-Tonnes	Au Grade (g/t)	Au (koz)
0.50	221	2.62	19
1.00	198	2.83	18
1.50	160	3.22	17
2.00	116	3.82	14
2.50	91	4.27	13

14.3.8 Mineral Resource Classification

The Mineral Resources for the MacLellan underground deposit are classified under the categories of Measured, Indicated and Inferred according to the guidelines as defined by the "CIM Definition Standards for Mineral Resources and Mineral Reserves", prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council on May 10, 2014.

Classification of the Mineral Resources reflects the relative confidence of the grade estimates. This is based on several factors including; sample spacing relative to geological and geostatistical observations regarding the continuity of mineralization, mining history, specific gravity determinations, accuracy of drill collar locations, quality of the assay data and other factors which can influence the confidence of the mineral estimation.

The classification parameters are defined in relation to the number of drill holes used to estimate the block grades and the block-composite separation distance. These classification criteria are intended to encompass zones of reasonably continuous mineralization.

The following classification parameters were applied to the MacLellan block model:

Measured Mineral Resources

There are no measured Mineral Resources for the MacLellan underground resource model

Indicated Mineral Resources

Blocks in the model which are within the 1.0 g/t Au solid that were informed by a minimum of 2 drill holes on the first or second estimation search pass (45 m x 45 m x 10 m).

Inferred Mineral Resources

Blocks in the model that do not meet the criteria for Measured or Indicated Resources and have been informed by a minimum of one drill hole on the third estimation search pass (100 m x 100 m x 20 m).

14.3.9 MacLellan Underground Mineral Resource Statement

The Mineral Resources, as of December 01, 2017, for the MacLellan Underground Resource have been estimated by Alamos at 1,161 kt grading an average of 4.37 g/t Au and 6.23 g/t Ag classified as Indicated Mineral Resources; with an additional 116 kt grading an average of 3.82 g/t Au and 3.43 g/t Ag classified as Inferred Mineral Resources. The Mineral Resources are stated above a 2.0 g/t Au equivalent cut-off and contained within potentially economic geologic underground solids. Mineral Resources are exclusive of Mineral Reserves.

The Mineral Resources are reported in accordance with NI 43-101 and have been classified in conformity with generally accepted CIM "Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines. The Mineral Resource estimate was completed by Mr. Jeffrey Volk, CPG, FAusIMM, Director of Reserves and Resources for Alamos. Mr. Volk is a Qualified Person within the meaning of Canadian Securities Administrator's National Instrument 43-101 (NI 43-101).

The effective date of this Mineral Resource estimate is December 01, 2017 and is based on drilling data finalized in December, 2017. The Mineral Resource statement for the underground portion of the MacLellan deposit is presented in Table 14-26.

Table 14-26 - MacLellan Underground Mineral Resource Statement, December 01, 2017

MacLellan Underground					
Category	**kTonnes**	**Au Grade (g/t)**	**Ag Grade (g/t)**	**Au koz**	**Ag koz**
Measured	0	0.00	0.00	0	0
Indicated	1,161	4.37	6.23	163	233
Total Measured and Indicated	***1,161***	***4.37***	***6.23***	***163***	***233***
Inferred	116	3.82	3.43	14	13

Notes:

- *The Mineral Resources are reported at an assumed gold price of US $1,400/ounce, and an assumed silver price of US $22.00/ounce;*
- *The Mineral Resource estimate was completed by Mr. Jeffrey Volk, CPG, FAusIMM, Director of Reserves and Resources for Alamos Gold Inc.;*
- *Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resources estimated will be converted into Mineral Reserves;*
- *Mineral Resources for the MacLellan Underground are stated above a 2.0 g/t Au cut-off. MacLellan Underground block grades are undiluted;*
- *Totals may not add due to rounding;*
- *Contained Au and Ag ounces are in-situ and do not include metallurgical recovery losses; and*
- *Mineral Resources are exclusive of Mineral Reserves*

14.4 Consolidated Project Mineral Resource Statement

The Mineral Resources, as of December 01, 2017, for the Consolidated LLGP have been estimated by Alamos at 5,967 kt grading an average of 2.12 g/t gold classified as Measured and Indicated Mineral Resources; with an additional 1,481 kt grading an average of 1.66 g/t gold classified as Inferred Mineral Resources. The Mineral Resources are stated above a 0.42 g/t Au equivalent cut-off for MacLellan and 0.62 g/t Au cut-off for Gordon, both contained within a potentially economically mineable open pit, and above a 2.0 g.t AU cut-off for MacLellan Underground. Mineral Resources are exclusive of Mineral Reserves.

Table 14-27 – Consolidated LLGP Mineral Resource Statement, December 01, 2017

	Total				
Category	**kTonnes**	**Au Grade (g/t)**	**Ag Grade (g/t)**	**Au koz**	**Ag koz**
Measured	2,119	1.86	5.31	127	362
Indicated	3,848	2.27	4.34	281	537
Total Measured and Indicated	***5,967***	***2.12***	***4.69***	***407***	***900***
Inferred	1,481	1.66	1.69	79	80

Notes:

- *The Mineral Resources are reported at an assumed gold price of US $1,400/ounce, and an assumed silver price of US $22.00/ounce;*
- *The Mineral Resource estimate was completed by Mr. Jeffrey Volk, CPG, FAusIMM, Director of Reserves and Resources for Alamos Gold Inc.;*
- *Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resources estimated will be converted into Mineral Reserves;*
- *Open pit Mineral Resources are stated as contained within potentially economically open pit above a 0.42 g/t AuEq cut-off for MacLellan and 0.62 g/t Au for Gordon, and includes external dilution at zero grade outside the 0.50 g/t Au solid;*
- *Mineral Resources for the MacLellan Underground are stated above a 2.0 g/t Au cut-off. MacLellan Underground block grades are undiluted;*
- *Totals may not add due to rounding;*
- *Contained Au and Ag ounces are in-situ and do not include metallurgical recovery losses; and*
- *Mineral Resources are exclusive of Mineral Reserves.*

14.5 Acid Rock Drainage Modeling

To aid in waste rock management for mine planning, ARD models were developed for both the MacLellan and Gordon Properties. Stantec performed ABA analysis on 187 drill core samples from MacLellan and 179 samples from Gordon.

The ABA testing included the following analyses of samples:

- Paste pH was measured by inserting a combination pH electrode into a sample paste using a ratio of 1:2, water-to-solid solution;
- Total sulphur was analyzed by the Leco induction furnace (Leco) with an infrared detector at or above 1,650 °C;
- AP was determined from measurement of all Sulphur species in a sample. Sulphate-sulphur was extracted by a 15% hydrochloric acid (HCl) digestion method and the leachate was measured by gravimetric method;
- Total Inorganic Carbon (TIC) was analyzed by the conversion of carbonate minerals to carbon dioxide using dissolution in hydrochloric acid and measured using colorimetric titration. Carbonate carbon was calculated from a concentration of TIC using conversion for molecular weight;
- Neutralization Potential (NP) was determined using the modified-Sobek method, with the addition of HCl over 24 hours at room temperature to maintain the pH between 1.5 and 2.0. The extract was then filtered and treated with hydrogen peroxide (H_2O_2) to oxidize ferrous iron to ferric iron prior to back titration with sodium hydroxide (NaOH); and

Samples were submitted to ALS for an ICP + ICPMS analytical method referred to as MS41L. The digestion method is the same as for Alamos drilling programs but there are a few additional elements reported (such as Zr) and detection limits are lower for some elements.

NP was calculated as follows:

$$NP\ (kg\ CaCO_3/t) = TIC\ (wt\%) \times 80.71$$

AP was conservatively calculated from total Sulphur as follows:

$$AP\ (kg\ CaCO_3/t) = Total\ sulphur\ (wt\%) \times 31.25.$$

The ABA tests showed that S total and S total-SHCl (S by HCl digestion) are comparable for MacLellan samples (ASL Internal Report, 2016)

The predicted values for AP and NP were composited into 5 m downhole intervals and were utilized to estimate AP and NP in both the MacLellan and Gordon block models. Neutralization Potential Ratios (NPR) were then calculated on a block by block basis using the formula NPR = NPpred/APpred. The model results were then provided to both Stantec and Q'Pit for use in mine planning and waste rock management planning.

15 MINERAL RESERVE ESTIMATES

15.1 Introduction

The estimate of the Mineral Reserves was carried out by Q'Pit within the guidelines of the NI 43-101 Standards of Disclosure for Mineral Projects.

The estimates for the Mineral Reserves are based on the Measured and Indicated Mineral Resources of the block model and the detailed final pit limit designs carried out by Q'Pit using the design parameters of the Project discussed in this section.

The Mineral Reserves for the LLGP are listed in Table 15-1 with the Au and Ag grade estimates based on the diluted grades of the block model.

Table 15-1 – LLGP Mineral Reserve Statement, December 01, 2017

	Class	Tonnage (Mt)	Au Grade (g/t)	Ag Grade (g/t)	Au oz Contained (x1000)	Ag oz Contained (x1000)
Gordon	Proven	2.31	2.82	n/a	210	n/a
	Probable	6.41	2.27	n/a	468	n/a
	Total Proven and Probable	8.72	2.42	n/a	678	n/a
MacLellan	Proven	9.55	1.91	5.01	586	1,539
	Probable	8.53	1.32	3.79	361	1,039
	Total Proven and Probable	18.08	1.63	4.43	947	2,578
Lynn Lake Gold Project	Proven	11.86	2.09	4.03	796	1,539
	Probable	14.94	1.73	2.16	829	1,039
Total Proven and Probable		26.80	1.89	2.99	1,625	2,578

Notes:

- *Mineral Reserves reported are in agreement with the CIM Definition Standards for Mineral Resources and Mineral Reserves*
- *The Mineral Reserve is estimated using metal prices of US $1,250/Au oz and US $15.00/Ag oz.*
- *Totals may not add up due to rounding.*
- *The estimates were carried out using cut-off grades of 0.69 Au g/t for Gordon and 0.47 Equivalent Au g/t for MacLellan and a metallurgical Au recovery of 89-94% for Gordon and 91-92% for MacLellan.*
- *The design parameters applicable are detailed in Section 15 of this report.*
- *The estimate of the Mineral Reserves was carried out under the supervision of Efthymios Koniaris, PhD., P.Eng., of Q'Pit Inc.*

15.2 Block Model

The geological models for the Gordon and MacLellan deposits were supplied by Alamos. The block size is 5 m x 5 m x 5 m. Stoped zones in the MacLellan deposit were excluded by assigning an adjusted block density for the affected block. Q'Pit estimated the fraction of the block stoped. For the purposes of the pit limit design, it was assumed that the fraction of the block stopes is backfilled with waste.

The information in the block model includes the following:

- Gold grade (g/t) (diluted);
- Silver grade (g/t) (diluted) (MacLellan only);
- Mineral Resource classification (Measured, Indicated, Inferred);
- In-situ density (t/m^3); and
- The adjusted block density (t/m^3).

For the purposes of the pit limit analysis and design and the related Mineral Reserve estimates, only the Measured and Indicated classifications of the Mineral Resources were used. Inferred Resources were considered as waste.

15.3 Material In-Situ Value Calculation Parameters

The information listed in Table 15-3 was used for the material in-situ valuation, estimates of cut-off grades and optimization with respect to the open pit limit design.

Table 15-2 – Cost and Revenue Mine Design Parameters

	Gordon	MacLellan
Au Price (US $/oz)	1,250	
Exchange Rate (US/CAD)	0.75	
Ag Price (US $/oz)	n/a	15
Royalty (%)	2.5	0
Off-site Refining and Transportation (US $/oz Au)	3.25	
Gordon Au Recovery (%) (for Au Grades +0.5 g/t)	100-((0.0978+0.0835ln(Au))/Au*100-0.9)[1]	
MacLellan Au Recovery (%)	100-((0.03+0.0587*Au)/Au*100-0.9)[1]	
Ag Recovery (%)	n/a	49
Mining and Camp Cost ($/t Mined)	3.28	3.03
Processing Cost ($/t milled)	14.39	
G&A and Camp (excluding mine) Cost ($/t milled)	5.45	
Tailings Dam Expansion ($/t milled)	1.29	
Gordon Highway Haulage and Camp Cost ($/t milled)	10.67	n/a
Low Grade Re-Handle Cost Allowance ($/t)	0.55	0.49
Total Process, G&A, Highway Haulage, and Re-handle Cost ($/t)	32.35	21.62

Notes:

1. *Au represents feed grade in g/t*

The Mineral Reserve cut-off grades were set on the basis of $0.00/t in-situ net value, calculated using the parameters listed in Table 15-2 and the block model gold and silver grades. For the calculation of the Mineral Reserve cut-off grade, the mining costs are excluded. For the purposes of the cut-off grade calculation, the mine operating cost for the ore as well as the waste is considered to be the same. Accordingly, no difference for the mine cost for ore and waste is applied in the calculation of the cut-off grades.

Due to the location of the mill at MacLellan, the mill feed sourced from Gordon requires highway haulage. The utilized economic model considers and allocates the highway haulage costs to the Gordon cut-off grade calculation as well as for the final pit limit analysis and design. The in-situ material valuation for the pit limit design and the cut-off grade model used for the Mineral Reserve estimates also considers the extra highway personnel with the allocation of a proportionally higher fraction of camp costs to the ore from Gordon.

15.4 Mine Recovery and Dilution

The gold and silver grades in the block model allows for edge dilution. The current geological model estimation methodology inherently introduces dilution, at zero grade, in the estimate of the block grades of 13% and 15% for MacLellan and Gordon respectively. Q'Pit is of the opinion that no additional dilution and mining recovery factors are applicable, in overall agreement with the practice in studies and operating properties in the experience of Q'Pit.

15.5 Cut-off Grade Calculation

Based on the parameters listed above, the Mineral Reserve cut-off grades for the Gordon and MacLellan deposits are calculated at 0.69 g/t Au and 0.47 g/t Au respectively, excluding the Ag credits. The silver credit is taken into account for the cut-off grades calculation for the MacLellan ore.

The Mineral Reserve cut-off grades were set based on an in-situ net value of $0.00/t of mill feed. This calculation leads to the following cut-off grade calculation formula:

$$g_{Au\_c} = (C_p + C_{ga} + C_{tmf} + Cst + C_{hwy} - CR_{ag}) / [r_{Au} * (P_{Au} - C_{RefAu}) * (1\text{-}Royalty_{Au})]$$

where:

g_{Au_c}: Gold cut-off grade

C_p : Processing Cost.

C_{ga}: General and Administration Cost

$C_{tmf\,:}$ The marginal cost of raising the tailings dam

C_{st}: Stockpile re-handle cost allowance

C_{hwy}: Highway loading and haulage (applicable for Gordon ores only)

CR_{Ag} : Silver Credit, applicable in MacLellan only

r_{Au} : Gold payables

P_{Au} : Gold price

C_{RefAu}: Transportation and refining costs

$Royalty_{Au}$: Gold royalty

The Silver Credit is further defined as:

$CR_{Ag} = g_{Ag}\_ * r_{Ag} * (P_{Ag} – CrefAg) *(1-Royalty_{Ag})]$ where:

$g_{Ag\,:}$ Silver grade

r_{Ag}: Payable silver

P_{Ag}: Silver price

C_{RefAg}: Transportation and refining costs

$Royalty_{Ag}$: Silver royalty

15.6 Wall Slope Angles and Bench Configuration

The Feasibility Study geotechnical investigation for the LLGP has been conducted by Golder (Golder Associates, 2016) and is discussed further in Section 16.

The geotechnical investigation considered a 10 m operating bench and double benching in the fresh rock with a berm every 20 m. The current study considers operating benches of 5 m and 10 m and a berm every 20 m. The wall slope design parameters used in this study are based on the recommendations in the geotechnical study for inter-ramp slope angles, with adjustments to the berm widths as required.

The recommended inter-ramp slope angles range from 52^o to 56^o in the fresh rock zones and from 46^o to 49^o in the fractured zones. The overburden slope was set at 2H:1V.

The geotechnical study recommends the use of a single bench configuration for the first 20 m in rock, considered to be the fractured zone. In addition, a 10 m berm was recommended at the base of the overburden as well as at the boundary between the fractured zone and the fresh rock. It is noted that at both sites the overburden and topography are variable. For the detailed pit designs, the fractured zone slope angles were applied for thicknesses ranging approximately from 20 m to 30 m at MacLellan and 15 m to 40 m at Gordon.

At MacLellan, a step out on the southeast part of the pit wall is recommended to be incorporated.

15.7 Ramp Width

The ramp width is set with reference to the application of 144 t or 136 t class mine trucks such as the Komatsu HD1500 or the Caterpillar 785 or similar. The ramp width is derived using as a basis 2.5 times the operating width of the mine truck plus allowance for berm and ditch.

For the primary trucks, the nominal in-pit ramp width for two-way traffic is set at 25 m with a grade at 10%. This ramp width and grade is used for the pit limit design at Gordon.

For the pit limit design at MacLellan, the application of a combination of two-way and single lane traffic is planned. The minimum ramp width for single lane traffic is calculated at 15 m for the primary truck class. The pit limit design also considers that the articulated truck fleet will be employed for the haulage of the material from the bottom of the MacLellan open pit. The ramp width is set at 25 m width and 10% grade for the top 200 m of mining. Thereafter, the design considers that the ramp width to decrease to 20 m and 15 m nominal width and retreat mining for the last two 10 m operating benches of the open pit will be utilized. The pit limit design also considers that the ramp grade will be increased to 12% for several benches and, for the last two 10 m operating benches from the pit bottom, to 15% grade. The latter considers that the articulated truck fleet will be used for haulage in the last two operating benches.

15.8 Constraints to the Final Pit Limit Analysis and Design

Both deposits are subject to constraints for the pit limit design.

The MacLellan open pit is constrained to the west by the Keewatin River. The setback distance was set to 60 m from the average high level of the river for the overburden crest, which leads to a setback of 85 m or more from the rock crest in the constrained zone. This constraint was introduced to the pit limit analysis by flagging and excluding all mineralization blocks potentially forming walls or conical shapes that violate the constraint of this setback distance.

Some of the final pit limit walls and ramp locations at MacLellan were adjusted, to minimize the interaction of final walls with the underground openings from past operations at MacLellan.

Similarly, the Gordon pit limit is constrained by two water bodies to the east and west. The Gordon mine site at present includes pit perimeter berms from past operations. The pit limit design was constrained to remain within the pre-existing pit perimeter berms in the west and east and southeast sectors. The proposed pit limit is outside the perimeter berms in the south, north and northeast sectors of the open pit.

15.9 Detailed Pit Designs for the Mineral Reserve Estimates

The Mineral Reserve estimates are based on detailed pit limit designs which were validated by a LOM mine plan. The detailed pit limit designs were carried out using the Q'Pit mine planning package.

The pit limit analysis and optimizations were derived based on the Lerchs and Grossmann theory. The pit limit analysis was carried out using software developed by and proprietary to Q'Pit. The pit limit analysis used:

- The Measured and Indicated Resources contained in the Mineral Resource block models for each deposit, discussed in Section 15.2;
- The parameters for the in-situ material valuation of Section 15.3; and
- The geotechnical design parameters as outlined in Section 15.6 as well as a model of the ramp configurations in the open pit limit. The optimization respects average overall slopes angles as well as inter-ramp slope angles.

The proposed pit limit design for Gordon is shown in Figure 15-1. A schematic comparison of the final pit limit design (black) for Gordon and the unsmoothed optimized pit (blue) used for the design is shown in Figure 15-2.

The Gordon final pit limit design is in close agreement with the unsmoothed optimized pit.

The proposed pit limit design for MacLellan open pits are shown in Figure 15-3 with a schematic comparison of the final pit limit (black) to the guide pit (blue and to the unsmoothed optimized pit (red) included in Figure 15-4.

The MacLellan pit limit design is slightly smaller than the excavation supported by the unsmoothed optimized pit.

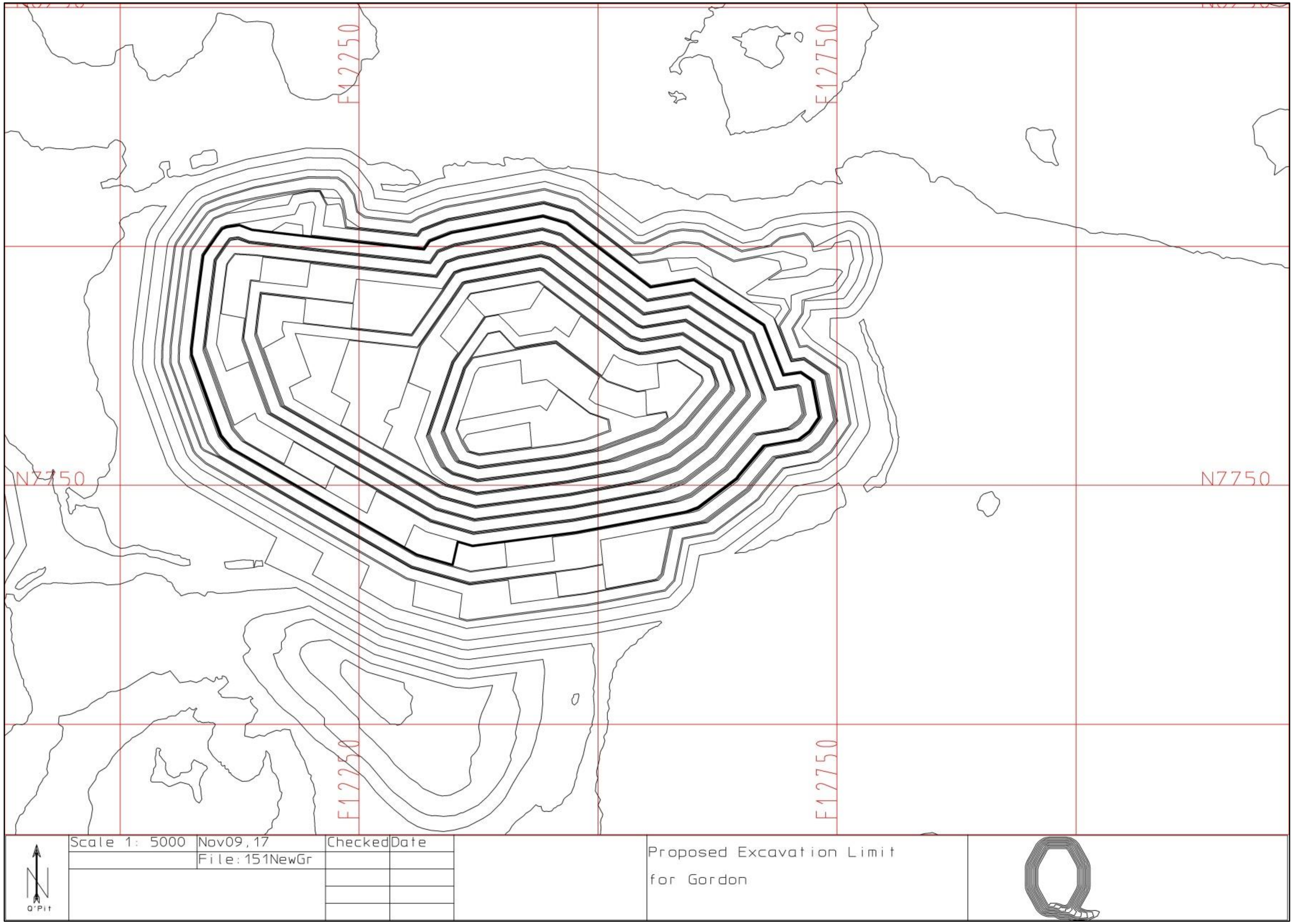


Figure 15-1: Mineral Reserve Pit Limit for Gordon

Source: Q'Pit (2017)

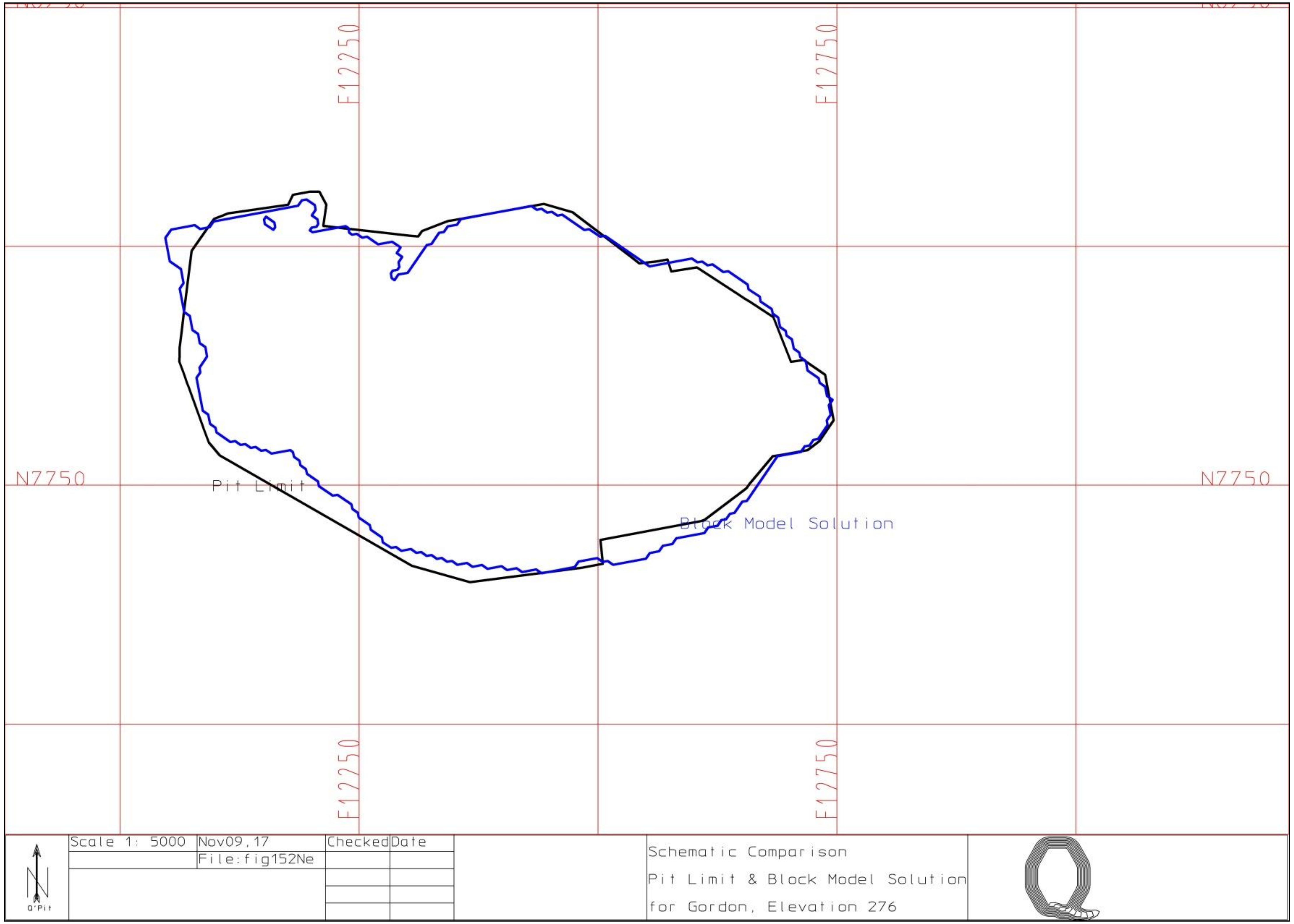


Figure 15-2: Utimate Pit Limit versus Unsmoothed Optimized Pit, Gordon, Elev. 276

Source: Q'Pit (2017)

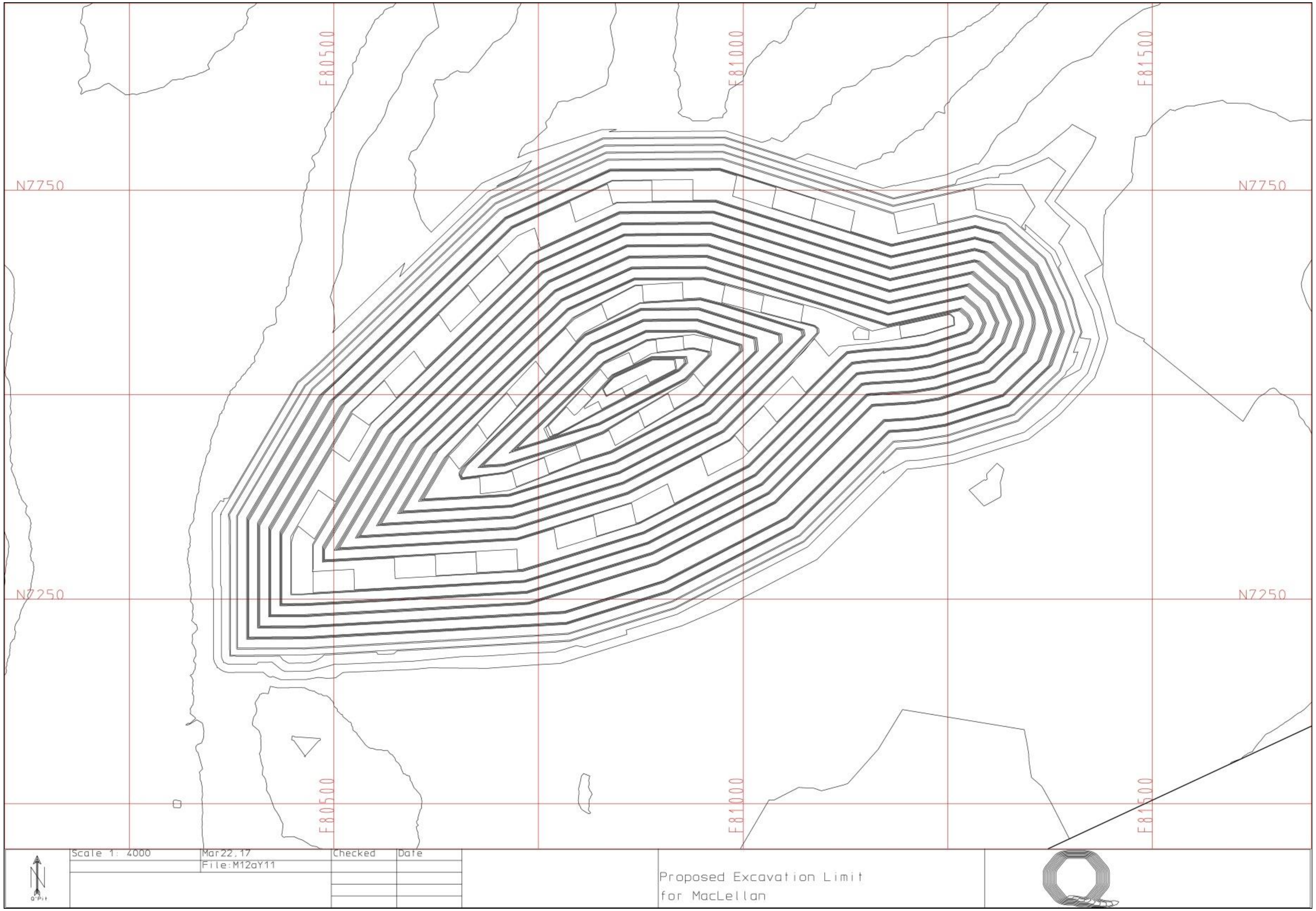


Figure 15-3: Mineral Reserve Pit Limit for MacLellan

Source: Q'Pit (2017)

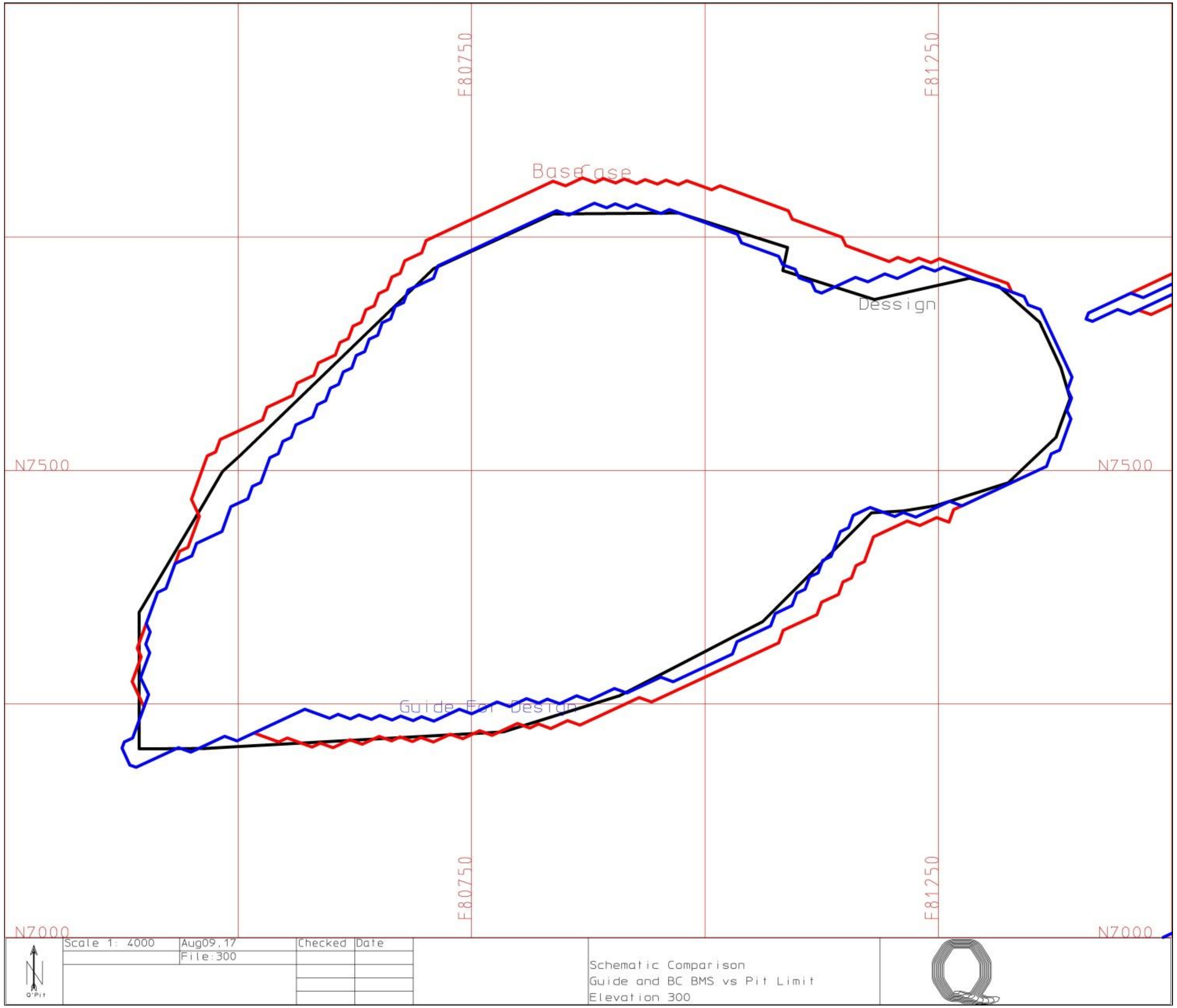


Figure 15-4: Ultimate Pit Limit versus Unsmoothed Optimized Pit, MacLellan, Elev. 300

Source: Q'Pit (2017)

15.10 Sensitivity of the Mineral Reserve due to the Open Pit Geometry and Size

A sensitivity analysis with respect to the geometry of the final pit limits for a wide range of variations in the optimization parameters was conducted.

The sensitivity analysis established that the presented Gordon pit limit can be considered robust for a relatively wide range of variation of the design parameters. The robustness of the Gordon pit limit is attributed in part to the imposed constraints on the east and southeast of the pit and, primarily, to the fact the proposed pit limit encompasses in the Mineral Reserve the major part of the Measured and Indicated Resources of the deposit.

In contrast, the geometry of MacLellan pit limit is sensitive to the design parameters. An iterative approach was required to determine the MacLellan final pit limit design. The proposed design is smaller in total volume than the pit limit supported by the Lerchs and Grossmann analysis. Q'Pit considers that the proposed MacLellan pit limit design can be deemed conservative and that it will remain valid for substantial adverse changes to the design parameters. Figure 15-4 shows the final pit limit design (black) as well as the pit limit location supported by the base case design parameters (blue) as well as the unsmoothed optimized pit (red) utilized for the design.

15.11 Sensitivity of the Mineral Reserve due to Changes in Cut-off Grades

The Mineral Reserve cut-off grades affect the Mineral Reserve estimates. The presented Mineral Reserve estimate is based on the information available and the applicable design parameters at the time this report was compiled. The MacLellan long-term re-handle cost is estimated at $0.97/t re-handled or $0.18/t of ore from the MacLellan pit. The Gordon ROM re-handle cost is estimated at $0.55/t. The applied re-handle cost is therefore conservative for MacLellan with respect to the reported Mineral Reserve tonnage.

Information affecting cut-off grade calculation and, by extension, the estimate of the Mineral Reserve include the metal prices and exchange rates as well as overall mine and process variable and fixed costs, the stockpile re-handle and the allocation of these costs to the process or the mine cost centers. The costs allocated to the process cost center and, by extension, any additions or exclusions from costs in this cost center, affect the Mineral Reserve cut-off grades.

Q'Pit considers that the utilized design parameters are appropriate and is not aware of any other factors which could materially impact the estimates of the Minerals Reserves of the LLGP presented in this report.

16 MINING METHODS

16.1 Introduction

The LLGP calls for the development of the Gordon and MacLellan deposits. Both deposits are planned to be developed using conventional open pit mining methods.

Both deposits were the subject of past development. Gordon was mined with the development of two relatively shallow open pit excavations, the East pit and the Wendy pit. The MacLellan site was the subject of underground development and production.

The Feasibility Study mine plan was carried out using a peak mine production rate of 27.0 Mt/y and a mine life of 10.4 years, in addition to the pre-production period. The plant site is to the southeast of the MacLellan open pit. The mill feed is planned to be delivered to the crusher pad adjacent to the plant site at a rate of 7,000 t/d or 2.56 Mt/y.

The operations at MacLellan commence at the beginning of pre-production (Year -2) with the mining of the in-pit and ex-pit borrow sources to provide rock for the construction of the initial roads, infrastructure and TMF.

The mine life at Gordon is six years plus one year in the pre-production period. Some of the lower grade ores will be stored at Gordon and will be retrieved after the end of mining activities at Gordon. The primary mining fleet will be relocated from Gordon to MacLellan between Year 4 and 6 of operations.

Concurrent operations of the Gordon and MacLellan open pits are planned for the first six years of operation. Thereafter, MacLellan is planned to be the only operating mine with some of the mill feed provided as needed from stockpiles.

The site specific operating conditions determine that a provision for several stockpile pads and stockpiles be made. Specifically, all the ore from Gordon is planned to be re-handled at the Gordon mine site for its transhipment to highway trucks and then re-handled again at the primary crusher at MacLellan. In addition, the ore production from the MacLellan open pit in excess of the plant capacity will be stored at the low-grade stockpile. The low-grade stockpile in turn is planned to be used to provide a portion of the mill feed at the plant in Year 7 and Year 8 when the ore presentation from the MacLellan open pit is less than the plant capacity.

16.1.1 Operating Mode

16.1.1.1 Base Case: Mine Operations by the Owner

The FS considers the application of the owner’s equipment, personnel and facilities for the mining operations as well as the highway haulage of ROM ore from Gordon to MacLellan.

The owner mining case is detailed in Section 16 and forms the basis for the operating and capital costs estimates detailed in Section 21, as well as provides the operating costs for the estimates of the Mineral Reserves for the Project in Section 15.

16.1.1.2 Alternative: Mine Operations by Contractor

Alamos has experience with the successful application of mining by a contractor from its Young-Davidson Mine (Young-Davidson) operations near Matachewan in Northern Ontario. Alamos has investigated the option of employing a contractor and it is considered that this remains a valid operating option.

16.2 Geotechnical

The geotechnical feasibility level pit slope designs for the LLGP (specifically the two planned open pits: MacLellan and Gordon) was completed by Golder in 2016. Subsequently, in 2017, the overburden slope design recommendations were provided, along with updated recommendations for the rock slopes based on new pit optimization information provided by Q'Pit.

The design for pit optimization was completed iteratively by Q'Pit with input from Golder to ensure that the geotechnical recommendations are met for the ultimate pit wall design.

16.2.1 Geotechnical Investigation Program

A field investigation program was conducted in 2015 as part of the FS of rock slopes. A total of five inclined deep rock boreholes (four at the MacLellan Property, and one at the Gordon Property) were advanced to depths ranging from 150 m to 300 m. To improve the bedrock characterization, televiewer surveys were conducted in eight exploration boreholes (five at the MacLellan Property, and three at the Gordon Property). A total of nearly 1,500 m of borehole wall data was collected to expand on the geotechnical borehole data set.

The overburden slope FS was completed in three phases of investigation, in 2015, 2016, and 2017. In 2015, a total of 18 geotechnical overburden boreholes were drilled (12 at the Gordon Property, and six at the MacLellan Property) to characterize overburden, historical dam fill materials, and shallow bedrock in the open pit areas. The borehole depths ranged from 3 m to 21 m, depending on overburden thickness.

In 2016, a total of four geotechnical overburden boreholes were drilled around the open pit areas (two at Gordon, and two at the MacLellan Property). The borehole depths ranged from 8 m to 15m, depending on the overburden thickness.

In 2017, two additional boreholes were drilled along the northeast end of the Gordon pit to further characterize the hydraulic conductivity of the shallow bedrock along an identified east-west trending fault zone.

Hydrogeology characterization was completed in overburden and shallow and deep bedrock around the ultimate pit areas by means of slug testing and packer testing. Short-term pumping test programs were conducted at MacLellan and Gordon Properties in 2015. Based on the short-term pumping test results, a long-term pumping test study was conducted at the Gordon Property in 2016. Results are summarized in Section 18.3.1.

In general, the geotechnical analyses (structural, strength, rock mass quality, and hydrogeology) indicate relative agreement with the limited historical data from the Project site, providing confidence in the current characterization.

16.2.2 Pit Wall Slopes

16.2.2.1 Overburden Slope Stability

Overburden in the MacLellan pit area is variable in thickness, and characterized by a generally northeast-southwest trending overburden trough (up to 17 m thick near the northeast edge of the pit and up to 20 m thick further to the northeast, beyond the planned pit perimeter). In general the overburden thickness thins both to the north and south of the pit, and consists of discontinuous peat/organics underlain by glaciolacustrine and alluvial deposits, underlain by sandy till and then bedrock. Permafrost conditions were noted along the east and south edges of the pit to depths of 1.0 m and 8.9 m, respectively.

In the Gordon pit area, the overburden is generally thin consisting of topsoil and/or peat, and/or sand and gravel fill underlain by discontinuous silty clay to clayey silt, underlain by discontinuous alluvial sand, and generally underlain by silty sand till overtop of bedrock.

Based on the soil materials, the recommended slope configuration is 2H:1V for both MacLellan and Gordon Properties. A 10 m wide berm is recommended at the overburden-bedrock interface to collect any sloughing material and allow access for slope maintenance and drainage control. There is potential to reduce the width to 8 m in areas of the pit away from the influence of water bodies.

The stability of the overburden slopes would be reduced by increased pore pressures, in particular from the nearby water bodies at both pit areas. Therefore, it will be important to maintain adequate drainage, and to take the necessary measures to prevent potential erosion or piping on the slope face. A minimum set-back distance of 45 m is recommended from the water elevation of the river and lakes to the crest of the bedrock slopes, as illustrated on Figure 16-1.

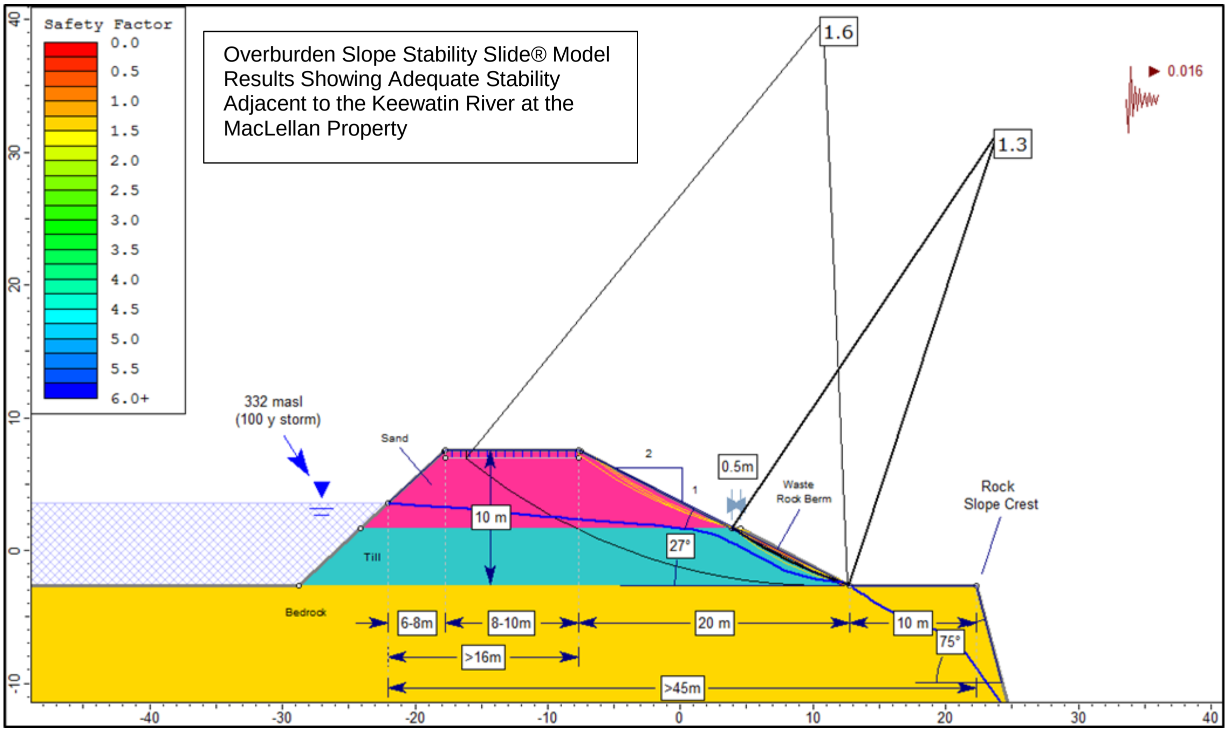


Figure 16-1: Slide® Model Results Showing Adequate Stability Adjacent to the Keewatin River

Source: Golder (2016)

16.2.3 Rock Slope Stability

The MacLellan pit area is characterized by mafic to intermediate volcanics, granodiorite, and mafic to intermediate schist units. The main rock masses that will be exposed on the final pit walls in both pit areas are estimated to be strong (UCS > 50 MPa) and of good to very good quality (typical RMR 70-80), with anticipated localized poor to fair (RMR 35 – 55) quality zones due to minor faults and shears. As a result, large-scale deep seated rock mass failure is not anticipated and the main consideration for rock slope failure on the planned pit walls would be kinematic controlled, in particular due to toppling and planar failure along foliation. The Keewatin River, located to the west of the pit area, may influence the groundwater pressures exerted on the pit walls.

The E-W fault zone currently identified at the MacLellan pit appears to be healed as breccias and favourably oriented with respect to the planned pit shell. Occurrence of other large-scale fault zones have not been interpreted for the proposed pit areas. In addition, intervals with significant broken core and/or fault gouge have not been observed based on the available drill core investigations.

Local instabilities may occur in zones of decreased rock quality, such as where the mineralized contact zone and/or the argillite banded iron formation daylights on the pit walls, or where blasting damage is concentrated. The recommended slope configurations, as outlined in Table 16-1 and Table 16-2 aim to limit the risk due to kinematic controls, and berm widths are planned to catch small scale failures due to planar, toppling, and wedge blocks.

The Gordon pit area is predominantly characterized by banded iron formation, with minor units including mafic to intermediate intrusives, and argillite iron formation. The intensity and lower frictional strength of the foliation in the iron formation rocks at Gordon lead to higher risk of planar failures along the foliation-parallel north and south pit walls. The proposed Gordon pit is surrounded by lakes and wetlands that may influence pore pressure and seepage into the pit, including Farley Lake to the east and Gordon Lake to the west.

Based on the structural assessment, the two pit areas have unique (though similar) structural characterizations, and were considered as separate structural domains. Based on the pit geometries, MacLellan pit was subdivided into eight design sectors and Gordon pit was subdivided into seven design sectors.

Though the rock mass is generally of good quality and strong, the influence of groundwater can exert pore pressures that may lead to pit slope instabilities, particularly from the influence of nearby water bodies. Slope stability modelling showed stable results for the recommended set-back distance of 45 m from the surface water bodies. However, the Slide® model showed saturated slope conditions due to the proximity of the surface water bodies, which will require dewatering and seepage control measures.

The pumping test results provided estimates of relatively low inflows into the MacLellan pit, suggesting groundwater control may be achieved through typical sump pump operations, with opportunity to utilize the current underground mining infrastructure in the early phases of pit development. At the Gordon Property, the pumping test results provided relatively high estimates of inflows in the proposed pit. To control groundwater at Gordon, a grouting program has been recommended to limit seepage.

Excavation of the MacLellan pit will intersect previous underground workings along the pit floor, with some of the historical drifts, stopes, and the shaft being at or in close proximity to the pit walls. In instances where it is suspected that the pit walls and/or floor is in close proximity to the underground workings, probe holes should be drilled in advance of blasting to confirm the size and thickness of the workings. It is anticipated that some of the openings may be collapsed with controlled blasting or backfilled from the pit floor.

16.2.4 Open-Pit Optimization and Design

Overburden slope configuration is recommended at 2H:1V, with a 10 m wide berm to be constructed at the overburden-bedrock interface to collect any surface sloughing and to allow access for slope maintenance and drainage control.

Based on the geotechnical assessment, the open pit rock slope design recommendations for MacLellan and Gordon are given in Table 16-1 and Table 16-2. It is noted that vertical bench separation of 20 m is considered double benching. For the deeper MacLellan pit, 15 m wide geotechnical berms are recommended in areas where the slopes are un-interrupted for a vertical height of 140 m or greater. The design also considers that the slopes will be excavated using controlled blasting (pre-split and trim) techniques for final walls and for slopes located underneath the ramps.

Table 16-1 – Recommended Feasibility Pit Slope Angles at MacLellan

Design Sector	Wall dip Direction	Bench Face Angle (BFA) (°)	Vertical Bench Separation (m)	Bench Width (m)	Inter Ramp Angle (IRA) (°)
Upper (20 m) More Fractured Bedrock Zone					
All Sectors	000° – 360°	75	10	6 to 7	46 to 49
Fresh Bedrock Zone					
I: North Wall	155° (135° to 180°)	80	20	10	56
IA: Northeast Wall	200° (180° to 220°)				
II: East Wall	245° (220° to 270°)	75	20	9	54
IIA: Southeast Wall	290° (270° to 310°)				
III: South Wall	335° (310° to 350°)	80	20	10	56
IIIA: Southwest Wall	010° (350° to 030°)				
IV: West Wall	065° (030° to 095°)	80	20	10	56
IVA: Northwest Wall	115° (095° to 135°)	75	20	9	54

Table 16-2 – Recommended Feasibility Pit Slope Angles at Gordon

Design Sector	Wall dip Direction	Bench Face Angle (BFA) (°)	Vertical Bench Separation (m)	Bench Width (m)	Inter Ramp Angle (IRA) (°)
Upper (20 m) More Fractured Bedrock Zone					
All Sectors	000° – 360°	75	10	6 to 7	46 to 49
Bedrock Zone					
I: North Wall	185° (170° to 210°)	80	20	10	56
IA: Northeast Wall	230° (210° to 255°)				
II: East Wall	300° (255° to 345°)	75	20	10	52
III: South Wall	010° (345° to 035°)	80	20	10	56
IIIA: Southwest Wall	060° (035° to 090°)	75	20	10	52
IV: West Wall	110° (090° to 130°)	80	20	10	56
IVA: Northwest Wall	150° (130° to 170°)				

16.2.5 Support Recommendations

The pit slope design recommendations for rock and overburden account for typical material properties, and are designed with berms to control and protect against rockfalls. Poor quality zones and unanticipated conditions must be allowed for in pit slope operations. Design of support systems for localized instabilities must be facilitated throughout the mining process.

Additional support by use of backfill may be required when encountering historic underground openings while advancing the MacLellan open pit. To minimize against the historic openings daylighting along the interim pit walls and access ramps, the planned Phase 2 slopes have been adjusted to take into account the historical shaft and stope openings.

A Water Management Plan will be developed to implement drainage ditches to divert the surface water from the overburden slope faces. Grading at the top of slopes should be carried out to prevent ponding and infiltration and to direct water away from the face of slopes wherever possible. Surface water that cannot be practically diverted from the slope should be controlled and collected at the crest and toe of benches and discharged appropriately with a series of lined ditches. Waste rock is recommended for overburden slope cover protecting against erosion, and as toe berms to further support the slopes.

Groundwater will be managed by locally installing sub-horizontal drains and by using sumps within the pit, and monitored by vibrating wire piezometers. In addition, deep dewatering system and/or grouting measures will be employed at the Gordon pit perimeter in close proximity to the adjacent lakes to reduce the groundwater inflow.

Routine digital mapping techniques using photogrammetry or LiDAR are recommended with some physical geological mapping of exposed bench faces to supplement and qualify digital mapping derived data. Regardless of how the mapping data is collected, physical bench mapping will be required to assess rock mass characteristics such as discontinuity roughness and alteration. Adequate geological or rock slope engineering personnel are required to review and interpret the mapping so that its implications for slope design can be addressed and remedial measures implemented in a timely manner.

Arrays of prisms will be installed at regular intervals as the pit deepens with two or more EDM (electronic distance measure total stations) survey stations. Alternatively, a laser scanner or a periodic radar survey of the pit may also be considered, in particular, as the MacLellan pit approaches the 280 m depth in the last two years of operation.

16.2.6 Recommendations for Ramp Locations

From a geotechnical perspective, the ramp placement will assist in rockfall hazard reduction by de-coupling otherwise continuous inter-ramp slopes. Geotechnical berms of 15 m width are recommended in areas where the uninterrupted inter-ramp stack is higher than 140 m (vertically).

16.2.7 Excavation Characteristics of Rock Types

Excavation characteristics can be derived from strength and quality of the units as assessed for geotechnical characterization. The typical intact strength of the dominant lithological units are provided in Table 16-3. Note that the mineralized zones of the rock mass, which will be exposed on interim pit walls, were not included in geotechnical assessment, which focused on the stability of the ultimate pit walls.

Table 16-3 – Summary of Intact Rock Strength Parameters

Rock Type	Location	UCS (Laboratory)		Hoek-Brown			Point Load Test				
		Average (MPa)	Count	σ_{ci} (MPa)^2	m_i^1	σ_{ti} (MPa)^3	Average $Is_{(50)}$ (MPa)	K^4	Density (g/cm^3)	Young's Modulus, E (GPa)	Poisson's ratio (μ)
Andesite	MacLellan	**126.9**	26	130	15	-8.9	**-**	-	2.99	28.2	0.16
Basalt	MacLellan	**84.2**	8	90	10	-9.0	**-**	**-**	2.90	20.5	0.13
Argillite IF and BIF +/ – Magnetic (Combined)	Gordon	**101.8**	8	100	9	-11.1	**6.09**	17	2.94	28.6	0.14
Argillite IF	Gordon – West Side	**62.45**	2	60	9	-6.7	**4.94**	13	2.81	22.7	0.15
BIF +/ – Mag	Gordon – North and East Sides	**183.1**	4	175	12	-14.6	**6.67**	26	3.07	34.6	0.14
Biotite Chlorite Amphibole Schist (Chlorite Schist and Intermediate Schist included in $Is_{(50)}$ average calculation)	MacLellan – All sides	**92.6**	12	100	10	-10.0	**9.04**	11	2.95	26.0	0.18
Diorite and Granodiorite	Gordon – South Side	**148.5**	6	140	23	-6.1	**6.19**	23	2.79	31.5	0.14
Intermediate Tuff, Mafic Tuff and Tuff Sediments	MacLellan – West Side	**122.5**	10	120	13	-9.2	**9.48**	13	2.97	32.5	0.14

Notes:

1. *mi = Hoek-Brown material coefficient;*
2. *σci = intact compressive strength;*
3. *σti = tensile strength;*
4. *K = correlation factor between UCS and the point load strength index IS(50). Where available, σ_{ci} was used instead of laboratory UCS to calculate K.*

16.3 Overview of the Planned Open Pit Operating Environment

The development of both deposits is planned with the use of conventional shovel truck open pit mining operations.

The bedrock at MacLellan is overlain by overburden consisting of peat, till, sandy till and gravels with thickness from 0 m to 17 m. There is limited overburden in Gordon due to the prior open pit operations, generally from 0 m to 5 m in thickness. The application of backhoe excavators and articulated trucks for the removal of the overburden at MacLellan is planned. The Gordon overburden is planned to be removed using in part the backhoe and articulated truck configuration or, when possible, the primary mine fleet.

The rock is planned to be pre-drilled with reverse circulation (RC) drilling to further delineate the mineralization zone and enhance selectivity by assisting in the decision to use a 5 m or 10 m operating bench height and the production drill patterns. The RC drilling is performed in advance, typically with a denser pattern in the mineralization zones than in the known waste zones.

The rock is subsequently drilled with the production drilling equipment. Double bench configuration interim and final pit walls are planned to be presplit.

Loading for the ore and waste rock is planned to be carried out with 300 t class hydraulic excavators and 11.6 m^3 FELs, paired with a primary mining truck size of 144 t.

The mill feed sourced from MacLellan is planned to be delivered to the primary crusher directly when possible. However, all the mill feed from Gordon as well as a substantial fraction of the mill feed from MacLellan is planned to be rehandled. The operating environment at the LLGP is outlined in the following as it affects mine planning, the equipment type and size selection for the open pit development and the related estimates mine capital and operating costs.

16.3.1 Pit Limit Design and Mine Plan Methodology

For the generation of LOM plans at the LLGP, primary objectives and constraints include:

- The start-up of operations at MacLellan Year -2, which permits the buildup of sufficient ore stockpiles to feed the process facility to capacity, given the available capacity of Gordon and the highway haulage fleet. In addition, it provides the rock required for roads, infrastructure, and the TMF;
- Operational parameters including minimum mining width, operating area interaction, rates of vertical advance and operation in the vicinity of underground voids at MacLellan;
- Feeding the process facility to capacity;
- The minimization of the required mining equipment capacity for each of the open pits as well as the combined mining equipment capacity. Related to this is the requirement to postpone increasing the mining equipment capacity at MacLellan until the mining of Gordon tails off or ceases;
- The minimization and, as much as possible, the balancing of highway haulage requirements for the transhipment of the Gordon ore to the process facility at MacLellan; and
- The depletion of the Gordon pit as early as possible due to the higher grade of the Mineral Reserve and the use of the stockpiles to present the higher grade to the mill earlier to enhance the financial performance criteria of the Project.

16.4 Excavation Size and Mine Life

The scope and scale of the open pit mine and the process facility is affected by and, as discussed in Section 15 also affects, the total pit tonnage and the total Mineral Reserve tonnage of the Project.

Table 16-4 includes a summary of key overall quantities on the excavation size for the Gordon and MacLellan open pits as well as some additional key global quantities related to the Mineral Reserve tonnage.

Table 16-4 – Pit Tonnages for the Lynn Lake Gold Project Open Pits

	Gordon	MacLellan	Total
Total In-Pit Tonnage (Mt)	58.60	163.38	221.98
Ore (Mt)	8.72	18.08	26.80
Waste Rock (Mt)	48.75	137.56	186.11
Overburden (Mt)	1.13	6.47	7.80
UG Voids Fill (Mt)	n/a	1.27	1.27
In-pit Pit Waste (Mt)	49.88	145.30	195.18
Ex-pit Borrow Waste (Mt)	0.17	1.65	1.82
Total Waste (Mt)	50.05	146.95	197.00
Process Capacity (Mt/a)	0.7 to 1.6	0.95 to 2.56	2.56
Mine Life (Years)	5.5	10.4	10.4
Pre-stripping (Years)	1	2	2
Peak Mine Rate (Mt/y)	13.50	24.72	27.05

16.5 Sequence of Open Pit Development

The Gordon pit contains ore of higher grade. As a result, it is desirable to develop and deplete this pit as early as possible. Smoothing the highway haulage requirements as well as considerations with respect to the rate of vertical advance and the mine capacity at Gordon, support a mine life of approximately 5.5 years for this deposit.

The FS mine plan calls for stripping/mining to initially start in the in-pit and ex-pit borrow sources at MacLellan. This is governed by the requirement for construction rock for the infrastructure as well as the requirement to present ore at the nominal plant capacity at start-up. The latter requires that ore production take place at both Gordon and MacLellan for the first five years of the operation.

The plant capacity and required ore presentation to the crusher, the mine operating life for both deposits, the number of phases, the applied mine capacity the mine operating life, the rate of vertical advance and interaction of operations between successive phases are interrelated parameters affected by and affecting each other.

16.6 Phasing

The phased development of both the Gordon and MacLellan open pit is planned. Gordon will be developed with two phases, and MacLellan will be developed with three phases.

16.6.1 Slope Angles for Internal Phase Walls

Q'Pit elected to utilize the single bench (10 m) wall configurations for a major part of Phase 1 of Gordon (49^o slope) and the entire Phase 1 of MacLellan (49^o and 46^o slopes). Similarly, a single bench configuration was used for the top several benches of Phase 2 of MacLellan (46^o and 49^o slopes). In addition, a 15 m wide berm in Phase 1 of Gordon and in Phase 2 of MacLellan was incorporated to provide for enhanced spillage control.

16.6.2 Phasing for Gordon

The Gordon open pit is located between two water bodies and encompasses two existing pits named the Wendy and East Pits. The waste dump, overburden storage and ore stockpiles are located to the south of the open pit.

For the phasing of Gordon, a target minimum mining width of 50 m for sustained phase mining and an operation with two primary loading units are planned.

The phase development for Gordon mines the higher grade and lower stripping ratio part of the Mineral Reserve in the west part of the pit as early as possible with Phase 1.

The FS mine plan indicates a mine life at 5.5 years plus one year of pre-stripping for Gordon, compared to 10.4 years and two years of pre-stripping at MacLellan. The open pit and deposit geometry, the planned open pit total depth, the expected target rate of vertical advance, the level of envisioned production, and the overall Project mine life dictate that Gordon be developed with at most two phases.

The phased development at Gordon is expected to enhance the opportunity to even out haulage requirements and drilling related to final wall control blasting, the latter becoming an issue as mining to the pit limit in some sectors is effectively required from the start-up of mining operations.

16.6.3 Phasing for MacLellan

The MacLellan open pit is located east of the Keewatin River. The location of the primary site access is planned to be from the southeast of the pit. The waste dump, overburden storage and ore stockpiles are located to the north of the open pit. The proposed open pit limit at MacLellan encompasses part of the historical underground mining openings. Phases are designed to reduce or minimize the exposure of stoped zones on the intermediate phase pit walls and ramps were placed preferentially away from the stoped zones when possible.

The target mine capacity of 14 Mt/y to 27 Mt/y supports the operation of two to three primary loading units. A target minimum mining width of 50 m and a typical target operating width of 60 m to 110 m or more are planned.

The target vertical advance is set at 50 m to 90 m per annum, depending on the number of 10 m and 5 m operating benches within each year and phase.

The development of MacLellan using three phases is planned, in addition to the in-pit borrow source excavation. For the placement of the ramps, phasing considered the reduction of operational interaction between successive phases and, specifically with the planned three phase development, the operational interaction between Phase 2 and Phase 3.

The size of Phase 3 is determined based on meeting the balance of several constraints and objectives, including the target vertical advance, the envisioned range of mill feed presentation per annum, the operating bench heights of 5 m and 10 m, the orebody geometry and the available mill feed per bench. In addition, Phase 3 must meet the requirement to accommodate the full primary mine fleet after the depletion of Gordon in Year 6.

For the location and final overall size of Phase 2, the potential for interaction of operations between Phases 2 and 3 was considered and the time of its depletion with the LOM mine plan. In addition, splitting the requirements for mining in the vicinity of the historical underground workings was considered as well as the location of the shaft.

The Proven and Probable Mineral Reserves released by each phase are listed in Table 16-5. It can be noted that the planned phase development presents the Mineral Reserve of the two open pits in a well ranked manner, in the sense that the lower stripping ratios and lower modelled cash costs are encountered earlier, except for Phase 2 of MacLellan.

The planned phase development achieves the presentation of the higher grade and lower stripping ratio of the Mineral Resource with Phase 1 for both Gordon and MacLellan. It can also be seen that Phase 1 of MacLellan presents 12% of the Mineral Reserve tonnage and 13% of the contained Au metal and that Phases 2 and 3 of MacLellan present the Mineral Reserve with Au grade and stripping ratio close to the respective average values for the open pit.

Table 16-5 – Summary of Key Quantities by Phase for Gordon and MacLellan

	Gordon				MacLellan			
Phase	Phase 1	Phase 2	Total Gordon	Borrow	Phase 1	Phase 2	Phase 3	Total MacLellan
Total Tonnes (kt)	16,184	42,452	58,636	2,425	12,313	57,935	90,710	163,384
Proven and Probable Mineral Reserves								
Mill Feed (kt)	2,785	5,938	8,723	0	2,089	5,849	10,142	18,080
Au (g/t)	2.68	2.30	2.42	0	1.82	1.62	1.59	1.63
Au Contained (oz x 1000)	240	438	678	0	122	305	520	947
Stripping Ratio	4.8	6.1	5.7	n/a	4.9	8.9	7.9	8.0
Cumulative Total Tonnes	28%	72%	100%	1%	8%	35%	56%	100%
Cumulative Ore Tonnes	32%	68%	100%	0%	12%	32%	56%	100%
Cumulative Contained Au Metal	35%	65%	100%	0%	13%	32%	55%	100%

16.7 Mine Planning

Mine planning was carried out in annual increments to the end of mine life for both deposits.

The mine plan calls for stripping/mining to initially start in the in-pit and ex-pit borrow sources at MacLellan in Year -2. The development of Gordon is envisioned to start the following year.

The mine plan places emphasis on the requirement to present the higher-grade mill feed from Gordon preferentially for the first 5.5 years of operation at a rate of up to 4500 t/d.

The mine capacity at Gordon and a stockpiling strategy were used as control variables to achieve smoothing the highway haulage requirements as much as possible, presenting the higher grade to the primary crusher as early as possible and completing the highway haulage operations as close as possible to the end of mining operations at Gordon.

The rate of maximum vertical advance was limited to 12 operating benches per phase per annum for MacLellan. This constraint is active in the latter years of operations, supporting the use of a stockpile to meet the constraint of feeding the process facility at its nominal capacity. In contrast, a lower vertical advance of up to 45 m vertically or nine 5 m operating benches is attained at Gordon.

Table 16-6 includes the estimates of key quantities for the Mineral Reserve presentation with the mine plan in annual increments.

The mine plan calls for peak mining rate of 13 Mt/y at Gordon and 24.7 Mt/y at MacLellan and a mine life of 10.4 years after the pre-stripping operations. The mine plan calls for total mining in a relatively narrow range of 20.5 Mt/y to 27.0 Mt/y for the first seven years of operation. The mine capacity from Year 7 onwards at MacLellan is affected by and affects the number of haulage trucks in the fleet and is limited by the rate of vertical advance.

Figure 16-2 presents the mill feed by source pit and the total Au grade profile by period. Figure 16-3 shows the tonnes mined by pit by period.

Maps indicating the planned mine and waste storage area advance for Years 1, 3, 5, 8 and 11 for Gordon and MacLellan are listed in Figure 16-4 to Figure 16-11.

Table 16-6 – Summary of Key Quantities for Lynn Lake Gold Project Mine Plan

Year	Y-2	Y-1	1	2	3	4	5	6	7	8	9	10	11	Total
Gordon Tonnes Mined (kt)	0	3,911	11,000	12,954	13,000	10,500	6,045	1,200	0	0	0	0	0	58,608
MacLellan Tonnes Mined (kt)	1,522	3,472	9,525	14,095	14,012	14,070	20,771	24,721	23,129	17,010	13,749	6,006	1,300	163,382
Total Tonnes Mined (kt)	1,522	7,383	20,524	27,049	27,012	24,570	26,816	25,921	23,129	17,010	13,749	6,006	1,300	221,991
Gordon														
Ore Tonnes Mined (kt)	0	120	1,647	1,651	1,805	1,738	1,406	356	0	0	0	0	0	8,723
Au Grade (g/t)	0	1.82	2.79	2.36	2.09	2.19	2.52	3.53	0	0	0	0	0	2.42
Au Ounces (oz x1000)	0	7	148	125	122	122	114	40	0	0	0	0	0	678
Waste Tonnes Mined (kt)	0	3,790	9,352	11,303	11,195	8,762	4,639	844	0	0	0	0	0	49,886
MacLellan														
Ore Tonnes Mined (kt)	0	251	1,060	1,227	906	1,213	2,242	2,168	1,866	1,813	2,507	2,110	715	18,080
Au Grade (g/t)	0	1.64	1.79	1.68	1.36	1.41	1.78	1.81	1.42	1.4	1.43	1.8	2.31	1.63
Au Ounces (oz x1000)	0	13	61	66	40	55	128	126	85	81	115	122	53	947
Ag Grade (g/t)	0	3.47	5.18	5.19	3.89	3.69	3.92	5.46	4.72	3.8	3.86	4.47	5.65	4.44
Ag Ounces (oz x1000)	0	28	177	205	113	144	283	380	283	221	311	303	130	2,578
Waste Tonnes Mined (kt)	1,523	3,221	8,464	12,867	13,106	12,857	18,529	22,553	21,263	15,197	11,241	3,895	585	145,302
Process Summary														
Process Nominal Capacity (t/d)	0	Variable	7,000	7,000	7,000	7,000	7,000	7,000	7,000	7,000	7,000	7,000	7,000	
Annual Mill Feed (kt)	0	322	2,555	2,555	2,555	2,555	2,555	2,555	2,555	2,555	2,555	2,555	930	26,803
Au Grade Contained (g/t)	0	1.59	2.51	2.28	1.91	2.06	2.63	2.04	1.27	1.18	1.42	1.59	1.91	1.89
Au Contained (oz x 1000)	0	16	206	187	157	169	216	167	104	97	116	131	57	1,625
Ag Grade Contained (g/t)	0	2.7	1.97	2.08	1.38	1.42	2.67	4.66	3.53	3.5	3.84	4.17	4.99	2.99
Ag Contained (oz x 1000)	0	28	162	171	113	117	220	383	290	288	315	343	149	2,578
Strip Ratio (Waste/Ore)	n/a	18.9	6.6	8.4	9	7.3	6.4	9.3	11.4	8.4	4.5	1.8	0.8	7.3

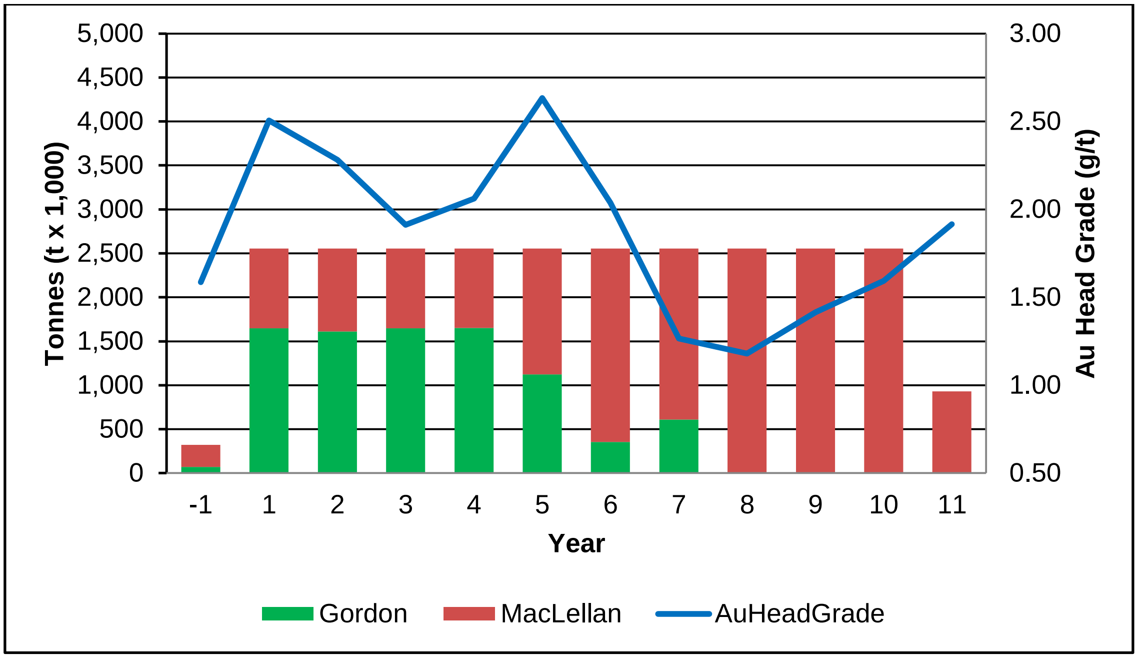


Figure 16-2: Mill Feed by Source Pit and Au Grade Profile

Source: Q'Pit (2017)

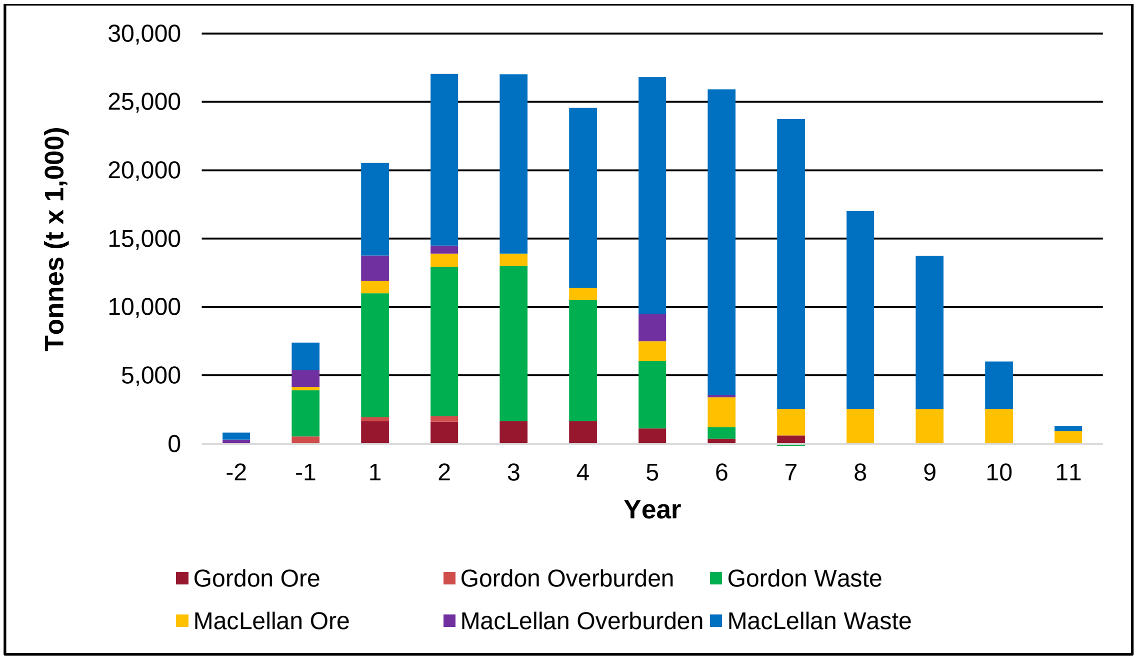


Figure 16-3: Material Mined by Pit by Period

Source: Q'Pit (2017)

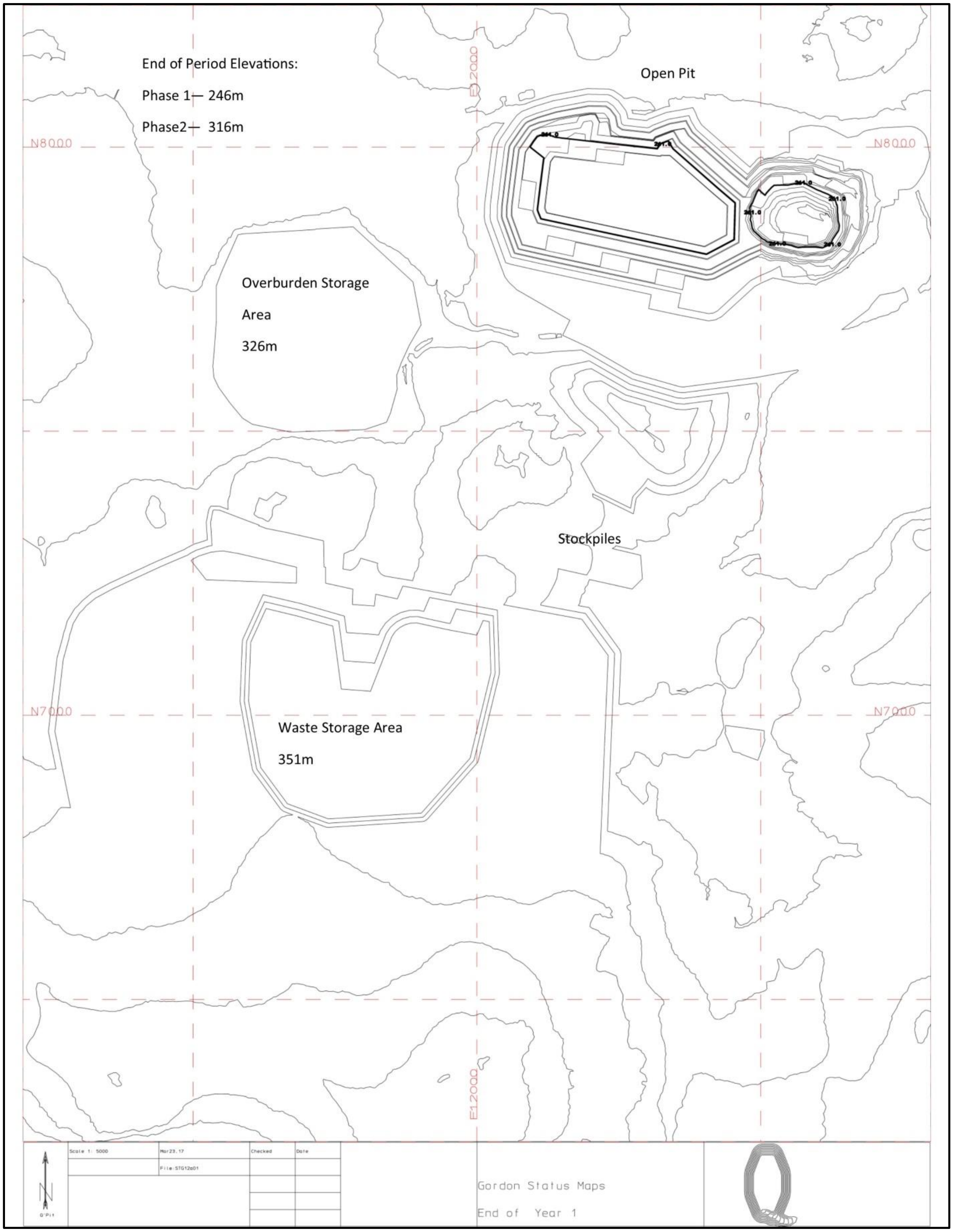


Figure 16-4: Gordon Planned Mine and Waste Dump Advance for Year 1

Source: Q'Pit (2017)

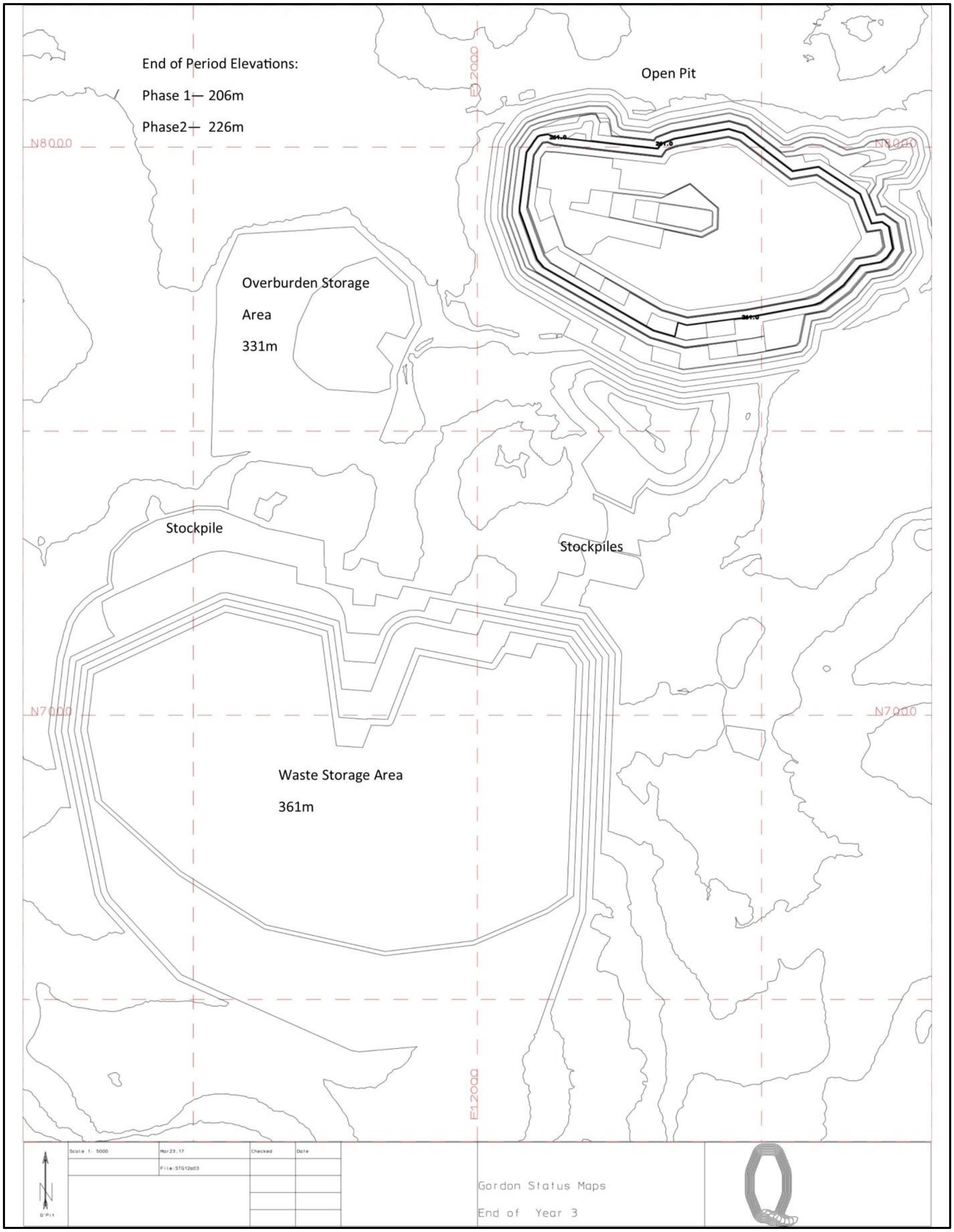


Figure 16-5: Gordon Planned Mine and Waste Dump Advance for Year 3

Source: Q'Pit (2017)

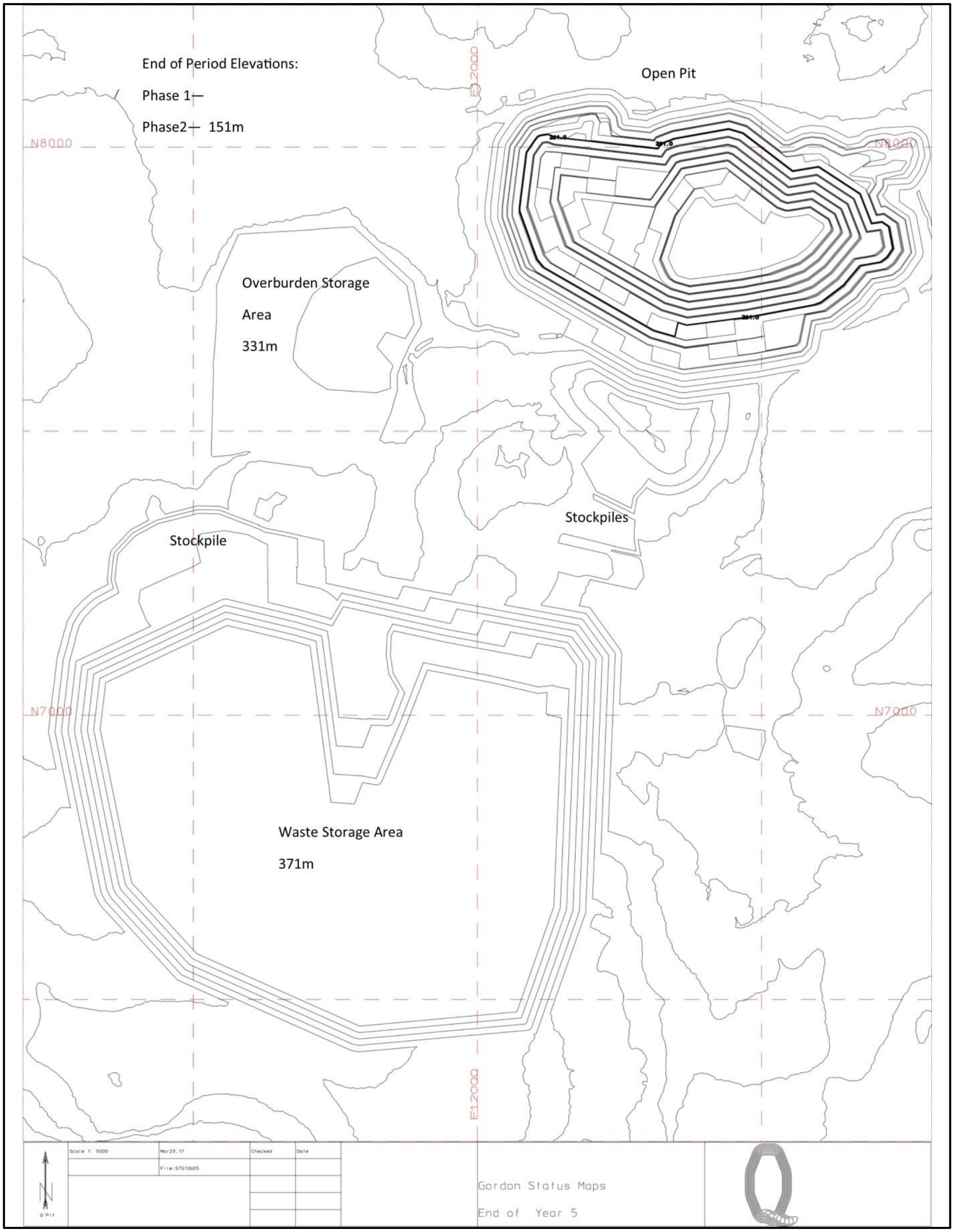


Figure 16-6: Gordon Planned Mine and Waste Dump Advance for Year 5

Source: Q'Pit (2017)

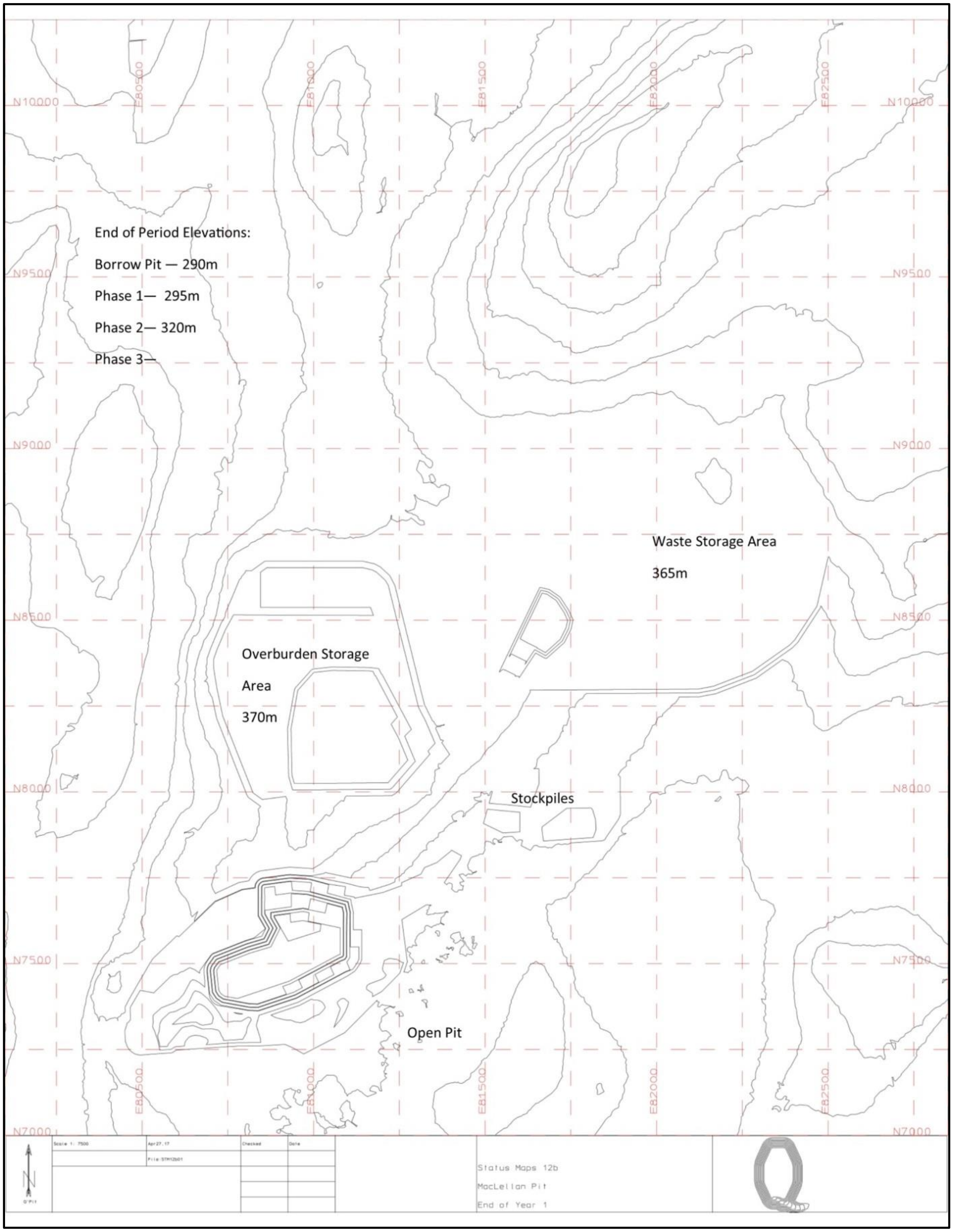


Figure 16-7: MacLellan Planned Mine and Waste Dump Advance for Year 1

Source: Q'Pit (2017)

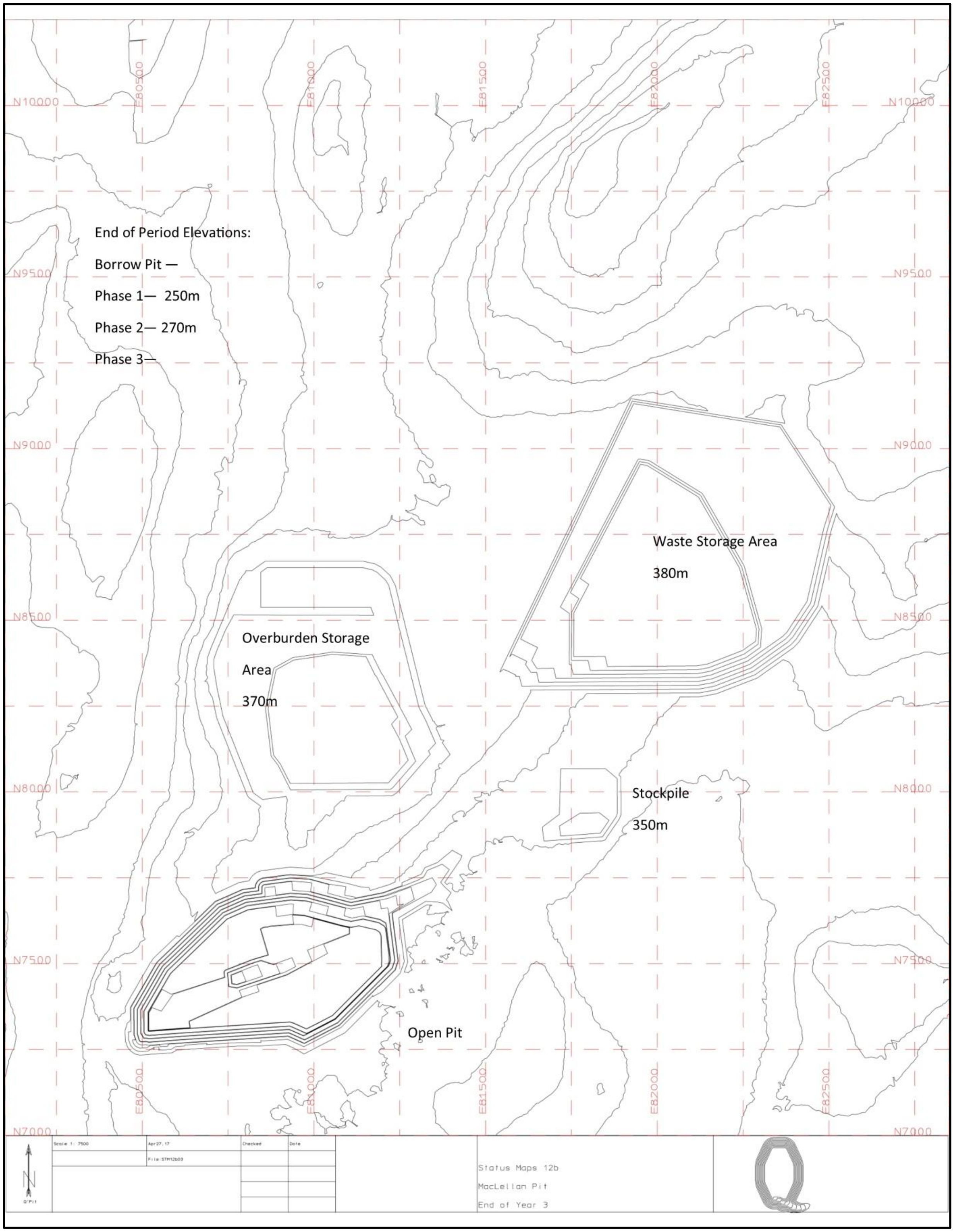


Figure 16-8: MacLellan Planned Mine and Waste Dump Advance for Year 3

Source: Q'Pit (2017)

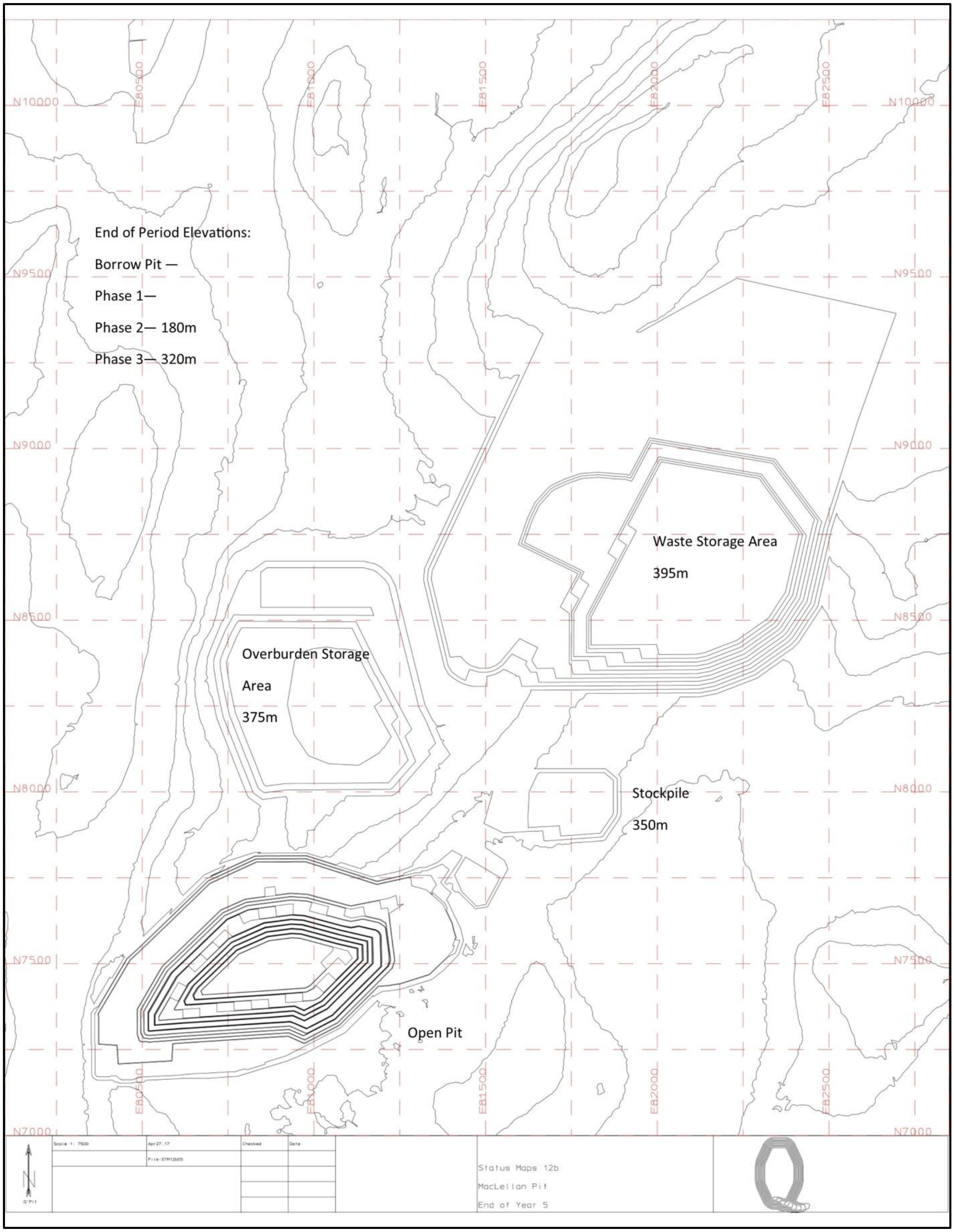


Figure 16-9: MacLellan Planned Mine and Waste Dump Advance for Year 5

Source: Q'Pit (2017)

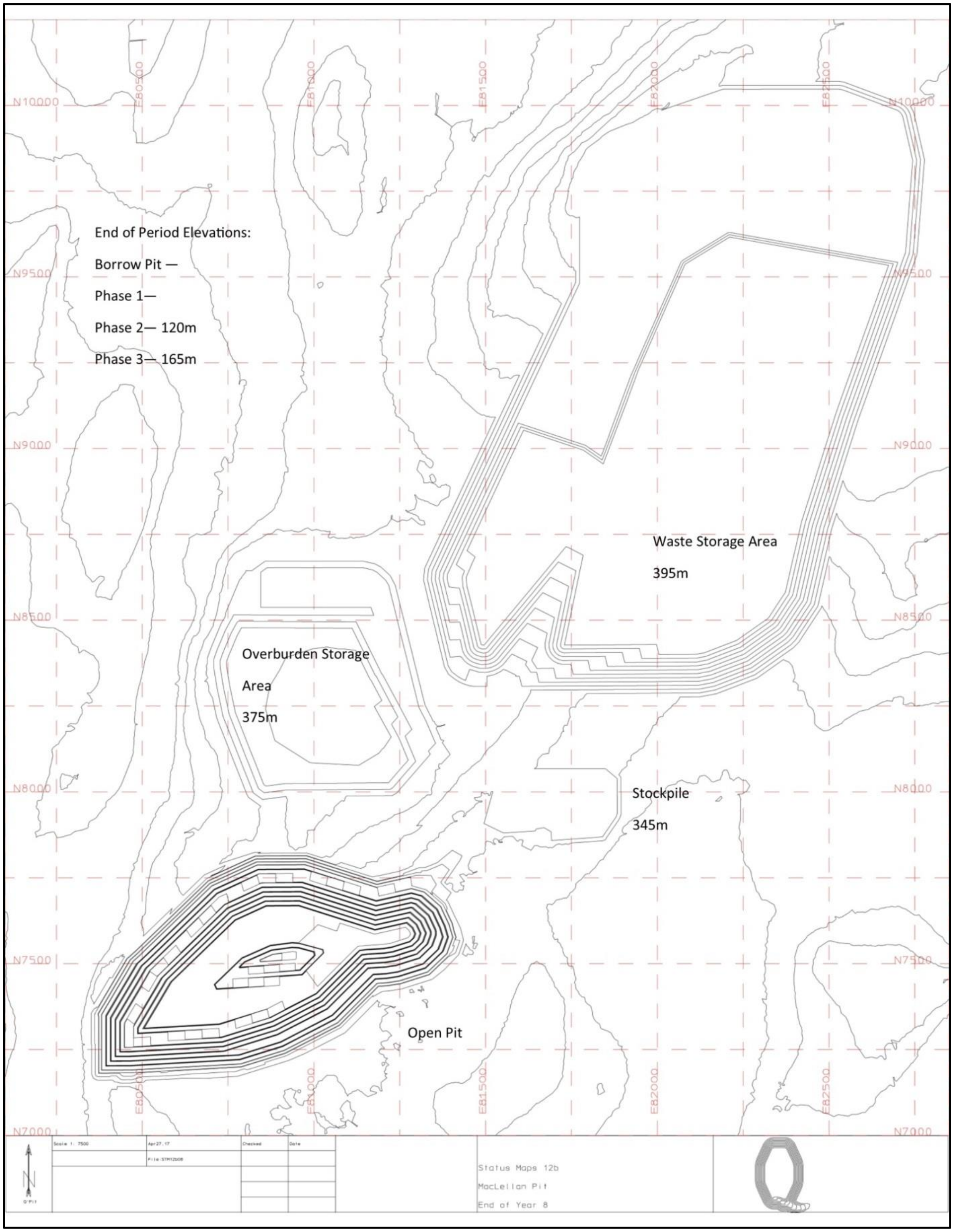


Figure 16-10: MacLellan Planned Mine and Waste Dump Advance for Year 8

Source: Q'Pit (2017)

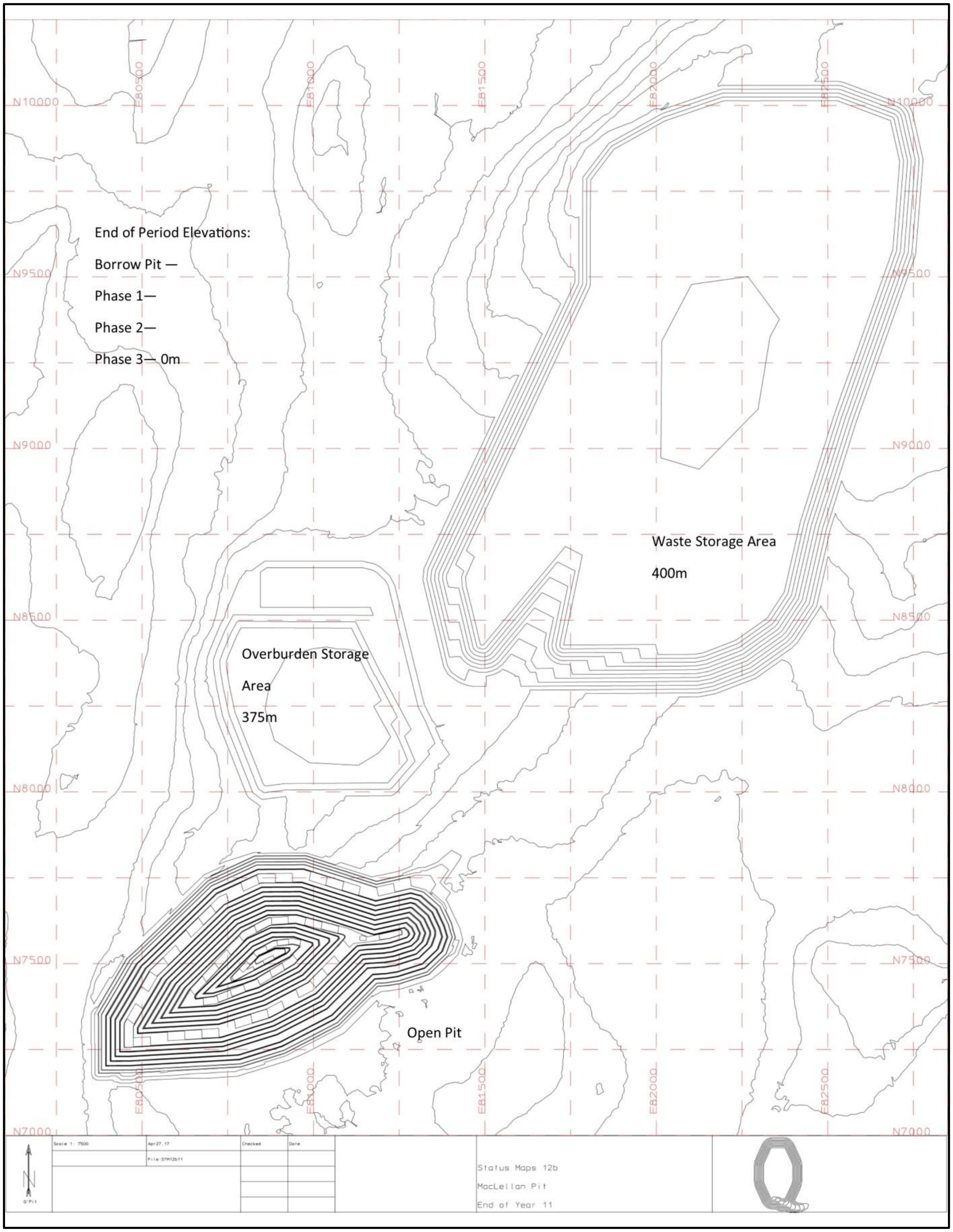


Figure 16-11: MacLellan Planned Mine and Waste Dump Advance for Year 11

Source: Q'Pit (2017)

16.8 Pre-Stripping

The mine production schedule considers a two-year pre-production period at MacLellan and a one-year pre-production period at Gordon.

During the pre-production period, a total of 12.2 km of in-pit and ex-pit roads are planned to be constructed at MacLellan and 3.8 km at Gordon. At both MacLellan and Gordon the requirements for construction rock are met by mining in two borrow sources, one in-pit and one ex-pit. The deposit at MacLellan is overlain by overburden, with thickness varying from 0 m to 17 m. The mineralization zone itself outcrops to the overburden rock contact. Pre-stripping accordingly involves primarily the removal of overburden. Due to the quantity of overburden, the overburden stripping at MacLellan starts in Year -2.

The mine plan calls for the removal of a total of 5.0 Mt of material during the pre-production at MacLellan of which 1.8 Mt is overburden. Pre-stripping at Gordon is set at 3.9 Mt.

In addition, the schedule considers mining of material from ex-pit borrow sources of 1.65 Mt at MacLellan and 0.17 Mt at Gordon.

16.9 Waste Storage

The Gordon open pit total excavation size is estimated at 58.6 Mt. In addition to the Mineral Reserve, the Gordon open pit is estimated to contain 1.1 Mt of overburden and 48.7 Mt of waste rock materials.

The MacLellan open pit total excavation size is estimated at 163.4 Mt. In addition to the Mineral Reserve, the MacLellan open pit is estimated to contain 138.8 Mt of waste rock and 6.5 Mt of overburden materials.

The Gordon rock waste dump, overburden storage pile and the mill feed stockpiles are located to the south of the open pit. The MacLellan rock waste dump, overburden storage pile and the mill feed stockpiles are located to the north and northeast of the open pit.

The waste dumps are designed using a final wall slope of 2.25:1. The construction of the waste dumps is planned in 10 m lifts in Gordon and up to 15 m lifts in MacLellan.

16.10 Stockpile Storage

The operations include material movement through stockpiles for both Gordon and MacLellan.

The short term stockpiles at Gordon are used for the transhipment of the material to highway trucks. In addition, there is a provision for a long-term stockpile at Gordon, which is used for storage of the lower grade ores as well as the storage of the mill feed during pre-production while the processing facility is not yet available. The entirety of the stockpiled material at Gordon is planned to be depleted by Year 7, one year after the mining operations at Gordon cease. The material movement schedule uses stockpiling to even out the loading and highway haulage requirements for the Gordon mill feed transhipment as well as for the presentation of the higher grade to the primary crusher as early as possible.

The stockpiles at MacLellan are similarly planned for the long-term storage of the lower grade ores, which are re-handled during the period of short supply of ore from the MacLellan open pit and after the end of the mine life. It was discussed earlier that the geometry of the orebody and the MacLellan open pit development lead to a period of two years during which the mill feed presentation from the open pit is less than the plant capacity. Accordingly, this sets the minimum amount of ore in a long-term stockpile required to provide the mill feed during this period. The maximum size of the long-term stockpile at MacLellan is 1.6 Mt. Similar to the material movement at Gordon, the current mine plan takes advantage of the requirement for stockpiling at MacLellan to manage the grade of the mill feed presentation to the primary crusher.

16.11 Run of Mine Stockpile

A provision for a ROM stockpile at the primary crusher at MacLellan has been made. This provides for the decoupling of operations when the primary crusher in not available as well as for the re-handle of the Gordon mill feed to the primary crusher. Re-handle during the first six years of operation will use a dedicated 4.6 m^3 FEL loader. After Year 6, ROM re-handle will be on an as needed basis.

The use of the ROM stockpile is planned to be reduced from Year 7 onwards, as MacLellan becomes the only open pit source of ore.

16.12 Voids Management

MacLellan has been subject to past underground (UG) development, primarily in the Main zone, but as well as the Niksu zone in the eastern part of the open pit. Mapped drifts are generally under 5 m x 5 m in dimensions. Stopes are vertical or sub-vertical with dimensions ranging from 20 m to 65 m in length and from 5 m to 15 m in width. Stopes are reportedly filled with sand. A shaft with dimensions of 3 m by 6.5 m will also be mined out with Phase 2 to elevation 135 m or approximately 200 m in depth. Typically, considering a 10 m operating bench in the vicinity of stopes and voids, the perimeter of each stope is expected to be delineated and encountered during operations for three to four operating benches and up to 13 benches for one of the stopes.

Several operations in Australia, Canada and the US conduct open pit mining operations around voids from past underground operations. In some open pit mines, the extent of the historical underground development is substantial, and in one case 90 km of underground development was encountered in an open pit mine in Northern Canada (Harnnett, 2017). The length of the MacLellan underground development inside the planned open pit, shown in Figure 16-12, consists of approximately 3.3 km of development drifts, access ramps and shafts and 6.7 km of stope perimeter.

The MacLellan underground operations are relatively recent and stopes and drifts were well mapped during operations and confirmed by exploration drilling. Specialized operating procedures and equipment will be required for void detection, delineation and operations near or through the historical underground development openings and voids.

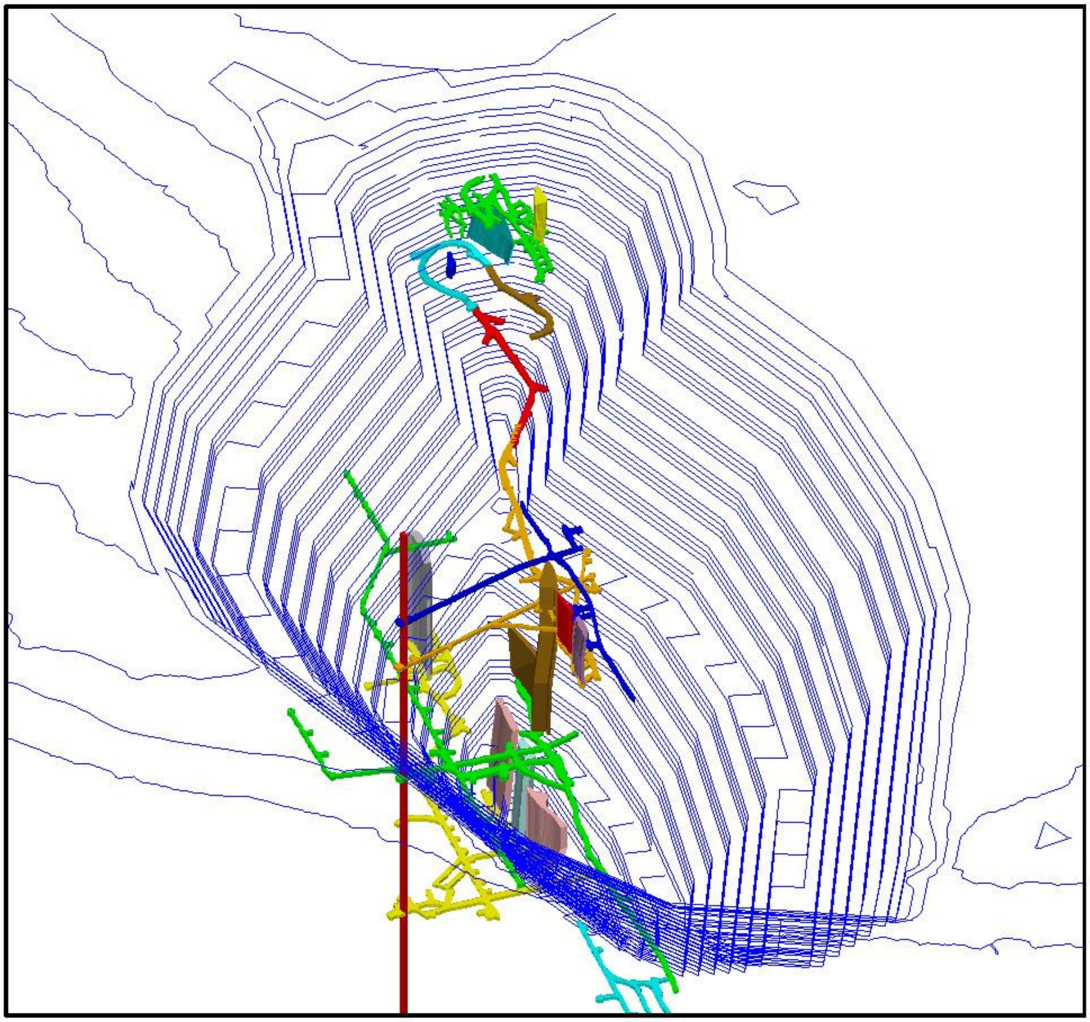


Figure 16-12: Underground Development within the MacLellan Pit

Source: Q'Pit (2017)

16.13 Operating Bench Height

For the equipment type and size selection in this study, it was considered that operating benches of 10 m height and 5 m height will be employed for both Gordon and MacLellan. The major part of the mineralization will be preferentially mined using a 5 m operating bench to enhance selectivity.

Mining for the top several (3 to 5) benches in Gordon is expected to take place using mainly a bench height of 10 m. This includes mining in the perimeter of the top benches of the legacy Wendy and East pits where the mineralization has been effectively depleted by prior operations.

A major part of the waste mining in Phase 2 and Phase 3 of MacLellan is planned to take place using a 10 m operating bench, to take advantage of the cost efficiencies of the higher operating bench. In addition, mining in the vicinity of the underground voids, irrespective of material type, is planned to take place typically employing 10 m operating benches and to require specialized methods, including drilling and blasting at variable drilling column heights, in some cases in excess of 10 m.

16.14 Mine Equipment Selection

16.14.1 Overview of the Equipment Type and Size Selection Considerations

The LLGP plans to use 300 t class hydraulic shovels such as the Cat 6030FS or the Komatsu PC-3000 with a 14 m^3 bucket and 11.6 m^3 front end loaders as the primary loading units, paired with 144 t rigid body trucks as the primary mine truck. The use of 60,000 lbf hydraulic top head drive drills as the primary production drilling unit, with down the hole (DTH) rock hammer configuration for 178 mm diameter drill holes is also planned.

For the overburden removal at MacLellan, the use of 75 t and 90 t class hydraulic excavator shovels with 4 and 6 m^3 buckets respectively are planned. The shovels will be combined with 39.5 t articulated trucks such as the Caterpillar 740B. It is expected that these units will also be used at the pit bottom where narrower and steeper ramps are incorporated in the design. The implementation of a multiple truck class size operation and the corresponding ramp widths has been widely applied in the development of open pit mines in Northern Canada.

For the overburden removal at Gordon, the use of two 75 t (4 m^3 capacity) class hydraulic excavators is planned. The same excavator was selected for the loading of the Gordon mill feed to highway trucks for transport to the plant site.

Highway trucks of 30 t capacity were selected, which are the largest capacity units permitted to travel on the provincial highway.

Key considerations for the selection of the primary and secondary equipment unit type and size include:

- The site specific operating conditions at the LLGP which include operating in severe climatic conditions in the winter and operating around underground openings in MacLellan;
- The remote location of the mine sites;
- The relative scale of the mining operations for both total tonnes mined and mill feed, the highway haulage, re-handle as well as, the operating life of the each one of the open pits and the overall Project;
- Operating with two or more primary loading units at each mine site;
- The characteristics of the rock to be mined;
- The expected selectivity requirements from the both Gordon and MacLellan deposits; and
- The expected impacts on the Project overall operating and capital costs.

The primary equipment requirements were estimated based on productivity calculations carried out by Q'Pit, with reference to the mine plan and conditions at the Gordon and MacLellan sites. The support equipment fleet was selected with reference to operations of similar scope and size.

Key parameters for the estimate of equipment requirements include operation for 360 days per year, allowing for 15 days lost due to adverse weather conditions, using two 12 hour shifts. A total of four hours per day (two hours per shift) are allocated to shift change, rest/lunch breaks and refueling, resulting in the use of a net 20 operating hours per day to be used for the equipment requirements estimates.

16.14.2 Loading and Hauling

The estimates for the requirements of the primary loading units is based on the quantity of the mine plan and the productivity derived as listed in Table 16-7.

Table 16-7 – Loading Equipment Productivity and Annual Production Estimates

	Face Shovel Cat 6030 and HD1500	Front End Loader Cat 993 and HD1500
Bucket Size (m^3)	14	11.6
Swing Cycle (seconds)	32	42
Bucket Fill Factor (%)	90	90
Swell Factor (%)	35	35
Passes	5	6
Loading Time (min)	3.42	4.95
Truck Load (dry t)	140	139.2
Truck Load Volume (m^3)	63	62.64
Spotting Time (min)	0.75	0.75
Productivity (t/h)	2,459	1,687
Productivity (t/d Operating)	49,171	33,745
Shovel Idle (%)	33	33
Production (t/d Operating)	32,944	22,609
Availability (%)	90	90
Direct Operating Hours (h/y)	6,480	6,480
Direct Loading Hours (h/y)	4,342	4,342
Nominal Planning Production (Mt/y)	10.67	7.33
Nominal Planning Production (t/d)	29,650	20,349

A simplified set of speeds was used for estimating haulage requirements, as listed in Table 16-8. These were used in conjunction with the haulage profiles derived based on the LOM mine plan layout and quantities from the LOM plan. For the purposes of the loading and hauling requirements estimates, the quantity of re-handle for operating floor preparation was set at 5% of the overburden volume and 2% for rock mining.

The queue time was set at 10% of the total cycle, considering that both operations have excess loading capacity.

Table 16-8 – Truck Speeds for Estimate of Haulage Requirements

Haulage Segment	Speed Loaded (km/h)	Speed Empty (km/h)
Ramp Uphill, grade 10%	10	30
Ramp Downhill, grade 10%	17	27
Flat on Operating Bench	25	25
Flat	35	40
Queue Time	10% of total cycle	

For the estimate of the equipment requirements, the target equipment availabilities used are shown in Table 16-9. These are based on experience from operations using equipment of similar size and scope. The availability and the target maximum use of availability are utilized for the estimate of required units and apply effectively only on the peak or maximum of the fleet requirements.

Table 16-9 – Primary Equipment Target Availability for Equipment Requirements Estimates

	Target Availability	Maximum Use of Availability
Hydraulic Shovels (300 t class)	85%	Variable, 60% to 90%
Front End Loaders (11.6 m^3)	85%	Variable, 60% to 90%
Backhoe Excavators (74 t to 90 t class)	80%	Variable, 60% to 90%
Mine Haul Trucks	85% to 90%	80% to 95%
Highway Trucks	90%	90%
Articulated Trucks	85%	Variable, from Idle to 90%
Drills	85%	Variable, <80%
Support Equipment	85%	Variable, <80%

16.14.3 Drilling and Blasting

The estimates of the requirements for drilling and blasting are based on the parameters listed in Table 16-10 and 16-11 for MacLellan and Gordon respectively. The parameters used for the drilling and blasting estimates for grade control drilling, presplitting and underground void ceiling collapse are shown in Table 16-12 and Table 16-13.

16.14.3.1 Drilling

The use of 178 mm drill holes is planned for production drilling. Presplitting, void collapse and delineation drilling will use 114 mm drill holes. All drilling is planned with a DTH hammer configuration. Presplitting considers spacing of 1.5 m and use of the same drill bit diameter of 114 mm. The grade control drilling will use a 114 mm reverse circulation configuration.

The material compressive strength is one of the primary factors in the evaluation and selection of the drill parameters. From the existing information, the uniaxial compressive strength for Gordon ranges from 62 MPa to 183 MPa and averages approximately 128 MPa. MacLellan materials have an average uniaxial compressive strength of approximately 113 MPa, ranging from 84 to 127 MPa. For the purposes of the drilling and blasting estimates, a conservative uniaxial compressive strength of 150 MPa was used.

An instantaneous penetration rate of 32.5 m/h for the 178 mm drilling and 50.7 m/h for the 114 mm drilling was derived for the Gordon Pit. Due to the lower UCS rock in MacLellan Pit, a higher by 10% penetration rate was used, resulting in an average of 37.3 m/h for the 178 mm drill bit and 58.3 m/h for the 114 mm drill bit. The fixed time for bit change and drill hole move time was estimated to be from 5.43 min/hole to 5.65 min/hole. The fixed time was incorporated into the estimate to determine an effective (overall) penetration rate ranging from 21.42 m/h to 25.40 m/h for Gordon and from 23.44 m/h to 28.28 m/h for MacLellan.

16.14.3.2 Blasting

Due to expected wet conditions at the mine sites, it is planned that bulk emulsion will be used for blasting at both Gordon and MacLellan. The emulsion is expected to be manufactured on-site or prepared on-site using a mixing facility.

During the pre-production period, while the emulsion facility is under construction, a packaged gel emulsion product will be used. Packaged product will continue to be used for pre-shearing and, at MacLellan, for underground void ceiling controlled collapse.

The blasting consumables for this project will be conventional products, including one lb boosters, non-electric delay detonators, detonating cords for tie-in patterns, delays and lead line.

Budget quotations were requested from two explosives manufacturers and costs were obtained for an on-site manufacturing facility as well as an on-site mixing facility only. The budget quotes indicate that emulsion unit costs for an on-site manufacturing facility is lower, however the facility construction cost and resulting fixed monthly charges are higher. It is unclear at this time which option is most cost effective. The selected location of the explosives preparation platform permits using either one of the two options.

The selected powder factors are 0.31 kg/t for ore and 0.34 kg/t for waste at Gordon and 0.31 kg/t for ore and 0.30 kg/t for waste at MacLellan.

Table 16-10 – Production Drilling and Blasting Parameters – MacLellan

Parameter	Unit	Ore Common Bit size Comparison 178mm (7") DTH	Ore Common Bit size Comparison 178mm (7") DTH	Waste Common Bit size Comparison 178mm (7") DTH	Waste Common Bit size Comparison 178mm (7") DTH
Bench Height	m	5.00	10.00	5.00	10.00
Hole Diameter	mm	178	178	178	178
Burden	m	4.90	5.40	4.90	5.40
Spacing	m	4.90	5.40	4.90	5.40
Area of Influence	m^2	24.01	29.16	24.01	29.16
Sub Drill	m	0.70	1.00	0.70	1.00
Stemming	m	2.25	3.00	2.25	3.00
Powder Column	m	3.45	8.00	3.45	8.00
Loading Density	kg/m	31.84	31.84	31.84	31.84
Emulsion Powder/hole	kg	109.83	254.69	109.83	254.69
Bulk Emulsion SG	SG	1.28	1.28	1.28	1.28
Rock SG	SG	2.958	2.958	2.947	2.947
Tonnes per Hole	tonne	355	863	354	859
Powder Factor	kg/tonne	0.31	0.30	0.31	0.30
Powder Factor	kg/m^3	0.91	0.87	0.91	0.87
Instantaneous Penetration Rates	m/h	37.3	37.3	37.3	37.3
Hole Change Time	min/hole	5.43	5.65	5.43	5.65
Effective Penetration Rates	m/h	23.44	28.28	23.44	28.28

Table 16-11 – Production Drilling and Blasting Parameters – Gordon

Parameter	Unit	Ore Common Bit size Comparison 178mm (7") DTH	 178mm (7") DTH	Waste Common Bit size Comparison 178mm (7") DTH	 178mm (7") DTH
Bench Height	m	5.00	10.00	5.00	10.00
Hole Diameter	mm	178	178	178	178
Burden	m	4.60	5.20	4.60	5.20
Spacing	m	4.60	5.20	4.60	5.20
Area of Influence	m^2	21.16	27.04	21.16	27.04
Sub Drill	m	0.70	1.00	0.70	1.00
Stemming	m	2.25	3.00	2.25	3.00
Powder Column	m	3.45	8.00	3.45	8.00
Loading Density	kg/m	31.84	31.84	31.84	31.84
Emulsion Powder/hole	kg	109.83	254.69	109.83	254.69
Bulk Emulsion SG	SG	1.28	1.28	1.28	1.28
Rock SG	SG	3.073	3.073	3.035	3.035
Tonnes per Hole	tonne	325	831	321	821
Powder Factor	kg/t	0.34	0.31	0.34	0.31
Powder Factor	kg/m^3	1.04	0.94	1.04	0.94
Instantaneous Penetration Rates	m/h	32.5	32.5	32.5	32.5
Hole Change Time	min/hole	5.43	5.65	5.43	5.65
Effective Penetration Rates	m/h	21.42	25.40	21.42	25.40

Table 16-12 – Presplitting, Grade Control and Void Delineation Drilling Parameters – MacLellan

Parameter	Unit	Pre-Production		Pre-Shear	Voids	Grade Control	
		Waste				Ore	Waste
		114mm (4 1/2") DTH	114mm (4 1/2") DTH	114mm (4 1/2") DTH	114mm (4 1/2") DTH	114mm (4 1/2") RC	114mm (4 1/2") RC
Bench Height	m	5.00	10.00	20.00	20.00	20.00	20.00
Hole Diameter	mm	114	114	114	114	114	114
Burden	m	3.00	3.20	1.00	2.50	5.00	10.00
Spacing	m	3.00	3.20	1.50	2.50	10.00	20.00
Incline Angle	deg.	90.00	90.00	90.00	90.00	60.00	60.00
Area of Influence	m^2	9.00	10.24	1.50	6.25	50.00	200.00
Sub Drill	m	0.70	1.00	0.50	1.00		
Stemming	m	2.25	3.00		6.00		
Powder Column	m	3.45	8.00		15.00		
Loading Density	kg/m	12.85	12.85		12.85		
Emulsion Blastex 89 x 400 (16.7 kg) Powder/hole	kg	44.35	102.83		192.82		
Bulk Emulsion SG	SG	1.26	1.26		1.26		
Rock SG	SG	2.947	2.947	2.947	2.95	2.975	2.947
Tonnes per Hole	tonne	133	302	88	368	2,576	10,209
Powder Factor	kg/t	0.33	0.34	–	0.52	–	–
Powder Factor	kg/m^3	0.00	0.00	–	0.01	–	–
Instantaneous Penetration Rates	m/h	52.4	52.4	58.3	52.40	52.4	52.4
Hole Change Time	min/hole	5.43	5.65	6.10	6.10	6.10	6.10
Effective Penetration Rates	m/h	28.6	36.2	45.2	41.80	41.4	41.4

Table 16-13 – Presplitting, Grade Control Drilling Parameters – Gordon

Parameter	Unit	Pre-Production		Pre-Shear	Grade Control	
		Waste			Ore	Waste
		114mm (4 1/2") DTH	114mm (4 1/2") DTH	114mm (4 1/2") DTH	114mm (4 1/2") RC	114mm (4 1/2") RC
Bench Height	m	5.00	10.00	20.00	20.00	20.00
Hole Diameter	mm	114	114	114	114	114
Burden	m	3.00	3.20	1.00	5.00	10.00
Spacing	m	3.00	3.20	1.50	10.00	20.00
Incline Angle	deg.	90.00	90.00	90.00	90.00	90.00
Area of Influence	m^2	9.00	10.24	1.50	50.00	200.00
Sub Drill	m	0.70	1.00	0.50		
Stemming	m	2.25	3.00			
Powder Column	m	3.45	8.00			
Loading Density	kg/m	12.85	12.85			
Emulsion Blastex 89 x 400 (16.7 kg) Powder/hole	kg	44.35	102.83			
Bulk Emulsion SG	SG	1.26	1.26			
Rock SG	SG	3.035	3.035	3.035	3.131	3.035
Tonnes per Hole	tonne	137	311	91	3,131	12,140
Powder Factor	kg/t	0.32	0.33	–	–	–
Powder Factor	kg/m^3	0	0	–	–	–
Instantaneous Penetration Rates	m/h	50.7	50.7	50.7	45.6	45.6
Hole Change Time	min/hole	$5.43	$5.65	$6.10	$6.10	$6.10
Effective Penetration Rates	m/h	28.1	35.3	40.5	37.0	37.0

16.14.3.3 Grade Control Drilling

The estimate for the grade control drilling and assay requirements and costs is based on sampling from RC drilling using a sample spacing of 5 m (X) x 10 m (Y) x 20 m (Z) in the ore zones and 10 m (X) x 20 m (Y) x 20 m (Z) in the waste zones for both the Gordon and MacLellan pits.

The use of both RC grade control and blast hole assays is planned for grade control. The grade control drilling is expected to also provide addition information for the void delineation. The instantaneous penetration rate for RC drilling is reduced by a factor of 10% compared to the DTH hammer drilling using the same diameter drill bit.

16.14.3.4 Voids Management Drilling

Voids management drilling consists of the voids ceiling collapse with vertical drilling and the stope perimeter delineation drilling with the application of inclined drill holes. The voids include the existing stopes as well as all development openings.

Inclined holes are planned for the delineation of the stope perimeter walls. For the purposes of the drilling requirements estimates, an allowance for drilling density of 15 m/m of stope perimeter length is planned.

16.14.4 Highway Haulage

The LLGP requires the haulage of ROM ore from Gordon to the process facility at MacLellan. The use of 30 t payload highway trucks is planned. The travel distance between the two mine sites is approximately 55 km.

Highway haulage of the ore from Gordon to the primary crusher at MacLellan is planned to be carried out by the owner. However, it is expected that the possibility to employ a contractor for this activity may be examined in subsequent studies. Additionally, the possibility of the application of larger capacity haul trucks, with Ministry of Transport approval, will be investigated to reduce the number of round trips required.

16.15 Primary and Support Equipment List

The planned equipment fleet is listed in Table 16-14 and Table 16-15 and is used as the basis for the operating and capital cost estimates for the LLGP.

During the initial road and site construction period, it is expected that some comparatively smaller size backhoe excavators and articulated and/or rigid body trucks will be utilized, sourced from the local rental market.

A schedule for the purchase of major equipment is presented in Table 16-6. Other than light vehicles it is not expected that any replacement equipment will be required during the life of the mine.

Table 16-14 – Primary Mine Equipment List by Application

Equipment	Example Model[1]	Capacity Class	Maximum Gordon Fleet[2]	Maximum MacLellan Fleet[2]	Units Purchased	Application
Hydraulic Shovel – 300 t	Cat 6030FS /PC3000	13.5 m^3	1	2	2	Production loading
Wheel Loader	Cat 993K	11.6 m^3	1	1	2	
Backhoe – 75t	Cat374D	4.6 m^3	3	2	4	Overburden, Gordon rehandle, pit bottom
Backhoe – 90t	Cat390	6 m^3	1	0	1	
Front End Loader	Cat 980	4.5 m^3	1	1	2	Rehandle, support
Haul Truck-Gordon	HD 1500	144 t	7	13	14	Production haulage per plan
Articulated Truck	Cat 740C	38 t	3	4	4	Overburden and pit bottom
Highway Trucks	HX620	31.5 t	23		23	Highway haulage
Blast Hole Drill	Atlas Copco PV235	60,000 lbf	2	3	4	Production drilling
Blast Hole Drill	FlexiROC D65	539 hp	1	1	2	Production/presplit drilling

Notes:

1. *Named models and manufacturers are only for reference, to identify sizes and costs, and are not intended as manufacturer and supplier endorsements or recommendations.*
2. *Some units will be relocated between the two sites.*

Table 16-15 – Support Mine Equipment List by Application

Equipment	Example Model[1]	Capacity Class	Max Fleet		Application
			Gordon	MacLellan	
Grader	Cat 16M	290 hp	1	1	Road/bench/dump/access road
Grader	Cat 14M	274 hp	1		Road/bench/dump/access road
Track Dozer	Cat D6	140 hp	1		Berm clean-up
Track Dozer	Cat D8T	310 hp	1	1	Initial roads, platforms,
					D10T2 replacement/support
Track Dozer	Cat D10T2	600 hp	1	1	Loading and dump areas
RT Dozer	834H	489 hp	1	1	Dumps and roads
Water Truck	Cat 775		1	1	Dual use as tow tractor, on-site
Small Water Truck	-	20,000 L	1	1	Gordon/MacLellan and cccess roads
Tow Truck/Flatbed	-	91 t	1		Highway use
Blast hole Stemmer	-	-	1	1	Gordon/MacLellan
Hydraulic Hammer	-	-	1	1	Attachments to Cat374/Cat390
Compactor	-	-	1		
Loader at Primary	Cat 980	4.5 m^3	1		Stockpile rehandle primary
Loader at transhipment area	Cat 374	4.6 m^3		1	
Backhoe	Cat 336F L	300 hp	1		
Bobcat	-	-	1		Attachments snow blower and auger
Crane	-	90 t		-	Equipment maintenance, field
					rentals as needed
Crane	-	120 t	1		
20t highway truck	-	-	2	2	General service fleet, rental as needed
Fuel/Lube truck	-	-	1	1	Field maintenance
Lift Truck/Flatbed	-	-	1	1	Field maintenance
Steam Cleaner on Trailer	-	-	1	1	Field maintenance
Forklift/Telehandler	-	-	1	1	Field maintenance
Ground Penetrating Radar	IDS Stream X	-		1	UG voids management, secondary breakage, geotechnical support
Excavator Mounted Long Reach Drill	-	-	1		
Welding Truck	-		1	1	Field maintenance

Equipment	Example Model[1]	Capacity Class	Max Fleet		Application
			Gordon	MacLellan	
Crew Bus	-	-			Supplied by contractor
Crew Change Vehicles	-	-	1	1	Personnel transport
Pickup trucks	-	-	12	8	Field transport
Tire Handler	-	-	1	1	Tire handling, stockpile rehandle, snow clean-up, tire maintenance, support tasks
Portable Lights	-	1,000 W	18	12	Pit, stockpile and waste dump lighting
Hot Box	-	-	1		
Pit Fire Truck	-	-	1	1	
Mine Rescue Truck	-	-	1	1	
Pit Ambulance	-	-	1	1	
Explosives Contractor	Not Specified				2/3 trucks and pickup trucks by/for explosives supplier use

Notes:

1. *Named models and manufacturers are only for reference, to identify sizes and costs, and are not intended as manufacturer and supplier endorsements or recommendations.*

Table 16-16 – Major Mine Equipment Purchase Schedule

Equipment	Example Model	Total	Y-2	Y-1	1	2	3	4	5	6	7	8	9	10	11
Hydraulic Shovel	Cat6030FS	2	-	-	2	-	-	-	-	-	-	-	-	-	-
Hydraulic Excavator	Cat 374D	2	1	1	-	-	-	-	-	-	-	-	-	-	-
Hydraulic Excavator	Cat 390D	1	1	-	-	-	-	-	-	-	-	-	-	-	-
FE Loader	Cat 993K	2	-	2	-	-	-	-	-	-	-	-	-	-	-
FE Loader	Cat 980	2	-	2	-	-	-	-	-	-	-	-	-	-	-
Haul Truck	HD1500	14	-	6	4	2	1	1	-	-	-	-	-	-	-
Articulated Truck	Cat 740B	4	4	-	-	-	-	-	-	-	-	-	-	-	-
Blasthole Drill	Pit Viper 235	4	-	2	2	-	-	-	-	-	-	-	-	-	-
Blasthole Drill	FlexiROC D65	2	1	1	-	-	-	-	-	-	-	-	-	-	-
Hydraulic Excavator	Cat 374D	1	-	-	1	-	-	-	-	-	-	-	-	-	-
Highway Truck	HX620	23	-	-	19	4	-	-	-	-	-	-	-	-	-
Grader	16M	2	1	1	-	-	-	-	-	-	-	-	-	-	-
Grader	14M	1	1	-	-	-	-	-	-	-	-	-	-	-	-
Track-Dozer	D6N	1	1	-	-	-	-	-	-	-	-	-	-	-	-
Track-Dozer	D8T	2	1	1	-	-	-	-	-	-	-	-	-	-	-
Track-Dozer	D10T2	2	1	1	-	-	-	-	-	-	-	-	-	-	-
RT-Dozer	834K	2	-	2	-	-	-	-	-	-	-	-	-	-	-
Backhoe	CAT374	1	-	-	1	-	-	-	-	-	-	-	-	-	-
Backhoe	CAT336FL	1	1	-	-	-	-	-	-	-	-	-	-	-	-
Water Truck	Cat 775	2	1	1	-	-	-	-	-	-	-	-	-	-	-
Pickup	-	70	10	15	15	0	0	15	-	-	10	-	-	5	-

17 RECOVERY METHODS

17.1 Proposed Process Flow Sheet

The unit operations used to achieve plant throughput and metallurgical performance are well-proven in the gold/silver processing industry. The flow sheet incorporates the following major process operations:

- Two-stage crushing and stockpile;
- Semi-autogenous grinding (SAG);
- Ball mill grinding and classification;
- Leaching and carbon-in-pulp (CIP) adsorption;
- Desorption and gold room;
- Tailings detoxification and disposal;
- Fresh and reclaim water supply; and
- Reagent preparation and distribution.

The overall process plant flow sheet is shown in Figure 17-1.

The solids throughputs for the following parts of the plant are:

- Crushing plant is 7,000 t/d or 490 t/h at 65% availability; and
- Process plant is 7,000 t/d or 317 t/h at 92% availability.

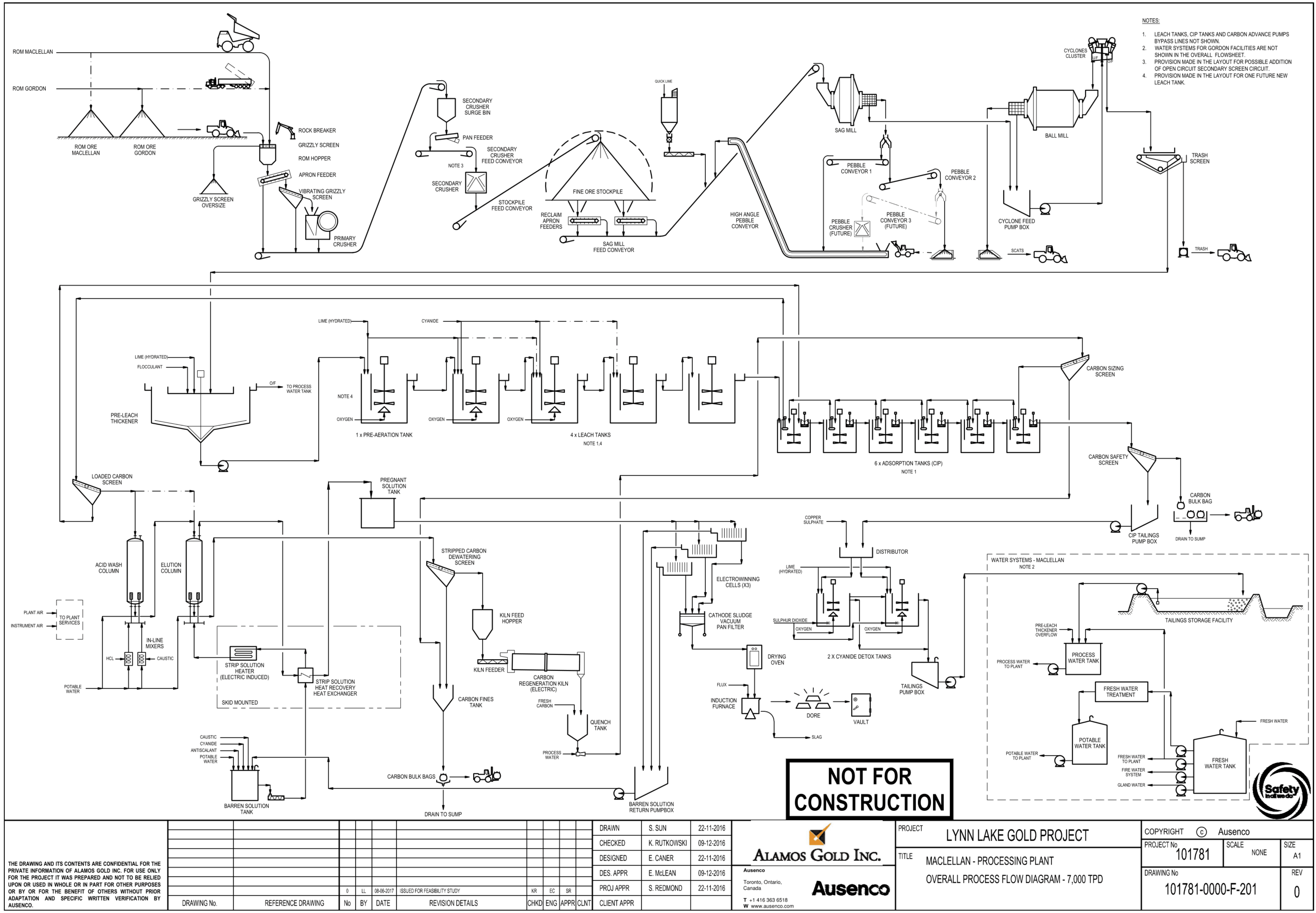


Figure 17-1: Overall Process Flow Diagram

Source: Ausenco (2017)

17.2 Process Design Criteria

The process design criteria summary is provided in Table 17-1.

Table 17-1 – Process Design Criteria Summary

Description	Units	Value
Ore Throughput	Mt/y	2.56
Plant Throughput, Average	t/d	7,000
Crushing Plant Availability	%	65
Crushing Plant Design Throughput	t/h	490
Treatment Plant Availability	%	92
Treatment Plant Design Throughput	t/h	317
Gordon Ore Gold Grade, LOM Average	g/t	2.42
MacLellan Ore Gold Grade, LOM Average	g/t	1.63
MacLellan Ore Silver Grade, LOM Average	g/t	4.44
Gordon, Overall Recovery Gold	%	92.9
Gordon, Overall Recovery Silver	%	n/a
MacLellan, Overall Recovery Gold	%	91.4
MacLellan, Overall Recovery Silver	%	49.0
Maximum Production, Gold, Year 5	oz Au	200,000
Maximum Production, Silver, Year 6	oz (Ag)	188,000
Crushing (Two Stage):		
Primary Crusher	Type	Single Toggle Jaw Crusher
Secondary Crusher	Type	Cone Crusher
Fine Ore Stockpile	h	16
Grinding:		
Primary Grinding (with pebble return to SAG)	Type	SAG Mill
Secondary Grinding (closed circuit with cyclones)	Type	Ball Mill
Product Particle Size, at 80% passing (P_{80})	µm	75
Pre-leach Thickener:		
Net Solids Loading	$t/m^2/h$	0.42
Underflow Pulp Density	% w/w	55
Pre-aeration, Leaching and Adsorption:		
Pre-aeration Residence Time	h	6
Gordon Oxygen Uptake Pre-aeration	mg/L	8.20
MacLellan Oxygen Uptake Pre-aeration	mg/L	6.40
Leaching Residence Time	h	23

Description	Units	Value
Gordon Oxygen Uptake Leaching	mg/L	1.40
MacLellan Oxygen Uptake Pre-aeration	mg/L	0.80
Total Adsorption Residence Time	h	7
Desorption and Regeneration:		
Desorption Process	Type	Pressure Zadra
Carbon Batch Size	t	6.0
Acid Wash and Elution Cycles per Day	#	1
Acid Wash and Elution Operating per Week	d/wk	7
Cyanide Destruction:		
Method		$O_2/SO_2/Cu^{2+}$
Residence Time	h	2
CN_{WAD} Not to Exceed	mg/L	10

17.3 Process Plant Description

17.3.1 Crushing and Stockpile

ROM ore from both MacLellan and Gordon open pits is discharged to the ROM ore hopper by front end loaders (FEL) or, haul trucks or highway trucks. The ROM ore hopper incorporates a horizontal static grizzly with 800 mm x 800 mm aperture to screen out lump oversize. The static grizzly oversize is reclaimed by a front end loader and stockpiled for rock breaking. The static grizzly undersize discharges into the ROM ore hopper. Ore is reclaimed at a controlled rate by an apron feeder and discharged onto a vibrating grizzly screen.

The oversize from the vibrating grizzly enters the single toggle 490 t/h (design) Metso C130 51” x 40” primary jaw crusher powered by a 160 kW motor. The jaw crusher close size setting (CSS) is 130 mm. The vibrating grizzly undersize, jaw crusher product and apron feeder fines combine and discharge onto the primary crusher transfer conveyor. A tramp magnet removes steel trash from the primary crushed ore as it transfers by conveyor to the secondary crusher.

The secondary crusher is a 490 t/h (design) cone crusher with CSS of 35 mm. The crusher is powered by a 450 kW motor. It is located inside the main process plant building, in the grinding area. A surge bin with ten minutes retention time, located above the crusher, ensures choke feed conditions. A pan feeder extracts ore from the surge bin to feed the secondary crusher via a conveyor. The fine ore product of P_{80} of 40 mm is conveyed by a stockpile feed conveyor to the fine ore stockpile (FOS). A weightometer measures the fine ore rate produced from the crushing circuit.

The FOS has a live volume of 5,300 t, providing 16 hours of mill feed. Two variable speed reclaim apron feeders provide two live draw-down pockets. The feeders provide duty/duty mill feed service whereby either one can handle nominal mill feed demand if the other unit is off line.

17.3.2 Grinding and classification

The grinding circuit consists of one single pinion 7.3 m x 3.6 m SAG mill followed by one single pinion 5.8 m x 8.7 m ball mill. The ball mill operates in closed circuit with a hydrocyclone cluster. The final product from the grinding circuit (cyclone overflow) has a target P_{80} 75 µm. The SAG mill is equipped with a single 3.1 MW low speed synchronous motor with variable frequency drive (VFD) to operate between 61% and 79% of critical speed. The ball mill is equipped with a single 5.1 MW low speed synchronous motor to operate at 75% of critical speed.

The reclaimed fine ore is conveyed by a covered SAG mill feed conveyor from the two FOS reclaim feeders under the stockpile to the SAG mill feed chute. The recycled pebbles from the SAG mill trammel oversize chute are re-introduced to the SAG mill via a high angle pebble conveyor that discharges onto the SAG mill feed conveyor. A weightometer, after the stockpile reclaim, measures the amount of fresh mill feed ore entering the SAG mill and a second weightometer to the SAG mill, after the recycled pebbles and lime addition points onto the conveyor, measures the total feed. The pebble recirculating load is a maximum of 15% by weight. Both, or either of the variable speed duty/duty reclaim feeders can supply the mill demand. Quick lime is metered to the SAG mill feed conveyor using a screw feeder. For both mills, process water is added at a controlled rate into the feed chutes to achieve a nominal pulp density of 71% solids.

The SAG mill is fitted with 20 mm discharge grates to allow slurry to pass through the mill and prevent pebble build-up in the mill. The SAG mill discharge product is screened on the SAG mill trommel with 10 mm aperture. Oversize from the SAG mill trommel is conveyed back to the SAG mill feed conveyor using a series of pebble conveyors. A provision is made in the layout to accommodate a future pebble crusher. SAG mill trommel undersize gravitates to the cyclone feed pumpbox, where it is combined with the ball mill discharge slurry from the ball mill trommel undersize.

Ball mill slurry discharge overflows onto a rubber lined trommel screen with trommel oversize discharging to a bunker for regular collection and disposal by a skid steer loader. The trommel undersize gravitates to the cyclone feed hopper where the slurry is diluted with process water and pumped with a single duty cyclone feed pump to the cyclone cluster. A density meter monitors and controls the amount of process water required to produce a target density to the cyclones. The cluster consists of eight cyclones: six operating and two standby.

SAG and ball mill grinding media (steel balls) supplied in trucks are unloaded into three ball storage compartments, each holding a specific diameter steel ball. A ball loading magnet (moving through an overhead ball bin crane) attracts and transports a collection of balls from the compartments to individual storage bins. From each storage bin the balls are discharged into a dedicated 1 t ball kibble, which in turn discharges to the mills to maintain the desired ball loads and power draw. The ball kibble is lifted by a 50 t bridge-crane and emptied into the mill ball discharge indexer.

The cyclones produce a ground overflow product of P_{80} 75 µm which is sampled and then gravitates to the linear trash screen. Oversize debris is removed and falls to a trash bin at ground level. The trash screen underflow gravitates to the pre-leach thickener feed box outside the west wall of the building.

Spillage in the grinding area is contained within a full concrete slab and bunded area. Three grinding area sump pumps are provided at low points in the bunded area to reclaim spillage and return this to the cyclone feed hopper. Two cameras monitor the sumps and cyclone pumpbox levels for operator intervention.

17.3.3 Leach and Adsorption

Trash screen undersize is thickened from 33% to 55% solids in a 31 m diameter high rate pre-leach thickener in preparation for downstream pre-aeration, leaching, and carbon-in-pulp (CIP). Hydrated lime slurry is added to the ground ore slurry in the feed box to raise the pH to 10.5. Flocculant is metered to the thickener feedwell.

Thickener overflow gravitates to the process water tank and is recycled for plant use while the underflow slurry is pumped with variable speed duty/standby pumps to the pre-aeration tank for slurry conditioning prior to leaching. The thickener is located outdoors sharing a common secondary containment area with the process water tank. The thickener bund with an overflow weir to the pre-aeration and leach tanks bund will contain the total slurry volume from the thickener.

Thickener underflow is pumped to the pre-aeration tank. Hydrated lime slurry can be added to the tank to raise the slurry pH if required. Oxygen is added to the tank via a distribution hat. Pre-aeration and leach tanks are equipped with a dual impeller mechanical agitator to ensure uniform mixing of slurry and oxygen. The pre-aeration tank overflows to the first leach tank or can be by-passed to the second leach tank. The pre-aeration tank is located in a dedicated bunded area.

The leach tank circuit consists of four tanks in series. Total live volume is 9200 m^3 which allows 23 hour residence time for a 317 t/h solids feed rate at the operating solids pulp density. Space has been set aside in the layout for an additional future leach tank, if required. The tanks are located outdoors to the northeast of the pre-leach thickener in a dedicated bunded area. Any tank can be removed from service using a bypass line. Slurry from the last tank in the leach train flows by gravity to the first CIP tank in the adsorption circuit or can be by-passed to the second CIP tank.

The pH is measured with probes installed in the leach tanks and hydrated lime slurry can be added as required. An online Leach Cyanide Analyser measures the free cyanide concentration in the first two leach tanks.

The CIP circuit consists of six adsorption tanks in series, each with a live volume of 500 m^3. The tanks are located inside the process building in a dedicated bunded area serviced by a CIP area sump pump. Pulp flows continuously from the first to the last adsorption tank, while carbon is pumped counter–currently, in pre-set intervals, from the last to the first tank. Loaded carbon is recovered from the first adsorption tank; stripped and regenerated carbon enters the adsorption circuit at the last tank.

The CIP tanks are arranged with the tops of all tanks at the same elevation. Mechanically swept, inter-tank pumping screens are used to move slurry to the next downstream tank, while preventing activated carbon from flowing with the slurry to the next CIP tank. Each tank is equipped with a dual impeller mechanical agitator to ensure uniform mixing of slurry and carbon. Any tank can be removed from service using a bypass line. Target carbon concentration in each of the tanks is 25 g/L.

A CIP tank top travelling 10 t gantry crane removes the screens to a dedicated bay/stand area for maintenance, routine cleaning and agitator maintenance. A spare inter-tank screen is available to allow rapid screen changeover.

Following elution (desorption) of the loaded carbon and thermal regeneration, the barren carbon is screened over the 1.2 mm square aperture barren carbon sizing screen and reports to the last CIP tank in the adsorption train. Fine carbon is recovered to the carbon fines tank.

The secondary containment for the CIP circuit provides for a minimum of one tank volume.

At the east side of the CIP area, the tailings slurry from the final CIP tank gravitates to the vibrating carbon safety screen to recover any carbon in the event of damage, wear or other issues with the CIP inter-tank screen. Carbon recovered on the 0.84 mm square screen is collected in a bulk bag that can be manually transferred for re-use. Tailings discharging from the carbon safety screen undersize gravitates to the CIP tailings pumpbox. From here, the tailings is pumped to the cyanide detoxification feed box where the flow is equally split to two parallel cyanide detoxification tanks located to the northeast of the CIP circuit.

A HCN Gas Detector detects any potential hydrogen cyanide gas in this area.

17.3.4 Cyanide Detoxification

The cyanide detoxification circuit reduces weak acid dissociable cyanide (CN_{WAD}) to a target value of 5 ppm and a not-to-exceed value of 10 mg/L for disposal. The detoxification process uses the conventional (air) $O_2/SO_2/Cu^{2+}$ process.

Slurry from the carbon safety screen undersize is pumped to two 500 m^3 cyanide detoxification tanks in parallel. The slurry residence time in the detoxification tanks is 2 h. For operational flexibility, these can operate in series if needed, or if one reactor is offline, there is 60 minutes residence time available. Increasing the sulphur dioxide (SO_2) addition rate offsets the temporary reduction in detoxification residence time.

Oxygen for the detoxification reaction is supplied to each tank via a dispersion cone mounted to the bottom of each. The tanks utilise high shear agitators to enhance oxygen dissolution in the slurry to meet the oxygen demand of the cyanide destruction process. SO_2 gas is injected into the tanks using dedicated injection lances. SO_2 provides the oxidizing agent required for the process. Copper sulphate solution is dosed into both tanks, providing the catalyst for the cyanide detoxification process. Acid generated as a by-product is neutralized with lime slurry, added from a ring main to each tank.

Two stage sampling is used to take a representative tailings sample after the slurry has been detoxified and prior to entering the tailings hopper.

A CN_{WAD} analyser automatically monitors slurry cyanide concentration. The detoxified slurry stream gravitates to the tailings pump box from where it is pumped through a single pipeline to the TMF by a variable speed tailings pump (one duty, one standby). The tailings slurry is then discharged at outlet points around the periphery of the facility. Pipe runs are designed to be self-draining to avoid dead legs. The HDPE tailings pipeline runs for approximately 3 km before reaching the TMF.

17.3.5 Desorption

Loaded carbon is processed in the desorption circuit comprising the acid wash column, elution column and regeneration kiln. The pregnant solution from elution is pumped through electrowinning cells. The barren eluate from the electrowinning cells is returned to the barren solution tank in the elution circuit and recycled through the elution column. Gold/silver sludge from the cathode wash and cell floor clean-up is filtered, dried and then smelted to produce doré for sale.

Loaded carbon from CIP tanks is recovered on the loaded carbon screen (oversize) and directed to a butyl rubber-lined, mild steel acid wash column with 6 t carbon capacity. The system also has the option to by-pass the acid wash column via a diverter gate and to discharge the loaded carbon from the screen into the elution column.

Carbon is washed with dilute hydrochloric acid (HCl). Concentrated acid (HCl at 32% w/w) is diluted with potable water in an in-line mixer to provide the required acid wash solution concentration of 3% w/v HCl.

Following acid solution contact, the carbon is rinsed with potable water and then with in-line diluted sodium hydroxide (NaOH) to remove residual acid and to neutralise the pH of the liquor in the carbon column. The neutralized acid solution is discharged to the tailings pumpbox. Washed carbon is then transferred to the elution column.

A separate, acid-proofed concrete bund is provided in the acid wash column area to ensure that all spillage is captured and kept separate from other process streams. A dedicated, acid-resistant sump pump is located at the low point of this bund. Spillage from this area is pumped using the acid area sump pump to the tailings pump box. Transfer and fill operations of the acid wash column are controlled manually. All other aspects of the acid wash and the pumping sequence are automated.

A pressure Zadra elution circuit has been selected for stripping of gold and silver from loaded carbon.

In the Zadra process a 1% w/v sodium hydroxide and 0.1% w/v sodium cyanide solution at 135°C is used to desorb gold and silver from the carbon. Gold and silver are recovered from the pregnant strip solution by electrowinning. The gold/silver depleted solution is then re-heated and recycled to elution.

The Zadra elution system comprises an elution column, barren solution tank, strip solution pump, two-speed positive displacement water pumps and a strip solution heater package. This equipment operates in a closed loop with three electrowinning cells located inside the gold room.

17.3.6 Regeneration

After completion of the elution process, stripped carbon is transferred to the stripped carbon dewatering screen. The screened carbon (screen oversize) is fed into the carbon regeneration kiln feed hopper. The screen undersize gravitates to the carbon fines tank. The carbon is then fed to the carbon regeneration kiln. This electrically powered kiln is a horizontal, rotary unit designed to regenerate 100% of the stripped carbon and operates nominally 20 hours per day.

Pre-drying of carbon occurs when warm kiln off-gas is drawn through the carbon pre-dryer. Non-process gases from the regeneration kiln are used to dry the feed carbon in the carbon pre-dryer hopper and are extracted to the atmosphere via the carbon regeneration kiln exhaust fan. The kiln operates at temperatures of 650 – 700°C. Carbon is heated to 700°C and held at this temperature for 15 min to allow reactivation to occur. Regenerated carbon discharges from the kiln to a quench tank to cool down from where it is transferred via an eductor to the carbon sizing screen.

The sizing screen oversize returns carbon to the last CIP tank in the train, while the quench water and fine carbon from the undersize are combined with the stripped carbon dewatering screen undersize in the carbon fines tank. Water overflows from this tank to the carbon fines area sump pump. When a sufficient amount of carbon has been accumulated in the tank, it is discharged by gravity to a nearby bag stand for disposal.

17.3.7 Gold Room

The following operations are carried out in the secure gold room area:

- Electrowinning; and
- Smelting.

Gold and silver recovery from pregnant solution is achieved by electrowinning. Three electrowinning cells are located on the mezzanine within the gold room. Each electrowinning cell houses 23 cathodes and is built in stainless steel with a polypropylene liner. The 0-2000 Amp rectifier associated with each electrowinning cell backs onto the gold room wall allowing easy access for operations and maintenance outside the secure area of the gold room. Fumes from the electrowinning cells are vented to atmosphere via a demister – to collect any mist, and a stack.

Stainless steel framed cathodes are lifted using the electrowinning area hoist to the upper floor of the gold room. Gold and silver plated on cathodes are recovered by high-pressure water spray to a sludge on the cell floor together with any cathode sludge which may have fallen to the cell floor during operation.

Cathode wash material and any accumulated cell sludge are drained from the electrowinning cell to a vacuum pan filter.

Excess water overflows to the floor sump whilst the cell sludge is filtered. The filter cake (gold/silver sludge) is manually loaded from the pan filter into trays on the electrowinning sludge trolley. The trays slide into the gold room drying oven, which heats the sludge to about 100°C to dry this material before smelting.

The cooled sludge is combined with fluxes (silica, nitre, borax and sodium carbonate) in the flux mixer. The fluxes are weighed according to a pre-determined recipe and manually added to the flux mixer. The sludge-flux mix is smelted in an electric induction furnace. The fluxes react with base metal oxides to form a low viscosity, free flowing slag whilst gold and silver remain as molten metals.

The melt is poured from the furnace into a cascade pouring table of doré moulds. The barren slag is separated from the precious metals and collected in slag trays at the bottom of the cascade tables. The doré in the moulds solidifies and is quenched in water, cleaned to remove slag, weighed, stamped for identification, sampled for analysis and stored in a safe while awaiting dispatch to a commercial refinery.

17.3.8 Reagents and Consumables

Reagents will be mixed in a separate contained area to the east end of the process plant and within the process plant building. Bunded areas control any spillage. Tank storage capacity has been generally sized based on reagent consumption rates to supply the process without any interruption, or according to available delivery volumes.

A separate reagent storage building south of the reagent mixing area has been sited to provide dry reagent storage. Reagents are transported by forklift to the light-vehicle access bay at the reagent make-up area.

Reagent consumptions are based on project specific test work or industry operating practice. Reagent costing is supported by recent vendor quotations. A summary of the estimated reagents and steel media rates are shown in Table 17-2.

Table 17-2 – Lynn Lake Estimated Reagent Consumption

Reagents	Form	Unit	Specific Consumption
Activated Carbon	Solid, granular, coconut	g/t feed	40
Sodium Cyanide	Solid, briquettes	kg/t feed	0.36
Quicklime	Solid, pebbled or granulated	kg/t feed	0.17
Sulphur	Solid, prills	kg/t feed	0.46
Copper Sulphate	Crystalline granules solid	kg/t feed	0.07
Hydrated Lime	Solid (fine powder)	kg/t feed	1.3
Antiscalant	Polymeric	L/t feed	0.03
Flocculant	Granular powder	g/t feed	25
Sodium Hydroxide	Liquid (50% w/w), solution	kg/t feed	0.07
Hydrochloric Acid	Liquid (32% w/w), solution	L/strip	700
Sulphamic Acid	Solid, crystals or powder	g/t feed	5.0
Borax	Powder	kg/smelt	24
Silica	Powder	kg/smelt	12
Sodium Nitrate (Nitre)	Powder	kg/smelt	2.0
Sodium Carbonate	Powder	kg/smelt	2.0
Grinding Media – SAG Mill	100 mm balls	kg/t feed	0.5
Grinding Media – Ball Mill	50 mm balls	kg/t feed	1.0

17.3.9 Services

17.3.9.1 MacLellan Site

Process Water

Two (one duty, one standby) horizontal centrifugal slurry pumps supply process water to the various consumers throughout the plant site, but predominantly to the grinding circuit. Both pumps are fed from a stand pipe which is connected to both the process water tank and the pre-leach thickener overflow. This standpipe reduces the chances of solids carryover in the thickener overflow from settling in the storage tank. The process water tank is constructed from mild steel and has a live volume ensuring 60 minutes of residence time. The reticulation pipework is heat-traced and thermally-insulated where required. A dedicated metering pump provides antiscalant dosing into the process water tank to prevent scaling downstream.

In the case where the tailings return water flow is temporarily reduced, the fresh water pumps can supplement the process demand.

Plant site runoff water will be collected in the storm water pond and will be pumped into the return water pipeline and directed to the process water tank as makeup water or directly to the TMF. The storm water pond is 3000 m^3 live capacity, and sized based on a 1 in 50-year event of 128 mm of precipitation in 24 h.

Gland Water

Two (one duty, one standby) positive displacement pumps are fed from the fresh water tank. The pumps are located within the water services pumphouse and supply the various slurry pumps throughout the plant site.

Oxygen

Oxygen will be supplied to the pre-leaching tank, the first two tanks in the leaching circuit and the detox area by a vendor-supplied vacuum pressure swing adsorption (VPSA) oxygen plant. There are two off-takes for oxygen supply; one to the leach area and the second to the detoxification circuit. The oxygen is reticulated at a pressure of 480 kPag.

17.3.10 Tailings Disposal

Tailings reclaim water and run-off water from precipitation collected in the TMF are pumped to the process water tank located next to the pre-leach thickener. The sections of the tailings and decant pipelines running between the plant site and the TMF run in HDPE lined trenches. Where the pipelines run along the dam crest wall, single pipes, located on a bench running along the upstream slope of the dam towards the inside of the wall, are used such that any leakage will drain into the dam.

17.4 Process Control Philosophy

17.4.1 General

The process control philosophy for the LLGP is similar to many mineral processing operations.

Field instrumentation provides input to programmable logic controllers (PLC) which are monitored by Process Control System (PCS). The PCS system is configured to provide outputs to alarms, to control the function of selected process equipment and to provide advisory comment to the plant operators. In addition, logging and trending functions are available to assist in analysis of the operating plant data.

The plant is provided with a central control room from which the status of major electrical and mechanical equipment can be monitored and major regulatory control loops can be monitored and adjusted via the operator control station (OCS).

Critical safety and equipment protection interlocks are hardwired. Control of process variables is via the OCS or discrete controllers in the field.

All drives can be stopped in the field at the local control station (LCS) located adjacent to the drive. The LCS will also have a remote–maintenance selector switch. Either selection allows the operator to stop or start the drive in the field with the only difference in maintenance mode, where the process interlocks are over-ridden. The safety interlocks are still active in this mode. This provides flexibility for testing duties.

All electrical drives, which can be started via an OCS, have three faceplate operational modes.

During normal operation, operators will have a choice of cascade, auto or manual mode. Cascade mode allows the drive control variable to be set by another controller. This mode allows cascading control loops to function. Auto mode allows for automated sequence control of the drive where control setpoints can be entered by the operator. Manual mode does not allow drives to be started in a sequence, but can still be started via the OCS. Process and safety interlocks are active for all modes.

PLCs are utilized to accept status signals from the electrical switchgear for monitoring drive status conditions on an OCS.

17.4.2 Control Philosophy

The general control strategy for the LLGP is:

- Control by the PCS for all areas where equipment requires remote start and stop, sequencing, and process interlocking;
- Motor controls for starting and stopping drives at LCS's using the PCS or hard wired, depending on the drive classification. All drives can be stopped from the local control station at all times. Local and remote starting is dependent on the drive type and control mode;
- Vendor PLCs for areas or items that are supplied as complete vendor packages. The Vendor PLC communicates alarms and status information to the PCS for recording and monitoring;
- Monitoring of operations on the PCS and recording of selected information for data logging and/or trending;
- Control loops in the PCS, except where the vendor's PLC directly controls the vendor package;
- Trip and alarm inputs to the PCS are fail-safe in operation, i.e., the signal reverts to the de-energized state when a fault occurs;
- Hard-wired safety interlocks for personnel safety;
- Software interlocks for process safety and equipment protection start and stop sequences for certain groups of equipment;
- Automation of critical process components and a high level of monitoring to minimise the possibility of human error;
- Uniform architecture, hardware and software configuration throughout all non-vendor controlled equipment;
- A main plant control room with two operator control stations (OCSs);
- An additional control room at the primary crushing station with a single OCS;
- An OCS in the plant metallurgist's office with process viewing and data trending capabilities only; and
- Closed-circuit television (CCTV) monitoring of key areas or transfer points.

17.5 Process Plant Layout

The LLGP process plant has been designed with an approximate dimensions of 59 m (wide) x 88 m (long), and provides indoor space for grinding, adsorption, desorption and carbon regeneration, refining, and reagents areas. It also houses offices, electrical rooms and a control room. The pre-leach thickener, pre-aeration tank, cyanide leaching tanks, cyanide detoxification tanks, and water services tanks are located outdoors in close proximity to the process plant. The reagent storage, warehouse, assay laboratory structures are located adjacent to the southeast corner of the process plant. Propane storage is located north of the pre-leach thickener. The main electrical substation is located to the west of the plant area. The oxygen plant and sulphur dioxide plant are adjacent to the leach tank area and the water services pumps are nestled inside the main process plant.

Figure 17-2 shows the Lynn Lake (MacLellan) process plant general arrangement. Figure 17-3 to Figure 17-9 show the process plant building general arrangement plans.

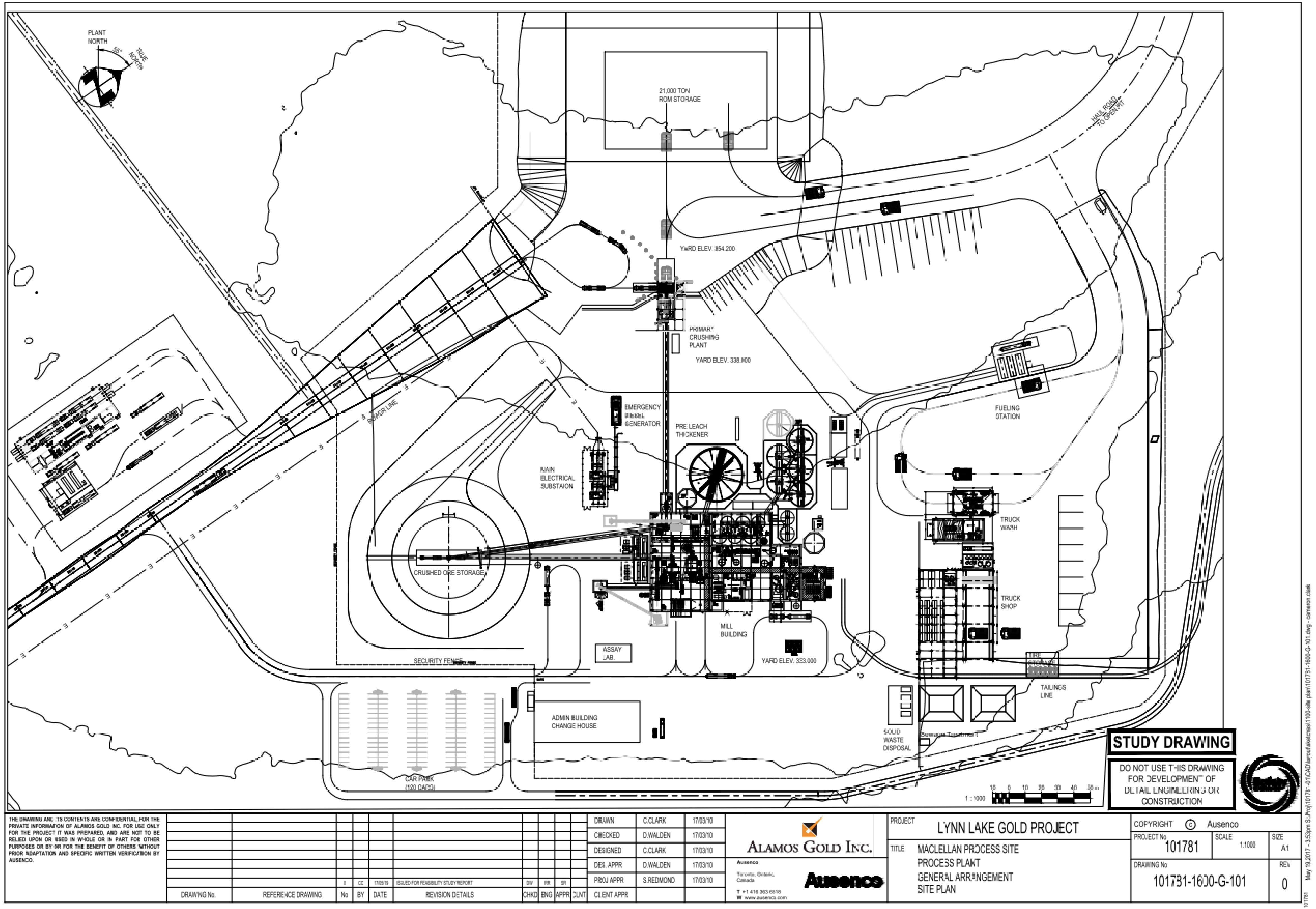
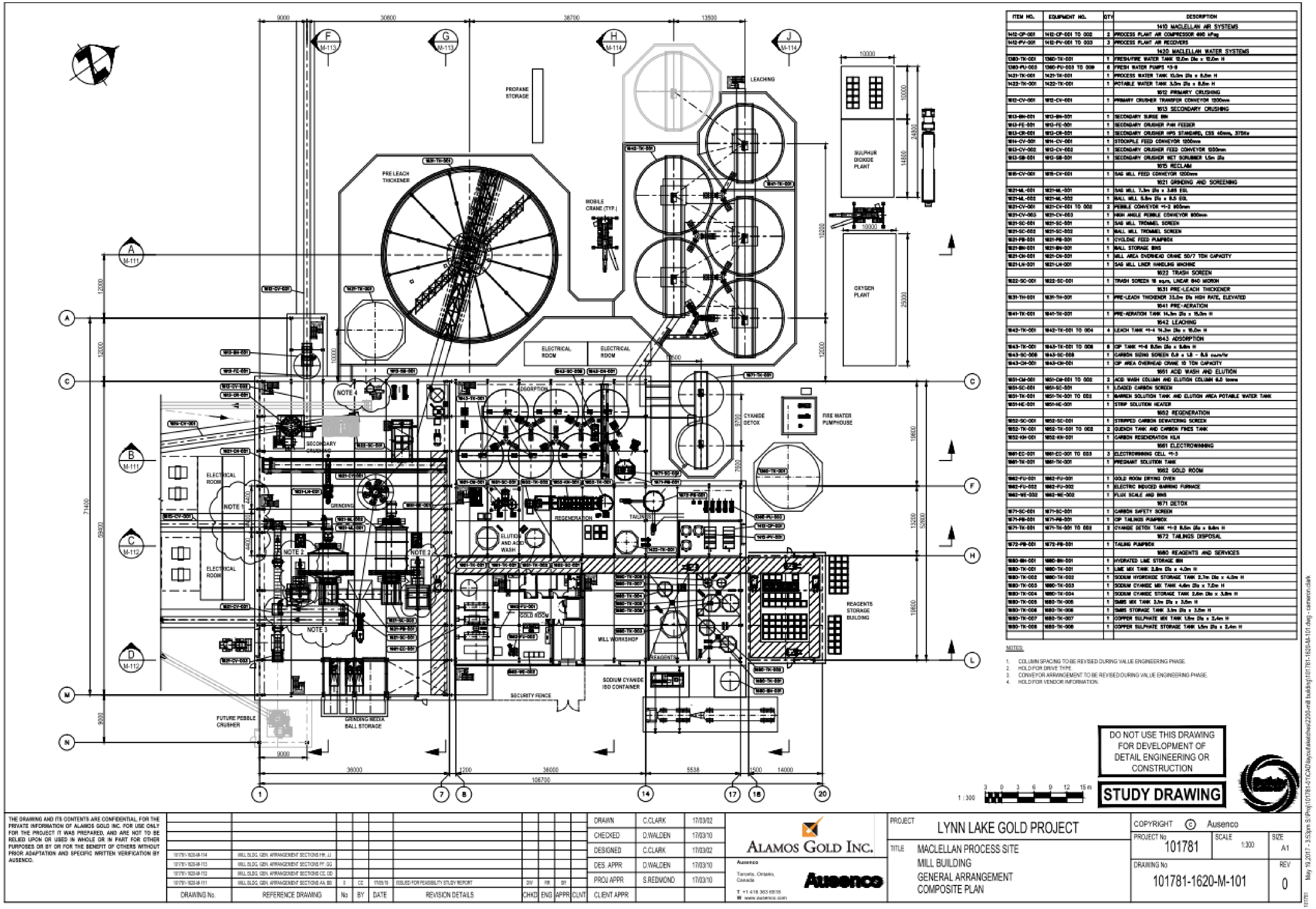


Figure 17-2: MacLellan Process Plant General Arrangement

Source: Ausenco (2017)



ITEM NO.	EQUIPMENT NO.	QTY	DESCRIPTION
			1410 MACLELLAN AIR SYSTEMS
1412-CP-001	1412-CP-001 TO 002	2	PROCESS PLANT AIR COMPRESSOR 490 kPag
1412-PV-001	1412-PV-001 TO 003	3	PROCESS PLANT AIR RECEIVERS
			1420 MACLELLAN WATER SYSTEMS
1360-TK-001	1360-TK-001	1	FRESH/FIRE WATER TANK 13.0m Dia x 13.0m H
1360-PU-003	1360-PU-003 TO 009	6	FRESH WATER PUMPS #3-9
1421-TK-001	1421-TK-001	1	PROCESS WATER TANK 10.0m Dia x 8.0m H
1422-TK-001	1422-TK-001	1	POTABLE WATER TANK 3.0m Dia x 8.0m H
			1612 PRIMARY CRUSHING
1612-CV-001	1612-CV-001	1	PRIMARY CRUSHER TRANSFER CONVEYOR 1200mm
			1613 SECONDARY CRUSHING
1613-BN-001	1613-BN-001	1	SECONDARY SURGE BIN
1613-FE-001	1613-FE-001	1	SECONDARY CRUSHER PAN FEEDER
1613-CR-001	1613-CR-001	1	SECONDARY CRUSHER HP5 STANDARD, CSS 40mm, 375kw
1614-CV-001	1614-CV-001	1	STOCKPILE FEED CONVEYOR 1200mm
1613-CV-002	1613-CV-002	1	SECONDARY CRUSHER FEED CONVEYOR 1200mm
1613-SB-001	1613-SB-001	1	SECONDARY CRUSHER WET SCRUBBER 1.5m Dia
			1615 RECLAIM
1615-CV-001	1615-CV-001	1	SAG MILL FEED CONVEYOR 1200mm
			1621 GRINDING AND SCREENING
1621-ML-001	1621-ML-001	1	SAG MILL 7.3m Dia x 3.65 EGL
1621-ML-002	1621-ML-002	1	BALL MILL 5.8m Dia x 8.5 EGL
1621-CV-001	1621-CV-001 TO 002	2	PEBBLE CONVEYOR #1-2 900mm
1621-CV-003	1621-CV-003	1	HIGH ANGLE PEBBLE CONVEYOR 900mm
1621-SC-001	1621-SC-001	1	SAG MILL TROMMEL SCREEN
1621-SC-002	1621-SC-002	1	BALL MILL TROMMEL SCREEN
1621-PB-001	1621-PB-001	1	CYCLONE FEED PUMPBOX
1621-BN-001	1621-BN-001	1	BALL STORAGE BINS
1621-CN-001	1621-CN-001	1	MILL AREA OVERHEAD CRANE 50/7 TON CAPACITY
1621-LH-001	1621-LH-001	1	SAG MILL LINER HANDLING MACHINE
			1622 TRASH SCREEN
1622-SC-001	1622-SC-001	1	TRASH SCREEN 18 sq.m, LINEAR 840 MICRON
			1631 PRE-LEACH THICKENER
1631-TH-001	1631-TH-001	1	PRE-LEACH THICKENER 33.0m Dia HIGH RATE, ELEVATED
			1641 PRE-AERATION
1641-TK-001	1641-TK-001	1	PRE-AERATION TANK 14.3m Dia x 15.0m H
			1642 LEACHING
1642-TK-001	1642-TK-001 TO 004	4	LEACH TANK #1-4 14.3m Dia x 15.0m H
			1643 ADSORPTION
1643-TK-001	1643-TK-001 TO 008	8	CIP TANK #1-8 8.0m Dia x 8.8m H
1643-SC-008	1643-SC-008	1	CARBON SIZING SCREEN 0.9 x 1.8 - 8.8 cu.m/hr
1643-CN-001	1643-CN-001	1	CIP AREA OVERHEAD CRANE 10 TON CAPACITY
			1651 ACID WASH AND ELUTION
1651-CM-001	1651-CM-001 TO 002	2	ACID WASH COLUMN AND ELUTION COLUMN 6.0 tonne
1651-SC-001	1651-SC-001	1	LOADED CARBON SCREEN
1651-TK-001	1651-TK-001 TO 002	1	BARREN SOLUTION TANK AND ELUTION AREA POTABLE WATER TANK
1651-HE-001	1651-HE-001	1	STRIP SOLUTION HEATER
			1652 REGENERATION
1652-SC-001	1652-SC-001	1	STRIPPED CARBON DEWATERING SCREEN
1652-TK-001	1652-TK-001 TO 002	2	QUENCH TANK AND CARBON FINES TANK
1652-KN-001	1652-KN-001	1	CARBON REGENERATION KILN
			1661 ELECTROWINNING
1661-EC-001	1661-EC-001 TO 003	3	ELECTROWINNING CELL #1-3
1661-TK-001	1661-TK-001	1	PREGNANT SOLUTION TANK
			1662 GOLD ROOM
1662-FU-001	1662-FU-001	1	GOLD ROOM DRYING OVEN
1662-FU-002	1662-FU-002	1	ELECTRIC INDUCED BARRING FURNACE
1662-WE-002	1662-WE-002	1	FLUX SCALE AND BINS
			1671 DETOX
1671-SC-001	1671-SC-001	1	CARBON SAFETY SCREEN
1671-PB-001	1671-PB-001	1	CIP TAILINGS PUMPBOX
1671-TK-001	1671-TK-001 TO 002	2	CYANIDE DETOX TANK #1-2 8.5m Dia x 9.4m H
			1672 TAILINGS DISPOSAL
1672-PB-001	1672-PB-001	1	TAILING PUMPBOX
			1680 REAGENTS AND SERVICES
1680-BN-001	1680-BN-001	1	HYDRATED LIME STORAGE BIN
1680-TK-001	1680-TK-001	1	LIME MIX TANK 2.8m Dia x 4.0m H
1680-TK-002	1680-TK-002	1	SODIUM HYDROXIDE STORAGE TANK 3.7m Dia x 4.0m H
1680-TK-003	1680-TK-003	1	SODIUM CYANIDE MIX TANK 4.6m Dia x 7.0m H
1680-TK-004	1680-TK-004	1	SODIUM CYANIDE STORAGE TANK 3.6m Dia x 3.8m H
1680-TK-005	1680-TK-005	1	SMBS MIX TANK 3.1m Dia x 3.6m H
1680-TK-006	1680-TK-006	1	SMBS STORAGE TANK 3.1m Dia x 3.6m H
1680-TK-007	1680-TK-007	1	COPPER SULPHATE MIX TANK 1.5m Dia x 2.4m H
1680-TK-008	1680-TK-008	1	COPPER SULPHATE STORAGE TANK 1.5m Dia x 2.4m H

NOTES:

1. COLUMN SPACING TO BE REVISED DURING VALUE ENGINEERING PHASE.
2. HOLD FOR DRIVE TYPE.
3. CONVEYOR ARRANGEMENT TO BE REVISED DURING VALUE ENGINEERING PHASE.
4. HOLD FOR VENDOR INFORMATION.

DO NOT USE THIS DRAWING FOR DEVELOPMENT OF DETAIL ENGINEERING OR CONSTRUCTION

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DRAWING No.	REFERENCE DRAWING
101781-1620-M-114	MILL BLDG. GEN. ARRANGEMENT SECTIONS HH, JJ
101781-1620-M-113	MILL BLDG. GEN. ARRANGEMENT SECTIONS FF, GG
101781-1620-M-112	MILL BLDG. GEN. ARRANGEMENT SECTIONS CC, DD
101781-1620-M-111	MILL BLDG. GEN. ARRANGEMENT SECTIONS AA, BB

No	BY	DATE	REVISION DETAILS	CHKD	ENG	APPR	CLNT
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DESIGNED	C.CLARK	17/03/02
DES. APPR	D.WALDEN	17/03/10
PROJ APPR	S.REDMOND	17/03/10
CLIENT APPR		

PROJECT: LYNN LAKE GOLD PROJECT

TITLE: MACLELLAN PROCESS SITE MILL BUILDING GENERAL ARRANGEMENT COMPOSITE PLAN

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Figure 17-3: MacLellan Mill Building General Arrangement, Composite Plan

Source: Ausenco (2017)

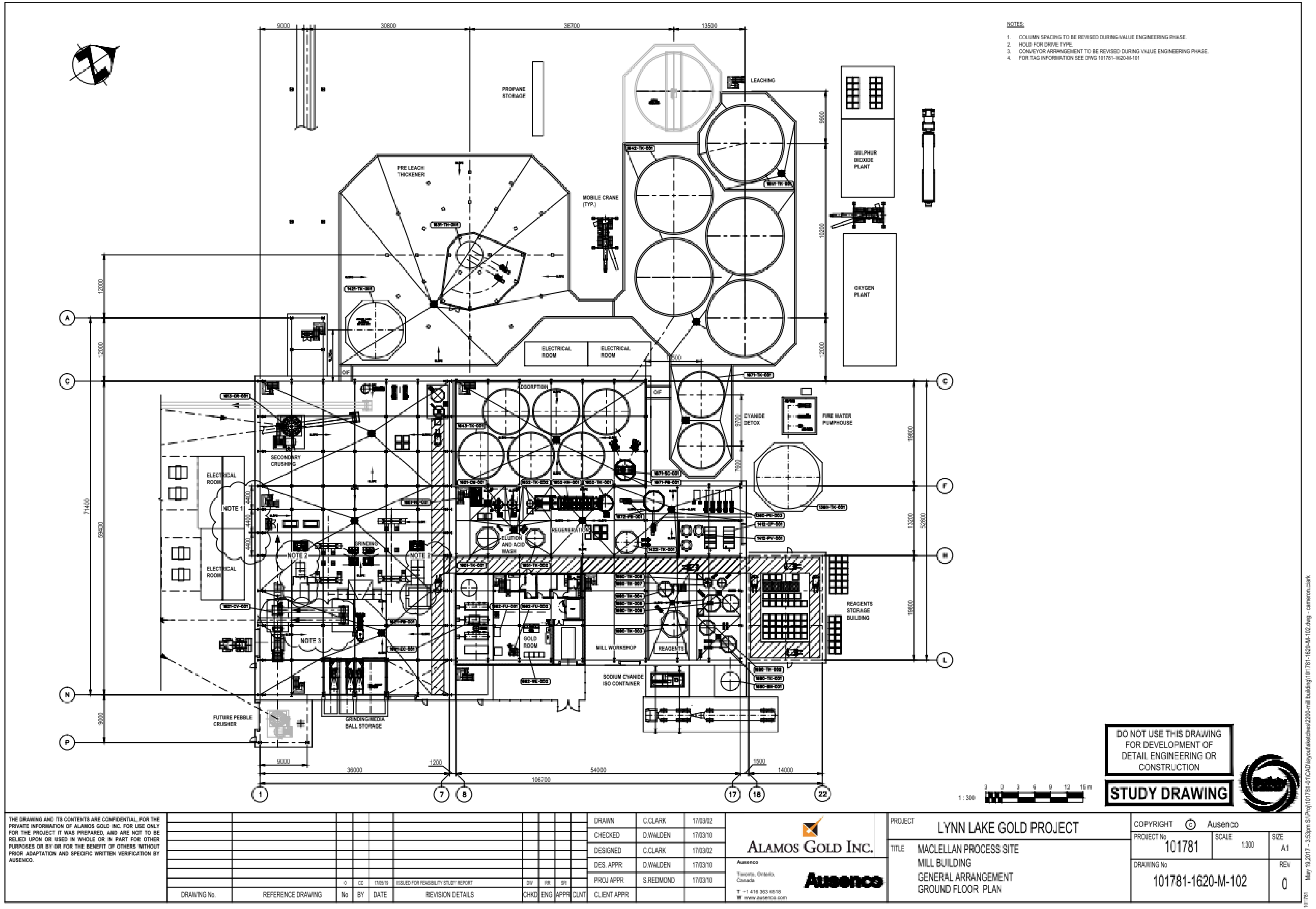

Figure 17-4: MacLellan Mill Building General Arrangement, Ground Floor Plan

Source: Ausenco (2017)

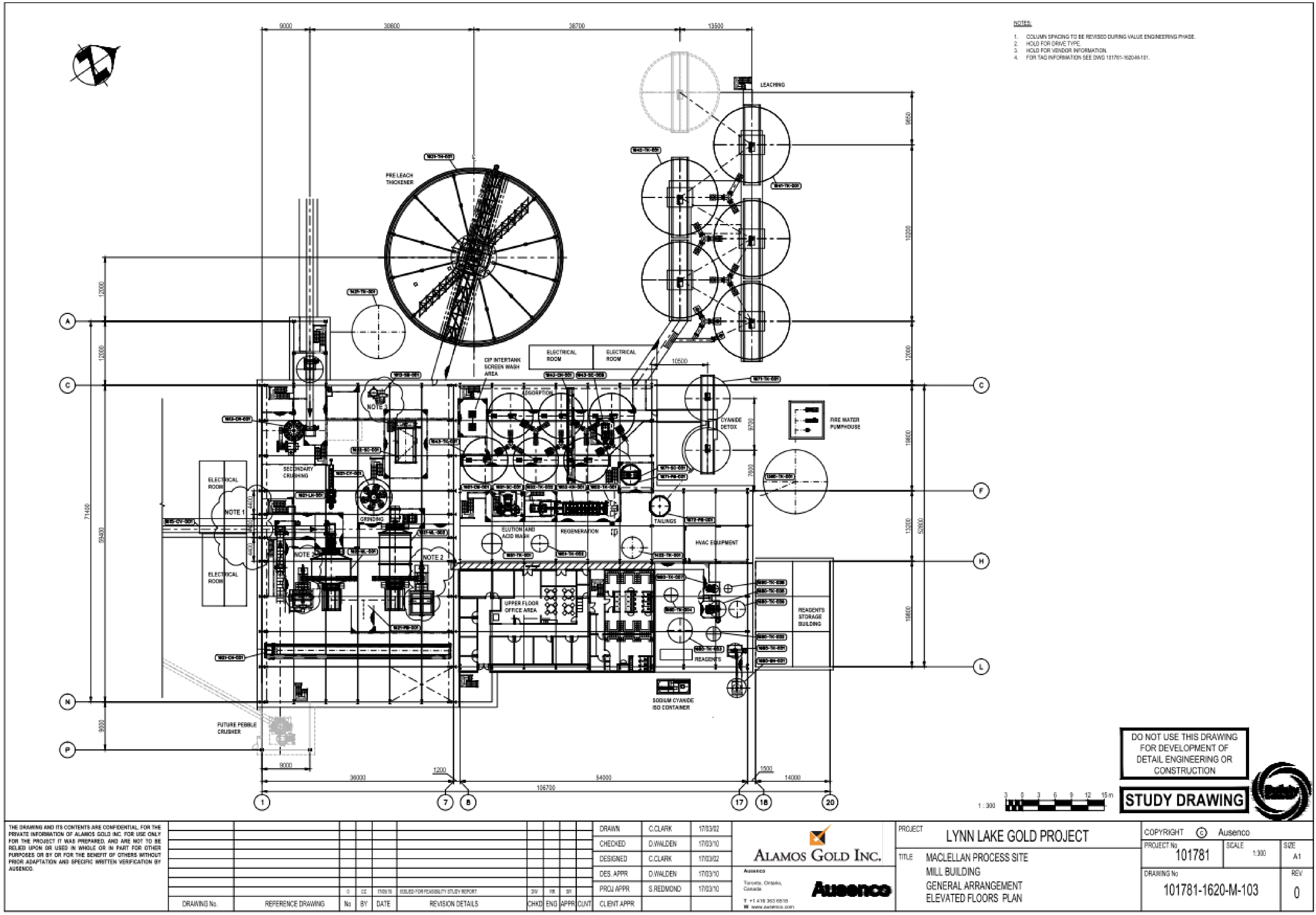

Figure 17-5: MacLellan Mill Building General Arrangement, Elevated Floors Plan

Source: Ausenco (2017)

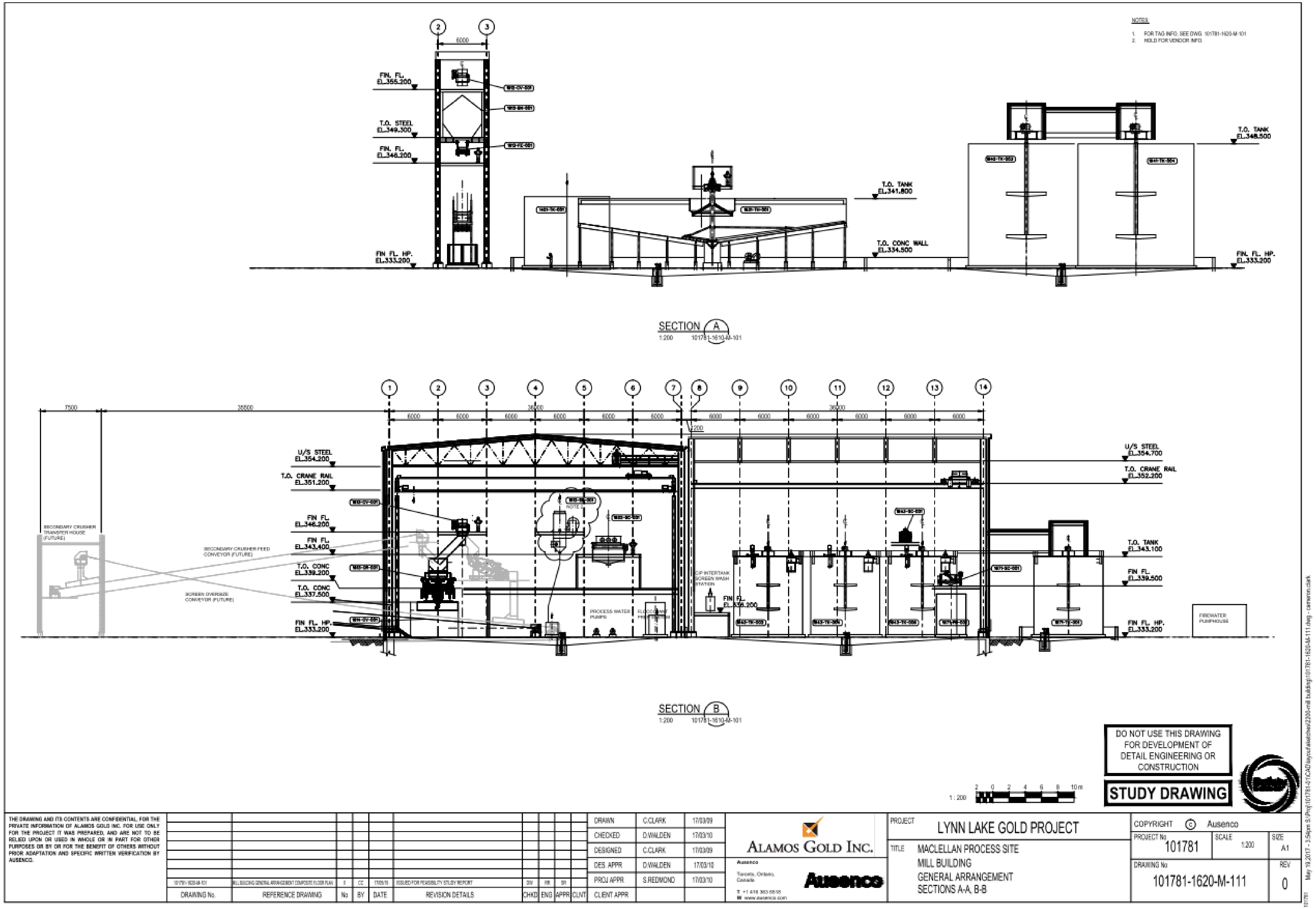

Figure 17-6: MacLellan Mill Building General Arrangement, A-A and B-B

Source: Ausenco (2017)

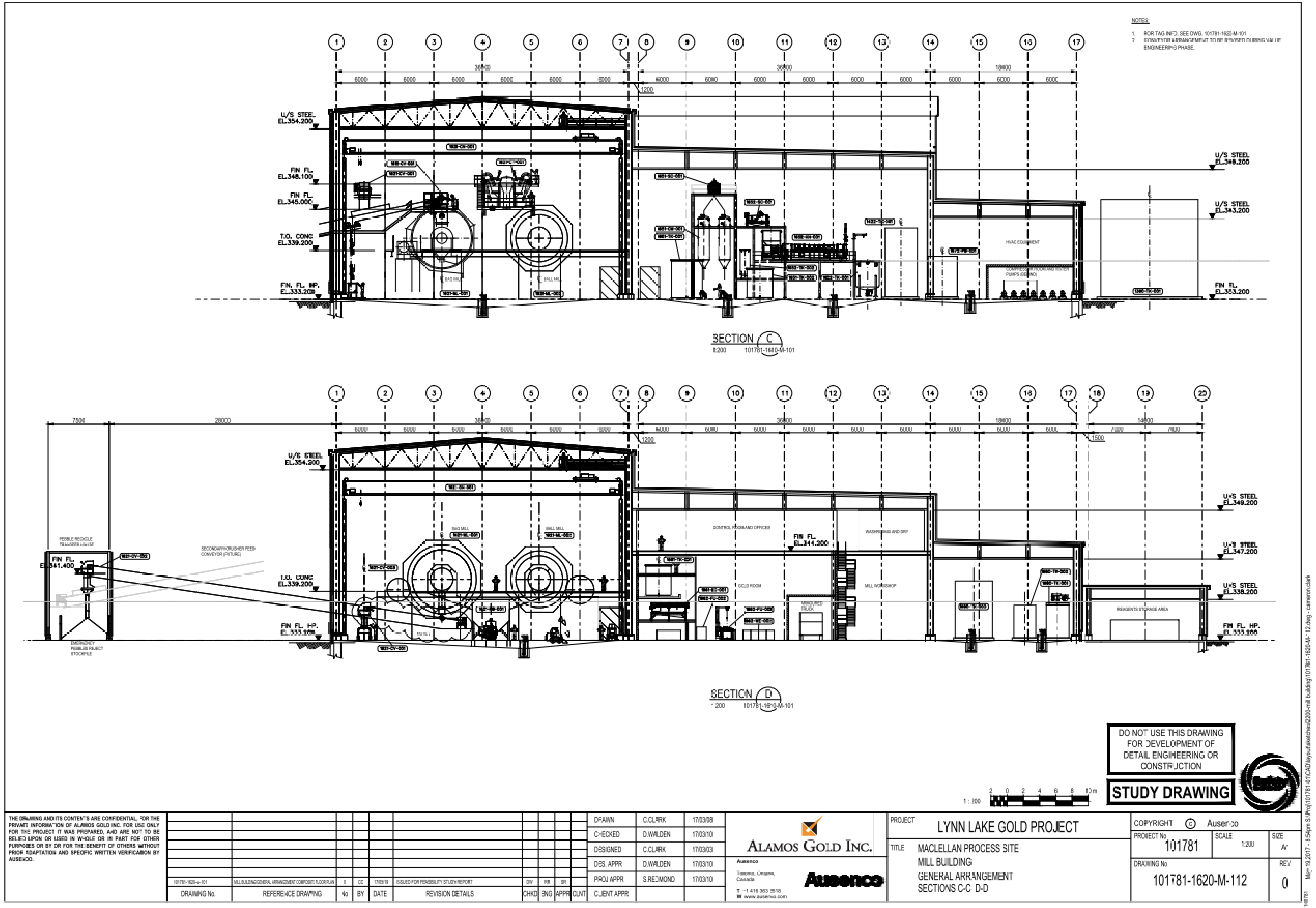

Figure 17-7: MacLellan Mill Building General Arrangement, C-C and D-D

Source: Ausenco (2017)

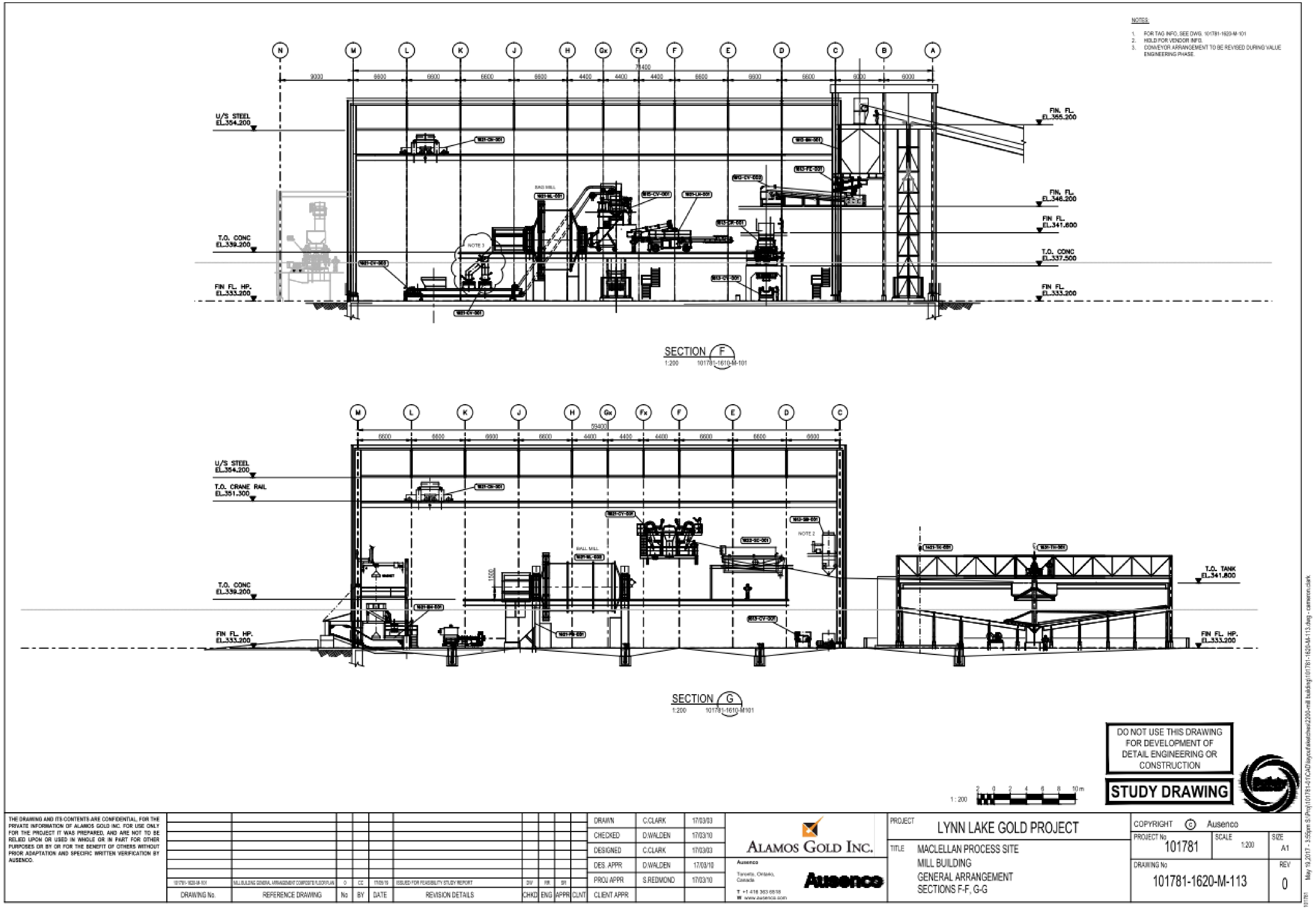

Figure 17-8: MacLellan Mill Building General Arrangement, F-F and G-G

Source: Ausenco (2017)

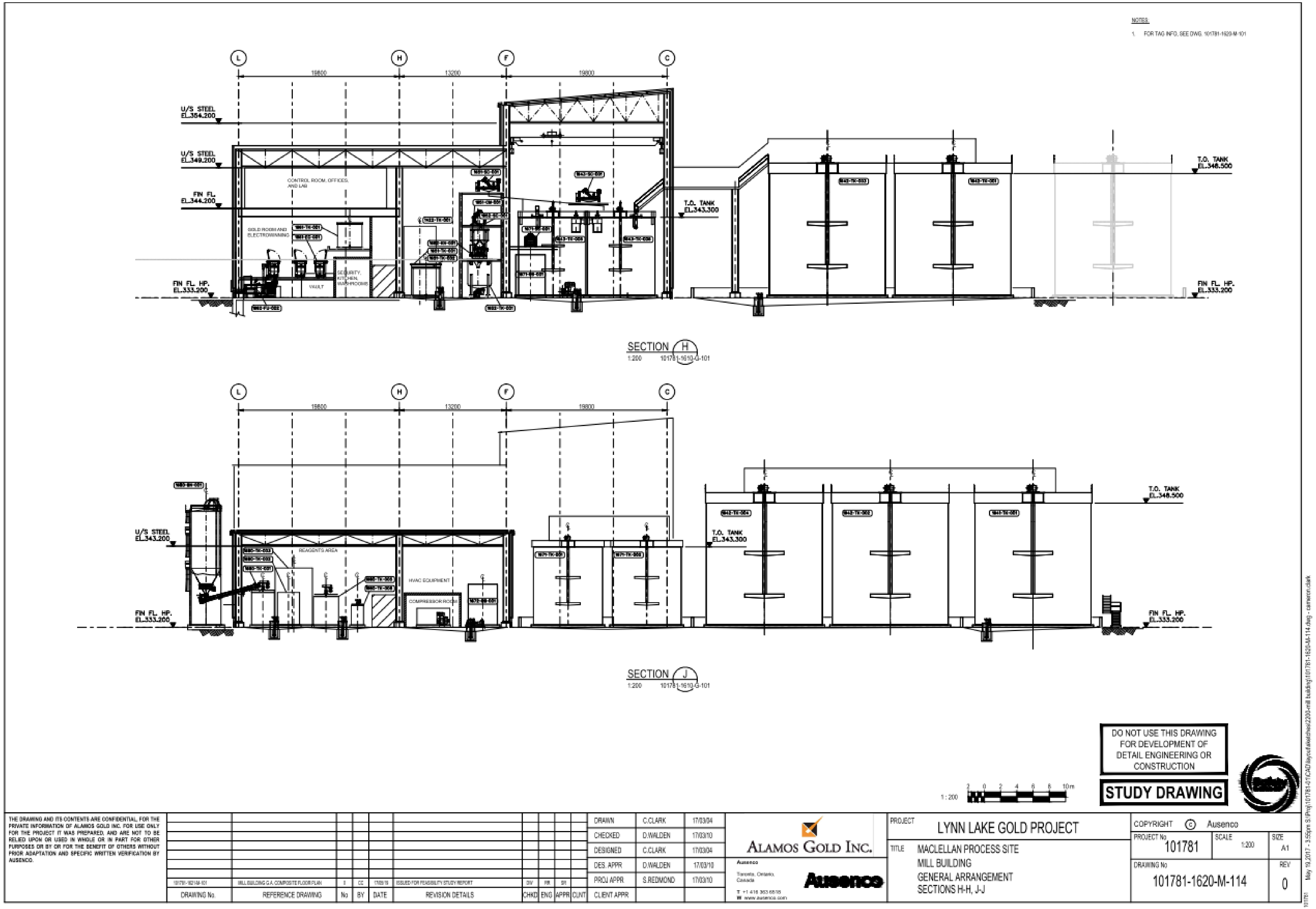

Figure 17-9: MacLellan Mill Building General Arrangement, H-H and J-J

Source: Ausenco (2017)

17.6 Major Process Equipment

The major process equipment is summarized in the Table 17-3.

Table 17-3 – Major Process Equipment, Lynn Lake Project

Area	Item	Description
Primary crushing	Jaw Crusher	51” x 40”, 160 kW, CSS 130 mm
Secondary Crushing	Cone Crusher	450 kW, CSS 35 mm
Stockpile	Apron Feeders	Two feeders, each capable of feeding 100% of full milling throughput of 317 t/h
Grinding	SAG Mill	3.1 MW, 7.3 m diameter with 3.6 m effective grinding length (‘EGL’)
	Ball Mill	5.1 MW, 5.8 diameter with 8.7 m EGL, in closed circuit with hydrocyclones, grinding to a P_{80} of 75 µm
Leach Feed Thickening	Thickener	31 m diameter high-rate thickener
Pre-aeration and leaching	Agitated Tanks	Five 2300 m^3 (live volume) carbon steel agitated tanks, 14.3 m diameter and 15 m height, with 93 kW rubber lined dual impeller agitator each
Gold Adsorption	Agitated Tanks	Six 500 m^3 (live volume) carbon steel agitated tanks, 8.5 m diameter and 9.6 m height, with 22 kW rubber lined dual impeller agitator each
Gold Desorption	Acid Wash and Pressure Zadra System	Batch type, 6 t circuit with one acid wash and one elution column, elution at 135°C and 400 kPa
Carbon Regeneration	Electrical Kiln	680 kW electrical kiln, 300 kg/h batch process, 700°C
Electrowinning	Electrowinning Cells	3 cells, each with 23 1.0 m x 1.0 m cathodes
Cyanide Detoxification	Agitated Tanks	Two parallel 500 m^3 (live volume) carbon steel agitated tanks, 8.5 m diameter and 9.6 m height, with 45 kW rubber lined dual impeller agitator each
Oxygen Production	Oxygen Plant	Vacuum pressure swing adsorption (VPSA), 400 Sm^3/h, 175 kW installed power
Sulphur Dioxide Production	Sulphur Dioxide Plant	Molten sulphur burning plant, 10.5 t/d at 98% SO_2 basis

Table 17-4 shows the summary of the proposed installed power by area, total installed power and normal/average operating demand for both, MacLellan and Gordon sites. The power for the MacLellan site will be supplied by Manitoba Hydro and the power for the Gordon site will be supplied by diesel generators.

Table 17-4 – Summary of Required Installed, Running and Average Power by Area

WBS	Area Description	Installed kW	Running kW	Average kW
MacLellan Site				
1300	MacLellan Onsite Infrastructure	444	39	37
1320	MacLellan Building	1,130	911	865
1360	MacLellan Fresh Water System	143	62	59
1400	MacLellan Utilities and Services	1,120	584	555
1610	Crushing, Storage and Reclaim	1,290	882	838
1620	Grinding	9,608	9,056	8,603
1630	Thickening	471	253	241
1640	Pre-Aeration, Leach, Adsorption	1,076	779	740
1650	Desorption and Regeneration	2,037	1,071	1,017
1660	Electrowinning and Goldroom	234	145	138
1670	Detox and Tailings Pumping	647	301	286
1680	Reagents	395	320	304
1700	Tailings Management Facility	135	75	69
	Subtotal MacLellan	18,730	14,475	13,751
Gordon Site				
2400	Gordon Site	464	217	206
	TOTAL	19,194	14,692	13,957

The annual average consumption is estimated to be 120,000 MWh/y for the MacLellan site and 1,800 MWh/y for the Gordon site.

17.7 Production Forecast

The forecast production for each year of operation was based on the recovery algorithms provided to the mine plan consultants, Q'Pit Inc. The recovery algorithms for gold comprised an extraction model based on a relationship of residue grade (Au-r) with feed grade (Au-f) established from analysis of the test data, less plant soluble and other losses.

- MacLellan, recovery Au % = 100 – ((0.03 + 0.0587 x Au-f) / Au-f x 100 – 0.9) ; and
- Gordon, recovery Au % = 100 – ((0.0978 + 0.0835 x ln(Au-f) / Au-f x 100 – 0.9).

Silver recovery assumed a fixed average extraction of silver due to the highly variable results in leach tests with no apparent trend, less plant soluble losses and other losses. As no significant silver is present in Gordon ores, silver was not accounted for in its production forecast.

- MacLellan, recovery Ag % = 51.9 – 2.9 = 49.0 ; and
- Gordon, recovery Ag % = 0.

Q'Pit calculated process recoveries on a weighted average basis for mine blocks in their mine plan on a period-by-period basis. For each block in the model, a recovered grade was calculated using the recovery algorithm provided. For every period, the grades used in the mine plan were weighted averages based on the sum-product of the tonnage and the grade divided by the total tonnage of the material in that block.

Forecast production by year based on the weighted average of recovered grades in the Q'Pit mine model are summarized in Table 17-5.

Table 17-5 – Annual Production Gold and Silver from MacLellan and Gordon Deposits

PIT	Units	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	TOTAL
GORDON														
Mill feed	,000 t	71	1,647	1,613	1,649	1,652	1,124	356	610	0	0	0	0	**8,723**
Grade (diluted) feed to mill	g/t Au	1.41	2.79	2.44	2.21	2.26	2.94	3.53	0.83	0.00	0.00	0.00	0.00	**2.42**
MACLELLAN														
Mill feed	,000 t	251	908	942	906	903	1,431	2,199	1,945	2,555	2,555	2,555	930	**18,080**
Grade (diluted) feed to mill	g/t Au	1.63	1.99	2.01	1.36	1.69	2.39	1.80	1.40	1.18	1.42	1.59	1.91	**1.63**
Grade (diluted) feed to mill	g/t Ag	3.5	5.5	5.6	3.9	4.0	4.8	5.4	4.6	3.5	3.8	4.2	5.0	**4.4**
TOTAL														
Mill feed	,000 t	322	2,555	2,555	2,555	2,555	2,555	2,555	2,555	2,555	2,555	2,555	930	**26,803**
Grade (diluted) feed to mill	g/t Au	1.59	2.51	2.28	1.91	2.06	2.63	2.04	1.27	1.18	1.42	1.59	1.91	**1.89**
Grade (diluted) feed to mill	g/t Ag	2.7	2.0	2.1	1.4	1.4	2.7	4.7	3.5	3.5	3.8	4.2	5.0	**3.0**
Au Recoverable	,000 oz Au	15.0	191.2	173.2	144.7	156.1	200.3	154.3	94.4	87.9	105.9	119.5	52.5	**1,495.1**
Ag Recoverable	,000 oz Ag	13.7	79.3	83.6	55.5	57.3	107.6	187.6	142.1	140.9	154.6	168.1	73.1	**1,263.4**
AVERAGE ANNUAL		**Y1-Y10**												
Production average (y1-10)	,000 oz Au	143												
Production average (y1-10)	,000 oz Ag	118												

Life-of-mine average recoveries for each deposit are:

- MacLellan Au = 91.4%;
- MacLellan Ag = 49.0%; and
- Gordon Au = 92.9%.

The overall gold recovery for the LLGP is 92.0%.

18 PROJECT INFRASTRUCTURE

18.1 Site Selection and Site Earthworks

The process plant is located immediately to the east of the MacLellan deposit, on relatively natural high ground. Figure 17-2 shows the approximate location of the process plant and associated administration offices and workshops.

The site was selected on a natural elevated area on proper soil/bedrock away from watersheds and to locate the ROM pad close to the open pit to minimize the haul distance and keep the ROM pad activities downwind of the administration area.

The factors considered for layout and site selection were as follows:

- Process plant located in an area safe from flooding;
- Locate the heavy equipment foundation on bedrock;
- Mining, administration and processing staff offices close together due to extreme cold weather;
- Location of the ROM pad near the open pit;
- Utilization of the natural high ground hill from which to build the ROM pad;
- Separating heavy mine vehicle traffic from non-mining light-vehicle traffic;
- The provision to have the change house next to the gatehouse to limit the distance employees have to walk from the bus drop-off point; and
- Having the ready line close to the change house.

18.2 General Access and On-Site Roads

18.2.1 Public Roads

PR 391 is a provincial road which provides access to the MacLellan and Gordon sites from the Town of Lynn Lake and also the City of Thompson, Manitoba.

18.2.2 MacLellan Access Road

MacLellan access road will be a new 2.5 km long and 8 m wide gravel surfaced road connecting the Plant site to PR 391, immediately east of Keewatin River. Plant and mine staff commuting, Gordon ore shipment and supply of material for the MacLellan mine operation will be from this road.

Figure 18-1 shows PR 391 and access roads.

18.2.3 Gordon Access Road

The Gordon access road is an existing two lane gravel road which will be upgraded for the mine operation period. This road will also be used for the haulage of ROM ore from Gordon site to the MacLellan process plant for the duration of six years.

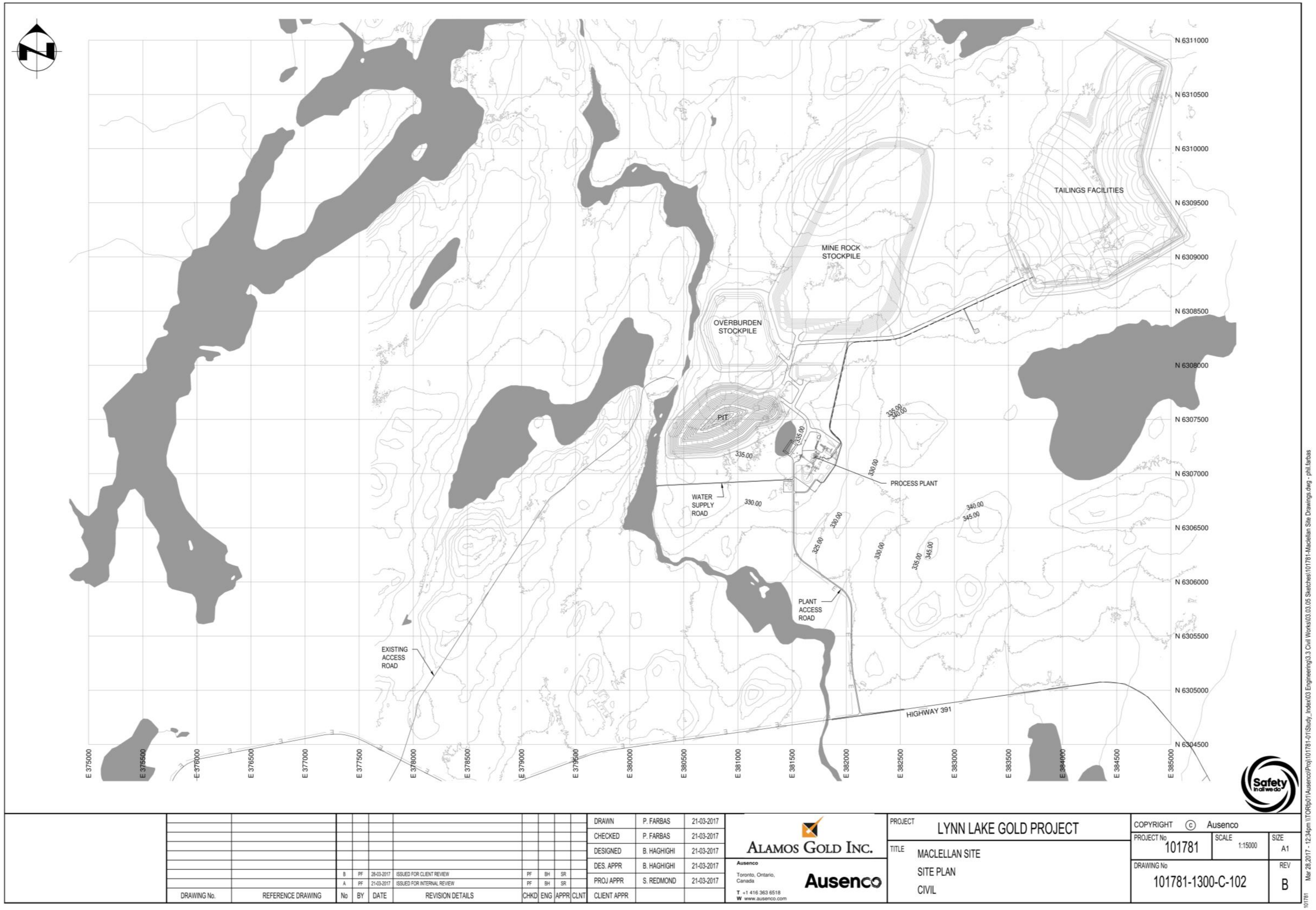

Figure 18-1: PR 391 and Access Roads

Source: Ausenco (2017)

18.2.4 Mine Haul Roads

Haulage of material from the open pit to the ROM pad and stockpile at the MacLellan and Gordon mines will be by off-highway mine trucks.

Mine haul roads are designed to be 25 m wide with 1.7 m high berms on each side, designed for off-highway mine trucks of approximately 144 t capacity, for both MacLellan and Gordon mines.

Haulage of ore material from the Gordon stock pile to the MacLellan ROM pad will be by road trucks via PR 391. The road haul truck will be side dump tractor trailers with approximately 30 t capacity.

18.2.5 Service Roads

The service roads to access other site facilities such as drainage sumps, reagent delivery, electrical substations, administration building, sewage treatment, explosive magazines, etc. will be 5 m wide gravel roads.

18.3 Water Management

18.3.1 Hydrogeology

18.3.1.1 MacLellan Hydrogeology

Groundwater levels have been monitored at 28 monitoring well nests and six drive-point piezometers across the MacLellan site, with continuous water level data collected at 20 monitoring well nests. Groundwater flow is strongly influenced by topography resulting in an overall flow direction from northwest to southeast across the site. In the area of the proposed TMF, groundwater flows from a high of 348 m AMSL in the northwest at Payne Lake to a low of 330 m AMSL in the southeast where Minton Lake and a number of unnamed lakes are located. Existing groundwater flow in the area of the proposed open pit is radial, resulting in a groundwater flow divide extending south of the proposed open pit with flow to the west directed toward the Keewatin River and flow to the east directed toward a tributary of the Keewatin River that is associated with a diffuse surface water drainage area west of Minton Lake. Artesian conditions (i.e., water levels above ground surface) were observed at eight of the bedrock monitoring wells and two overburden wells. Seasonal groundwater level variations were observed to vary from 0 to 3.4 m for the wells monitored.

The hydraulic conductivity of the overburden and bedrock at the MacLellan site was assessed by single well response tests at 12 monitoring wells completed in the overburden, 18 monitoring wells completed in shallow bedrock, and short term pumping tests conducted in 13 exploration boreholes in 2015. Results of the pumping test were assessed with additional results from the packer testing conducted at four boreholes as part of the geotechnical investigation program.

The estimated hydraulic conductivity of the bedrock ranges from approximately 10^{-9} to 10^{-4} m/s, with the upper 50 m of fractured bedrock showing higher conductivity than lower units. The shallow (upper 50 m) bedrock ranges from 10^{-8} to 10^{-4} m/s conductivity, while the lower bedrock ranges from 10^{-9} to 10^{-7} m/s. The distribution of hydraulic conductivity measured in bedrock is shown on Figure 18-2.

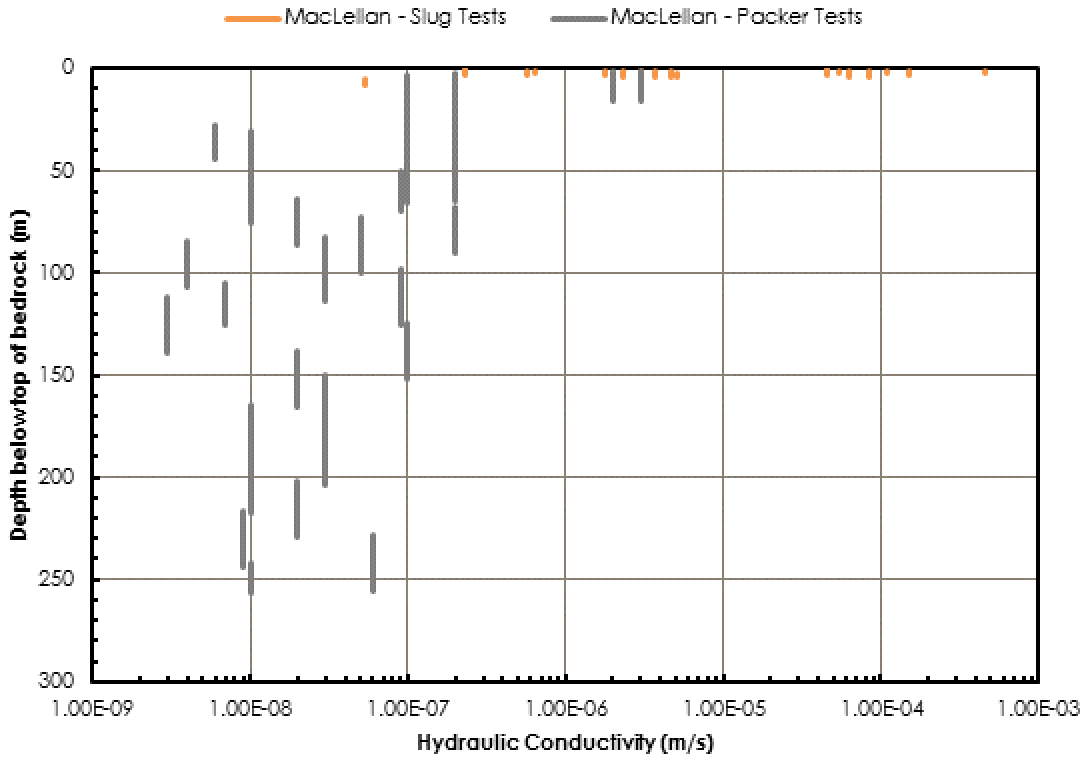

Figure 18-2: MacLellan Bedrock Hydraulic Conductivity Results

Source: (Stantec Consulting Ltd., 2017)

The overburden was found to have hydraulic conductivity ranging from 10^{-7} to 10^{-5} m/s. Based on the initial groundwater modelling, it is anticipated that groundwater inflows may be adequately controlled by pumping from sumps within the open pit. The overburden trough (a northeast-southwest trending bedrock valley, where overburden thickness is increased) may require additional controls to divert the groundwater away from the pit crest.

18.3.1.2 Gordon Hydrogeology

Groundwater levels have been monitored at 17 monitoring well nests and four drive-point piezometers across the Gordon site, with continuous water level data collected at 12 monitoring well nests. Groundwater flow is strongly influenced by topography, which results in radial flow from the topographic high located south of the proposed open pit. Existing groundwater flow from this topographic high is radial with a portion of groundwater flow directed towards Susan Lake in the south, Pump Lake in the east, Farley and Gordon Lakes in the north and a tributary of Gordon Lake in the west. Groundwater flow converges from the north and south in the area of the proposed open pit and Gordon and Farley Lakes.

The Gordon site is characterized by zones of high hydraulic conductivity, as initially investigated by others (W.A. Meneley Consultants 1988). The results of the 2015 geotechnical investigation packer testing completed by Golder provided input for short term pumping tests which were subsequently conducted later in 2015. Based on the results of these studies, a further long-term pumping test program was carried out at the Gordon site in 2016. Golder oversaw the drilling and installation of four pumping wells: two near Gordon Lake, and two near Farley Lake.

The estimated hydraulic conductivity of the bedrock ranges from approximately 10^{-9} to 10^{-4} m/s, with the upper 50 m of fractured bedrock showing higher conductivity than the lower units. The hydraulic conductivity of the bedrock in the vicinity of the lakes was estimated to be approximately an order of magnitude higher than the surrounding shallow bedrock. The upper 50 m of fractured bedrock generally has higher hydraulic conductivity than the bedrock below 50 m. The overburden was found to have hydraulic conductivity ranging from 10^{-7} to 10^{-5} m/s. Figure 18-3 provides a summary plot of the hydraulic conductivity results for the bedrock at Gordon.

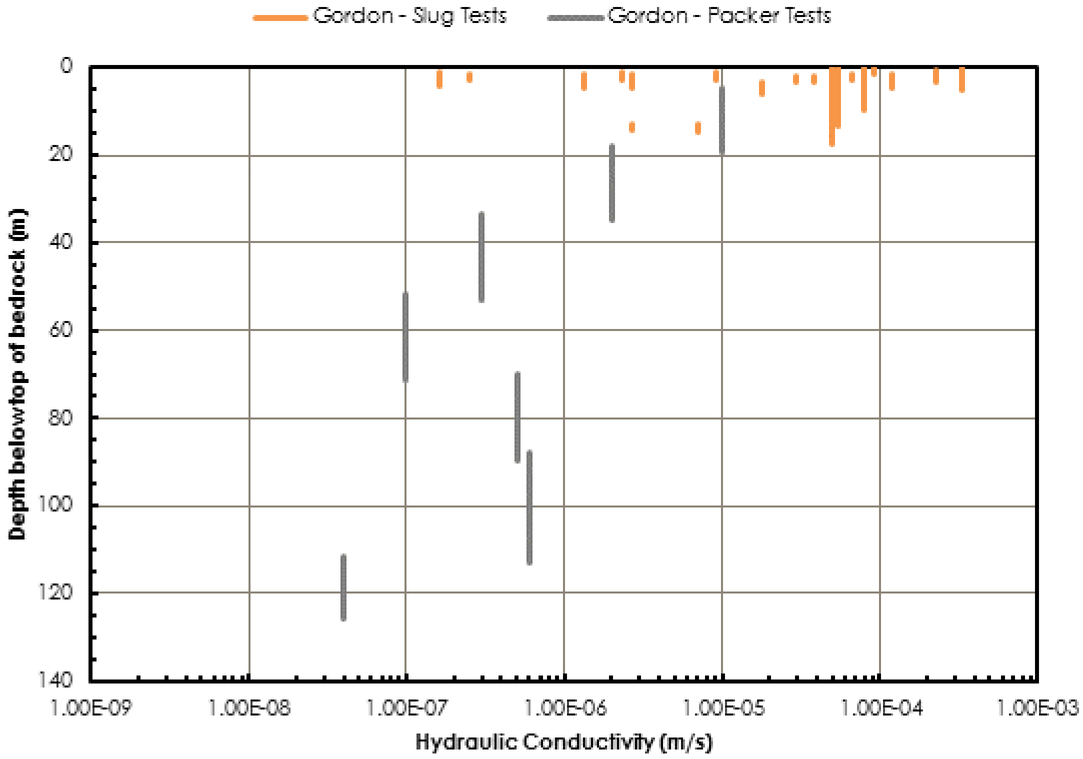

Figure 18-3: Gordon Bedrock Hydraulic Conductivity Results

Source: (Stantec Consulting Ltd., 2017)

A preliminary 2D HydroGeoSphere numerical model was used by Golder to simulate the steady-state groundwater and surface water flow systems that could be expected to develop around the pits by assuming subsurface material properties and distributions, and model boundary conditions that were derived and conceptualized from the field investigation program. A 3D FEFlow model was used by Stantec to simulate the steady state operation of the open pit including the use of shallow (i.e., 50 m deep) dewatering wells which have been proposed to mitigate pit inflows from Farley and Gordon Lakes. The modelling completed by Stantec indicates a long-term average groundwater inflow rate into the open pit of 0.024 m^3/s, with 0.200 m^3/s pumped from dewatering wells. The discharge from the dewatering wells is assumed to be directed toward Gordon and Farley lakes.

18.3.2 MacLellan Site Water Balance

Water balance modelling was carried out for the operations stage to estimate the amount of water collected at the site, the amount of water available in the TMF for reclaim, the discharge volumes to the environment (i.e., site excess water), and the storage volume requirements for sizing the TMF and the collection pond. The water balance was developed on linked Microsoft Excel spreadsheets to simulate the site wide water balance for the MacLellan site. It also includes a mass balance model, which was used for estimating solids settled in the collection pond prior to discharge to the environment.

The water balance simulates the transfer of water between the various mine facilities of the MacLellan site on a monthly basis over a one year period. The inflows to the system are the runoff from precipitation, flows associated with processing the ore, and seepage (i.e., groundwater inflow) into the MacLellan open pit. Evaporation from pond surfaces and water retained in the deposited tailings are accounted for as a loss to the system. The model integrates flows across the site to quantify net inflows, pumping requirements and discharge volumes to the environment for the purpose of Feasibility Study level design.

The site wide water balance was prepared to simulate two phases of the Project mine life, which correspond to the start–up and the ultimate configurations of the site. The same operating data is assumed for both the start-up and ultimate configurations. Three precipitation conditions were evaluated: mean annual (i.e., average year), 100-year wet annual and 100-year dry annual.

The water balance flow logic (Figure 18-4) was set up based on the water management concept described in Section 18.3.9. In addition, the following was considered in the development of the McLellan site water balance:

- The quality of water collected in the proposed site collection pond meets the standards for direct discharge to the environment. Only total suspended solids (TSS) settling is required prior to discharge;
- There is no discharge of water from the TMF to the environment. The TMF has sufficient capacity to temporarily store water during periods of time when the TMF inflows exceed the water losses and the amount of water reclaimed to the mill;
- The discharge of water from the site collection pond to the environment only takes place during non-winter months;
- In the winter a certain volume of water stored in the TMF is in the form of ice and therefore unavailable for reclaim to the mill;
- All seepage through the TMF dams is collected and pumped back to the TMF (i.e. no net seepage loss); and
- Tailings water pumping to the TMF and reclaim water from the TMF to the mill occurs year-round.

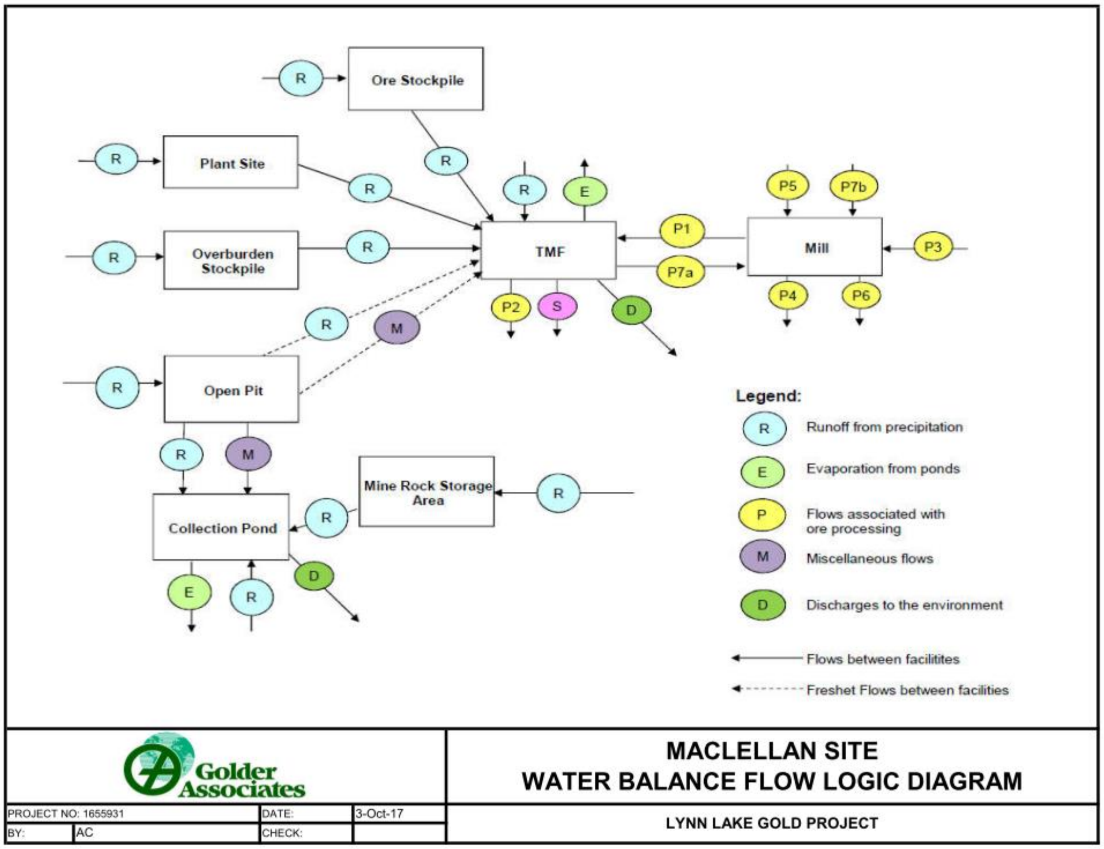

Figure 18-4: MacLellan Site Water Balance Flow Logic Diagram

Source: Golder (2017)

The modelling results are summarized in Table 18-1 and Table 18-2 for the start-up and ultimate configurations, respectively, as annual inflows and outflows associated with the TMF.

Table 18-1 – Tailings Management Facility Water Balance Summary – Start-up Configuration

Flows		Annual Precipitation Conditions 100-y Dry	Mean	100-y Wet
		Annual Volume (Mm^3/year)		
TMF Inflows	Surface Runoff	0.66	0.98	1.39
	Tailings Water	2.27	2.27	2.27
	Mine Dewatering	0.14	0.14	0.14
	Total Inflows	3.07	3.39	3.80
TMF Outflows	Water Retained in Deposited Tailings	0.85	0.85	0.85
	Evaporation	0.43	0.36	0.29
	Reclaim Water from TMF	1.79	2.18	2.29
	Excess Water Stored	0.00	0.00	0.37
	Total Outflows	3.07	3.39	3.80
Mill Make-up Water Deficit (reclaim water required from an external source)		0.50	0.11	0.00

Table 18-2 – Tailings Management Facility Water Balance Summary – Ultimate Configuration

Flows		Annual Precipitation Conditions 100-y Dry	Mean	100-y Wet
		Annual Volume (Mm^3/year)		
TMF Inflows	Surface Runoff	0.64	0.94	1.34
	Tailings Water	2.27	2.27	2.27
	Mine Dewatering	0.14	0.14	0.14
	Total Inflows	3.05	3.35	3.75
TMF Outflows	Water Retained in Deposited Tailings	0.85	0.85	0.85
	Evaporation	0.23	0.19	0.15
	Reclaim Water from TMF	1.97	2.29	2.29
	Excess Water Stored	0.00	0.02	0.45
	Total Outflows	3.05	3.35	3.75
Mill Make-up Water Deficit (reclaim water required from an external source)		0.32	0.00	0.00

For the start-up configuration, there is insufficient water in the TMF to supply the mill water demand year-round under mean annual or dry annual precipitation conditions. Under these conditions, the mill will require water from other sources (e.g., the Keewatin River) to meet water requirements during the winter. The mill make-up water deficit would range from 0.11 Mm^3/y under the mean annual precipitation conditions to 0.50 Mm^3/y under the 100-year dry annual precipitation conditions.

For the ultimate configuration, annual inflows to the TMF approximately equal the annual mill water demand under mean annual precipitation conditions (there is a slight excess that must be stored in the tailings pond according to the water balance results). Under dry annual precipitation conditions, the mill will require water from other sources to meet water requirements during the winter. The mill make-up water deficit would be approximately 0.32 Mm^3/y under the 100-year dry annual precipitation conditions.

Based on the maximum accumulation in the tailings pond under the 100-year wet annual precipitation conditions, operating pond volumes of 1.86 Mm^3 and 1.4 Mm^3 are required for the start-up and ultimate configurations, respectively.

The modelling results for the MacLellan collection pond are summarized in Table 18-3 as annual inflows and outflows associated with the site collection pond.

Table 18-3 – MacLellan Collection Pond Water Balance Summary – Ultimate Configuration

Flows		Annual Precipitation Conditions		
		100-y Dry	Mean	100-y Wet
		Annual Volume (Mm^3/year)		
Inflows	Runoff	0.60	0.88	1.25
	Mine Dewatering	0.51	0.51	0.51
	Total Inflows	1.11	1.39	1.76
Outflows	Evaporation	0.01	0.01	0.01
	Water Discharged	1.10	1.38	1.75
	Total Outflows	1.11	1.39	1.76

18.3.3 Gordon Site Water Balance

Water balance modelling was carried out for the operations stage to estimate the amount of water collected at the site, pumping requirements and discharge volumes to the environment, and the storage volume requirements for sizing the collection pond. The water balance for the Gordon site was developed on linked Microsoft Excel spreadsheets and also includes a mass balance model, which was used for estimating solids settled in the collection pond prior to discharge to the environment.

The flow model simulates the transfer of water between the various mine facilities at the Gordon site on a monthly basis over a one year period. The inflows to the system are the runoff from precipitation and seepage (i.e., groundwater inflow) into the Gordon open pit. Evaporation from the pond surface is accounted for as a loss to the system.

The site wide water balance simulated the ultimate configuration of the site under three precipitation conditions: mean annual (i.e., average year), 100-year wet annual and 100-year dry annual.

The water balance flow logic (Figure 18-5) was set up based on the water management concept described in Section 18.3.11. In the development of the Gordon site water balance it was assumed that the discharge from the collection pond to the environment (i.e., to Farley Lake) only takes place during non-winter months, as pit inflows are expected to be minimal during the winter, and that the mine water quality meets standards for direct discharge to the environment. Therefore, water treatment is not required prior to discharge, with settling of TSS being the main function of the collection pond.

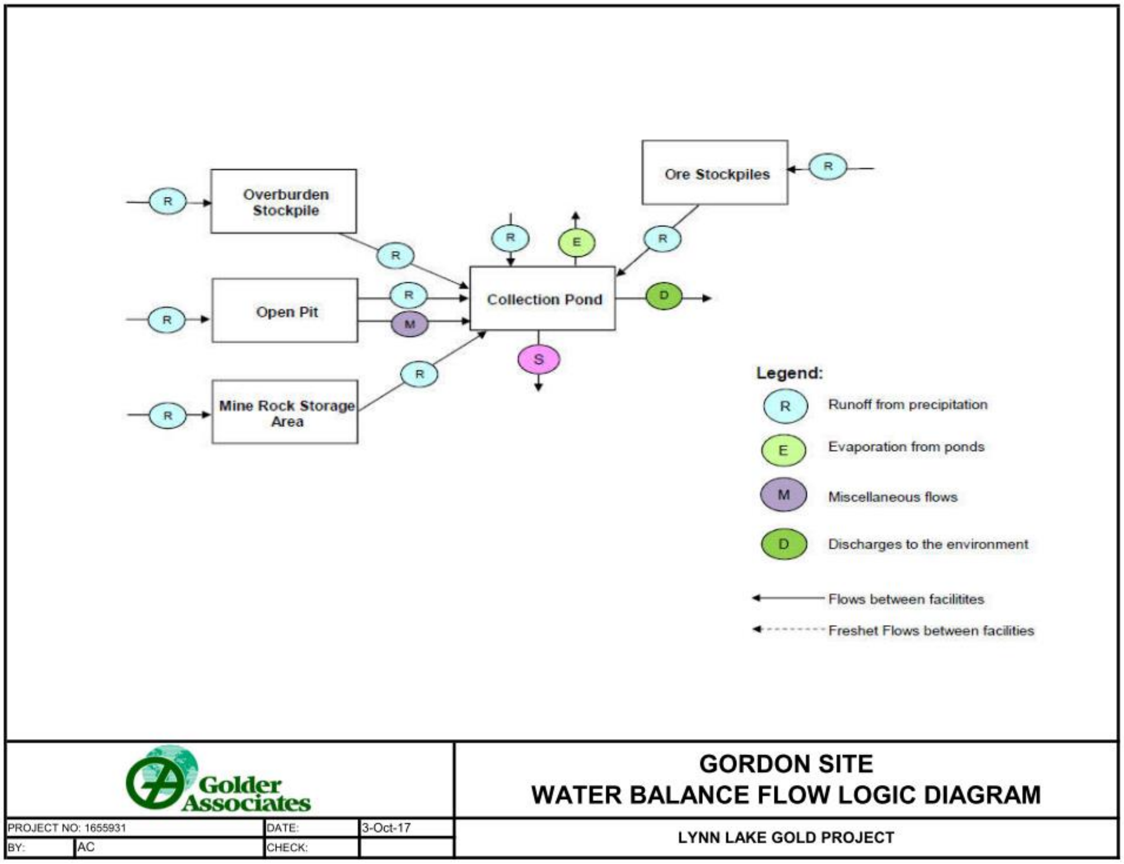

Figure 18-5: Gordon Site Water Balance Flow Logic Diagram

Source: Golder (2017)

The modelling results are summarized in Table 18-4 as annual inflows and outflows associated with the site collection pond.

Table 18-4 – Gordon Collection Pond Water Balance Summary

Flows		Annual Precipitation Conditions		
		100-y Dry	Mean	100-y Wet
		Annual Volume (Mm^3/y)		
Inflows	Runoff	0.43	0.63	0.89
	Mine Dewatering	0.44	0.44	0.44
	Total Inflows	0.87	1.07	1.33
Outflows	Evaporation	0.01	0.01	0.01
	Water Discharged	0.86	1.06	1.32
	Total Outflows	0.87	1.07	1.33

18.3.4 MacLellan Fresh Water Supply and Distribution

The fresh water for the processing facilities will be pumped, via two submersible pumps (one duty, one standby), from the Keewatin River through a buried 2.4 km 250 mm HDPE pipeline to the fresh/fire water tank located within the plant site. The pipeline is buried to prevent freezing. The storage tank is constructed from mild steel and has a total live volume of 1,050 m^3. A portion, corresponding to two hour retention time, is utilized as the fire water reserve and the remaining portion has a one hour capacity to feed the various fresh water pumps. Two (one duty, one standby) horizontal centrifugal pumps supply fresh water to various consumers throughout the plant site such as reagent mixing and SAG and ball mill cooling water make-up. Two (one duty, one standby) additional horizontal centrifugal fresh water pumps feed fresh water to the process water tank to make up the volume of available process water. A fifth, horizontal centrifugal fresh water pump supplies water to the fresh water treatment plant producing potable water to all the Project installations. The reticulation pipework is heat-traced and thermally-insulated where required. Maximum fresh water requirements at MacLellan will be 312 m^3/h for plant start-up and the first year of operation. Normal fresh water requirements will be 40 m^3/h once enough water is available from the TMF for reclaim.

18.3.5 Gordon Fresh Water Supply and Distribution

The fresh water supply is pumped, via two submersible pumps (one duty, one standby), from the Gordon Lake through a buried 100mm HDPE pipeline to the fresh/fire water tank located at the site. The pipeline is buried to prevent freezing. The storage tank is constructed from mild steel and has a live volume of 700 m^3. A portion, equivalent to two hour live volume, is utilized as the fire water reserve and the remaining portion has a one hour capacity to feed the various fresh water pumps. Two (one duty, one standby) horizontal centrifugal pumps supply fresh water to various consumers. The reticulation pipework is heat-traced and thermally-insulated where required. Normal fresh water requirements for truck shop and truck wash make up water will be 10 m^3/h.

18.3.6 MacLellan Potable Water System

Potable water supply will be provided for the processing plant by treating filtered fresh water. A vendor supplied potable water treatment plant is used to generate the potable water. Fresh water is pumped through an ultra-violet unit for sterilization and filters for removal of contaminants and particles not acceptable in water for human consumption. The plant capacity will be 92 m^3/d.

Two (one duty, one standby) horizontal centrifugal pumps are fed from a 107 m^3 covered tank located within the water services pump house and supply safety showers and main distribution lines throughout the plant site.

18.3.7 Gordon Potable Water System

Two (one duty, one standby) horizontal centrifugal pumps will be fed from a covered 21 m^3 tank and supply safety showers and main distribution lines throughout the Gordon site. The tank is filled via tanker truck from the MacLellan potable water treatment.

18.3.8 Reclaim Water

The main source of process water will be reclaim water from the TMF. Water will be reclaimed with three (two duty, one standby) submersible return water pumps located on a single floating barge. Reclaim water is transferred through a single 4.6 km 250 mm HDPE pipeline to the process water tank at the main plant. The reclaim water pipeline will be a buried to prevent freezing. Normal reclaim water flow will be 272 m^3/h.

A tee with automatic valves in the pipeline directs excess water back to the TMF when the process water tank becomes full or there is plant site run-off bleed-off required. The tee is located near the barge end of the pipeline.

18.3.9 Sewage Collection

The sewage from the MacLellan site buildings will be collected in manholes and will be conveyed by gravity through buried PVC piping to a 60 m^3/d sewage treatment plant located at the MacLellan site.

The collected sewage from the Gordon site buildings will be conveyed by gravity through buried PVC piping to two septic tanks at the truck shop and administration building. It will then be trucked to MacLellan for processing at the MacLellan sewage treatment plant.

18.3.10 MacLellan Surface Water Management

The overall water management concept is to divert non-contact water to reduce the amount of contact water to be managed at the MacLellan site. Contact water is conveyed to:

- The TMF, where water is temporarily stored for recirculation to the mill; or
- The collection pond, where sediment control can be provided prior to the release of water to the environment.

Figure 18-6 shows the ultimate footprints of the mine facilities located at the MacLellan site and the water management infrastructure, including non-contact water diversion ditches, contact water collection ditches, sumps, culverts and the site collection pond.

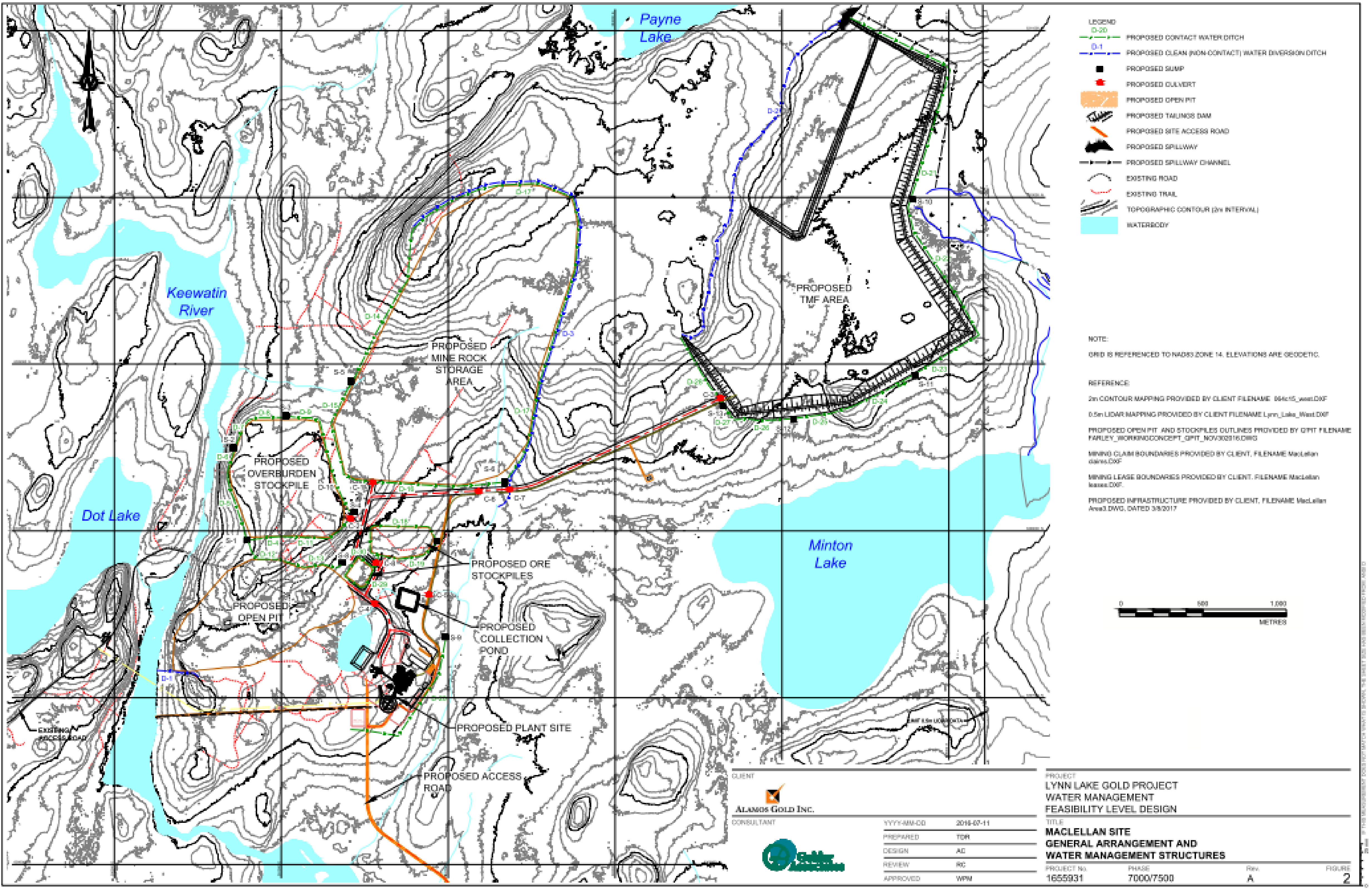

Figure 18-6: MacLellan Site General Arrangement and Water Management Infrastructure

Source: Golder (2017)

There are three non-contact water diversion ditches. These are located at the open pit, the Mine Rock Storage Area and the TMF. The diversion ditches are designed to convey the peak flow resulting from the 1 in 25-year rainfall storm event.

The diversion ditch along the southwest corner of the open pit (D-1) flows into the Keewatin River. The diversion ditch for the Mine Rock Storage Area (D-3) flows from north to south, under the TMF access road via a culvert (C-7) and discharges into an existing stream south of the access road. The diversion ditch located on the west side of the TMF (D-2) flows from south to north and discharges into a wetland area north of the TMF.

Contact water collection includes the following:

- Mine water, comprised of dewatering flows from the MacLellan open pit;
- Tailings water from the process plant;
- TMF water comprised of direct precipitation on the tailings pond, surface runoff from the tailings beach and seepage through the TMF dams is collected and pumped back to the TMF; and
- Site runoff and seepage water from the process plant area, Mine Rock Storage Area and stockpiles.

General site surface runoff from disturbed areas will be allowed to drain naturally to collection ditches that will convey the water to collection sumps. The water from the Mine Rock Storage Area will be pumped from the collection sumps to the collection pond for storage prior to discharge to the environment (towards the Keewatin River). The water from the plant site and the ore and overburden stockpiles will be pumped from the collection sumps to the TMF. Water collected in the TMF will be stored for recirculation to the mill. There will be no discharge of water from the TMF to the environment during operations.

The water collection ditches are designed to convey the peak flow resulting from the 1 in 25-year rainfall storm event. The collection sumps are sized to manage the peak flow resulting from the 1 in 25-year rainfall storm event through a combination of water storage and pumping while maintaining a minimum freeboard of 0.3 m.

Dewatering flows from the MacLellan open pit will be pumped to the site collection pond during the non-winter months, except during the spring freshet (April and May) when it has been assumed that 75% will be pumped to the TMF and the remaining 25% to the site collection pond.

Seven culverts are required to convey flows under access roads. The culverts are sized to convey the peak flow resulting from the 1 in 100-year rainfall storm event.

The sizing of water management ponds was completed at feasibility level considering the following:

- Minimum pond depth for operational purposes;
- Maximum pond depth based on:
    - Maximum operating volume; or
    - Maximum storage required in combination with a discharge pumping rate; or
    - Retention time to promote settling of solids.

- Storage of the Environmental Design Flood (largest design runoff event that can be stored and does not result in an unscheduled discharge of water to the environment); and
- Minimum freeboard.

In addition, the volume of water that is lost to ice in the winter was considered when sizing the TMF pond.

The TMF is equipped with an emergency spillway to allow safe routing of the design precipitation event to maintain a minimum freeboard and prevent dam overtopping. The design event was selected as 1/3 between the 1 in 1000-year and the probable maximum flood (PMF), according to the CDA Dam Safety Guidelines (Canadian Dam Association, 2013).

The maximum storage available in the collection pond is dictated by space constraints. Additional settling will be required through mitigation techniques such as flocculation or physical methods (e.g., silt curtains and/or baffles) to avoid exceeding the discharge limits set under the Metal Mining Effluent Regulations (MMER, 2017) for TSS concentrations.

18.3.11 MacLellan Water Treatment

Water treatment during the construction phase is not required as the water quality of the initial dewatering sites do not exceed Metal Mining Effluent Regulation (MMER) limits based on the baseline monitoring program. During operation, preliminary results of runoff from mine rock, overburden, and ore stockpiles and dewatering from the open pit sump also do not suggest any MMER exceedances. No discharge is anticipated during operation of the TMF.

By the time of mine closure, material from the ore stockpile will have been removed and processed. The MacLellan open pit will be flooded to avoid ARD and Metal Leaching (ML) related with the open pit surfaces. PAG and NPAG mine rock will be blended at MacLellan during operation to avoid ARD at closure. Monitoring of runoff and groundwater associated with historical mine rock and overburden storage do not show MMER exceedances or signs of acidification after approximately 30 years of storage. Accordingly, no treatment of discharge from overburden and mine rock is anticipated for the MacLellan site during closure and post-closure. Any discharge from the TMF will be directed to the open pit during closure. The tailings surface will be covered as part of the mine closure which will limit ARD/ML and is expected to improve TMF pond quality to levels acceptable for direct discharge to the environment. After improvement of water quality in the TMF pond, the pit lake will be permanently stratified and filled. When the pit lake is full, the discharge chemistry is expected to be calcium, magnesium and sulphate rich with some elevated metal concentrations (e.g., iron, manganese, copper, and lead); similar to the water quality currently observed in the existing MacLellan shaft and deep bedrock exploration boreholes. Additional polishing can be done by discharging the water from any mine components through engineered or natural wetlands, which will provide passive treatment.

18.3.12 Gordon Surface Water Management

The overall water management concept is to divert non-contact water to reduce the amount of contact water to be managed at the Gordon site, and collect the contact water for conveyance to the collection pond where sediment control can be provided prior to the release of water to the environment. Figure 18-7 shows the ultimate footprints of the mine facilities located at the Gordon site and the water management infrastructure, including a non-contact water diversion ditch, contact water collection ditches, sumps, culverts and the site collection pond.

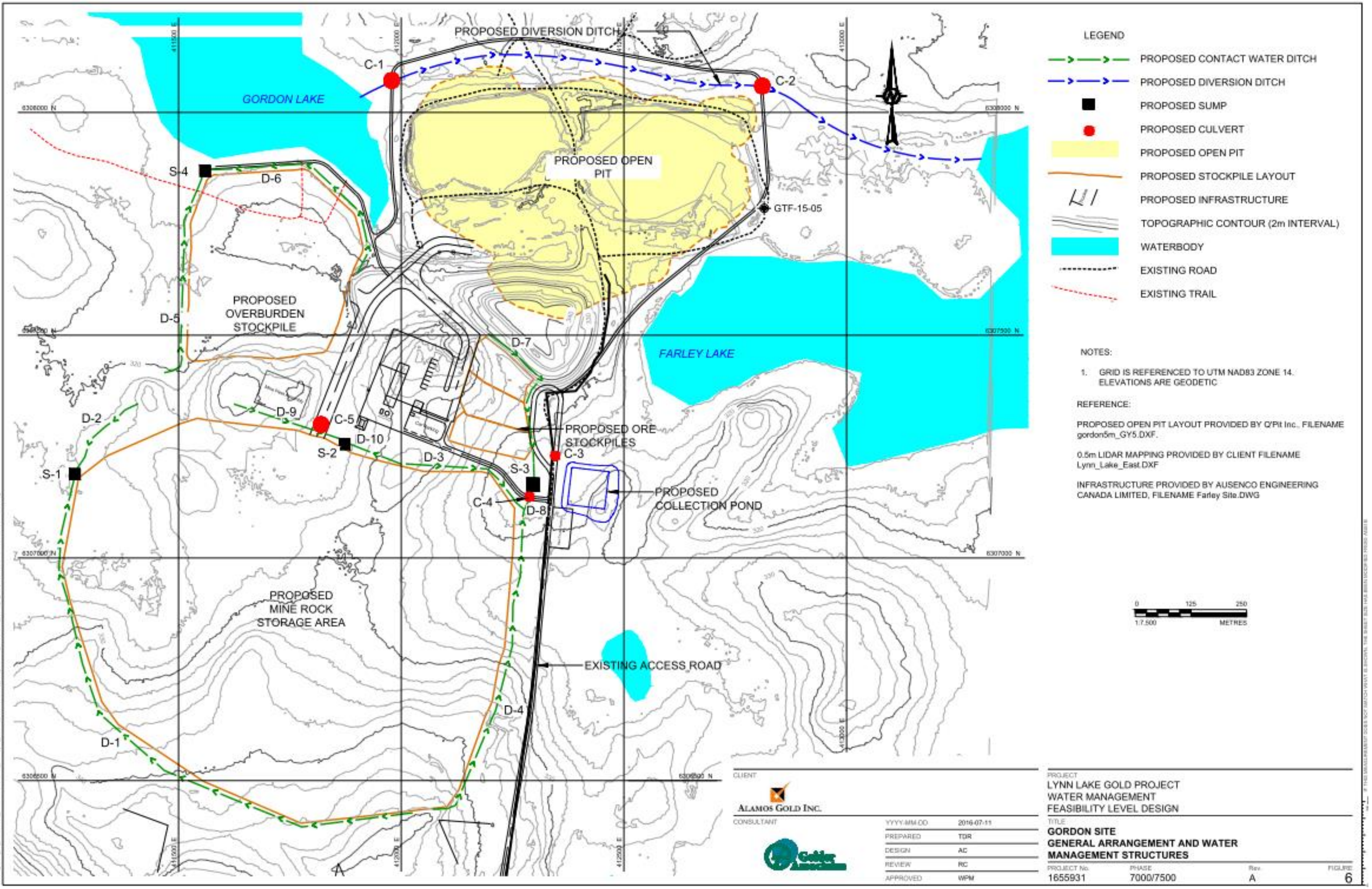

Figure 18-7: Gordon Site General Arrangement and Water Management Infrastructure

Source: Golder (2017)

The existing diversion channel that currently connects Gordon and Farley lakes will have to be re-aligned to accommodate the expansion of the open pit, resulting in a new diversion ditch to be built north of the open pit. This is the only non-contact water diversion ditch included in the water management system of the Gordon site. The diversion ditch is designed to convey the peak flow resulting from the 1 in 25-year rainfall storm event.

Contact water collection includes the following:

- Mine water, comprised of dewatering flows from the Gordon open pit; and
- Site runoff water, comprised of surface runoff from the stockpiles and seepage resulting from infiltration of precipitation through the stockpile surfaces.

General site surface runoff from disturbed areas will be allowed to drain naturally to collection ditches that will convey the water to collection sumps. The water will be pumped from the sumps to the collection pond for storage prior to discharge to the environment. Dewatering flows from the open pit will be also pumped to the site collection pond. Water collected in the pond will be discharged to Farley Lake.

The water collection ditches are designed to convey the peak flow resulting from the 1 in 25-year rainfall storm event. The collection sumps are sized to manage the peak flow resulting from the 1 in 25-year rainfall storm event through a combination of water storage and pumping while maintaining a minimum freeboard of 0.3 m.

Dewatering wells will be installed around the pit perimeter to reduce the inflow of water to the pit. Water from these wells will be discharged directly to the environment. Dewatering flows from the Gordon open pit sumps will be pumped to the collection pond during the non-winter months (approximately from April through October).

Five culverts are required to convey flows under access roads. The culverts are sized to convey the peak flow resulting from the 1 in 100-year rainfall storm event.

The sizing of the collection pond was completed at feasibility level considering the following:

- Minimum pond depth for operational purposes;
- Maximum pond depth based on:
  - Maximum storage required in combination with a discharge pumping rate; or
  - Retention time to promote settling of solids.
- Storage of the Environmental Design Flood (largest design runoff event that can be stored and does not result in an unscheduled discharge of water to the environment); and
- Minimum freeboard.

The maximum storage available in the collection pond is dictated by space constraints. Additional settling may be required through mitigation techniques such as flocculation or physical methods (e.g., silt curtains and/or baffles) to avoid exceeding the discharge limits set under the Metal Mining Effluent Regulations (MMER, 2017) for TSS concentrations.

18.3.13 Gordon Water Treatment

During construction, discharges from the dewatering of the existing pit lakes will be required. Dewatering wells will also be installed to divert water from entering the existing pits. Current water quality in the existing pit lakes and surrounding groundwater environment do not exceed MMER limits. Accordingly, there is no apparent requirement for treatment prior to discharge to the environment.

During operation runoff from the mine waste rock, overburden, and ore stockpiles and from the Gordon open pit sumps will be directed to a collection pond. Kinetic testing does not show any exceedances of MMER limits in leachates from mine rock, overburden, or ore. The ARD onset time for portions of PAG ore and PAG mine rock is beyond the current expected period of operation at the Gordon site. Preliminary water quality modeling results indicate that concentrations in the collection pond will not exceed MMER limits during operation. On this basis, treatment of discharge from the Gordon site collection pond is not expected to be required.

By the time of mine closure, material from the ore stockpile will have been removed and processed. The Gordon open pit will be flooded to avoid potential ARD/ML associated with the pit surfaces. The water quality in the resulting pit lake is not expected to exceed MMER based on observed concentrations in the existing (historical) pit lakes. During operation, PAG and NPAG mine rock will be blended to avoid ARD. Monitoring of runoff and groundwater near the historical mine rock and overburden storage does not show MMER exceedances or signs of acidification after approximately 20 years of storage. Accordingly, no treatment of discharges from the pit lake, overburden, and mine rock at the Gordon site is anticipated to be required during closure and post-closure.

18.3.14 MacLellan Pit Dewatering

Groundwater at the MacLellan open pit will be primarily managed by sump pumps. If localized seepage were to occur, then sub-horizontal drains will be installed. This will minimize the potential build-up of pore pressure behind the slope faces. Groundwater levels will be monitored by vibrating wire piezometers. This water will be primarily pumped to the collection pond to allow for settling of solids prior to discharge.

A conventional sump pump dewatering system will be in operation at the MacLellan open pit. The system includes three diesel driven pumps and HDPE piping running on the wall surface, anchored and buried at bench and road crossings. The dewatering includes inflow water from pit walls and storm water runoff, the quantities of which depend on the stage of pit development and will increase to a maximum rate of 443 m^3/hr at the final stage of pit development.

18.3.15 Gordon Pit Dewatering

Dewatering measures, by means of pumping wells, will be employed at the Gordon open pit perimeter in close proximity to the adjacent lakes to reduce the groundwater inflow into the pit. A total of 13 dewatering wells will be installed to a depth of 50 m, with a nominal well spacing of 100 m, to intercept both groundwater flows and near surface flows from both Gordon Lake and Farley Lake. The easternmost system consists of seven pumping wells, each pumping at 14 L/s, while the westernmost system consists of six pumping wells, each pumping at 7 L/s.

A combination of drawdown wells and direct pit dewatering sump pumps in stages will be in operation at Gordon open pit.

The open pit dewatering pumping system includes two diesel driven pumps and HDPE piping running on the wall surface, anchored and buried at bench and road crossings. The dewatering includes inflow water from pit walls and storm water runoff, the quantities of which depend on the stage of open pit development and will increase to a maximum rate of 259 m^3/hr at the final stage of pit development.

18.4 Power Supply and Distribution

18.4.1 Electrical Power Source

Major electrical power will be required at the MacLellan site as all process facilities and major infrastructure buildings are situated there. The MacLellan site is located approximately 7 km northeast of the Town of Lynn Lake. The electrical power to Lynn Lake is supplied by Manitoba Hydro via Manitoba Hydro Line 6 which is terminated at the Lynn Lake Copper Substation. Line 6 runs from the Laurie River Generating Station to Lynn Lake. At the present time this line is being operated at 69 kV.

The Project will require the following:

- Changing Line 6 from 69 kV operation to 138 kV operation;
- Converting the Copper St. Station at Lynn Lake from 69 kV to 138 kV;
- Modifications at the Laurie River Terminal Substation for 138 kV connection to Line 6;
- Construction of a 138 kV tap on Line 6 near the Town of Lynn Lake; and
- Construction of a 7 km 138 kV overhead line to the MacLellan site, terminating at an Alamos owned end of line structure with a 138 kV isolating switch.

Through consultation with Manitoba Hydro, it has been determined that there is a weak power supply source for Line 6 at the 78 km mark from Copper Sub-station in Lynn Lake. For this reason, large Ball and SAG mill motors must be started with variable frequency drives (VFDs) to minimize the anticipated voltage drop on the line during motor start-up. This is discussed further in Section 18.4.7.

The Gordon site has low electrical power demand. Power requirements at the Gordon site will be met by two 300 kW diesel generators in duty/standby configuration. Power distribution will consist of 4.16 kV overhead lines.

18.4.2 Power Demand Estimates

The average and peak power demands for both sites are shown in Table 18-5. Total loads have been calculated using the Project's process plant mechanical equipment list and other building power requirements. Appropriate demand and utilization factors have been applied to the connected electrical loads to arrive at peak and average demands. The calculated peak power demand was then increased by 10% for the MacLellan site to provide an assessment of peak demand that the Manitoba Hydro power supply system would have to sustain during operation.

Table 18-5 – Average and Peak Power Demand

Site	Description	Demand
MacLellan Mine	Average Power Demand	13.8 MW
	Peak Demand	16.9 MW
	Manitoba Hydro Indicated Peak	18.6 MW
Gordon Mine (diesel generator)	Average Power Demand	200 kW
	Peak Demand	300 kW

18.4.3 Main Substation at MacLellan Site

At the MacLellan site main sub-station the 138 kV utility supply will step-down to 6.9 kV for plant wide power distribution. This substation will be located close to the SAG and Ball Mills, as they are the major electrical power consumers. The two 15/20 MVA oil type substation transformers are sized to carry the maximum power required by the MacLellan site. This includes future growth and redundancy in the event a single transformer is temporarily out of service.

18.4.4 Site Power Distribution

The electrical load centers, in the form of factory built electrical rooms (e-houses) are strategically located close to clusters of motors and other loads to minimize cable lengths. Factory prefabricated e-houses were selected to minimize the field installation labour hours, as a result expediting the construction schedule.

Power is being distributed to the process plant using 6.9 kV feeder cables, as appropriate, based on distance from the power source.

The overhead 6.9 kV distribution lines will supply power to the outlying pumping stations and service buildings.

18.4.5 Redundancy

For the site electrical power distribution system, redundancy has been the prime consideration at all voltage levels. Full redundancy in transformers and distribution breakers/ cables will provide continuity of operation in the event of transformer or main feeder cable failure.

18.4.6 Standby / Emergency Power Supply

The MacLellan site is provided with a 1 MW standby diesel generator sized to supply critical process loads and life safety systems. The standby diesel generator is located close to the main substation and connected to the 6.9 kV main substation switchgear. In this way, a single generator set is able to supply standby power to all facilities using the normal power distribution system.

At the Gordon site two 300 kW diesel generators in duty/standby configuration are included to provide the same reliability for emergency and life safety loads.

18.4.7 Ball and SAG Mill Starting System

Soft starting of mill motors will be required to minimize voltage drop. A common variable frequency drive (VFD) will sequentially start the mill motors. First, the ball mill will be powered with the VFD and its speed ramped up slowly. Once the ball mill motor reaches rated speed, a by-pass breaker will be closed to run the ball mill motor from the source supply in turn by-passing the VFD. The VFD will then become available for the starting and running of the SAG mill. The SAG mill will then continue running with VFD and its speed will be controlled by the Process Control System.

Synchronous VFD mill motors equipped with an active front end (rectifier) is installed to provide dual function support to the incoming power system. The active front end VFD will provide a soft start capability for both the Ball and SAG mill motors, mitigating large voltage drops during mill motor starting and offers power factor improvement at the MacLellan substation. By running the VFD in leading power factor mode, the power factor at the supply end of the main substation will be kept close to unity (1.0 PF). Even considering sustaining a full demand load by the process plant’s smaller motors at poor power factor (< 0.8 PF), which is typical for this process which utilizes a large number of induction motors.

The slow speed synchronous motors with a VFD mill drive system will be more expensive than the standard high speed induction motor system with gear boxes, but there is an associated major cost and operational advantage when overall power system improvements are taken into consideration. The requirement for significant capacitor banks will be eliminated and the slow speed synchronous system will remove the need for gear boxes, thus reducing maintenance with improved reliability and efficiency.

18.4.8 Construction Power

The permanent diesel generators required during operations will be deployed to provide construction power. The MacLellan site will have a 1 MW generator, while Gordon will have two 300 kW generators in duty/standby configuration. The standby generator at Gordon will be portable, allowing it to serve as back-up power for both sites. Contractor power requirement will be 600 V, so a small portion of permanent 6.9 kV or overhead lines will be built at each site to bring power to contractor trailers and construction areas.

18.5 Plant Control System

18.5.1 Process Control System (PCS)

The key objective of the control system is to provide an automated plant wide system that is uniform in architecture, hardware and configuration software throughout all process plant areas.

Control of the plant will be through the Human Machine Interfaces (HMIs) located in the crushing area control room, and the main control room. PCS controller panels will be located in each e-house, networked with fibre optic cables, providing remote access.

The control system is designed to be redundant down to the I/O level, with software updates and certain hardware changes to be done while the plant remains in operation. The servers for the control system will be located in the main control room. Local uninterrupted power supply (UPS) will maintain operation for 60 minutes to compensate for power surges and outages.

A workstation is provided in the maintenance workshop for maintenance monitoring of the process control system. The Asset Management System and Engineering Server are also available on this HMI.

The Process Control System shall be fully configured and Factory Acceptance Test (FAT) tested before shipment to the site. This includes the communication to third party PLC systems and controls, weigh scales, and analysers. Third party PLC's must be transparent and fully integrated to the control system.

The Gordon site has minimum control for plant services (water, air and fuel). The control will be done with a local controller located in the electrical room. It will not communicate with the MacLellan site.

18.5.2 Field Instruments and Valves

Field instrumentation and valves will be wired to either the Process Control System or the specialty PLC's. All analogue devices will have Highway Addressable Remote Transducer (HART) protocol to monitor each device and setup the configuration. An Asset Management System is provided to capture and store call HART transmitter configuration and changes. Each analogue device will be supplied in a pre-calibrated state ready for installation.

Wireless instruments have been considered based on certain instrument locations and applications. Certain instruments and analysers will be supplied with mechanical equipment.

18.5.3 Closed-Circuit Television

Process video cameras will be installed in seventeen locations to assist the operator's view of the process. These cameras are viewed on a separate display in the primary crushing area control room, and the main control room.

18.6 Communication System

The Town of Lynn Lake is presently serviced with communications systems insufficient to support the needs of the LLGP. This includes the hardwired trunk lines that exist to Lynn Lake. Primary Rate Interface (PRI) trunk lines will continue to be used for primary voice communication, and backed up through alternate links.

Voice communications within each mine site will be on a subnet or hardwired connections, with a microwave link to Lynn Lake for interface to the Bell Canada system. Several trunk lines on a back-up satellite link will also be available. These trunk lines are not prioritized for voice.

A dedicated commercial satellite network will be employed to handle the data and Internet needs of the mine site and its employees. Satellite bandwidth of 10-20 Mbps and a block of 125 GB of data per month is required.

The satellite link at the MacLellan site will be located next to the site offices. The site office will provide a controlled environment space to house the equipment to be connected to the site networks.

The satellite link at the MacLellan site will communicate via microwave links to the Gordon mine site and to the Alamos office in Lynn Lake. The microwave link will operate at 50 Mbps bandwidth. Microwave towers shall be provided and installed in each of the three locations – MacLellan, Gordon and the Alamos office in Lynn Lake.

The network will have a teleport with cable provider service to the head office in Toronto, Ontario. The MacLellan site will be the communication hub to Toronto.

Site towers will also be used for the open pit mine communication. It is assumed that each tower is 30 m high and self-supported. Each tower requires a foundation, which will be a concrete base or screwed anchors. Towers located outside of Alamos property may require a land lease.

Additional recreational internet services will be provided to employees in the accommodations camp via the satellite link and then distributed on the subnets within each mine site/town.

These links and interconnections are handled by Alamos. These include the telephone, Business IT connections, and security systems.

18.7 Mine Management System

Operations are expected to use radio communication. An allowance of $200,000 is included in the initial capital budget for the initial purchases. Replacement of radios is covered under the miscellaneous area of the mine operating costs.

18.8 Compressed Air Systems

18.8.1 MacLellan Compressed Air System

The compressed air system at the MacLellan site is comprised of two 750 kPa rotary screw compressors (one duty, one standby). A portion of the air will be filtered and dried using a desiccant air dryer and stored in a dedicated instrument air receiver for distribution to pneumatically actuated instruments. The plant air will be distributed to utility stations for maintenance.

A separate 750 kPa rotary screw compressor and receivers will be dedicated to the primary crusher to eliminate the need for long utility pipe runs. A portion of the air will be filtered and dried using a desiccant air dryer and stored in a dedicated instrument air receiver for distribution to pneumatically actuated instruments.

18.8.2 Gordon Compressed Air System

The compressed air system at the Gordon site is comprised of two (one duty one standby) 750 kPa rotary screw compressors and a wet receiver. This system will be for untreated compressed air distributed to utility stations for equipment maintenance. No instrument quality air is required at the Gordon site.

18.9 Fuel Supply, Storage and Distribution

MacLellan will have two fuel distribution stations. One station will service the mining fleet with three high flow pumps using dyed diesel. The other will have one gasoline pump for the light vehicle fleet and two high flow diesel pumps for the road haul trucks using un-dyed diesel.

Gordon will have one fuel distribution station which will service the mining fleet with two high flow pumps using dyed diesel.

18.9.1 MacLellan Diesel Fuel Stations

Mining fleet fuel will be provided by a dedicated fuel station located close to the maintenance shop and truck wash facilities. Three tanks each of 120,000 L capacity will be used to store mine equipment dyed diesel fuel.

Diesel and gasoline fuel storage and distribution for road trucks and light vehicles will be by a dedicated fuel station at the road haul truck access road. Two tanks each of 75,000 L capacity will be used to store un-dyed diesel and one tank for gasoline.

All tanks will be double walled horizontal cylindrical with piping system in concrete trenches as secondary containment.

18.9.2 Gordon Diesel Fuel Station

Mine truck fuel will be provided by a dedicated fuel station at the maintenance shop and truck wash facilities. Two tanks each of 120,000 L capacity will be used to store mine vehicle dyed diesel fuel. All tanks will be double wall cylindrical with piping system in concrete trenches as secondary containment.

18.9.3 Gasoline

Gasoline fuel storage and distribution at the MacLellan site for light vehicles includes one double walled tank with the capacity of 5,000 L and a single pumping dispenser. The gasoline fuel tank and dispensers will be located on a common concrete pad with the haul trucks diesel fuel station.

18.9.4 Propane

Propane will be used for heating purpose and maintenance shop requirements. Propane at the MacLellan site will be stored in two horizontal vessels, each 20,000 L capacity. Propane at the Gordon site will be stored in a 20,000 L horizontal vessel.

18.10 Vehicle Wash-down Facilities

Two mine truck wash facilities will be needed for the LLGP; one at MacLellan and one at Gordon. The road haul trucks will be periodically washed in the MacLellan wash facility.

All wash facilities will be enclosed, include monitors using circulated water with minimum make up water. Oil separator units will be installed at each truck wash. The wash facilities are further discussed in Section 18.11. Figure 18-8 depicts the overall size and layout of the MacLellan truck wash facility.

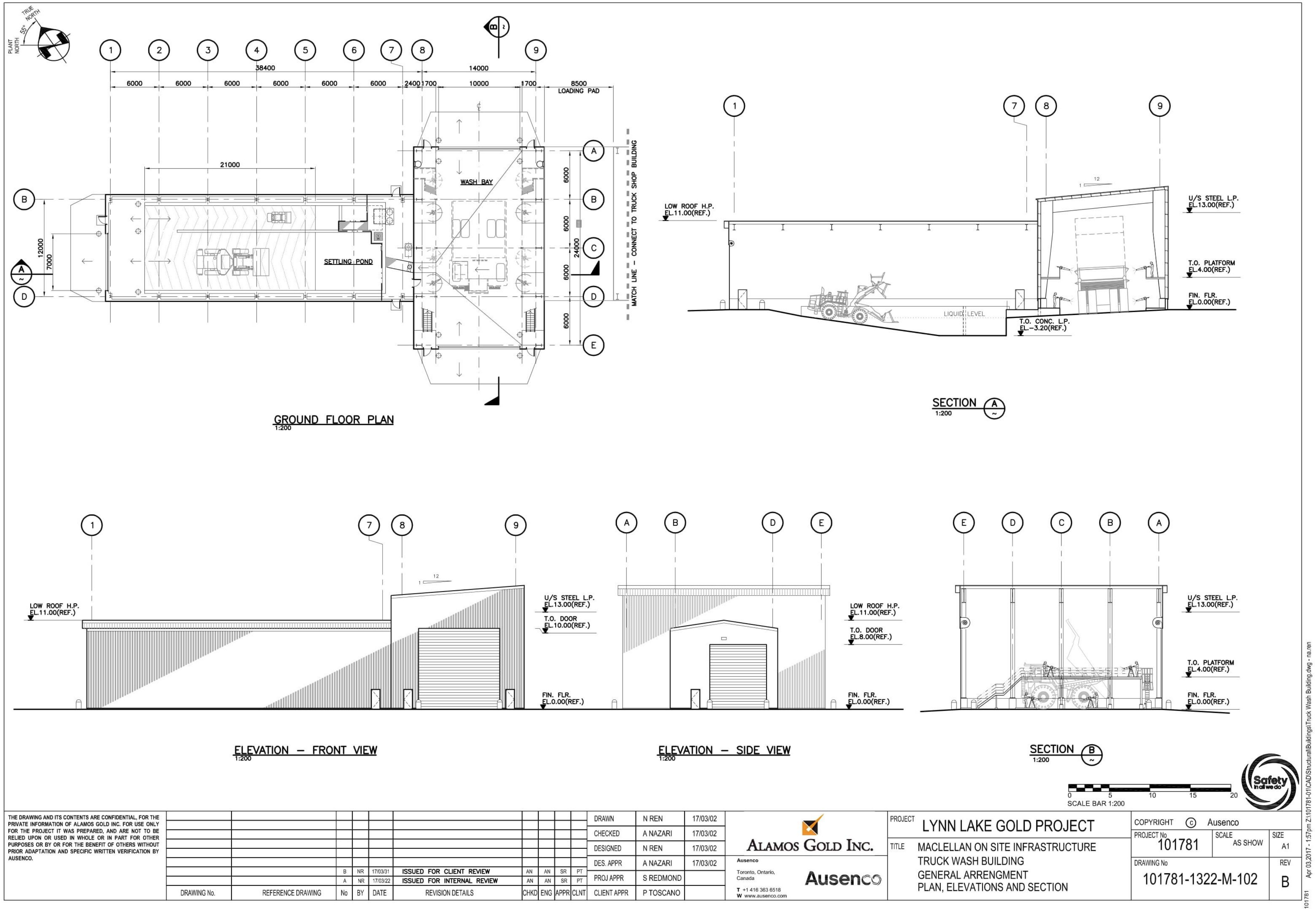

Figure 18-8: MacLellan Truck Wash Facility General Arrangement

Source: Ausenco (2017)

18.11 Buildings

18.11.1 Primary Crushing Building – MacLellan

The primary crusher building at MacLellan will be 30 m (long) x 16 m (wide), located north of process plant, and directly south of the ROM pad Mechanically Stabilized Earth (MSE) retaining wall. The building will house the primary crusher control room, rock breaker, stationary grizzly, primary crusher hopper, apron feeder, vibrating grizzly feeder, primary jaw crusher, chutes and platework, dust collector and compressor room. The equipment will be located on the ground floor, and on two main operating floors and multiple equipment access platforms. The process equipment will be serviced by overhead crane. The building construction will be conventional structural steel framing with reinforced concrete foundations, and the buildings will be fully enclosed with pre-insulated sandwich panel roof and wall cladding.

18.11.2 Process Plant Building – MacLellan

The process plant building at MacLellan will be approximately 59 m (wide) x 88 m (long), located south of primary crusher building, and west of the coarse ore storage stockpile/reclaim. The building will be divided into four sections: grinding, CIP, elution/gold room/control room, and reagent storage.

The grinding area will be 26 m (wide) x 59 m (long), and will have a ground floor, one elevated concrete floor and multiple equipment access platforms. The building will house the secondary crusher feed bin, secondary cone crusher, SAG mill, ball mill, trash screen, cyclone, cyclone feed pump, pebble conveyors, ball bunker, and liner handler. The process equipment will be serviced by an overhead crane.

The CIP area will be 20 m (wide) x 36 m (long), and will house six 8.5 m diameter CIP tanks including tank platforms. The area will be serviced by an overhead crane for accessing tank agitators and filters/strainers.

The elution/gold room/control room area will be 32 m (wide) x 54 m (long) with two stories. The first floor will house the acid and elution columns, carbon regeneration kiln, potable water tank, tailings pumps, compressor room, reagent mixing tanks, mill workshop and gold room. The gold room will be a 18 m (wide) x 24 m (long) secure room within the process plant, with Concrete Masonry Unit (CMU) blockwork construction, and will house the electrowinning cells, barren solution pump box, pan filter, furnace, vault and security room. The second floor in this area will be 20 m (wide) x 24 m (long) and will house the plant main control room, metallurgical laboratory, lunch room, offices, men's and women dry, and wash rooms. The second floor construction will be pre-fabricated modular construction type, placed on an elevated concrete slab floor.

The reagent storage area will be 20 m (wide) x 24 m (long) at the east end of the process plant building and will be used for storing reagents including cyanide, activated carbon, carbon sulphate, flocculants, antiscalant, and SMBS. Outdoor storage adjacent to the reagent storage area is reserved for additional storage as required.

The process plant building construction will be of conventional structural steel framing with reinforced concrete foundations, and the building will be fully enclosed with pre-insulated sandwich panel roof and wall cladding.

18.11.3 Administration Office – MacLellan

The administration office at the MacLellan site will be a 25 m (wide) x 65 m (long) single story building located west of the process plant, and south of the coarse ore stock pile. The building will house offices, meeting rooms, work stations, lunch rooms, wash rooms, men's and women's dry, lockers, and showers. The administration building floor plan is shown in Figure 18-9.

The building will be of pre-fabricated modular construction, placed on precast concrete block footings.

18.11.4 Administration Office / Gate House – Gordon

The administration office/gate house at Gordon site will be approximately 20 m (wide) x 20 m (long) single story building. The building will house two offices, meeting room/lunch room, wash rooms, men's and women's dry, lockers, and showers.

The buildings will be of pre-fabricated modular construction, placed on precast concrete block footings.

18.11.5 Mine Truck Shop / Warehouse – MacLellan

The mine truck shop/warehouse building at the MacLellan site will be 45 m (wide) x 66 m (long), and will be located east of the process plant building, and south of mine truck wash building. The building will be divided into two sections: mine truck maintenance area and warehouse area.

The maintenance area will be 30 m (wide) x 66 m (long), and will have four maintenance bays, a repair shop, compressor room, offices, lunch room, washroom, spare part staging room, and a lubricant storage area. The maintenance bays will be serviced by an overheard crane.

The warehouse area will be 15 m (wide) x 66 m (long), and will house storage racks, and an open area for large part storage. Outdoor storage adjacent to the truck shop has been reserved for tire storage.

The building will be a pre-engineered metal building with reinforced concrete foundations, and insulated roof and wall metal cladding. Figure 18-10 depicts the layout and general arrangement of the MacLellan truck shop/warehouse building.

18.11.6 Mine Truck Shop – Gordon

The mine truck shop building at Gordon will be 16 m (wide) x 60 m (long), and will house two mine truck bays and a maintenance area around the trucks. The building will be of fabric building type construction, comprised of steel frames as structural support and an architectural membrane cover.

18.11.7 Road Haul Truck Maintenance

Road haul trucks will be maintained in the MacLellan mine truck shop.

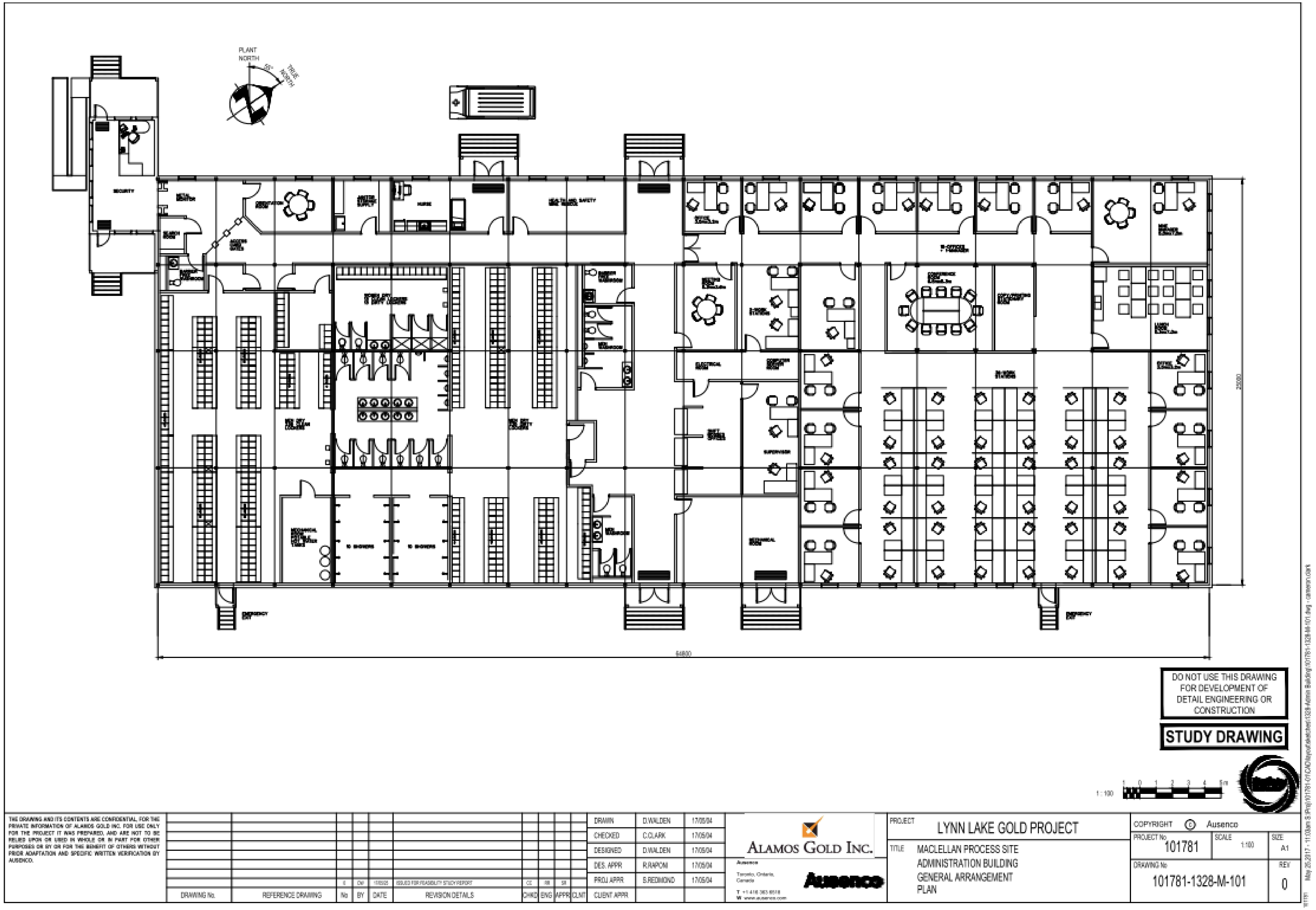

Figure 18-9: MacLellan Administration Building Floor Plan

Source: Ausenco (2017)

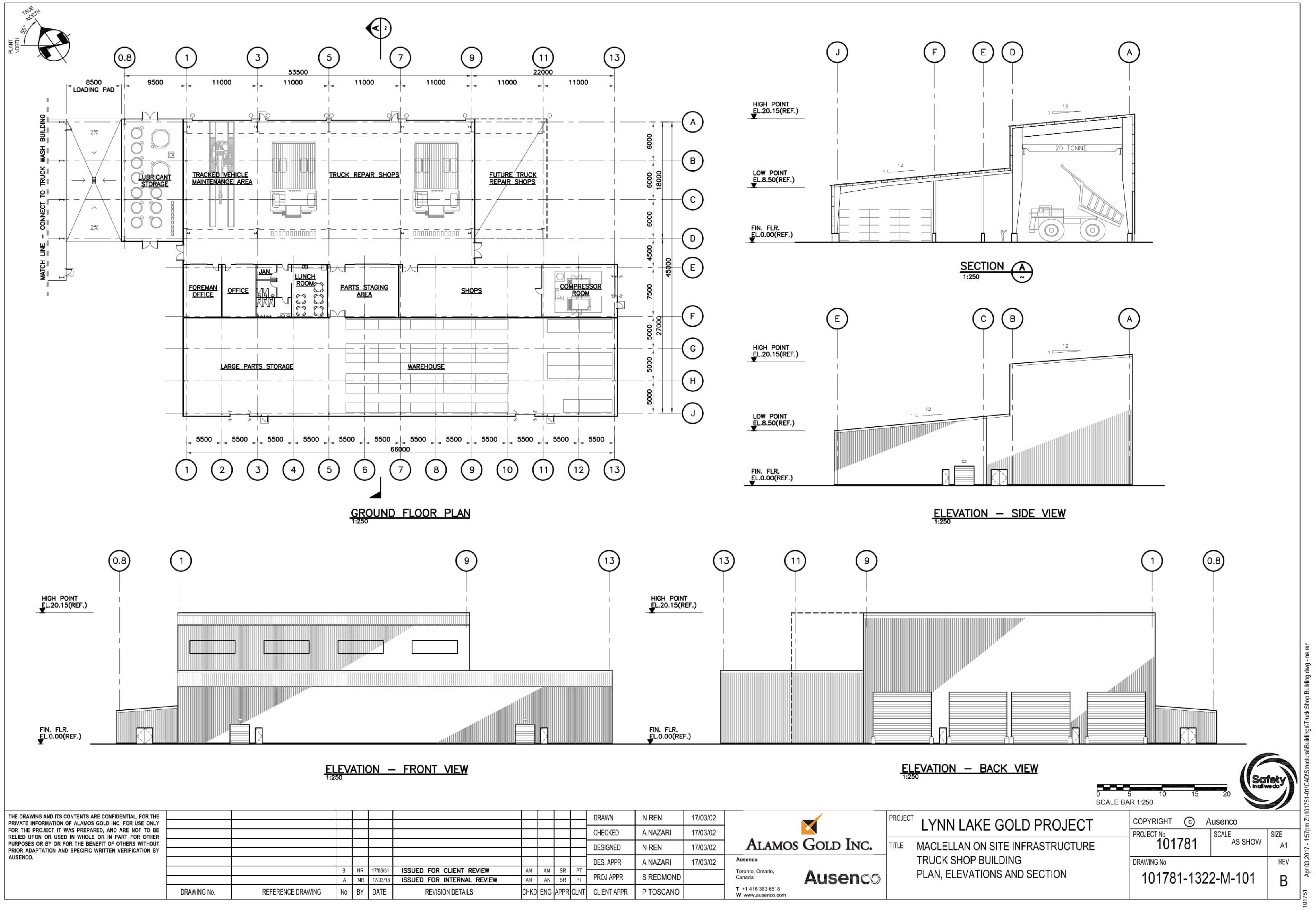

Figure 18-10: MacLellan Truck Shop/Warehouse Facility General Arrangement

Source: Ausenco (2017)

18.11.8 MacLellan Mine Truck Wash

The mine truck wash building at MacLellan will be 28 m (wide) x 30 m (long). The building will be a pre-engineered metal building with reinforced concrete foundations and concrete holding pond/dive-in sump, insulated roof, and wall metal cladding. The building will be located east of the process plant, and north of the mine tuck shop/warehouse building.

The building will be divided into two sections; a wash bay and a mud-settling pond. The truck wash system will have eight manually operated water cannons, four adjacent to the front and rear haul truck axels and four elevated 4 m above grade. The wash floor will be sloped to the mud-settling pond which is equipped with a skimmer for oil and grease removal and a water filtration system for continuous recycling of wash water.

18.11.9 Gordon Mine Truck Wash

The mine truck wash building in Gordon will have similar size, footprint and foundations as the mine truck wash at MacLellan, except for the structure which will be of fabric building type construction, comprised of steel frames as structural support with an architectural membrane cover.

18.11.10 Road Haul Truck Washing

Road haul trucks will be periodically washed in the MacLellan mine truck wash bay. Operating procedure will be developed to limit the interaction between mine and road going vehicles.

18.11.11 Assay Laboratory – MacLellan

A temporary assay laboratory will be installed at the MacLellan site during the pre-operations phase to provide gold assaying as well as environmental testing The temporary structure will be constructed of interconnected sea containers and house all of preparatory and analytical equipment, as well as dust collection systems, required to analyze for Au, Ag, Hg, As, Cu, sulphide sulfur and total sulphur. This equipment will be transferred to the permanent laboratory building, which will be of pre-fabricated modular construction place on concrete slab-on-grade. The permanent assay laboratory will be 12 m (wide) x 24 m (long) single story building located south of process plant.

18.11.12 Sulphur Burning Plant – MacLellan

The sulphur burning plant at MacLellan will be 10 m (wide) x 15 m (long), and located east of process plant, and will house the sulphur burning equipment used for the cyanide destruction system.

The building construction will be a pre-fabricated folding metal building type with concrete slab floor/foundations.

18.11.13 Powder Magazine – MacLellan

The powder magazine at MacLellan will be 8 m (wide) x 12 m (long), and will be used for storing, mixing and preparation of explosives. Explosives and accessories will be transported to the Gordon site on an as needed basis.

The building construction will be of pre-fabricated folding metal building type with concrete slab floor/foundations.

18.12 Fire Protection

Fire protection for both MacLellan and Gordon sites will be carried out by independent fresh/fire water tanks and ring mains.

Each fresh/fire water tank will be equipped with a standpipe which will ensure that a fire water inventory with at least 500 m^3 is always available. This volume is equivalent to two hours of fire water supply. Fresh water will replenish the tank when the automatic level control activates the fresh water pumping system.

The fire-water pumping skid consists of an electric pump and a backup diesel driven pump to assure continuous and adequate flow for fire-fighting. These pumps will deliver fire water to a network of sprinklers, hydrants and hose reels located throughout the plan sites. A smaller electrically drive jockey pump will maintain pressure in the fire water ring main.

Automated fire detection and fire protection will be installed in the crushing, grinding and process plant buildings, the interconnecting conveyor galleries and tunnels and Project infrastructure such as the warehouse and fuel storage areas and other areas as required. Firewalls and fire rated floors to limit the spread of fire, high temperatures and smoke will be provided as required. Emergency exists will be mounted in all buildings with appropriately illuminated exist signs.

Supplemental hand-held fire extinguishers, each suitable for each area, will be mounted throughout the buildings including MCCs, control rooms, transformers areas and fuel storage locations.

First response firefighting activities will be conducted by the mine rescue team utilizing on-site water trucks and EMS equipment. Primary firefighting activities will be handled by the local fire authority of the Town of Lynn Lake.

18.13 Site Security and Health Services

Alamos will be responsible for overall site security. The individual contractors will provide site security for their own respective assets during construction.

During construction, first aid facilities will be supplied by the EPCM contractor. First-aid personnel will provide transport to Lynn Lake hospital when required.

During operations, first aid facilities will be supplied by a dedicated first aid/mine rescue office in each of the site administration offices. Site security personnel will be trained as EMS first responders, and when required, provide transfer to Lynn Lake hospital.

18.14 Tailings Management Facility

The location of the TMF at the MacLellan site was selected in a scoping level siting study (Golder Associates Ltd., 2015a) and is located approximately 3 km northeast of the proposed MacLellan Pit and plant site areas, and 500 m north of Minton Lake, as shown on Figure 18-11.

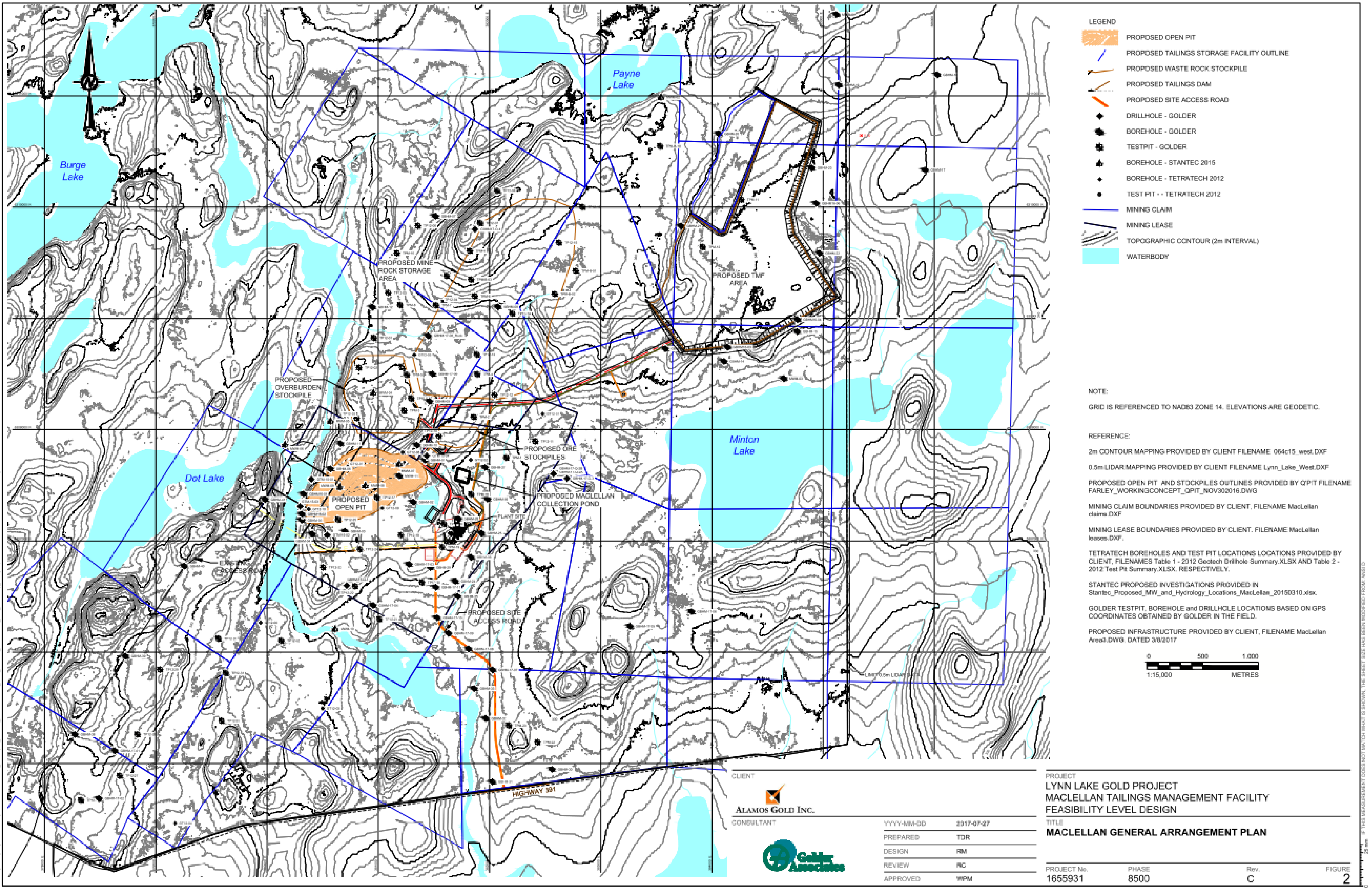

Figure 18-11: Lynn Lake Gold Project MacLellan Site

Source: Golder (2017)

18.14.1 Background Information

The site is located within the sporadic to discontinuous permafrost zone, where permafrost is typically found in 10% to 50% of the land area and in a relatively low seismic hazard region, known as the "stable central region." The mean annual air temperature is – 3.2 °C, ranging from a mean monthly temperature of 22 °C in June to – 29 °C in January. There is an annual average of 98 frost-free days. On average, there are 141 days with precipitation per year with an average annual precipitation of 478 mm (318 mm as rain and 160 mm as snow-water-equivalent). The site is vegetated with evergreen trees.

The proposed TMF location lies in a sub-watershed that drains to the south towards Minton Lake, which drains to Cockeram Lake to the south.

Three stages of subsurface investigations for the TMF were initiated by Golder during the course of the Feasibility Study, one in 2015, one in the summer of 2016 and another in the winter of 2017 (Golder Associates Ltd., 2017b).The subsurface conditions encountered in the low lying areas of the proposed TMF typically can be characterized by a peat / topsoil, overlying a thin layer of glaciolacustrine clay, overlying a layer of sand to silty sand, transitioning into a silty sand till at depth prior to encountering bedrock.

18.14.2 Operating Data

The sizing and the flow modelling for the TMF are based on the planned annual mill throughput averaged over 365 days per year. This is defined as the nominal value.

Operating data is required for the sizing of the TMF and flow (water balance) modelling. The key operating data for the sizing of the design of the TMF are listed below:

Mineral Reserve and Resource	30.0 Mt
Average planned annual mill production	2.56 Mt/y
LOM Tailings production	30.0 Mt
Tailings/ore ratio	100% by weight
Mill availability	92 % of the year
Factor of safety on design	1.1
Tailings discharge slurry density (S)	50 % solids by weight
Water content of the ore going into the mill	< 5 %
Slurry density of the tailings leaving the mill	96.6 %
Clean make-up water required in the mill (design)	0 m^3/h
Losses in the mill to spillage and evaporation	0 m^3/h

Golder has assumed:

Void ratio (e) of the deposited tailings	1.0 (volume of voids / volume of solids)
Tailings specific gravity	3.0

Key calculated data:

Life of mine	10.4 years

Dry Density of deposited tailings (γ_d)	1.50 t/m^3
Volume of deposited tailings	20.0 Mm^3

It should be noted that the tailings facility has been designed to allow for flexibility in the development of the TMF and its water management strategies should the amount of Mineral Reserve increase over the life of mine or water that accumulates within the facility freezes and takes away from the available capacity. In the long-term, it is expected that any water that freezes within the facility will eventually thaw but this might be after closure of the facility.

The TMF will receive tailings water, runoff from precipitation on the tailings collecting watershed as well as runoff from the mill site, transportation corridors, waste stockpiles and mine water.

18.14.3 Design Criteria

18.14.3.1 Dam Hazard Classification

The TMF dams were classified as “High” based the Canadian Dam Association Dam Safety Guidelines (Canadian Dam Association, 2014). It is understood that the Province of Manitoba defers to the guidelines developed by CDA to establish the dam safety design criteria. The guidelines provide recommendations on the classification of dams with respect to the consequences associated with a presumed dam failure. These consequences are to be evaluated in terms of incremental consequences over and above the consequences of the given event if the dam failure had not occurred.

The TMF dams are classified as “High” during the operations based on the following (Golder Associates Ltd., 2016f):

- Population at Risk – temporary workers in the tailings area;
- Loss of Life: ten or fewer;
- Environmental Impacts – significant loss of wildlife/fish habitat along Minton Lake and potentially further downstream; and
- Economic losses – associated with possible washout of PR 391. Large financial costs for cleanup and remediation downstream.

18.14.3.2 Inflow Design Flood

Based on CDA recommendations for the selection of an appropriate inflow design flood (IDF) (Table 18-6), the IDF of the tailings facility during operations should be determined based on an annual exceedance probability (AEP) of 1/3rd between the 1 in 1000-year flood and the probable maximum flood (PMF). For the passive closure phase, the IDF is raised to 2/3 between the 1 in 1000-year flood and the PMF.

Table 18-6 – CDA (2014) Minimum Inflow Design Floods for Dams and Dykes

Dam Class	Inflow Design Flood (IDF) Annual Exceedance Probability – Operating Phase	Inflow Design Flood (IDF) Annual Exceedance Probability –Closure Passive Care Phase
Low	1 / 100	1 / 1,000
Significant	1 / 100 to 1 / 1,000	1/3 between 1 / 1,000 and PMF[1]
High	1/3 between 1 / 1,000 and PMF[1]	2/3 between 1 / 1,000 and PMF
Very High	2/3 between 1 / 1,000 and PMF	PMF
Extreme	PMF[1]	PMF

Notes:

1. *PMF is the Probable Maximum Flood*

18.14.3.3 *Seismic Design*

Table 18-7 provides AEP for earthquakes for the various dam classes for both operations and closure as per CDA (2014).

Table 18-7 – CDA 2014 AEP Earthquakes for Dams and Dykes – Operation and Closure Phases

Dam Class	Annual Exceedance Probability (AEP) Earthquake – Operations Phase	Annual Exceedance Probability (AEP) Earthquake – Closure Passive Care Phase
Low	1 / 100	1 / 1,000
Significant	1 / 100 to 1 / 1,000	1 / 2,475
High	1 /2,475	1/2 between 1 / 2,475 and 1 / 10,000 AEP or MCE
Very High	1/2 between 1 / 2,475 and 1 / 10,000 AEP or MCE[1]	1 / 10,000 AEP or MCE
Extreme	1 / 10,000 AEP or MCE	1 / 10,000 AEP or MCE

Notes:

1. *MCE is the Maximum Credible Earthquake*

The design earthquake for a “High” classification tailings dam is the 1 in 2,475-year event (CDA, 2014). The peak ground acceleration (PGA) under this event is 0.031. For passive closure the design earthquake considered would be ½ between the 1 in 2,475-year event and the 1 in 10,000-year event. Through extrapolation of the data provided in Table 18-7 above, this would correspond to a PGA of 0.07.

18.14.3.4 *Tailings Pond Sizing*

The tailings pond will collect runoff and tailings water. The sizing of the tailings pond is described in Section 18.3.2.

18.14.3.5 Tailings Pond Discharge

A reclaim pump station will be used to pump water to the mill. Pumping requirements are discussed further in Section 18.3.8.

18.14.3.6 Dam Stability

Table 18-8 presents the minimum factors of safety for slope stability of the tailings dams to be adopted during design based on the CDA (2014) guidelines.

Table 18-8 – Factors of Safety for Dam and Dyke Slope Stability (CDA, 2014)

Loading Condition	Minimum Factor of Safety
Short-term (immediately after construction)	1.3
Long-term steady state (once the facility is operating)	1.5
Rapid drawdown (upstream slope where applicable)	1.2 to 1.3
Pseudo-static	1.0
Post-earthquake	1.2

18.14.4 Tailings Management Facility Design

18.14.4.1 Deposition Modelling

Table 18-9 provides details on the TMF capacity, maximum elevation of the tailings discharge point, the tailings pond volume, and the dam elevations for each stage.

Table 18-9 – Staged Tailings Facility Requirements with Dam Construction

TMF Details		Unit	Start-up	Year 6	Ultimate	Total
Year of Operation		Years	1-2	3-6	7-10.4	10.4
TMF Tailings Storage Capacity (per stage)		Mm^3	3.4	6.9	9.7	20.0
TMF Tailings Storage Capacity (total)		Mm^3	3.4	10.3	20.0	20.0
Tailings Pond Volume	Max. Operating	Mm^3	1.86	1.65	1.42	NA
	EDF	Mm^3	0.21	0.21	0.21	NA
	Total	Mm^3	2.07	1.86	1.63	NA
Dam Crest Elevation		m	347.0	348.7-352.5	350.4-359.0	NA

Figure 18-12, Figure 18-13 and Figure 18-14 provide the tailings deposition and TMF configuration for the starter, Year 6 and ultimate stages of the mine life, each of which are described below:

Start-up:

Tailings deposition will start in the TMF from the south and east dams. The TMF can accommodate 3.4 Mm^3 tailings during the first two years of mine production. The maximum discharge elevation at the end of this stage is 346.5 m.

From the onset of the deposition, a pond will form at the toe of the containment dam and be pushed away from the dam as deposition progresses. The pond will always be pushed towards the west end of the TMF, with water being pumped back to the mill on an as needed basis. The invert elevation of the spillway was estimated to be at 346 m while the TMF retains both maximum operating water volume and environmental design flow (EDF) volume.

At the onset of this stage, an internal dyke will be constructed along the west side of the TMF to separate the tailings from the pond for better water reclaim and management. The internal dyke is proposed to be constructed of inert waste rock in two stages: first, to an elevation of 347.5 m which is sufficient for the start-up and up until the end of Year 6; second, to an elevation of 350.5 m for the ultimate stage. The intake manhole will also require raising during the second stage.

Year 6:

Tailings deposition will continue in the TMF from the south, east and west ends. The TMF can accommodate an additional 6.9 Mm^3 of tailings for another four years up to elevation 353.5 m (10.3 Mm^3 total). The capacity of the tailings pond for this stage was considered to contain the maximum operating water volume (1.65 Mm^3) plus the EDF volume (0.21 Mm^3). The invert elevation of the spillway was estimated to be at 347.1 m while the TMF retains both maximum operating water volume and EDF volume.

Ultimate:

For the remainder of the mine life, the tailings will be deposited from the east, south and west dams while the tailings pond is pushed towards the internal dyke. The capacity of the tailings pond for this stage was considered to contain the maximum operating water volume (1.42 Mm^3) plus the EDF volume (0.21 Mm^3). The invert elevation of the spillway was estimated to be at 349.4 m while the TMF retains both maximum operating water volume and EDF volume.

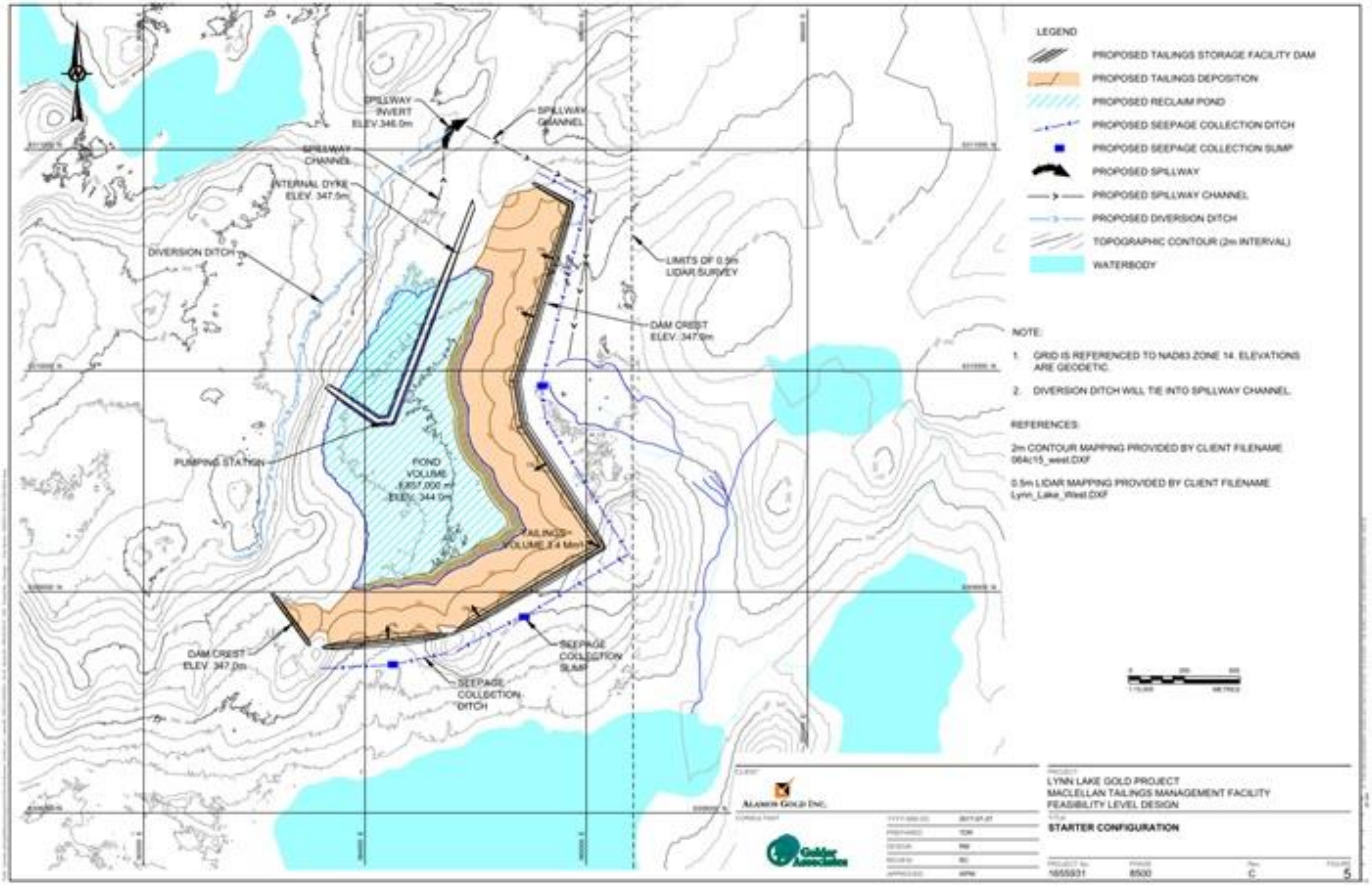

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Figure 18-12: Tailings Deposition and TMF Configuration for Start-up

Source: Golder (2017)

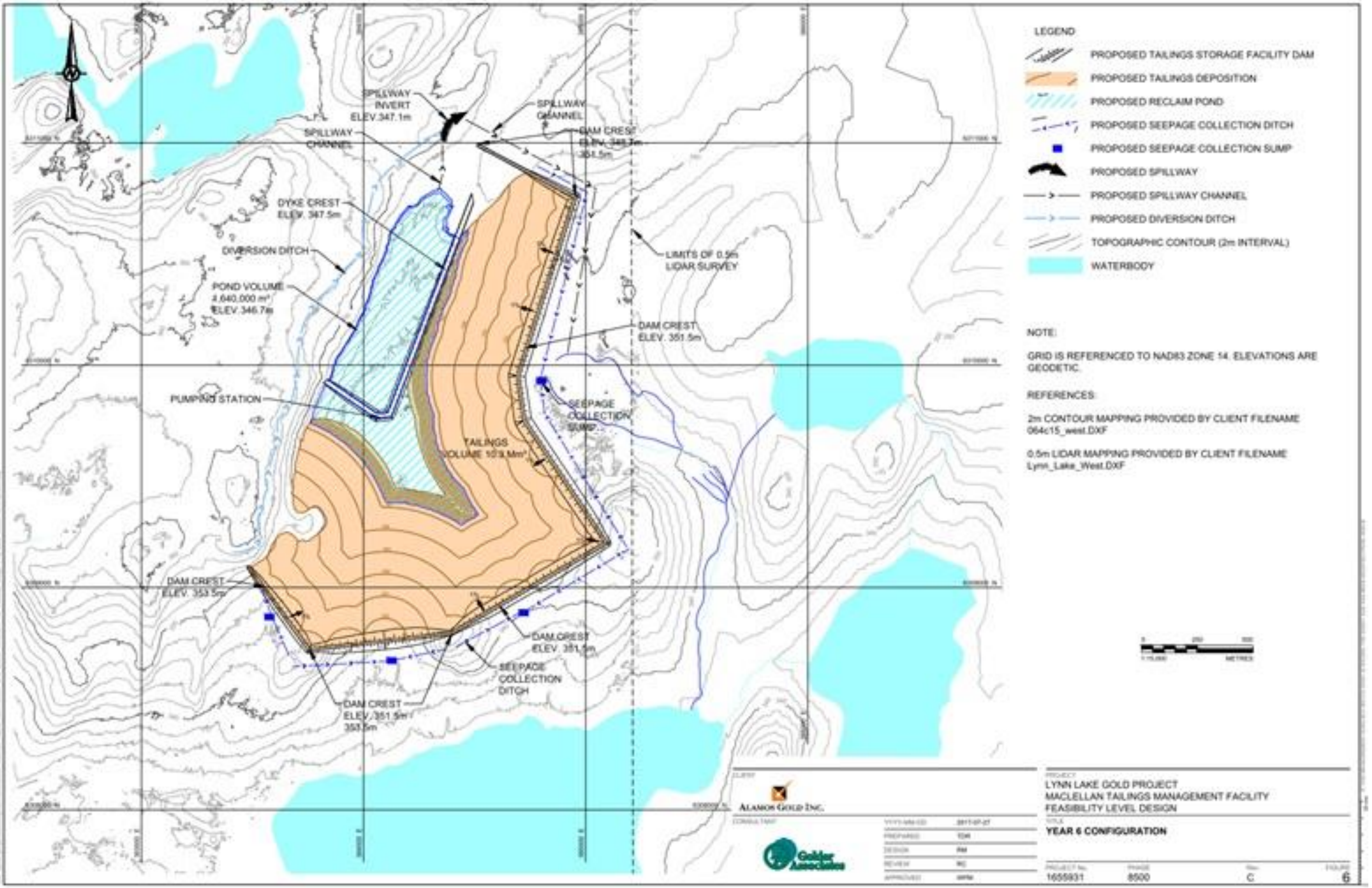


Figure 18-13: Tailings Deposition and TMF Configuration for Year 6

Source: Golder (2017)

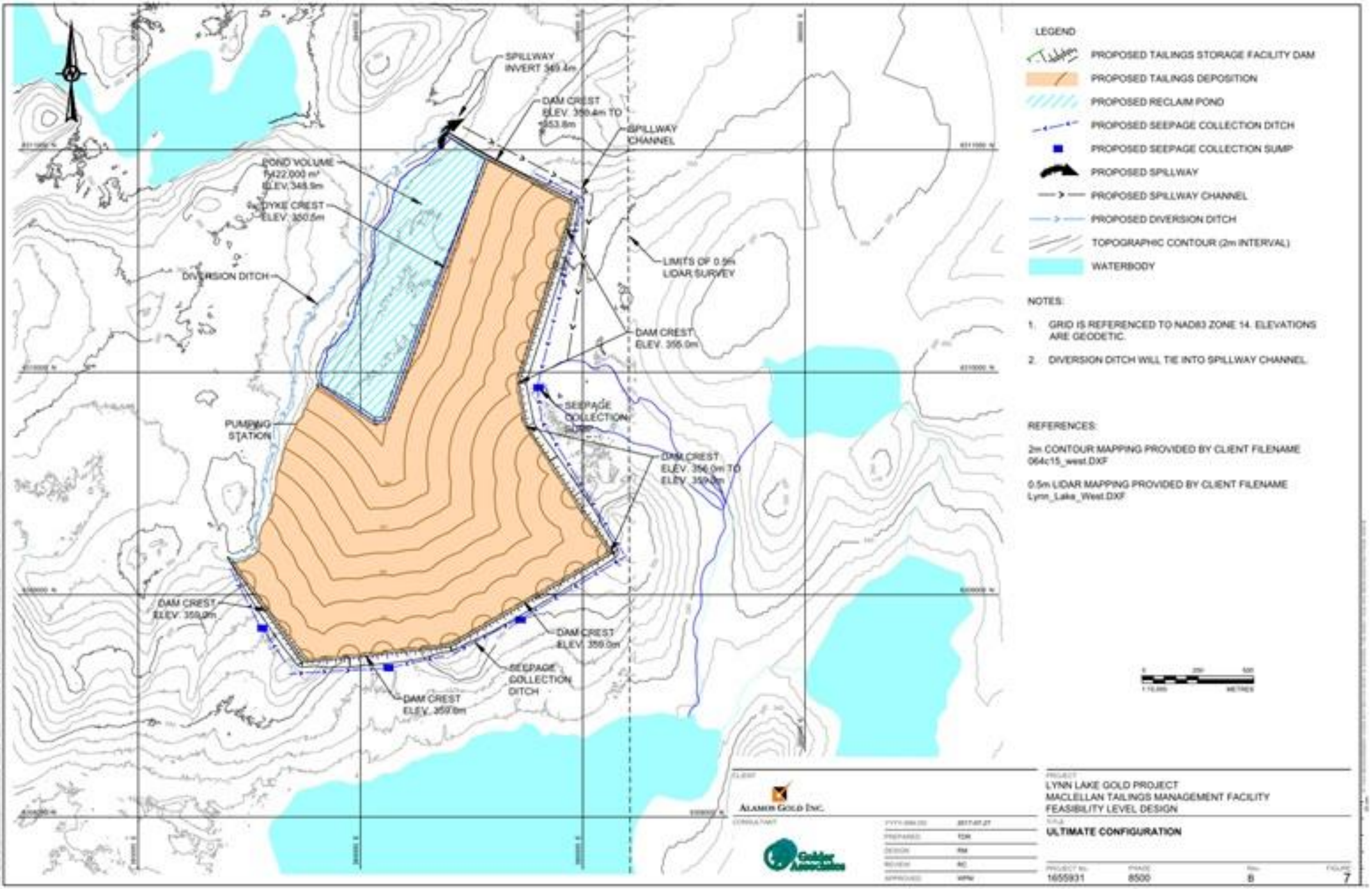


Figure 18-14: Tailings Deposition and TMF Ultimate Configuration

Source: Golder (2017)

18.14.4.2 TMF Dam Design

The key design levels derived from the tailings deposition and water management plans described above are summarized in Table 18-10. Figure 18-15 and Figure 18-16 illustrate the key design levels for the tailings pond for the start-up and ultimate configurations.

Table 18-10 – Key TMF Dam Design Levels

Key Design Levels	Unit	Start-up	Year 2	Year 6	Ultimate
Maximum Perimeter Dam Crest Elevation	m	347.0	353.5	357.5	359.0
Internal Dyke Crest Elevation	m	347.5	347.5	350.5	350.5
Emergency Spillway Invert Elevation	m	346.0	347.1	348.3	349.4
Maximum Flood Level	m	346.0	347.1	348.3	349.4
Minimum Operating Water Level	m	341.0	342.0	342.5	343.0
Maximum Operating Water Level (MOWL)	m	344.0	346.7	347.8	348.9
Maximum Tailings Level at Discharge Point	m	346.5	353.0	357.0	358.5

A typical cross-section of the TMF dams based on the design levels is shown on Figure 18-17. Except for the internal dyke, all perimeter dams have the same principal design features, as presented in Figure 18-17.

The tailings dams will be raised in stages to minimize the construction requirements over the life of mine and coincide with the storage capacity requirements.

- The first stage has been sized to contain two years of tailings deposition and will be required prior to start-up. This will involve the construction of the dams to elevation of 347.0 m. The dams will then be raised three more times to reach the ultimate elevation. All future raises of the dams are by the downstream method. The internal dyke will be constructed to an elevation of 347.5 m during the starter stage;
- The second stage of dam raise has been designed to contain four more years (from Year 3 to Year 6) of tailings deposition, with construction required during Year 2 of operations. This will involve the construction of the dams to a maximum elevation of 353.5 m;
- The third stage of dam raise is to contain tailings up to Year 9, with construction required during Year 6 of operations. This will involve the construction of the dams to a maximum elevation of 357.5 m. The internal dyke will be raised to its ultimate elevation of 350.5 m at this stage; and
- Prior to the end of Year 9, the ultimate dam will be constructed to a maximum elevation of 359.0 m to contain the remainder of the life of mine tailings.

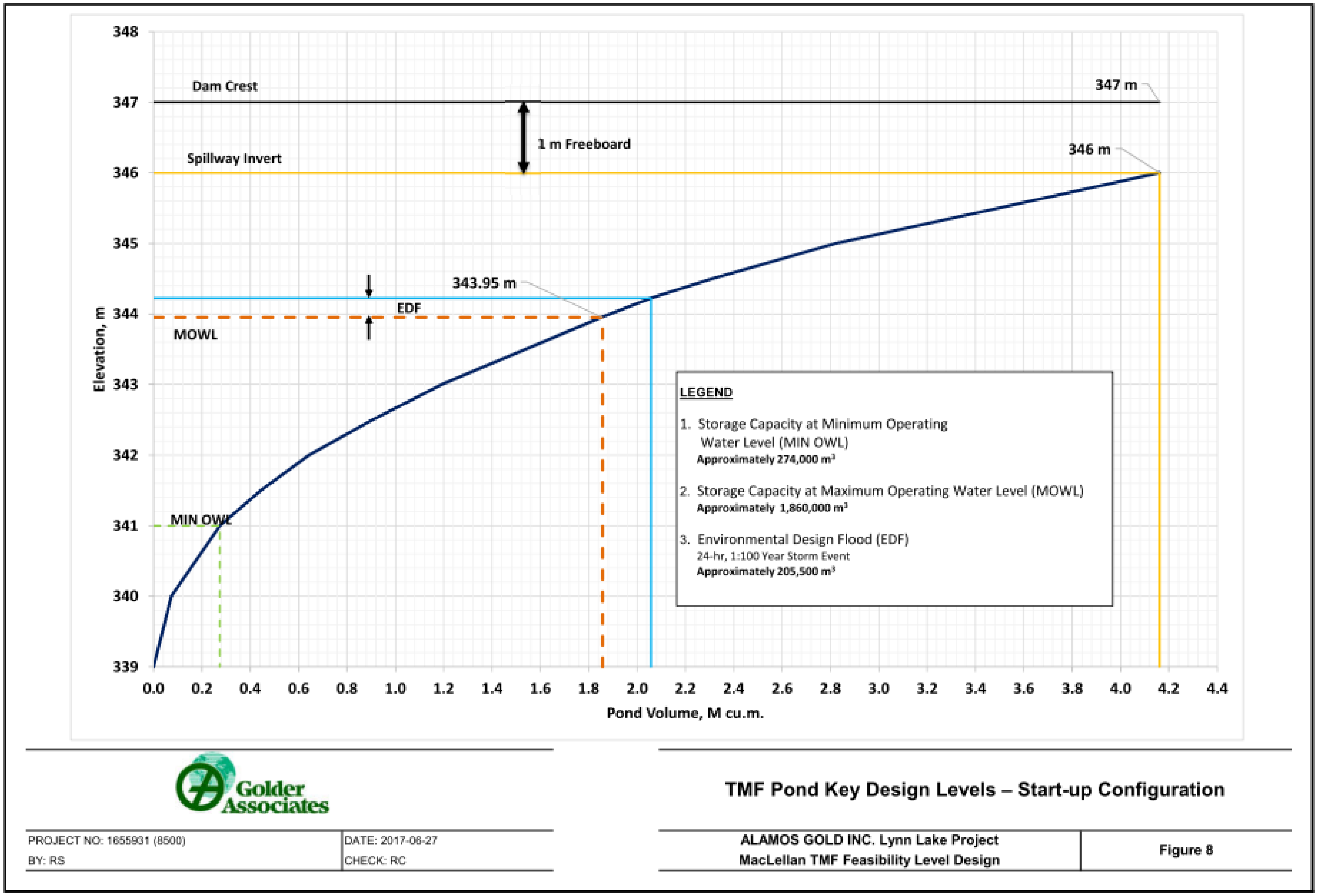


Figure 18-15: Key Design Levels for the Tailings Pond for Start-up Configuration

Source: Golder (2017)

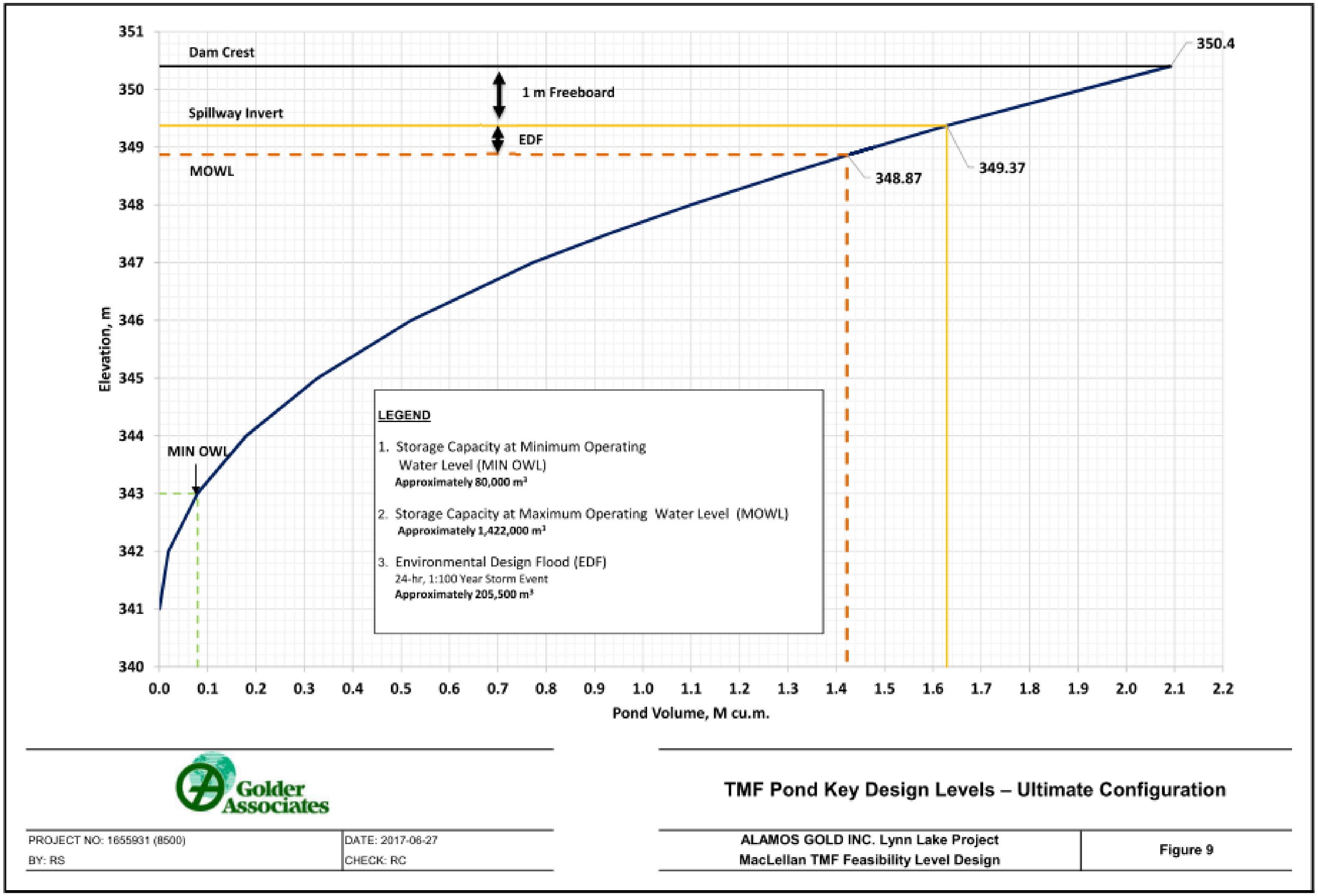


Figure 18-16: Key Design Levels for the Tailings Pond for Ultimate Configuration

Source: Golder (2017)

The perimeter tailings dams will be founded on bedrock or till. All tailings dams will have a crest width of 10 m with an upstream slope of 3H:1V and downstream slope of 2H:1V. The tailings dams will be constructed using rockfill materials with an upstream slope lined with a geomembrane (HDPE) as the main water retaining element. The geomembrane liner will be installed on a sand bedding (Zone 1) which will be underlain with 0.5 m of filter sand (Zone 2) and 1 m of transition material (Zone 3). To protect the liner, similar zoning is required on the upstream face of the slope before rockfill placement. The rockfill consists of a clean, non-acid generating, relatively free draining materials.

The liner will be anchored to the foundation via a concrete plinth, where the dam is founded on bedrock, or within an anchor trench, where the dam is founded on till. For the dams founded on bedrock, slush grouting will be provided on the bedrock, in combination with some dental concrete work to even out the bedrock surface. For the dams founded on dense till, a cut-off wall consisting of a cement-bentonite mixture will be provided along the entire length of the dam alignment to minimize the seepage. The depth and thickness of the cut-off wall is assumed to be 10 m and 1.5 m at this level of study.

At this current stage, bedrock injection grouting is considered not to be required based on the observed hydraulic conductivities. During the excavation of the foundation and exposing of the bedrock, it may be required to undertake injection grouting along some sections of the dam alignment if large fractures or fault zones are noted. Additional drilling during the detailed design stage should be used to confirm any bedrock grouting requirements.

A 0.3 m thick layer of road surfacing (well graded 2 inch minus material) will be placed and compacted along the dam crest to allow for light vehicle traffic during operations.

18.14.4.3 Emergency Spillway

An emergency spillway is required to prevent dam overtopping. The spillway will be raised progressively to correspond with raising of the TMF dams. The spillway will have a width of 5 m at the invert with 3H:1V side slopes and will be lined with 0.9 m of rip-rap erosion protection (D_{50} = 450 mm). The spillway outlet channel will also be lined with 0.9 m of rip-rap, with side slopes of 3H:1V. The outlet channel is provided with a stilling basin to promote energy dissipation and prevent erosion and scouring due to high flow velocities.

18.14.4.4 TMF Decant Design

A large capacity water pump is required to pump water from the tailings pond back to the process plant. The maximum pumping rate was estimated to be 260 m^3/h for both start-up and ultimate stages. The water pump needs to be located on the crest of the internal dyke, as shown on Figure 18-18. In this situation, water can get into the pumping well through intake pipes from both sides of the dyke where the tailings pond will be formed at early stages of deposition. The diameter of the intake pipes was estimated to be 350 mm based on the maximum pumping rate.

The pump station was designed to be constructed at the crest of the internal dyke at its start-up configuration with an elevation of 347.5 m. By the end of Year 6, the pump station needs to be raised as the internal dyke will be raised to its ultimate elevation of 350.5 m. It is anticipated that the bottom intake at the south side of the pump station will be plugged with tailings around the midpoint of the life of the TMF. A shut-off valve or gate should be installed between the manhole and intake pipe to prevent migration of tailings fines into the manhole and plugging of the pumps.

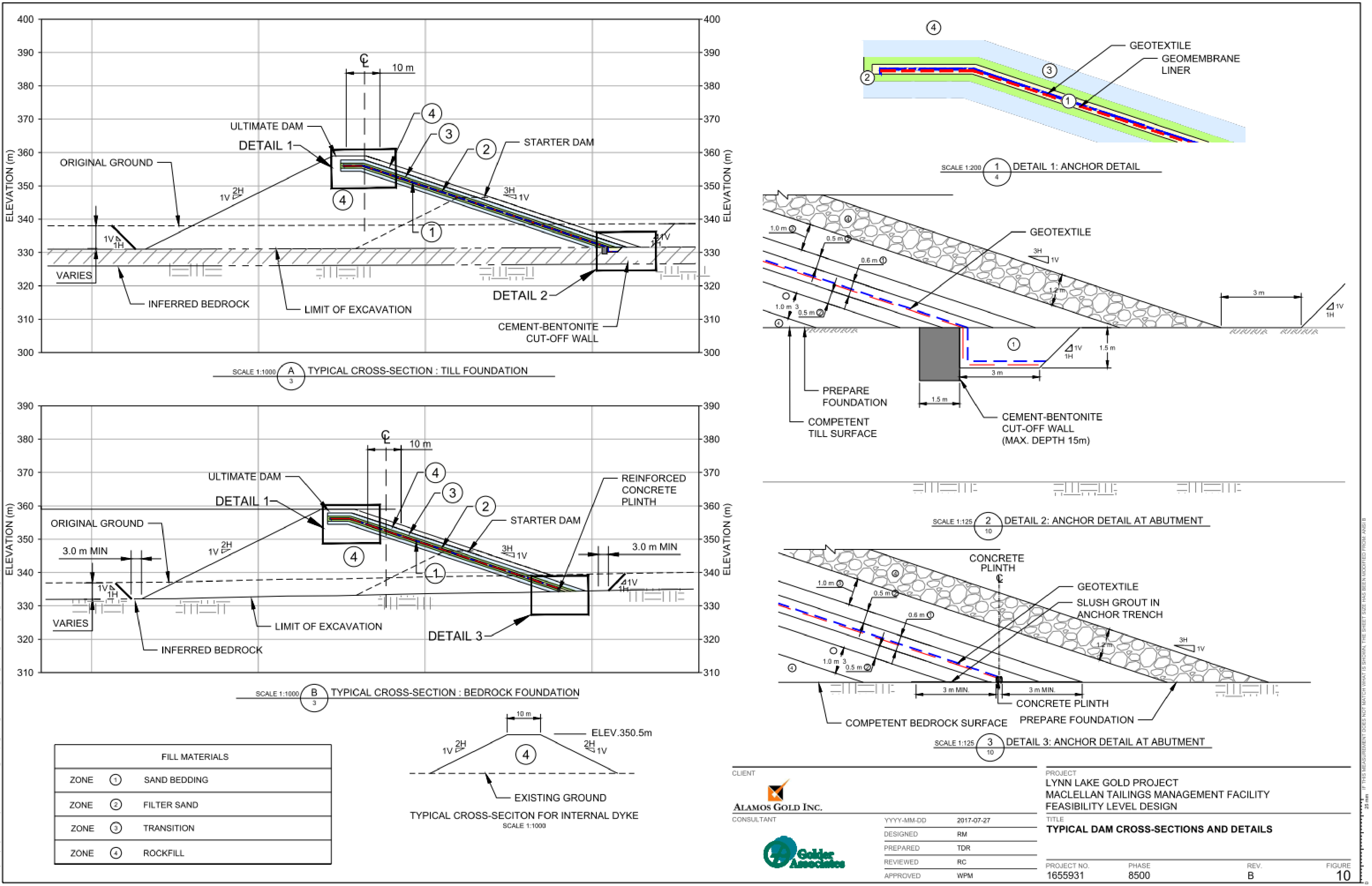
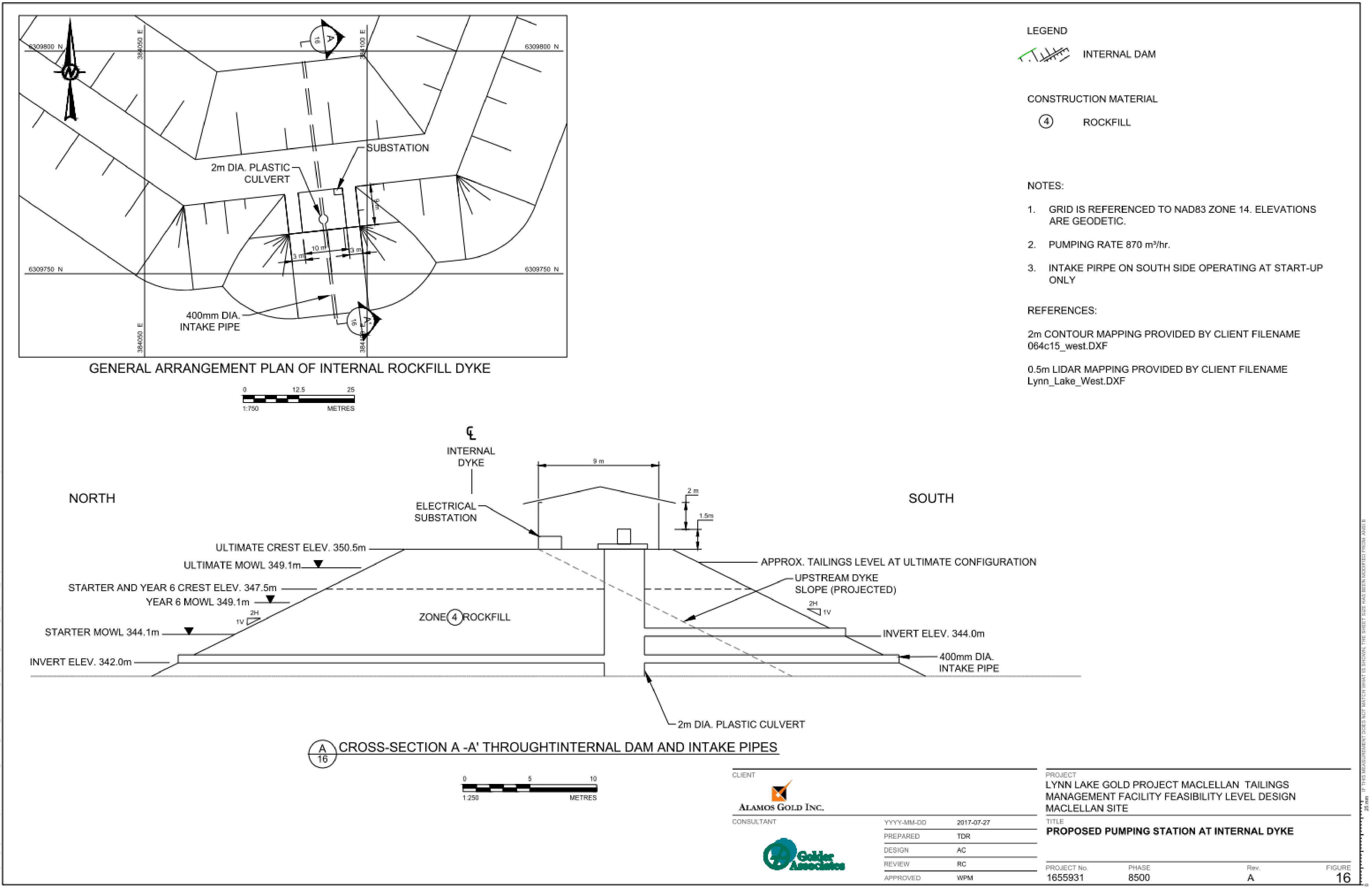


Figure 18-17: Typical TMF Dam Cross-Sections

Source: Golder (2017)



LEGEND

INTERNAL DAM

CONSTRUCTION MATERIAL

④ ROCKFILL

NOTES:

1. GRID IS REFERENCED TO NAD83 ZONE 14. ELEVATIONS ARE GEODETIC.
2. PUMPING RATE 870 m³/hr.
3. INTAKE PIRPE ON SOUTH SIDE OPERATING AT START-UP ONLY

REFERENCES:

2m CONTOUR MAPPING PROVIDED BY CLIENT FILENAME 064c15_west.DXF

0.5m LIDAR MAPPING PROVIDED BY CLIENT FILENAME Lynn_Lake_West.DXF

CLIENT
ALAMOS GOLD INC.

CONSULTANT
Golder Associates

YYYY-MM-DD	2017-07-27
PREPARED	TDR
DESIGN	AC
REVIEW	RC
APPROVED	WPM

PROJECT
LYNN LAKE GOLD PROJECT MACLELLAN TAILINGS MANAGEMENT FACILITY FEASIBILITY LEVEL DESIGN MACLELLAN SITE

TITLE
PROPOSED PUMPING STATION AT INTERNAL DYKE

PROJECT No.	PHASE	Rev.	FIGURE
1655931	8500	A	16

Figure 18-18: Proposed Tailings Pond Pumping Station Layout

Source: Golder (2017)

18.14.5 Slope Stability Analysis

Stability analyses were completed to assess the performance (i.e. factor of safety) of the dams under both static and dynamic (seismic) loading conditions. The factor of safety is the ratio of the forces tending to resist failure over the forces tending to cause failure. The minimum factors of safety discussed in Section 18.13.3.6 were used to assess acceptable dam performance for both static and pseudo-static loading conditions.

The slope stability analyses were carried out using the SLOPE/W module of the commercially available software package, GeoStudio 2007 Version 7.17, developed by GEO-SLOPE International Limited of Calgary, Alberta. The selected method of analysis was the Morgenstern-Price Method with a half-sine function to model the inter-slice forces. This method is based on limit equilibrium mechanics in which the solution satisfies both force and moment equilibrium.

The typical dam cross-sections shown in Figure 18-17 were used in the stability analyses. The dams will be founded on dense till or bedrock and constructed with rockfill. Therefore, liquefaction of the dam materials and the foundation soils under an Earthquake Design Ground Motion (EDGM) seismic event is not expected to occur.

For assessment of dam stability during operation, the analyses were completed with the tailings pond at its EDF level for the ultimate configuration. Given that the tailings pond will not be resting against the upstream side of any of the tailings dams, loading imposed by rapid drawdown was not analyzed.

Dam static stability for the passive closure condition was not analyzed given that no change in the dam cross-section will occur and water levels within the tailings are expected to decrease over time, thereby resulting in a more stable condition. However, the dam stability under the seismic loading for the passive closure scenario was completed with the tailings pond at its EDF level.

18.14.5.1 Stability Analysis Results

The results of the static and pseudo-static stability assessments are summarized in Table 18-11. Based on these results, the dams are expected to be stable under the loading conditions assumed in the analyses.

Table 18-11 – Dam Stability Assessment

Stability Assessment	Case	Duration	Calculated Minimum Factor of Safety	Required Minimum Factor of Safety
Static	Dam founded on dense till	Operation	1.90	1.5
	Dam founded on bedrock		1.65	
Pseudo-static	Dam founded on dense till	Operation	1.75	1.0
		Passive Closure	1.58	
	Dam founded on bedrock	Operation	1.53	
		Passive Closure	1.39	

Figure 18-19 and Figure 18-20 present the typical static and pseudo-static stability analysis results for the ultimate dam founded on till (South Dam at GBHM-14) at its EDF level, respectively.

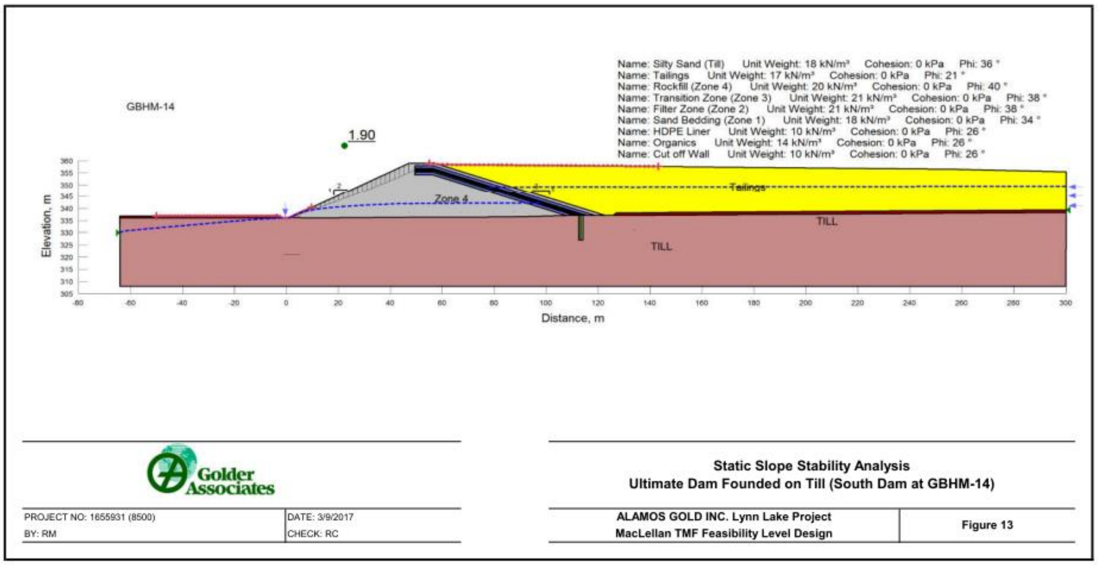


Figure 18-19: Typical Static Slope Stability Analysis

Source: Golder (2017)

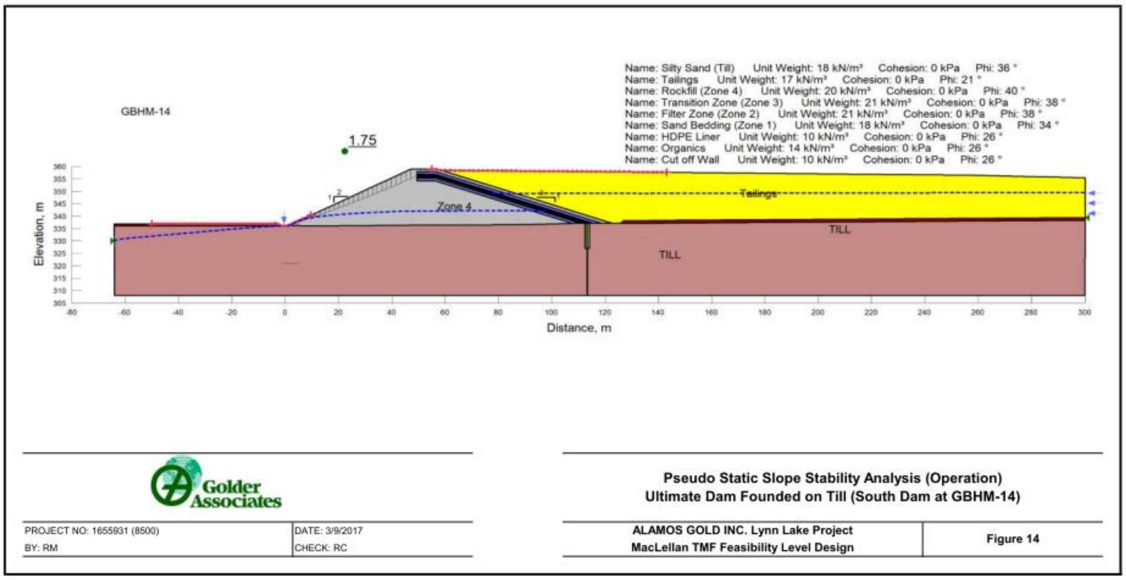


Figure 18-20: Typical Pseudo Static Slope Stability Analysis

Source: Golder (2017)

18.14.6 Construction Sequence

The general construction sequence for the TMF starter dams is anticipated to last about 18 months and would consist of the following:

- Clearing of entire TMF footprint (merchantable timber) – one month;
- Grubbing of surface and stripping/excavation of unsuitable soils along the dam foundation footprints with active dewatering to maintain stability of excavations – two months;
- Subgrade Preparation – includes installation of the soil-bentonite cut-off wall, compaction of any till foundation, and cleaning, slush grouting, and application of any dental concrete for bedrock foundations – three months;
- Dam fill placement and compaction – eight months;
- Installation of geomembrane liner (Likely done in two or three stages as the dam fill is being placed) – occurs in tandem with fill placement – likely two weeks per stage;
- Construction of decant structure likely occurring towards the end of the construction period – two months;
- Spillway and spillway channel construction – one month; and
- Wrap up of any leftover items and demobilization – one month.

Future dam raises are projected in Years 2, 6, and 9 with construction windows lasting approximately eight months for each.

18.14.7 Geochemistry

The tailings will consist of approximately 57% NPAG material produced from ores from both sites. ARD is not expected during operation. In the tailings pond, MMER limits could be exceeded for total cyanide, copper and nickel during operation. Un-ionized ammonia, mercury, silver, iron, cadmium, and arsenic might also be parameters of concern based on ageing tests. Seepage from the tailings may have concentrations exceeding the MMER limits for cyanide. The current plan is to collect and pump seepage back to the TMF. Cut-off walls under the TMF dykes will reduce the risk of groundwater contamination. The current plan is to design the TMF as a zero-discharge facility during operation. At closure, acidic conditions may develop in pockets of PAG tailings after eight or more years of exposure based on laboratory NP depletion rates. Under acidic conditions, potential exceedances of MMER limits for nickel and copper and Canadian Water Quality Guideline (CWQG) exceedances for arsenic, cadmium, copper, nickel, lead and zinc are possible. Even at neutral pH concentrations of arsenic and cadmium may exceed the Canadian Water Quality Guidelines for Freshwater Aquatic Life (CWQG-FAL) in runoff from exposed tailings. The risk of ARD and metal leaching from the tailings will be managed at closure using covers.

18.15 Accommodation

Four accommodation strategies were evaluated to provide for the FIFO workforce during the two-year pre-production and 10.4-year operational life of the Project:

1) Mine site camp within the MacLellan Property;

2) Town site camp within the Town of Lynn Lake;

3) Re-use of existing town housing stock within the Town of Lynn Lake; and

4) Integrated camp plus triplex units within the Town of Lynn Lake.

The preferred option, 2 – town site work camp within Town of Lynn Lake, was selected as it provides the greatest schedule advantage and low capital and operating costs. Additionally, the FIFO personnel can be better integrated within the Lynn Lake community and have access to local business and services.

The camp will be located within the Town of Lynn Lake before the construction phase to be used continuously throughout pre-production and operations. Camp infrastructure will be tied into existing Town facilities (power grid, potable water and wastewater) with Alamos partially contributing to infrastructure upgrades and improvements where required.

The camp will be supplied, erected, and operated (including catering, housekeeping and recreation) by a third party for the LOM.

18.15.1 Accommodation Size

The camp facility will be comprised of a 300-bed purchased camp plus 100 bed leased camp for the two-year pre-production period. Camp bed count was determined by the peak FIFO workforce size in pre-production and operation phases. It was assumed that 5% of the total workforce will be sourced locally, thus not requiring accommodation. Total labour force as well of full time equivalents (FTE) expected on-site at a given time (accounting for FIFO rotations) is summarized below:

- Pre-productions peak – 541 total labour force with 308 FTE in camp (92 spare beds);
- Operations peak – 519 total labour force with 253 FTE in camp (47 spare beds); and
- Operations TMF lift – 531 total labour force with 266 FTE in camp (34 spare beds).

Short term workforce spikes associated with TMF construction/lifting are in line with the peak operations workforce.

18.15.2 Housing Description

Accommodations are proposed to be provided in 30 to 44 person dormitories and are single occupancy. Rooms are approximately 6.5 m^2 with a mix of ensuite or jack-and-jill style shared washrooms. Kitchen/dining, recreational, and laundry facilities are shared, and linked to the dormitories through climate controlled corridors.

18.16 Mobile Equipment and Light Vehicles

A variety of plant mobile equipment is required for the operation.

A list of the plant mobile equipment for the LLGP is shown in Table 18-12. Mine mobile equipment is covered in Section 16.15.

Table 18-12 – Processing Plant Mobile Equipment List

Equipment	Capacity	Number
Single Cab 4WD Pickup	-	6
Twincab 4WD Hiab Truck	8 t	1
Bus[1]	22 seat	1
All Terrain Crane	40/50 t	1
Plant Forklift	4 t	2
Warehouse Forklift	4 t	1
Front End Loader	10 t	1
Skid-Steer Loader	1 t	1
Tip Truck	3 t	2
Telescopic Handler	3 t	1
Boom/Scissor Lift	-	2
Fusion Butt Welder	90/315	1
Portable Compressor	-	1
Diesel Welder	-	2
Portable Pumps Generator	200 kVA	1
Gravel truck	10 m^3	2
Graders	12 ft	2
Water Truck	15,000 L	1
Potable water Trailers	5,000 L	4
Franna Crane	15 t	1
Yard Crane – 20t Hydraulic	20 t	1
Pick-n Carry Mobile Crane	-	1
Backhoe	11 t	1
Forklift/Telescopic Handler	20 t	1

Notes:

1. *Site bus, principal employee transport between accommodation and mine sites is contracted*

19 MARKET STUDIES AND CONTRACTS

19.1 Market Studies

No market studies have been conducted by Alamos or its consultants in relation to the gold and silver doré which will be produced by the LLGP. Gold and silver are freely traded commodities on the world market for which there is a steady demand from numerous buyers.

19.2 Contracts

There are no refining agreements or sales contracts currently in place that are relevant to this Technical Report.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

This section provides a summary of the results of the environmental studies and consultation efforts to support the Project to date. Environmental baseline studies were initiated for the Project in March 2015 and were used to identify environmental constraints during the development of the layouts and designs for the Project. This includes consideration of siting and layout of Project infrastructure as well as consideration of design alternatives from an environmental management and approvals perspective.

The environmental setting described herein refers to a “General Project Area”, which does not have strictly defined spatial boundaries but generally includes the area indicated on Figure 5-1. This area is generally representative of the local environmental context for the Project.

The General Project Area (Figure 5-1) supports two communities: the Town of Lynn Lake and the Black Sturgeon Reserve. These communities are connected to each other by PR 391 which runs southeast from Lynn Lake to Thompson, Manitoba. Consultation with Indigenous communities and other stakeholders (community members, agencies, interested parties) is an integral part of the Project and has been undertaken throughout Project planning to date and will continue as the Project progresses.

No relevant regional environmental study(s) is (are) known to have been conducted in the General Project Area. There are similarly no known applicable plans pertaining to water use (including groundwater), resource management, or conservation.

20.2 Environmental Approval Process

There are several federal and provincial regulatory requirements that may apply to the Project, including an environmental assessment (EA) and other environmental permitting obligations.

Under the *Canadian Environmental Assessment Act*, 2012 (CEAA 2012), federal EAs are possibly required for ‘designated projects’ consisting of one or more physical activities specified in the *Regulations Designating Physical Activities* (the Regulations). The Canadian Environmental Assessment Agency (the CEA Agency) is responsible for the administration of federal EAs for metal mines under CEAA 2012 and has formally determined that a federal EA is required for the Project.

Provincially, the Classes of Development Regulation (CD Regulation) under *The Environment Act* of Manitoba identifies which developments must undergo a provincial EA and obtain a licence in accordance with the Act prior to construction, alteration, or operation. According to Section 3(5) of the CD Regulation, the Project is classified as a Class 2 development. This class applies to mines and milling facilities. The Project may also involve one or more water development and control activities that are considered Class 2 developments under Section 3(9) of the CD Regulation, such as stream channel alterations that affect fish mobility and fish habitat. The Project is not expected to involve any of the water development activities listed as Class 3 development triggers under Section 4(4) of the CD Regulation.

The Environmental Approvals Branch of Manitoba Sustainable Development (MSD) has advised that it considers the proposed Project activities at the Gordon and MacLellan sites to constitute separate “developments” that will require separate licences under *The Environment Act* of Manitoba. The Environmental Approvals Branch will allow both sites to be assessed in a single Environmental Impact Statement (EIS) under the provincial EA process.

A Project Description was submitted to the CEA Agency on July 19, 2017 to initiate the federal EA process under CEAA 2012 and inform the provincial EA process under *The Environment Act* of Manitoba. The Project Description was prepared in accordance with the federal *Prescribed Information for the Description of a Designated Project Regulations, the corresponding Guide to Preparing a Description of a Designated Project under the Canadian Environmental Assessment Act, 2012* (the Guidelines; the CEA Agency 2015), and in consideration of the provincial *Environment Act Proposal Report Guidelines* (EAP Guidelines; MCWS 2015). On November 01, 2017, following a public comment period for the Project Description and Draft Guidelines, Final Guidelines for the Preparation of an Environmental Impact Statement were issued for the LLGP by the CEA Agency.

A single EIS document is currently being prepared that will be submitted to satisfy federal and provincial EA requirements. The Project design, including implementing the identified mitigation measures, is not anticipated to cause significant adverse environmental effects, including effects from accidents and malfunctions, effects of the environment on the Project and cumulative effects. No issues have been identified to date that are expected to materially affect the ability of Alamos to extract minerals from the Project. This will be confirmed through the environmental assessment and permitting phases of the Project development and may require additional design modifications or mitigation measures to be implemented. As discussions with agencies are still ongoing, and the environmental assessment for the Project has just been initiated, the extent of changes to the Project cannot fully be captured in this report.

20.3 Baseline Data

The description of the environment provided below is a summary based on extensive environmental baseline studies that have been completed to support the Project to date. The purpose of these studies was to characterize the natural, social, economic, cultural and built aspects of the environment that may be potentially impacted by the Project. The corresponding technical data reports that accompanied these studies are identified in Table 20-1. Some high-level information obtained during these studies has been incorporated into the description of the environmental setting below.

Table 20-1 – Technical Data Reports Associated with Environmental Baseline Studies

Technical Data Report	Data Sources
Acoustic Baseline	Review of desktop information and collection of field data (i.e., noise monitoring)
Air Quality Baseline	Review of desktop information and collection of field data (i.e., active [continuous] particulate monitoring and dustfall monitoring)
Ambient Lighting Baseline	Review of desktop information and collection of field data (i.e., light survey and photographs)
Amphibian Baseline	Review of desktop information and collection of field data (i.e., habitat assessment, amphibian survey, and visual encounter survey)
Benthos and Sediment Baseline	Review of desktop information and collection of field data (i.e., periphyton sampling, phytoplankton sampling, zooplankton sampling, benthic invertebrate sampling, sediment sampling, and water quality sampling)
Bird Baseline	Review of desktop information and collection of field data (i.e., breeding bird survey, common nighthawk survey, waterbird survey, raptor nest survey, and barn swallow survey)
Climate and Meteorology Baseline	Review of desktop information
Fish Habitat, Distribution, and Tissue Analysis Baseline	Review of desktop information and collection of field data (i.e., fish habitat assessment, fish population sampling, fish spawning survey, and fish tissue sampling)
Geochemistry Baseline	Review of desktop information and characterization of materials, which will be exposed by the project (ore, waste rock, overburden, high-and low-grade ore and solid tailings derived as part of a metallurgical study)
Heritage Resources Baseline	Review of desktop information and collection of field data (i.e., pedestrian transects, shovel testing, and photographs)
Human Health and Ecological Risk Assessment Baseline	Review of desktop information and collection of field data (i.e., soil sampling, vegetation sampling, and small mammal sampling)
Hydrogeology Baseline	Review of desktop information and collection of field data (i.e., borehole drilling, hydraulic response testing, water level monitoring, and water quality monitoring)
Hydrology Baseline	Review of desktop information and collection of field data (i.e., snow survey, hydrometric monitoring, levelling survey, channel geometry survey, and bathymetric survey)
Mammal Baseline	Review of desktop information and collection of field data (i.e., aerial track survey, camera trap survey, ground-based tracking survey, beaver lodge survey, and bat survey)
Socio-Economic Baseline	Review of desktop information
Soil and Terrain Baseline	Review of desktop information and collection of field data (i.e., soil and terrain inspections and aerial reconnaissance)
Vegetation Baseline	Review of desktop information and collection of field data (i.e., ground plot survey [vegetation type characterization], wetland classification, and rare plant survey)

Other Project-specific studies that have been or are being undertaken include a Traditional Knowledge/Traditional Land and Resource Use study, and a Transportation Assessment. Further modelling programs for select subject areas (e.g., atmospheric dispersion, surface water quantity and quality and groundwater flows) are being completed in support of the EA.

The EIS will provide further details regarding the methods and results of studies completed in support of the Project.

20.3.1 Atmospheric Environment

20.3.1.1 Air Quality

Baseline dustfall measurements at the Gordon and MacLellan sites are well below dustfall objectives from Ontario and British Columbia (Manitoba does not have a dustfall objective). Average particulate matter baseline concentrations ($PM_{2.5}$ and PM_{10}) are also well below the Canada-Wide Standard for $PM_{2.5}$ (30 µg/m^3) and the Manitoba Ambient Air Quality Guideline for PM_{10} (50 µg/m^3) (CCME 2011; MSD 2005) at the sites and Black Sturgeon Reserve, although the presence of air emissions from forest fires biases this baseline (e.g., during June and early July 2015). Existing air quality is reflective of the remote location of the Project and the current lack of industrial activities in the area. Existing dust levels are attributed to traffic on unpaved roads and other human activities such as the use of wood stoves and open fires.

20.3.1.2 Ambient Sound

Isolated hourly sound levels (L_{eq}) in a representative remote, unpopulated region of the General Project Area (i.e., Gordon site) were found to range between 22.7 decibel A-weighting (dBA) and 39.1 dBA during the day and between 20.3 dBA and 41.7 dBA during the night. The baseline acoustic environment in remote areas is characterized by wind noise, occasional aircraft flyovers, vegetation rustling, wildlife (birds) and insect noise. Elevated noise levels observed at night are attributed to wildlife activity. The Lynn Lake Airport does not receive regularly scheduled commercial flights; however, aircraft flyovers from occasional air charter flights contribute to baseline ambient sound in remote regions of the General Project Area.

Isolated hourly L_{eq} results for a sparsely populated region of the General Project Area (i.e., Black Sturgeon Reserve) ranged between 33.0 dBA and 46.6 dBA during the day and between 23.2 dBA and 45 dBA during the night. The acoustic environment in sparsely populated areas is characterized by local activities (vehicle traffic, as well as general and recreational human activity and children playing), occasional aircraft flyovers, vegetation rustling, dog barking, and wildlife and insects noise.

Noise monitoring in a representative rural area (i.e., the cottage area within Burge Lake Provincial Park located west of the MacLellan site) identified isolated hourly L_{eq} results ranging between 31.3 dBA and 49.5 dBA during the day and between 22.7 dBA and 43.3 dBA during the night. The acoustic environment in rural areas is characterized by residents' activities, local traffic, watersport and recreational activities, occasional aircraft flyovers, vegetation rustling, wildlife, insects, and water ripple noise. Additional human sources of baseline noise are also related to traffic along PR 391 (which traverses the southern portion of the MacLellan site).

20.3.1.3 Ambient Light

The ambient light environment within the General Project Area is typical of light levels in remote towns and villages at higher latitudes. Baseline measurements are consistent with other small towns and villages where light pollution is typically not a priority for control. Sky glow is routinely influenced by the presence of Aurora Borealis (i.e., northern lights).

Dark sky is available within a few kilometers of Lynn Lake and the Black Sturgeon Reserve. The light that affects these communities is the light that is generated within them, not by the overlap of other sources, such as industry, outside of the urban areas.

20.3.2 Water Resources

20.3.2.1 Surface Water Hydrology

The General Project Area lies within four subwatersheds of the Granville Lake River Watershed: Hughes River, Lower Keewatin River, Lower Lynn River, and Cockeram Lake.

Surface water around the Gordon site drains southward into the Hughes River, via Swede and Ellystan Lakes, which in turn discharge into Barrington River and Southern Indian Lake on the Churchill River. Around the MacLellan site, water flows south into the Keewatin River and southeast through Cockeram Lake and Sickle Lake before discharging into Granville Lake on the Churchill River, upstream of Southern Indian Lake.

Gordon Lake is located at the top end of the watershed and west of the historical mine area that formerly drained eastward to Farley Lake via Gordon Creek. As part of historical mining activities, a diversion channel was constructed between Gordon and Farley lakes, north of the historical East and Wendy open pits. The East and Wendy pits are flooded and are not directly connected to the diversion channel or Gordon or Farley Lakes.

The Keewatin River, Lynn River, Goldsand Lake, and Cockeram Lake are some of the largest waterbodies in the Lower Keewatin River, Lower Lynn River, and Cockeram Lake subwatersheds. The subwatershed on the west side of the MacLellan site flows towards the Keewatin River which ultimately converges with the Lynn River before entering Cockeram Lake.

Five lakes surround the proposed TMF at the MacLellan site, including Payne Lake (which drains into the Keewatin River) and Lobster, Minton, and two unnamed lakes (which drain into an unnamed river that ultimately discharges to Cockeram Lake in the south). The Keewatin River flows southeast from Cockeram Lake, through Sickle Lake before discharging into Granville Lake on the Churchill River, upstream of Southern Indian Lake.

Results from the two years of surveys indicate that the highest flows typically occur during the spring period in response to snowmelt in the Project area. Peak flow sometimes occurs later in the melt season in response to rainfall events. In the 2016 water year, the highest recorded flow was 26.7 m^3/s in the Keewatin River on July 16, 2016. The lowest recorded flows occurred during the late winter, prior to the onset of spring snowmelt.

The regional hydrological analysis determined regional annual mean discharge, annual runoff, annual peak flows, monthly flows and distribution, and monthly low flows. The annual runoff calculated at the five regional stations ranged from 156 mm to 203 mm with an average of 178 mm. The average annual runoff from the regional stations (178 mm) was used to estimate mean annual discharge for stations on the Keewatin and Lynn rivers. The annual mean discharge values measured at the local stations were comparable to the values calculated in the regional analysis.

Evidence of beaver activity was noted throughout the General Project Area, particularly in streams and at lake outlets. In these areas, beaver dams have reduced flow and increased water levels upstream.

20.3.2.2 Groundwater Hydrogeology

The conceptual hydrostratigraphic understanding of the General Project Area consists of overburden, deposited through a series of glacial processes that overlies bedrock. Overburden thickness is controlled by bedrock topography with thicker overburden deposits associated with topographic lows in the bedrock surface and thin to absent overburden in areas of bedrock topographic highs. Overburden geology was characterized as glaciolacustrine sediments overlying glacial sand till, which generally overlies bedrock. Isolated pockets of glaciofluvial sediments were observed. A thin veneer of organic soils was observed at surface with thicker deposits observed in low lying areas.

Groundwater flow in the General Project Area is strongly influenced by topography, which results in localized groundwater flow from topographic highs with groundwater discharge to wetland areas or surface water features. The topography is controlled by top of bedrock, which is irregular as a result of north dipping beds that strike east-west. Groundwater flow across the MacLellan site is generally toward the southeast with some radial flow in the area of the proposed open pit. At the Gordon site, groundwater flow is controlled by a topographic high located south of the proposed open pit that results in radial flow. Groundwater flow converges on the proposed open pit at the Gordon site.

The hydraulic conductivity of overburden ranged over three orders of magnitude with higher hydraulic conductivity estimates associated with glaciolacustrine nearshore deposits (10^{-6} m/s to 10^{-5} m/s) and lower hydraulic conductivity estimates associated with glaciolacustrine offshore deposits (10^{-7} m/s to 10^{-6} m/s) and till (10^{-7} m/s to 10^{-6} m/s). The hydraulic conductivity of the bedrock decreases with depth, with the upper portions being the most transmissive due to increased weathering and/or fracturing. The shallow bedrock had the greatest variation in hydraulic conductivity estimates, over five orders of magnitude (10^{-8} m/s to 10^{-3} m/s), and is reflective of the variation in weathering and frequency and extent of bedrock fractures. At depths greater than 50 m below the top of bedrock, hydraulic conductivity estimates varied over three orders of magnitude (10^{-8} m/s to 10^{-6} m/s) and were generally three to four orders of magnitude lower than the upper range observed in the shallow bedrock. Packer testing and groundwater flow modelling have suggested that bedrock within the vicinity of the east and Wendy Faults at the Gordon site are an order of magnitude higher than the surrounding bedrock.

Groundwater quality in the General Project Area was compared to the Manitoba Water Quality Standards, Objectives, and Guidelines (MSOG) for drinking water and the Canadian Drinking Water Quality Guidelines (CDWQG). Background water quality was good and generally met the MSOG and CDWQG except for dissolved arsenic, iron, and manganese. At one background monitoring well, the concentration of sulphate was one to two orders of magnitude greater than other background monitoring wells and is reflective of natural mineralization and/or reduction-oxidation reactions that may be occurring. Groundwater quality associated with historical mining activities at the MacLellan site exceeded the drinking water guidelines for iron and manganese, similar to background, as well as sulphate and uranium. At the Gordon site, groundwater quality associated with historical mining activities exceeded the drinking water guidelines for dissolved As, Mn, and Fe, similar to background, as well as sulphate and dissolved Cd.

Groundwater quality in the General Project Area was also compared to the more stringent MSOG and Canadian Water Quality Guidelines (CWQG) for the protection of freshwater aquatic life (FAL). Background groundwater quality met the MSOG-FAL and CWQG-FAL except for F and dissolved As and Fe in addition to dissolved Cu and Al at the MacLellan site and dissolved Zn at the Gordon site. Groundwater quality associated with historical mining activities at the MacLellan site exceeded the MSOG-FAL and CWQG-FAL for the same parameters as background groundwater quality in addition to dissolved Al and dissolved U. At the Gordon site, groundwater associated with historical mining activities exceeded the MSOG-FAL and CWQG-FAL for the same parameters as background groundwater quality in addition to dissolved Cd, Ni, Se, U and Zn.

20.3.2.3 Geochemistry

The following summarizes the results of the geochemical characterization completed for each mine material associated with the Project:

Overburden

At both sites, overburden is not expected to generate ARD based on kinetic testing and monitoring of historical overburden storage. Leaching of F, Al and Cu above CWQG-FAL is predicted by kinetic tests, but no exceedances of CWQG for any of these parameters was observed in association with the historical overburden storage at the Gordon site.

Mine Rock

Approximately 72% and 78% of the mine rock from the MacLellan and Gordon open pits, respectively, will be NPAG. The rest is represented by PAG and uncertain rock having a risk to generate ARD after closure. The risk of ARD will be addressed through the blending of PAG and NPAG rock. There is no evidence of ARD observed downstream of the historical rock storage sites at the Gordon site, where PAG and NPAG were blended and covered with overburden and soil.

A high metal leaching potential was identified for As and a moderate leaching potential was identified for Al, Cd, Mo, and Cu in rock from the MacLellan site based on kinetic tests. Mine rock from the Gordon site showed a moderate leaching potential for F, As, Se, Cd, Cr, Al, and Cu. Among these elements, only As and Se exceeded the CWQG-FAL in ponds located downstream of historical rock storage areas at the Gordon site. At these locations, CWQG exceedances were also observed for Fe, NH_3, and NO_2. Ongoing water quality modeling will refine the list of potential parameters of concern and determine if metal leaching from waste rock should be addressed.

Ore

Ore from MacLellan and Gordon open pits will contain 52% and 66% of NPAG materials, respectively. The rest is represented by PAG and uncertain materials. ARD is not likely to be an issue with blended ore stockpiles during operation, considering minimum ARD onset time of 14 years compared to the residence time of ore in stockpiles.

The ore from the MacLellan site has a high leaching potential for As based on humidity cell tests. A moderate leaching potential was determined for Al, F and Ag at the Gordon site and for Ag, Cd, Pb, and Al at the MacLellan site. Water treatment during operation is not currently predicted based on the water quality modeling that is currently ongoing. At closure, any remaining ore will require management to prevent future ARD and metal leaching.

Tailings

The geochemistry of the tailings (ARD and ML potential) is discussed in Section 18.14.7.

20.3.3 Aquatic Environment

20.3.3.1 Water Quality

Most of the lakes near the Gordon and MacLellan sites are shallow (less than 4 m deep) and do not stratify during the summer. Background surface water quality generally reflects geochemistry of the Precambrian Shield. Lakes and streams are typically low in dissolved ions (< 80 mg/L total dissolved solids), soft (hardness < 75 mg/L as $CaCO_3$), and neutral to slightly acidic in pH. Some parameters (e.g., dissolved oxygen, pH, total P, Al, Cr, and Fe) are naturally elevated and occasionally do not meet water quality guidelines.

At the outlet of Gordon Lake, the 2015 to 2017 dataset showed no notable changes in water quality from background conditions. This suggests that drainage from the inactive mine site (i.e., surface runoff from existing mine rock and overburden storage dumps and seepage from the adjacent former open pit) does not affect water quality in Gordon Lake. Water quality data indicate elevated levels of some metals and other ions (e.g., alkalinity, hardness, specific conductance, Ca, Cl, Mg, K, Na, sulphate, As, Cu, Fe, Ni, and U) in the existing open pits and in Farley Lake compared to background concentrations in the Gordon site area, however, concentrations of these parameters were similar to background by Swede Lake, the next lake downstream from Farley Lake. In general, the Hughes River subwatershed, within which the Gordon site is located, has the following parameters in concentrations that exceed MSOG-FAL and/or CWQG-FAL: total and dissolved organic carbon, total P, Fe, and Al. These exceedances are likely the result of lithology in the case of aluminum and sulphate, the presence of mineralized rock in the case of Cu and Ni, and the proliferation of beaver dams, muskeg bogs, and low relief in the case of organic carbon.

The inactive MacLellan site does not appear to affect water quality in the Keewatin River, as there were no identifiable increased concentrations of water quality parameters between the sites upstream and downstream of the site (upstream of the Lynn River confluence). Sulphate and chloride concentrations and Al, Cu, Ni, Cd, Co and Zn concentrations were higher in Eldon Lake, the Lynn River, in the Keewatin River downstream of the Lynn River, and in Cockeram Lake (the first lake downstream of the inactive MacLellan mine site and the unrelated former East Tailings Management Area on the Lynn River) than in other lakes and streams not downstream from these facilities, including the Keewatin River upstream from the Lynn River confluence. Mean Cu, Ni, Fe, and Zn concentrations were higher than CWQG-FAL in Eldon Lake, the Lynn River, and in Cockeram Lake. These exceedances are generally attributable to past mining activity near Lynn Lake. Other guideline exceedances in the MacLellan site area, including total P, Fe, and Al, reflect background conditions.

20.3.3.2 Fish and Fish Habitat

Based on the results of field surveys conducted in 2015 and 2016, a total of 17 fish species are known to occur in the lakes and streams near the Project mine sites (Table 20-2). Small-bodied fish species are most prevalent in streams and small, shallow lakes including: brook stickleback (*Culaea inconstans*), ninespine stickleback (*Pungitius pungitius*), log perch (*Percina caprodes*), trout perch (*Percopsis omiscomaycus*), emerald shiner (*Notropis atherinoides*), spottail shiner (*Notropis hudsonius*), longnose dace (*Rhinichthys cataractae*), lake chub (*Couesius plumbeus*), and slimy sculpin (*Cottus cognatus*). Large-bodied fish species are more prevalent in larger, deeper lakes and include: northern pike, walleye, yellow perch, lake whitefish (*Coregonus clupeaformis*), burbot (*Lota lota*), cisco (*Coregonus artedi*), white sucker (*Catostomus commersoni*), and longnose sucker (*Catostomus catostomus*). Larger lakes, such as Cockeram Lake, typically support a greater diversity of fish and fish habitat than smaller lakes in the General Project Area. Northern pike are the most widespread large-bodied species in the lakes of the General Project Area, while brook stickleback are the most widespread small-bodied species in the lakes and streams.

Table 20-2 – Fish Species Known to Occur in Waterbodies near the Project Mine Sites

Waterbody	Location	Fish Species Confirmed to be Present During Field Surveys Conducted in 2015 and 2016
Gordon Site		
Farley Lake	Adjacent and to the east (downstream) of proposed open pit	Northern pike, yellow perch, white sucker, and brook stickleback
Gordon Lake	Adjacent and to the west (upstream) of proposed open pit	White sucker and brook stickleback
MacLellan Site		
Cockeram Lake	South (downstream) of proposed mine infrastructure and PR391	Northern pike, walleye, yellow perch, lake whitefish, white sucker, trout perch, emerald shiner, spottail shiner, lake chub, log perch, and ninespine stickleback
Dot Lake	West of and across Keewatin River from proposed open pit	Brook stickleback
East Pond	Adjacent to and between proposed open pit and proposed ore milling and processing plant	Brook stickleback
Keewatin River	Adjacent and to the west of proposed mine infrastructure	Northern pike, yellow perch, lake whitefish, cisco, burbot, white sucker, longnose sucker, lake chub, longnose dace, trout perch, brook stickleback, and slimy sculpin
Lobster Lake	Northeast of proposed TMF	Brook stickleback and northern pike
Minton Lake	South (downstream) of proposed TMF	Brook stickleback and northern pike
Payne Lake	North of proposed mine rock storage area and proposed TMF	Brook stickleback

No aquatic species of conservation concern (SOCC) have been documented or are expected in the General Project Area based on known fish species distributions, including those listed as special concern, threatened, or endangered under the federal *Species at Risk Act* (SARA; Government of Canada 2016b), recommended for listing under SARA by the Committee on the Status of Endangered Wildlife in Canada (COSEWIC; 2016), listed as threatened or endangered under Manitoba's *The Endangered Species and Ecosystems Act* (MB ESEA) (Government of Manitoba 2016), or listed as S1-S3 by the Manitoba Conservation Data Centre (MB CDC; 2015).

Most lakes in the General Project Area are shallow with soft substrates (e.g., sand or muck). Hard substrates (e.g., boulders or cobbles) are less common but present in some locations. Aquatic vegetation and cover in the littoral zone is abundant in most of the lakes in the General Project Area.

The Gordon site is located in the headwaters of the Hughes River watershed, and no large rivers flow through the study area. Farley Creek, the outlet of Farley Lake, is the largest stream potentially affected by the Project. Short cascades and a proliferation of beaver dams present seasonal or temporary barriers to fish passage in several of the streams near the Gordon site.

The Keewatin, Cockeram, and Lynn rivers and their tributaries connect the lakes near the MacLellan site. The Keewatin River is the largest river in the General Project Area, connecting Goldsand and Burge Lakes upstream of the MacLellan site to Cockeram Lake downstream of the mine site. The Lynn River is a tributary to the Keewatin River upstream of Cockeram Lake.

Stream habitats are generally low gradient, interspersed with short cascades that do not generally create barriers to migration. Beavers are active in the General Project Area and their dams present seasonal or temporary barriers to fish passage on several tributaries to the rivers and lakes.

20.3.3.3 Fish Tissue

Metals concentrations in northern pike muscle and whole body tissues from the Gordon and MacLellan sites are generally below the MMER and Health Canada guidelines for protection of aquatic life and human consumption. This includes total mercury for which average concentrations were below the human health guideline of 0.5 µg/g (wet weight).

Concentrations of selenium in northern pike muscle and whole body tissues were approximately one order of magnitude lower than the 4 µg/g (wet weight) guideline used in British Columbia (BCMoE 2014).

20.3.3.4 Sediment Quality

Arsenic and chromium concentrations exceed federal or Manitoba sediment quality guidelines in Farley and Gordon lakes, respectively. Iron and aluminum concentrations were also elevated in Gordon and Farley lakes compared to upstream reference sites. Sediment metal concentrations were otherwise below guideline values at the Gordon site.

Elevated concentrations of Cr, Cu, As, and Zn that exceed federal and/or Manitoba sediment guidelines have been measured in sediment at the inflow of the Keewatin River into Cockeram Lake. This location is downstream of the MacLellan site, downstream of the former East Tailings Management Area (located on the banks of the Lynn River), and downstream of other historical mining activities and other anthropogenic influences from the Town of Lynn Lake. These data suggest past contamination of sediments in North Cockeram Lake. Other metals, for which no guidelines currently exist, such as Al and Fe, were also elevated at several locations, including Goldsand Lake located upstream of the MacLellan site. This suggests some natural enrichment from the surrounding geology.

20.3.4 Terrestrial Environment

20.3.4.1 Terrain and Soils

The Project is located within the Churchill River Upland Ecoregion of the Boreal Shield Ecozone (Smith et al. 1998). It falls under the South Indian bedrock plateau subdivision of the Kazan Upland (Bostock 1970) which covers about 35,000 km^2 of mostly hilly, till veneered bedrock terrain, and intervening low areas of organic terrain (Klassen 1986). The General Project Area terrain ranges from level to moderately sloping, with most slopes ranging from 0 to 15%.

Within the Churchill River Upland Ecoregion, Dystric Brunisols are the dominant soils on sandy acidic till, while Gray Luvisols are dominant on well to imperfectly drained clay deposits (Smith et al. 1998). Granitic rock outcrops are co-dominant in the area. Appreciable areas of shallow and deep organic Mesisols, Fibrisols and Cryosols are associated with basin bogs, peat plateau and veneer bogs (Smith et al. 1998). Gray Luvisols, and to a lesser extent Static and Turbic Cryosols, are common on clayey lacustrine deposits along the Churchill River and around Southern Indian Lake, while Eutric Brunisols occur on silty fluvioglacial ridges and on calcareous loamy till.

At the Gordon site, the dominant soils are well drained, coarse-textured Eluviated Dystric Brunisols of the Fay Lake soil series and very poorly drained Terric Fibric Organic Cryosols of the Wuskwatim soil series. At the MacLellan site, the dominant soils are imperfectly drained, coarse-textured Gleyed Eluviated Dystric Brunisols of the Hat Lake soil series and very poorly drained, Terric Fibric Organic Cryosols of the Wuskwatim soil series.

The Permafrost Distribution Map of Canada (Heginbottom et al. 1995) indicates that the Project is located within the sporadic to discontinuous permafrost zone, where permafrost is typically found in 10% to 50% of the land area.

20.3.4.2 Vegetation and Wetlands

The Project area is characterized by black spruce-dominated forests on mineral soils and in poorly drained peatlands. Tamarack is typical on wetter peatland sites, while drier sites are forested white birch (*Betula papyrifera*), jack pine, and occasionally white spruce (*Picea glauca*). Jack pine stands occur on upland sites, while white birch is found throughout the Ecodistrict. Field surveys documented 200 plant species within the General Project Area.

Common dandelion (*Taraxacum officinale*) and quack-grass (*Elymus repens*) are considered noxious weeds in Manitoba and were observed in the General Project Area (Government of Manitoba 2015).

The Project is in the High Boreal wetland region which is characterized by permafrost and non-permafrost wooded bogs and patterned fens (Halsey et al. 1997). An estimated 37% of the High Boreal wetland region is covered in wetlands, comprised of (in order of dominance): permafrost wooded bogs, wooded fens with internal lawns, patterned open fens, non-patterned open fens, and wooded fens with internal lawns, along with small percentages of swamps and marshes. Eleven wetland types have been recorded within the General Project Area.

SOCC are those species listed as special concern, threatened, or endangered under SARA (Government of Canada 2016b), recommended for listing under SARA by COSEWIC (2016), listed as threatened or endangered under MB ESEA (Government of Manitoba 2016), or ranked as S1-S3 by the MB CDC (2015). The General Project Area is located within the known range of 23 plant SOCC. Four of these are known to occur within the General Project Area: quillwort (*Isoetes lacustris*), small water-lily (*Nymphaea tetragona*), northern woodsia (*Woodsia aplina*), and shrubby willow (*Salix arbusculoides*). None of the 23 plant SOCC are listed under SARA, nor are any of Manitoba's SARA-listed plant SOCC expected to occur based on the habitat types found in the General Project Area.

20.3.4.3 Amphibians

Three species of amphibian have the potential to breed within the General Project Area: boreal chorus frog (*Pseudacris maculata*), wood frog (*Rana sylvatica*), and northern leopard frog (*Lithobates pipiens*). Baseline field surveys confirmed the presence of breeding habitat for boreal chorus and wood frogs; both are widely dispersed throughout the General Project Area. Northern leopard frog is a SOCC (listed as special concern under SARA) and the historical range of northern leopard frog includes the General Project Area; however, there are no recent records of their presence and none were observed during baseline studies.

20.3.4.4 Birds

Based on the Manitoba Breeding Bird Atlas (MB BBA 2016), 198 bird species have the potential to breed in the General Project Area. Of these, 62 are waterbirds, four are upland game birds, 18 are raptors, and 114 are passerines (i.e., songbirds) or near-passerines (e.g., woodpeckers). Common waterbird species observed during baseline studies were mallard (*Anas platyrhynchos*), ring-necked duck, Canada goose (*Branta canadensis*), and common loon (*Gavia immer*). Common songbirds were swamp sparrow (*Melospiza Georgiana*), rubycrowned kinglet (*Regulus calendula*), Tennessee warbler, dark-eyed junco (*Junco hyemalis*), and yellow-rumped warbler (*Setophaga coronate*).

The General Project Area is located within the known range of nine bird SOCC. Of these nine bird SOCC, three are confirmed breeders in the area: common nighthawk (*Chordeiles minor*, listed as threatened under SARA and MB ESEA), olive-sided flycatcher (*Contopus cooperi*, listed as threatened under SARA and MB ESEA) and barn swallow (*Hirundo rustica*, listed as threatened by COSEWIC). Trumpeter swan (*Hirundo rustica*, listed as endangered under MB ESEA), horned grebe (*Podiceps auritus*, listed as special concern by COSEWIC), and rusty blackbird (*Euphagus carolinus*, listed as special concern under SARA) may occur based on the availability of suitable breeding habitat; however, yellow rail (*Coturnicops noveboracensis*, listed as special concern under SARA), short-eared owl (*Asio flammeus*, listed as special concern under SARA and threatened under MB ESEA), and bank swallow (*Riparia riparia*, listed as threatened by COSEWIC) are less likely to occur based on lack of suitable habitat in the General Project Area.

20.3.4.5 Mammals

Baseline data indicates the General Project Area is home to American marten, American red squirrel (*Tamiasciurus hudsonicus*), beaver (*Castor canadensis*), black bear, Canadian lynx (*Lynx canadensis*), fisher (*Martes pennant*), grey wolf (*Canis lupus*), mink (*Neovison vison*), moose, red fox (*Vulpes vulpes*), river otter (*Lontra canadensis*), snowshoe hare (*Lepus americanus*), weasel (*Mustela sp.*), wolverine (*Gulo gulo*), eastern red bat (*Lasiurus borealis*), hoary bat (*Lasiurus cinereus*), little brown myotis (*Myotis lucifugus*), various small rodents (e.g., voles). Moose and black bear are some of the important game species harvested by local resource users.

The General Project Area is located within the known range of five mammal SOCC. Two of the five mammal SOCC (i.e., little brown myotis, listed as endangered under SARA and MB ESEA, and wolverine, listed as special concern by COSEWIC) have been documented in the General Project Area. Northern myotis was not detected during bat baseline surveys, yet it has the potential to occur in the General Project Area due to the availability of suitable bat roosting and foraging habitat.

The General Project Area lies within the MSD Kamuchawie Caribou Management Unit. MSD has yet to delineate any boreal woodland caribou (Rangifer tarandus caribou, listed as threatened under MB ESEA and SARA) herd ranges or provide an estimate of herd size in this management unit (Manitoba Boreal Woodland Caribou Management Committee 2015; pers. comm. 2015a). There are, however, boreal woodland caribou herds delineated south of the General Project Area and south of the Churchill River (COSEWIC 2011).

The barren-ground caribou (Rangifer tarandus groenlandicus, listed as a threatened by COSEWIC) range extends to approximately 45 km north of the General Project Area, Beverly and Qamanirjuaq Caribou Management Board (BQCMB) . Discussions with local resource users and MSD, and results of previous studies (MinGold Resources Inc. 1989 and Tetra Tech 2013) indicate it is unlikely that boreal woodland or barren-ground caribou occur in the General Project Area (pers. comm. 2015a; pers. comm. 2015b; pers. comm. 2015c).

Subsequent conversations with the Regional Director for MSD (Northeast Region which includes Lynn Lake) and the BQCMB Senior Biologist confirmed that the current caribou range does not extend near the Project area (including the regional assessment area that will be included in the EA).

20.3.5 Human and Socio-Economics

20.3.5.1 Socio-Economic Context

There are two population centres in the General Project Area: the Town of Lynn Lake and Black Sturgeon Reserve. The Town of Lynn Lake has approximately 650 residents (Lynn Lake Mayor and Council 2014; pers. comm. 2015d). The population of the Black Sturgeon Reserve is 38, all of whom are among the 430 registered members of Marcel Colomb First Nation (MCFN) (INAC 2017). The Town of Lynn Lake was built in the mid-20th century, primarily to serve the mining industry. Since the closure of the Black Hawk mine, the region has sought to develop its tourism industry, which is based largely around fishing and hunting.

The Town of Lynn Lake is accessible by PRs 391, 394, 396, 397 and 399. PR 391 connects the Town of Lynn Lake and Black Sturgeon Reserve with the Town of Leaf Rapids and City of Thompson. PR 391 also provides access to all-weather gravel access roads to both the Gordon and MacLellan sites. There is currently no rail service to the Town of Lynn Lake. The Lynn Lake Airport is accessed by fishing charters, RCMP, health services, and mining-related activities (pers. comm. 2015e).

The Town of Lynn Lake provides solid waste services for residents and businesses. Waste from the town and Black Sturgeon Reserve is disposed at the Lynn Lake Landfill. Water for the town comes from West Lynn Lake. The water treatment plant has incurred operational problems that have led to water quantity and quality issues (pers. comm. 2015d). A boil water advisory has been in effect for the Town of Lynn Lake since 2012 (MSD 2017).

MCFN operates its own water treatment plant and sewage lagoon on the Black Sturgeon Reserve. Both were built recently and are in good working condition (pers. comm. 2015f). Hughes Lake is the source of drinking water supplies for MCFN's treatment plant.

Education services within the General Project Area are provided through Frontier School Division, Area 1, which provides both in-class teaching services as well as distance education for senior years and career programs (Frontier School Division). West Lynn Heights School serves the Town of Lynn Lake and Black Sturgeon Reserve.

The General Project Area is in the service delivery area for the Northern Health Region. The Lynn Lake Hospital is in the Town of Lynn Lake. The hospital shares health care resources with the Leaf Rapids Health Centre. For medical emergencies and specialist appointments, residents are transported by medevac to Thompson or Winnipeg.

The Town of Lynn Lake has 24-hour emergency medical services with one ambulance capable of providing patient transport to the Thompson General Hospital. The Lynn Lake RCMP is responsible for initial search and rescue at the outset of an emergency, with a specialized search and rescue team deployed to follow up. Lynn Lake, Leaf Rapids, Thompson, South Indian Lake, Nelson House and Cross Lake have volunteer search and rescue teams. The Lynn Lake Fire Department is a volunteer-run service that serves both the Town of Lynn Lake and Black Sturgeon Reserve (the MCFN has a fire truck but lacks trained operators).

20.3.5.2 Land Use

The Gordon and MacLellan sites are in a remote area approximately 37 km and 7 km northeast of the Town of Lynn Lake, respectively. The Black Sturgeon Reserve is located approximately 12 km southwest of the Gordon site. The nearest known permanent, seasonal, or temporary residences to the Project are a:

- Trapper cabin located approximately 3.5 km southeast of the Gordon site, on the north shore of Swede Lake; and
- Remote cottage located approximately 4.5 km southwest of the Gordon site, on the north shore of Simpson Lake.

The land use site in closest proximity to the MacLellan site is a landfill located approximately 3 km to the southwest.

Both mine sites are surrounded by vegetated land, forest cover, scattered lakes, watercourses, and wetlands, and located within areas of discontinuous permafrost cover.

The Town of Lynn Lake is the self-proclaimed ‘Sportfishing Capital of Manitoba’. Outdoor recreation activities are popular with both residents and visitors to the region and include sportfishing, hunting, boating, swimming, camping, cross-country skiing, and snowmobiling (Lynn Lake Mayor and Council 2014). There are two provincial parks within 20 km of the Town of Lynn Lake: Burge Lake and Zed Lake. Sand Lakes Provincial Park is approximately 40 km north of the Gordon site.

There are several municipal recreation facilities in the Town of Lynn Lake; however, their use is limited by lack of proper operation and maintenance. The arena has the potential to support curling, skating and hockey in the winter, and basketball, volleyball, floor-hockey, roller-skating, and badminton during other months. There is also an unsupervised beach, a public library, and a mining museum that is open by appointment. The former Royal Canadian Legion Hall in the Town of Lynn Lake is privately owned and used as a gathering place (pers. comm. 2015d). There are no recreation facilities on the Black Sturgeon Reserve (pers. comm. 2015f).

Municipal jurisdictions may adopt development plans and zoning by-laws to guide land use decisions within their respective boundaries. The following municipal development plan and zoning by-law apply to the Town of Lynn Lake:

- Town of Lynn Lake Development Plan No. 1329-2009; and
- The Local Government District (LGD) of Lynn Lake By-law No. 675.

The current Town of Lynn Lake Development Plan No. 1329-2009 identifies the MacLellan mine site as being designated a “Limited Development” area. Mineral exploration and development is encouraged in the Limited Development land use area under the Town of Lynn Lake Development Plan. There is no applicable development designation under a development plan for the Gordon mine site as it is located outside of municipal jurisdiction on unorganized Crown land.

Outside the built-up settlement area (townsite) of Lynn Lake, most of the land in the municipal boundary of the town is zoned as “LD – Limited Development District” under the LGD of Lynn Lake By-law No. 675, including the MacLellan mine site. Mining and quarrying are permitted uses in the Limited Development land use district under the By-law. There is no applicable zoning under a zoning by-law for the Gordon mine site as it is located outside of municipal jurisdiction on unorganized Crown land.

Provincial Land Use Policies (PLUPs) under the Provincial Planning Regulation No. 81/2011 reflect the provincial government's interest in land and resource use and sustainable development. The PLUPS apply to all lands subject to *The Planning Act of Manitoba* in the absence of adopted development plans. PLUPs are also given full consideration when undertaking planning activities and land use decision-making on Crown lands. Schedule 3 of the PLUPs include Policy Area 8: Mineral Resources, which expresses the provincial interest in mineral resources development.

20.3.5.3 Heritage Resources

The Gordon site was reviewed for heritage potential. The proposed ore and overburden stockpile locations were not considered to have high heritage resource potential based on predictive modelling undertaken, including extent of previous disturbance. Field assessments at this location did not record heritage resources.

Development within the MacLellan site is primarily located in areas that would have limited human activity given the nature of the terrain and general lack of navigable and potable water. Locations along the Keewatin River would have been more conducive for human occupation and resource harvesting. The one exception is the upland area where exposed quartz veins may have been quarried for stone tool manufacture. One such site was identified north of the proposed TMF and consisted of quartz flakes shallowly buried beneath the organic overburden. It is possible that additional sites are present at this and other upland locations. However, based on the site extent defined by shovel tests, these sites do not encompass a large area.

20.3.5.4 Current Use of Lands and Resources for Traditional Purposes by Indigenous People

A Traditional Knowledge/Traditional Land and Resource Use (TK/TLRU) Study is currently underway (interview phase complete) in collaboration with MCFN. MCFN has one reserve, the Black Sturgeon Reserve, which is the First Nation community located nearest to the Gordon (approximately 12 km) and MacLellan (approximately 24 km) sites.

While there is current use of lands and resources for traditional purposes by Indigenous peoples in the General Project Area, details of the TK/TLRU Study cannot be discussed until released by MCFN. MCFN is considered likely to be affected by the Project due to the proximity of its Reserve to the Project, identified traditional activity areas, and in consideration of the water flow directions and predominant wind direction.

Desktop research was conducted regarding the current use of lands and resources for traditional purposes by other Indigenous groups.

20.4 Potential Impacts and Proposed Mitigation Measures

20.4.1 Potential Project-related Environmental Interactions

Table 20-3 provides an overview of potential Project-related changes to environmental components that are directly linked or necessarily incidental to regulatory (federal and provincial) decisions that enable the Project to proceed, and associated effects on health and socio-economic conditions, physical and cultural heritage, and resources of historical, archaeological, paleontological, or architectural significance. The scope of the environmental assessment will focus on the potential adverse environmental effects of the Project on the Valued Components (VCs) identified in the last column of Table 20-3. VCs are environmental attributes associated with the Project of special value or interest to Indigenous peoples, regulatory agencies, the Proponent, resource managers, scientists, key stakeholders, and/or the general public.

Table 20-3 – Potential Project Environmental Interactions

Environmental Component(s) of Concern	Potential Environmental Interactions (Without Mitigation or Management)	How Potential Environmental Interactions will be Addressed in EIS
Fish, Fish Habitat, and Aquatic Species	The Gordon and MacLellan sites contain several fish-bearing watercourses and waterbodies. Routine Project activities could result in changes to fish and fish habitat due to the following potential interactions: • The Project has potential to adversely affect fish if Project-related hydrological and/or hydrogeological changes affect the quality or quantity of fish habitat. • Liquid discharges from the Project have potential to adversely affect fish habitat and fish health if they cause a reduction in water quality in receiving waters frequented by fish. • Construction, excavation, and dewatering and/or infilling of waterbodies have potential to cause injury or mortality to fish. • Alterations to stream channels have potential to cause injury or mortality to fish, as well as to affect fish mobility and fish habitat. • If any blasting occurs near fish-bearing waters, shock waves from the detonation of explosives have potential to cause injury or mortality to fish. An accidental spill or release to the environment originating from a Project activity or component would have potential to result in changes to fish and fish habitat, including: • Injury, mortality, and/or reduced health for fish. • Reduced availability and quality of fish habitat (including water quality).	• Potential Project-related environmental effects on fish and fish habitat will be assessed primarily in the context of the Fish and Fish Habitat VC, but will also be indirectly considered in the context of the Surface Water VC. • The assessment will include the identification of standard and VC-specific mitigation measures to reduce or eliminate Project-related environmental effects; characterization of residual Project-related environmental effects; and determination of the significance of residual Project-related environmental effects. • The EIS will also consider accidental events and assess the potential effects of an accidental spill or release to the environment on the Fish and Fish Habitat VC and Surface Water VC. • The assessment will be based on desktop information, the professional judgement of the EA Study Team, and the results of relevant environmental baseline studies carried out in support of the EIS, including associated baseline field data (e.g., Fish and Fish Habitat, Distribution, and Tissue Analysis; Benthos and Sediment; Water Quality; Hydrology; Hydrogeology; and Geochemistry technical data reports).
Migratory Birds	The Gordon and MacLellan sites may provide habitat for various species of migratory birds. Routine Project activities could result in changes to migratory birds as defined in Section 2(1) of MBCA due to the following potential interactions with the environment: • If conducted during the breeding bird season, site preparation activities (e.g., clearing and grubbing) have potential to cause injury or mortality to migratory birds, their nestlings, and their eggs, as well as to damage or destroy their nests. Project construction also has potential to result in alteration or loss of habitat for migratory birds. • Noise, vibration, and air emissions (e.g., dust) during Project construction and operation have potential to adversely affect habitat quality for migratory birds and could cause behavioural effects (e.g., avoidance/displacement). • Artificial night lighting during Project operation has potential to attract and/or disorient	• Potential Project-related environmental effects on migratory birds will be assessed primarily in the context of the Wildlife and Wildlife Habitat VC. • The assessment will include the identification of standard and VC-specific mitigation measures to reduce or eliminate Project-related environmental effects; characterization of residual Project-related environmental effects; and determination of the significance of residual Project-related environmental effects. • The EIS will also consider accidental events and will assess the potential effects of an accidental spill or release to the environment on the Wildlife and Wildlife Habitat VC. • The assessment will be based on

Environmental Component(s) of Concern	Potential Environmental Interactions (Without Mitigation or Management)	How Potential Environmental Interactions will be Addressed in EIS
	nocturnally migrating birds, and could cause an increased risk of injury or mortality from exhaustion and/or collisions with Project infrastructure. An accidental spill or release to the environment originating from a Project activity or component would have potential to result in changes to migratory birds, including: • Injury, mortality, and/or reduced health for migratory bird species. • Reduced availability and quality of migratory bird habitat.	desktop information, the professional judgement of the EA Study Team, and the results of environmental baseline studies carried out in support of the EIS, including associated baseline field data (e.g., Birds, Acoustics, and Ambient Lighting technical data reports).
Health and Socio-Economic Conditions for Indigenous and Non-Indigenous Peoples	The Gordon and MacLellan sites have potential to be used by various Indigenous and non-Indigenous land and resource users. Routine Project activities could result in the following changes to the environment that have potential to affect health and socio-economic conditions for Indigenous and non-Indigenous peoples: • Project activities and components have potential to affect the availability of lands and resources for commercial or recreational fishing and hunting/ trapping activities and/or other recreational uses currently carried out by Indigenous and non-Indigenous peoples. • Project-related requirements and the influx of Project personnel could increase the demand for local services and infrastructure, thereby potentially affecting the quality or availability of these amenities for Indigenous and non-Indigenous residents of the Town of Lynn Lake and other surrounding communities. • The Project has potential to adversely affect human health if liquid discharges from the Project degrade the quality of drinking water resources or if Project-related hydrological and/or hydrogeological changes affect the quality or quantity of drinking water resources. • Air, noise, and light emissions from the Project have potential to disturb nearby human receptors and pose a nuisance. • Emission and dispersion of chemicals from Project activities have the potential to affect air quality, as well as soil and surface water quality (through deposition), which could potentially affect human health (e.g., through conta-mination of drinking water resources or species of fish, wildlife, or plants that are consumed by Indigenous or non-Indigenous peoples). • The expenditures and employment associated with Project activities will affect local, regional, and provincial economic conditions through all phases of the Project. In addition to having positive economic effects, the Project could adversely affect labour and economy, for example by contributing to local or regional	• Potential Project-related environmental effects on health and socio-economic conditions for Indigenous and non-Indigenous peoples will be assessed in the context of the following VCs: Labour and Economy, Community Services and Infrastructure, Land and Resource Use, Traditional Land and Resource Use, and Human Health. • The assessment will include the identification of standard and VC-specific mitigation measures to reduce or eliminate Project-related environmental effects; characterization of residual Project-related environmental effects; and determination of the significance of residual Project-related environmental effects. • The EIS will also consider accidental events and will assess the potential effects of an accidental spill or release to the environment on these VCs. • A Human Health and Ecological Risk Assessment (HHERA) will be undertaken as the Project progresses. The HHERA will be completed using standard risk assessment protocols. • The assessment will be based on desktop information; the professional judgement of the EA Study Team; the results of the HHERA; the results of a Project-specific Transportation Impact Study; and the results of environmental baseline studies carried out in support of the EIS, including associated informant interviews and baseline field data (e.g., Socio-Economics, Acoustics, Air Quality, and Ambient Lighting technical data reports).

Environmental Component(s) of Concern	Potential Environmental Interactions (Without Mitigation or Management)	How Potential Environmental Interactions will be Addressed in EIS
	labour shortages or interacting negatively with the economic activities of other sectors, such as tourism or forestry.	
Physical and Cultural Heritage, and Resources of Historical, Archaeological, Paleontological, or Architectural Significance for Indigenous and Non-Indigenous Peoples	Archaeological and heritage resources have potential to occur on the Gordon and MacLellan sites. Routine Project activities could result in the following changes to the environment that have potential to affect the physical and cultural heritage of Indigenous or non-Indigenous peoples, and/or to affect any structure, site, or thing of historical, archaeological, paleontological, or architectural significance to Indigenous or non-Indigenous peoples: • Although the Project will be designed to avoid ground disturbance at sites where resources of cultural, historical, archaeological, paleontological, or architectural significance are known to be located, there is potential for Project-related ground disturbance (including excavation and blasting) to occur where previously unrecorded resources may be present. Such resources, if present, could be disturbed, damaged or destroyed by the Project. An accidental spill or release to the environment originating from a Project activity or component could result in changes to the environment that could affect physical and cultural heritage, or resources of historical, archaeological, paleontological, or architectural significance for Indigenous and non-Indigenous peoples.	• Potential Project-related environmental effects on physical and cultural heritage and resources of historical, archaeological, paleontological, or architectural significance for Indigenous and non-Indigenous peoples will be assessed in the context of the Heritage Resources VC. • The assessment will include the identification of standard and VC-specific mitigation measures to reduce or eliminate Project-related environmental effects; characterization of residual Project-related environmental effects; and determination of the significance of residual Project-related environmental effects. • The EIS will also consider accidental events and, will assess the potential effects of an accidental spill or release to the environment on the Heritage Resources VC. • A Heritage Resources Impact Assessment (HRIA) was completed for the Project in 2012, and a Heritage Resources environmental baseline study, including a field program, was completed in 2015 in support of the EIS.
Current Use of Lands and Resources for Traditional Purposes by Indigenous Peoples	The Gordon and MacLellan sites have potential to be used for traditional purposes by Indigenous land and resource users. The Project may therefore require access to, use or occupation of, or the exploration, development and production of lands and resources currently used for traditional purposes by Indigenous peoples. Routine Project activities could result in the following changes to the environment that have potential to affect the current use of lands and resources for traditional purposes by Indigenous peoples: • Project activities and components have potential to affect the availability of lands (including travel routes) and resources currently used by Indigenous peoples for traditional purposes such as fishing, hunting/trapping, and gathering. • The influx of Project personnel could increase the recreational demand for lands and resources that are currently used by Indigenous peoples for traditional purposes, thereby potentially affecting the quality or availability of these lands and resources for Indigenous peoples. • The Project has potential to adversely affect the	• Potential Project-related environmental effects on the current use of lands and resources for traditional purposes by Indigenous peoples will be assessed in the context of the Traditional Land and Resource Use VC. • The assessment will include the identification of standard and VC-specific mitigation measures to reduce or eliminate Project-related environmental effects; characterization of residual Project-related environmental effects; and determination of the significance of residual Project-related environmental effects. • The EIS will also consider accidental events and, in particular, will assess the potential effects of an accidental spill or release to the environment on the Traditional Land and Resource Use VC. • A Project-specific TK/TLRU Study is currently being completed in support of

Environmental Component(s) of Concern	Potential Environmental Interactions (Without Mitigation or Management)	How Potential Environmental Interactions will be Addressed in EIS
	quality or availability of fish species of traditional importance to Indigenous peoples (including species that are currently fished by Indigenous harvesters for traditional purposes) if liquid discharges from the Project degrade the quality of fish habitat. • Air, noise, and light emissions from the Project have potential to disturb wildlife species of traditional importance to Indigenous peoples and affect their movement, thereby potentially affecting their availability for current use by Indigenous peoples (e.g., hunting/trapping). • Emission and dispersion of chemicals from Project activities have the potential to affect air quality, as well as soil and surface water quality (through deposition). Thus, the Project has potential to adversely affect the quality or availability of fish, wildlife, and plant species of traditional importance to Indigenous peoples (including species that are currently fished, hunted/trapped, and gathered by Indigenous peoples for traditional purposes) if the Project results in the degradation of their habitats or the contamination of these resources. An accidental spill or release to the environment originating from a Project activity or component would have potential to result in changes to the environment that could affect the current use of lands and resources for traditional purposes by Indigenous peoples.	the EIS, with participation from Indigenous peoples in Lynn Lake, Pukatawagan, Winnipeg, and Regina. • The assessment will be based on desktop information, the professional judgement of the EA Study Team, the results of the TK/TLRU Study, and the results of the Socio-Economics and Heritage Resources technical data reports, including associated interviews and baseline field data.
Environmental and Human Health Effects Under Provincial Jurisdiction	In addition to the potential environmental and human health effects discussed above with respect to fish and fish habitat, migratory birds, health and socio-economic conditions, archaeological and heritage resources, and Indigenous traditional use (many of which fall under both federal and provincial jurisdiction), routine Project activities also have potential to result in the following other environmental effects under provincial jurisdiction: • Site preparation activities (e.g., clearing and grubbing) have potential to cause injury or mortality to provincially regulated non-migratory birds, their nestlings, and their eggs, as well as to damage or destroy their nests. Provincially regulated small mammals and amphibians may also be susceptible to potential injury or mortality during site preparation activities. Project construction also has potential to result in alteration or loss of habitat for provincially regulated non-migratory birds and other provincially regulated wildlife in general. • Site preparation activities (e.g., clearing and grubbing) will cause mortality to provincially regulated plants and have potential to cause alteration or loss of wetland habitat. • Noise, vibration, and air emissions (e.g., dust) during Project construction and operation have	• Potential Project-related environmental and human health effects under provincial jurisdiction will be assessed in the context of the following VCs: Atmospheric Environment, Groundwater, Surface Water, Fish and Fish Habitat, Vegetation and Wetlands, Wildlife and Wildlife Habitat, Labour and Economy, Community Services and Infrastructure, Land and Resources Use, Heritage Resources, Traditional Land and Resource Use, and Human Health. • The assessment will include the identification of standard and VC-specific mitigation measures to reduce or eliminate Project-related environmental effects; characterization of residual Project-related environmental effects; and determination of the significance of residual Project-related environmental effects. • The EIS will also consider accidental events and will assess the potential effects of an accidental spill or release to the environment on these VCs.

Environmental Component(s) of Concern	Potential Environmental Interactions (Without Mitigation or Management)	How Potential Environmental Interactions will be Addressed in EIS
	potential to adversely affect habitat quality for provincially regulated non-migratory birds and other provincially regulated wildlife, and could cause behavioural effects (e.g., avoidance/ displacement). • The Project has potential to cause an increased risk of injury or mortality for provincially regulated non-migratory birds and other provincially regulated wildlife due to collisions with Project vehicles. • Artificial night lighting during Project operation has potential to attract and/or disorient nocturnally active provincially regulated non-migratory birds, and could cause an increased risk of injury or mortality from exhaustion and/or collisions with Project infrastructure. Non-migratory birds attracted to the Project site by artificial night lighting, if any, could also be exposed to other threats such as predation or interactions with Project vehicles and equipment. Other provincially regulated nocturnal wildlife (e.g., bats) may also be susceptible to potential effects from artificial night lighting. • Project-related solid and liquid wastes have potential to attract provincially regulated non-migratory birds and other provincially regulated wildlife, where they may be exposed to threats such as predation or interactions with Project vehicles and equipment. • Open pit mining during the operational phase of the Project has potential to affect groundwater quantity (i.e., groundwater discharge levels and discharge to surface water features), flow, and quality. The exposure and weathering of some mine materials may result in acid generation and/or leaching of contaminants causing degradation of surface water runoff and groundwater quality. • Deposition of chemicals of potential concern from dust onto soil from Project activities has the potential to affect soil quality and surface water quality. This change in soil quality can directly affect ecological receptors that interact either directly or indirectly with this soil. The change in chemical concentrations in soil may alter their concentrations in vegetation, and prey species. These changes in media concentrations disseminate through the food web, and can potentially produce effects in organisms that ingest these media. • Discharges and runoff from Project operations may release chemicals of potential concern into groundwater and surface water. A change in surface water quality may affect ecological receptors that use surface water from the local assessment area as a source of drinking water. An accidental spill or release to the environment originating from a Project activity or component could	• The assessment will be based on desktop information; the professional judgement of the EA Study Team; the results of the HHERA, HRIA, and TK/TLRU Study; and the results of the various environmental baseline studies identified above for each environmental component of concern in this table, including associated interviews and/or baseline field data. The assessment will also consider the results of additional relevant environmental baseline studies, including associated baseline field data not identified above (i.e., Vegetation and Wetlands and Soil and Terrain technical data reports).

Environmental Component(s) of Concern	Potential Environmental Interactions (Without Mitigation or Management)	How Potential Environmental Interactions will be Addressed in EIS
	result in environmental effects under provincial jurisdiction, including many of those discussed above with respect to fish and fish habitat, migratory birds, health and socio-economic conditions, archaeological and heritage resources, and Indigenous traditional use, as well as: • Injury, mortality, and/or reduced health for provincially regulated non-migratory birds and other provincially regulated wildlife. • Reduced availability and quality of habitat for provincially regulated non-migratory birds and other provincially regulated wildlife. The potential environmental effects described above for birds and other wildlife, fish, and vegetation could affect secure species as well as SOCC protected under provincial legislation.	

20.4.2 Valued Components

VCs to be assessed in the EA include the atmospheric environment, surface water, groundwater, wildlife and wildlife habitat, fish and fish habitat, vegetation and wetlands, labour and economy, community services and infrastructure, land and resource use, heritage resources, traditional land and resource use and human health. These VCs have been selected in consideration of:

- The interactions discussed in Table 20-3;
- Regulatory guidance and requirements;
- Issues raised by regulatory agencies, Indigenous groups, key stakeholders, and the public.
- Technical aspects of the Project (i.e., the nature and extent of Project components and activities);
- Existing environmental conditions in the study area and interconnections between the biophysical and socio-economic environment;
- Experience and lessons learned from similar mining projects; and
- Professional judgment.

20.4.3 Potential Mitigation and Environmental Management Measures

Potential mitigation and environmental management measures have been developed for all VCs based on preliminary Project planning and design. These mitigation measures will be considered and refined as Project design and engineering progress, and will be informed by the outcomes of the EA process (including the results of EA-related modelling, as well as the results of public and Indigenous engagement carried out in support of the EA).

Opportunities for the reduction of potential adverse environmental effects are being and will continue to be considered in the design and engineering of Project components and the planning, scheduling, and carrying out of activities during all phases of the Project. Currently proposed mitigation measures are anticipated to result in compliance with applicable environmental legislation and regulatory requirements, including the *Fisheries Act* and MBCA.

Preliminary plans for mitigation and environmental management include development and implementation of the following Project-specific environmental management and monitoring plans, and consultation with applicable federal and provincial regulators and engagement with potentially affected Indigenous communities regarding these plans:

- Air Quality Management Plan and Greenhouse Gas Management Plan;
- Beaver Dam and Beaver Activity Management Plan;
- Bird Nest Mitigation Plan (if activities that could result in incidental take cannot be avoided);
- Closure Plan;
- Emergency Response and Spill Prevention and Contingency Plans;
- Erosion and Sediment Control Plan;
- Explosives Management Plan;
- Fish Habitat Offsetting Plan (if serious harm to fish that are part of or support a CRA fishery cannot be avoided);
- Groundwater Monitoring Plan;
- Surface Water Monitoring and Management Plan;
- Vegetation Management Plan and Weed Management Plan;
- Waste Management Plan; and
- Wildlife Monitoring and Management Plan.

Alamos will plan for communication of Project activities, locations and timing throughout construction, operation, and closure to affected Indigenous communities, land and resource users, interest groups, the provincial government, and local authorities leading up to construction and throughout the life of the Project.

20.5 Rehabilitation

A Closure Plan will be developed and implemented, in accordance with the *Mine Closure Regulation* under *The Mines and Minerals Act* of Manitoba and associated General Closure Plan Guidelines (MGET), to remove unneeded facilities and restore the Gordon and MacLellan sites following the completion of mining activities. The primary objective of reclamation and closure activities will be to establish self-sustaining physical, chemical, and biological stability of the sites, and to meet desired end land functions and uses. The Closure Plan will be updated as necessary to reflect the environmental requirements in place at the time of closure.

At the end of operations of the Project, the main features will include the open pits, mill processing facilities, offices, storage areas, TMF, and mine rock storage areas. Reclamation measures expected for each of the main features are described below. Progressive reclamation activities will be carried out where possible throughout the mine life; however, most decommissioning and reclamation work will take place once mining has been completed.

The main elements of the reclamation plan are comprised of:

- Reclamation of mine access roads not needed for post-mining land access, with contouring to restore natural drainages and roadways revegetated;
- Recontouring of disturbed areas to blend in with surrounding topography and to re-establish natural drainage patterns;
- Removal of water management features that are no longer required, such as water treatment systems, ponds, and ditches. This will include: recontouring/spreading of pond berms; backfilling of ponds and ditches; and re-establishing natural drainage patterns;
- Reclamation of mine rock storage areas with suitable covers as needed, revegetation, and establishment of stable drainage conditions;
- Allowing the open pits to fill with water to form pit lakes and directing the overflows to established drainages;
- Implementation of public safety measures around the pits (e.g., fencing or rock berms);
- Management of site runoff from developed areas, including from the ore milling and processing plant site, mine rock storage areas, TMF, and open pits, to meet federal and provincial regulatory requirements for downstream water quality;
- Installation of a suitable cover and revegetation of TMF and establishment of drainage to provide long-term erosion control;
- Removal of equipment and facilities from Gordon and MacLellan sites, together with above-ground concrete structures; and
- Re-vegetation of disturbed areas with plant species that are suitable for reclamation and the end land uses of the area. The goals of reclamation vegetation will be to: prevent erosion and sedimentation to protect aquatic resources; prevent invasive plant establishment; and re-establish a land use that is of value for wildlife and/or humans (including Indigenous peoples), and mitigates the residual environmental effects of the Project on the environment.

For closure cost see Section 21.5.4.

20.6 Environmental Monitoring

As part of the environmental assessment process, a monitoring framework will be developed for all subsequent phases of the Project. This framework will include both compliance and environmental effects monitoring during construction, operation and closure. Monitoring will follow applicable provincial and federal guidance, and regular reports will be prepared and submitted to the relevant agencies as and when stipulated by the licenses and permits issued for the Project. The compliance monitoring programs will be designed to assess whether the Project has been implemented in accordance with commitments made in the EIS. The effects monitoring program will be designed to verify the predictions of key environmental effects made during the environmental assessment.

20.7 Environmental Principles

Environmental protection and management measures will be adopted to guide the planning, design, construction, operation, and decommissioning, reclamation, and closure of the Project. These include:

- Where possible, siting facilities to avoid sensitive areas such as watercourses, wetlands, important habitat types, and areas of high archaeological potential; and where unavoidable, reducing the size and number of natural features that may be affected;
- Where possible, siting facilities within, instead of across, watershed boundaries to reduce the number of potentially affected waterbodies;
- Reducing the ‘footprint’ of Project facilities and activities, to the extent practical, to reduce the amount of disturbed land and disturbed water resources;
- Adhering to regulated standards for air and water emissions, for storage or disposal of solid wastes, and for handling and disposal of hazardous materials;
- Adhering to regulated and/or industry design and management standards to address environmental risks such as seismicity, unusual weather events, flooding, and erosion;
- Preparing an Environmental Protection Plan (EPP) for construction activities that is included in, and enforced through, construction contracts;
- Preparing and implementing an Environmental Management Plan (EMP) during operation for ongoing monitoring and management of, for example, land and soil resources, water, air and water quality, noise and vibration, hazardous materials and waste, and occupational and community health and safety;.
- Preparing and maintaining an Emergency Response Plan for the Project;
- Planning the mine for closure and having a Closure Plan, including the provision of security to the provincial Crown for performance of rehabilitation work; and
- Planning and financing activities to compensate for unavoidable adverse effects on environmental resources such as aquatic habitats.

The location of Project components will be finalized based on engineering feasibility studies and environmental considerations. To the extent possible, Project facilities will be sited to avoid or reduce interactions with watercourses/waterbodies, important habitat types, and areas of high archaeological potential. Where avoidance is not possible, mitigation measures will be developed in consultation with the applicable regulatory authorities and Indigenous communities.

20.8 Community Principles

Alamos has commenced public, stakeholder and Indigenous consultation and engagement activities and these efforts will be ongoing throughout the Project planning and permitting phase and implemented throughout construction, operation, and eventual mine closure with the objective of:

- Addressing public, stakeholder, and Indigenous community concerns to the extent possible during the design, construction, operation, and closure of the Project; and
- Promoting local benefits, including employment and business opportunities, to the extent practical.

21 CAPITAL AND OPERATING COSTS

21.1 Capital Cost Estimate Input

The capital cost of the Project has been estimated based on the scope of work defined in the sections below. The parties below have contributed to the preparation of the capital cost estimate in specific areas, as listed.

- Ausenco:
  - Crushing;
  - Process;
  - Truck shops, washbays, admin buildings, etc.;
  - Utilities;
  - On-site infrastructure;
  - Off-site infrastructure;
  - Indirect costs; and
  - Contingency.
- Golder:
  - TMF;
  - TMF starter dam; and
  - Ditches, drainage and sumps.
- Stantec:
  - Water management; and
  - Closure.
- Q'Pit:
  - Pre-production mining; and
  - Mine fleet.

21.2 Capital Cost Estimate Summary

The estimate conforms to AACE Class 3 guidelines for a FS Estimate with a -10% to +15% accuracy.

Table 21-1 provides a summary of the estimate for overall initial capital cost. The costs are expressed in Q2 2017, Canadian dollars and include all mining, site preparation, process plant, dams, first fills, buildings, roadworks, and off-site infrastructure.

Table 21-1 – Summary of Capital Costs

Description	Gordon Mine Total Cost ($'000s)	MacLellan Mine Total Cost ($'000s)	Both Mines Total Cost ($'000s)
Direct Cost:			
Mining Initial Capital Lease Payment (Common)			$23,767
Mine Pre-production and Miscellaneous	$20,295	$37,687	$57,982
Process Plant		$96,424	$96,424
Utilities and Services	$2,424	$20,378	$22,802
Tailings Management		$22,785	$22,785
On-site Infrastructure	$9,048	$46,881	$55,929
Off-site Infrastructure:			
Camp and Infrastructure			$15,694
Manitoba Hydro Line			$17,000
Subtotal Direct Costs	$31,767	$224,155	$312,383
Indirect Cost:			
EPCM and Consulting Services			$28,930
Vendor Representatives			$647
Freight			$4,668
Temporary Construction Facilities and Utilities			$4,882
Pre-production Camp Operations			$21,041
First Fills and Opening Stocks			$2,866
Initial Spares			$7,860
Provincial Sales Tax			$12,629
Subtotal Indirect Costs			$83,523
Subtotal Direct + Indirect			$395,906
Project Contingency			$38,432
Sub Total Directs + Indirects + Contingency			$434,338
Owner Cost			$16,280
TOTAL INITIAL CAPITAL COST			**$450,619**

The estimate is based on an EPCM execution approach as outlined in Section 24.1.

The following parameters and qualifications were considered:

- The estimate was based on Q2 2017 pricing;
- Mining equipment is leased;
- No allowance has been made for exchange rate fluctuations; and
- There is no escalation added to the estimate.

Data for the estimates have been obtained from numerous sources, including:

- Feasibility level engineering design;
- Mine schedules;
- Topographical information obtained from site survey;
- Geotechnical investigations;
- Budgetary equipment quotes;
- Budgetary unit costs from local contractors for civil, concrete, steel, electrical and mechanical works; and
- Data from similar recently completed studies and projects.

Major cost categories (permanent equipment, material purchase, installation, subcontracts, indirect costs and Owner's costs) were identified and analyzed. To each of these categories on a line item basis, percentage of contingency was allocated based on the accuracy of the data, and an overall contingency amount was derived in this fashion.

21.3 Capital Estimate

21.3.1 Initial Mine Capital

The initial mining capital cost includes the cost of equipment lease payments during the pre-production period, both primary fleet and support equipment, as well as the pre-production stripping and other owner's costs for pre-production activities such as roads and platforms preparation and peripheral facilities such as the explosives magazine and explosives production or mixing facility.

The initial mining capital cost estimate is shown in summary Table 21-2 and totals $81.7M.

Table 21-2 – Summary of Initial Mining Capital Costs

Initial Mining Capital	($'000s)
MacLellan Pre-production Mining	37,687
Gordon Pre-production Mining	20,295
Total Pre-production Mining	57,982
Initial Mobile Equipment Capital Lease Payment	23,767
Total Initial Mining Capital	81,749

The capitalized pre-production mine costs are estimated at $58.0M, excluding contingency. A contingency of 10% of pre-production cost ($5.8M) is included in project contingency. The pre-production period is designated as Year -2 and Year -1. Pre-production mining costs include the equipment and personnel costs for the in-pit and ex-pit borrow source mining, preparation of the mine roads and platforms and pre-production mining, preparation of the mine roads and platforms and pre-production mining.

The major equipment costs were based on budgetary level quotes which were requested for selected primary and support equipment items from dealers representing major original equipment manufacturers which have presence in the area of the Project. For certain items such as service trucks, pickups, fire truck, ambulance, etc., costs were assigned on the basis of information from past studies and other current references. Typically, major equipment budget quotes included transportation and commissioning costs.

The capital costs include a provision for miscellaneous start-up and pre-production one time capital items and charges. These items include the explosives mixing and/or production facilities construction cost, the explosives magazine setup costs, engineering software, survey and computer equipment, ambulance and mine rescue equipment, the ground penetrating radar equipment and other non-manned equipment items. Some recurring capital items such as dewatering pumps estimated by others are also in miscellaneous capital.

The mine mobile equipment purchase cost during preproduction for both sites is estimated at $79.6M, excluding PST and initial spare parts. The owner considers that the mine equipment purchases will be financed by a capital lease. The project capital deployment requirements were estimated by the owner based on a four year lease term at a 5% interest rate. Canada has a well developed market with several capital groups as well as OEMs participating in the financing of mobile mine equipment. Considering the prevailing interest rates, the capital deployment schedule can be considered conservative. The initial lease payment of $23.8 M in Year -1 is included with the project initial capital cost. The subsequent lease payments are included with the sustaining capital.

21.3.1.1 Freight and Equipment Commissioning Costs

Mine OEM provided budget level estimates for freight and assembly for the primary and support mine equipment. For items without quotes an additional 4.5% for freight and 3.75% for crainage and equipment erection was added. The equipment commissioning costs range from less than 2% for small equipment and up to 10% for the hydraulic shovel, averaging 5.8% for the equipment purchases for the LLGP. For the mine equipment, this cost has been included with the initial equipment capital cost.

21.3.1.2 Spare Parts for Mine Equipment

Spare parts initial inventory for all new mine equipment is accounted for with an allowance of 5% of the total capital for mobile mine equipment, excluding miscellaneous capital items. Over the life of the mine, this cost is estimated at $6.3M. Of this cost, $3.6M corresponds to the equipment requirements for the preproduction period and is included with the project indirect cost. For the purposes of costing, this inventory is assumed consumed during the last year of the mine operations and the respective costs allocated as a credit to the mine operating cost.

21.3.2 Process Plant

The gold plant and associated facility estimates have been prepared on a commodity basis (i.e., divided into earthworks, concrete, structural, etc.) and reported by area (i.e., crushing, milling, etc.). The quantities have been based on a first principles approach.

The estimate is based on the majority of the work being carried out under fixed price or re-measurable unit price contracts under a normal development schedule. No allowance is included for contracts on a cost plus or fast-track accelerated schedule basis. The erection of tankage, structural, mechanical, piping, electrical, instrumentation, and civil works will be performed by experienced contractors, using local to Manitoba labour.

21.3.3 Tailings Management Facility

Material quantities for the TMF and its water conveyance structures were provided by Golder. These included quantities for:

- Clearing and grubbing of the TMF footprint;
- Foundation excavation and subgrade preparation;
- Dam fill placement and installation of the HDPE liner;
- Construction of the spillway and seepage collection ditches; and
- Construction of the decant structure for mill reclaim.

All quantities provided by Golder were based on neat design lines with no added contingency. Ausenco applied contingencies to each line item.

21.3.4 On-site Infrastructure

The following infrastructure will be built:

- MacLellan administration building;
- MacLellan mine truck wash building;
- MacLellan mine truck shop building;
- Gordon administration building;
- Assay lab buildings;
- On site roads; and
- Power distribution.

21.3.4.1 Roads

Internal roadways were priced based on construction from crushed waste rock available from site and any naturally available materials. A dedicated mobile aggregate crushing plant will be utilized to prepare the granular and construction material for site roads during construction.

21.3.5 Off-Site Infrastructure

A budget estimate of $17.0M has been provided by Manitoba Hydro for supply of a 138 kV Hydro power line to MacLellan Mine site. This estimate is based on the design, procurement and construction of 7 km of new line from the Town of Lynn Lake to MacLellan and the upgrading of existing Manitoba Hydro infrastructure between the Laurie River generating station and the Town of Lynn Lake. This is based on the scope and definition of work agreed between Manitoba Hydro and Alamos and is described in Section 18.4.1 in greater detail.

21.4 Basis of Estimate

21.4.1 Direct Costs

Direct costs are quantity based and include all permanent equipment, bulk materials, subcontracts, labour, contractor indirects and growth associated with the physical construction of the facilities.

21.4.1.1 Commodity Take-offs

Bulk material take-offs to a feasibility level were developed from arrangement drawings. Rates were obtained from quotations obtained from local contractors. These rates include the appropriate gang rate for the commodity and the actual cost of the permanent materials. Local freight associated with contractor-supplied material is included in the unit rates.

No imported fill is required. Aggregate material is available via an on-site crushing plant, provided by the contractor (supply and operation). All aggregate will be provided “free–issued” at the site of the crusher. Excess cut material can be stockpiled on-site or consumed for future construction.

21.4.1.2 Labour Rates

Labour rates have been built-up from first principles for different trades (welders, boilermakers, roofers, pipefitters, millwrights, storeman, crane operator, etc.). These rates have been based on the Manitoba labour collective agreement (industrial sector).

These labour rates include the following:

- Base hourly rate; and
- Contribution rate from collective agreement – industrial sector:
  - Vacation, holiday and sick leave pay;
  - Premiums;
  - Safety, health and welfare;
  - Compensation for safety clothing and equipment; and
  - Social benefits and funds.

Contractor indirect costs for structural, mechanical, piping, electrical and instrumentation have been developed with the assistance of well-established local construction contractors within Manitoba. Earthworks and concrete have been based on unit rates from contractors within Manitoba. Distributable costs have been allocated by percentage in the estimate on a man-hour basis and are inclusive of the following:

- Salaries, salary burden, allowances and benefits for the contractor’s indirect labour, supervisory and management staff;
- Staff recruitment and travel expenses;
- Living out allowances;
- Mobilisation and demobilisation;
- Temporary buildings and facilities at site specifically for and used by the contractor;
- Workshop equipment and supplies;

- Vehicles and equipment used by staff during construction;
- Construction equipment including cranes up to 100 t;
- Small tools and consumables;
- Site office overheads, such as stationery, communications, light and power, first aid, security, etc.;
- Head office costs/contribution;
- Financing charges;
- Insurances; and
- Profit.

21.4.1.3 Equipment Costs

Multiple quotes were sourced for all the mechanical equipment, with the exception of some minor equipment which were sourced from Ausenco's database. The budget quotes cover over 85% of the overall mechanical equipment supply cost.

21.4.1.4 Freight

Plant equipment and bulks freight cost has been obtained from budget quotes inclusive of the freight component. The freight component has been split, and included as indirect freight cost. If freight cost was not quoted, then a percentage allowance has been considered.

21.4.1.5 Duties and Taxes

All applicable duties have been included.

All taxes are excluded unless otherwise stated.

21.4.2 Indirect Costs and Owner's Costs

Indirect costs include items that are necessary for Project completion, but not related to the direct construction cost.

These items are summarized in the subsections below.

21.4.2.1 Temporary Facilities and Services

Temporary facilities and services are items which are not directly attributable to the construction of specific physical facilities of the plant or associated infrastructure, but which are required to be provided during the construction period to support the construction and have been estimated in detail.

These costs include:

- EPCM office complex, HS&E services, security services, site vehicles, refuelling, bus transportation, recurring Project costs, maintenance services, provision of temporary roads, temporary power, water, effluent disposal and other facilities as required; and
- Heavy lift construction cranes. These represent cranes over and above what the construction contractor provides. These are cranes greater than 100 tonne capacity.

21.4.2.2 EPCM and Consulting Services

The engineering, procurement, Project and construction management budget has been compiled by the identification of resources over a defined schedule. The EPCM and consulting services estimate includes the following:

- Engineering;
- Procurement;
- Construction management;
- Travel expenses;
- Home office expenses;
- Site Office Expenses;
- Commissioning support: and
- Other consulting services (geotechnical, environmental, shipping logistics, surveys and QA/QC).

21.4.2.3 Vendor Representatives

The cost for vendor representatives, for both installation supervision and for the commissioning component, is included and are based on vendors recommended support that were provided in the quotations and incorporated where applicable. Where these were not provided in the quotation but still required, a historical percentage of equipment supply cost was included based on Ausenco's database.

21.4.2.4 Camp – Pre-Production Operations

The full supply and operation of a construction and workforce camp has been quoted by a third-party vendor. The number of construction and operations personnel required to stay on-site has been used in generating the approximate number of beds and catering. Operation of the camp, personnel, and maintenance of facilities for a two year period (Year -2 and Year -1) has been included.

21.4.2.5 Initial Spares

Spares include capital spares, one year of operational spares, and commissioning/start-up spares for the process plant and infrastructure. Mining equipment spares are included in the mining capital.

A spares list was developed from the budget quotations provided. Where spares were not priced in the quotation, a percentage of the equipment cost was applied to each spares category.

21.4.2.6 First Fills and Opening Stocks

An estimate for first fills has been included for all reagents. Grinding media and liners for the SAG and ball mills have been included.

21.4.2.7 Manitoba Provincial Sales Tax (PST)

Manitoba Provincial Sales Tax (PST) is included under the indirect costs.

21.4.2.8 Owner's Costs

Owner's costs include the following:

- Land;
- Owner's team including construction, startup and commissioning;
- Pre-production process and administrative costs, offset by pre-commercial production gold sales;
- Recruiting, training and site visits;
- IT and communications; and
- Insurance, finance, legal and Lynn Lake office.

21.4.2.9 Escalation

Escalation is excluded from this estimate.

21.4.3 Estimate Growth, Estimate Contingency and Accuracy

21.4.3.1 Growth Allowance

Allowances on quantities are included in direct costs to cover the anticipated growth from the Feasibility Study phase to execution.

The design growth allowance is a subjective amount added to the material take offs (MTO's) based on the degree of engineering completed and a comparison to historical data. The CAPEX growth allowance is presented in following in Table 21-3:

Table 21-3 – Growth Allowances

Description	Growth Allowance (%)
Civil and Earthworks	11%
Concrete	5%
Structural Steel	9%
Architectural	5%
Mechanical Equipment	5%
Mechanical Bulks	6%
Piping	7%
Electrical Equipment	9%
Instrumentation	4%
Power Line(1)	-

Notes:

1. *No growth was available from Manitoba Hydro for the upgrade estimate.*

21.4.4 Estimate Contingency

Contingency is a cost element to cover unknown items that are expected to occur within the defined scope of the Project but which cannot be properly defined at the current stage of the

Project. It should be assumed that the contingency will be spent. The contingency allowance specifically excludes cost arising from scope changes, Project risk factors and all other items that are excluded from the capital cost estimate. The Project contingency is meant to cover the normal inadequacies that are inherent in any project estimate due to the dynamic nature of project engineering and construction.

Contingency was calculated on a line-by-line basis on both direct and indirect costs. Contingency was built up from equipment, labour and estimated quantity based on the level of engineering certainty and source of cost. The overall contingency resulted in $38.4M.

21.4.5 Exclusions

The following costs and scope will be excluded from the CAPEX:

- All facilities not identified in the summary description of the Project;
- Scope changes;
- Any additional work that is required as a result of conditions, both subsurface conditions and conditions in and around the plant, that were not known as of the base date of the estimate (e.g., latent plant conditions and unknown geotechnical conditions), including any costs incurred in establishing and confirming as-built information over and above that defined in the estimate:
- Fees or royalties relating to use of certain technologies or processes;
- Costs incurred to accelerate the work;
- Financing charges and interest during construction;
- Currency exchange fluctuations after June 2017;
- Credits for salvage value of any demolition, modification work, residual construction materials, vehicles, and temporary buildings Costs associated with decontamination of site and associated facilities; and
- All costs associated with weather interruption of construction operations.

21.5 Sustaining Capital

The sustaining capital cost summary is presented in Table 21-4.

21.5.1 Mine Sustaining Capital

Primary and support mine equipment lease payments after the preproduction period are included with the sustaining capital costs. The equipment requirements were presented in Section 16 of this report. The purchase expenditure profile is shown in Table 21-5. The cost of the equipment is considered incurred in the period of requirement. It is noted that no equipment replacement is expected to take place in the short life of the open pits, other than for the pickup trucks and pumps.

The MacLellan sustaining capital cost includes $7.3M for the highway haulage equipment, including one backhoe, 23 highways trucks and two pickup trucks. As discussed in Section 21.3.1, mine equipment capital deployment is based on a four-year lease at 5% interest rate, including the equipment purchases from preproduction and up to Year 4 of operations, excluding the provision for the initial spare parts. The sustaining capital includes lease payments from Year 1 to Year 7.

Table 21-4 – Total Sustaining Capital Costs

Sustaining Capital Costs – Description ($ x,1,000)	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	TOTAL
Sustaining Capital Leases (mining mobile equipment)	$35,497	$37,823	$38,692	$16,131	$4,401	$2,075	$1,205					$135,824
Mining and Spare Parts	$1,965	$342	$146	$202	$658		$788			$394		$4,495
Mining Sustaining Capital	$37,462	$38,165	$38,838	$16,333	$5,059	$2,075	$1,993			$394		$140,319
Tailing Dam Cost		$9,097				$5,098			$8,869			$23,064
Water Wells		$2,232										$2,232
Water Management System	$1,117	$144	$45	$708		$572						$2,586
Light Fleet Replacement					$554							$554
Closure Cost MacLellan			$379	$474	$1,185	$2,133	$3,721	$4,195	$5,333	$6,281	$976	$24,677
Closure Cost - Gordon		$99	$406	$594	$990	$1,211	$136					$3,436
Other Sustaining Capital	$1,117	$11,572	$830	$1,776	$2,729	$9,014	$3,857	$4,195	$14,202	$6,281	$976	$56,549
Grand Total (x1000)	$38,579	$49,737	$39,668	$18,109	$7,788	$11,089	$5,850	$4,195	$14,202	$6,675	$976	$196,868

Table 21-5 – Sustaining Capital for Mining Equipment

Sustaining Capital Costs – Description ($ x1,000)	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	TOTAL
Sustaining Capital Leases (mining mobile equipment)	$35,497	$37,823	$38,692	$16,131	$4,401	$2,075	$1,205					$135,824
Mining Support Equipment	included in capital leases						$750			$375		$1,125
Mining Miscellaneous	included in capital leases				$658							$658
Spare parts (5%)	$1,965	$342	$146	$202			$38			$19		$2,712
Mining Sustaining Capital Cost	$37,462	$38,165	$38,838	$16,333	$5,059	$2,075	$1,993			$394		$140,319

21.5.2 TMF

The tailings dam will require progressive rising in the life of the Project. The lifts will occur in Years 2, 6 and 9 indicated in Table 21-4.

21.5.3 Water Management

An allowance of $2.4M is included in Year 2 for the supply and installation of water wells including pipes, pumps, drilling and instruments.

An allowance of $2.6M for earthworks will be required to maintain the Water Management System, as shown in Table 21-4.

21.5.4 Closure

Closure costs have been distributed in accordance with Provincial Regulation (Manitoba Mine Closure Regulation 67/99), Section 24 Table 1. Provision for mine closure of $28.1M includes:

- Demolition and disposal of process and on site infrastructure;
- Draining and covering of the TMF;
- Covering of the mine rock stockpile; and
- Removal and covering of roads and culverts.

21.6 Operating Costs

21.6.1 Operating Cost Summary

The operating cost estimate is presented in Q2 2017, Canadian dollars (CAD). The estimate was developed to have an accuracy of ±15%.

The estimate includes mining, processing and G&A/Accommodation costs. Credits associated with doré royalties and silver are provided by Alamos, and applied to partially offset OPEX costs.

The total operating-cost estimates for the LOM for the ore processing at the MacLellan/Gordon facilities are provided in Table 21-6. The overall LOM operating cost is $1,273.2M over 10.4 years, or $48.08/t of ore milled.

Table 21-6 – LOM Operating Cost Summary

Cost Centre	M$	$/t milled	% of Total
Mining	627.3	23.69	49
Highway Haulage (Gordon hauled ore averaged over full mill tonnage)[1]	85.9	3.25	7
Processing	382.8	14.46	30
General and Administration	106.9	4.04	8
Accommodation and Transport	81.2	3.07	6
External Refining	4.8	0.18	0
Subtotal OPEX	1,289.0	48.68	100
Royalties and Ag Credits	(15.8)	(0.60)	-
Total OPEX	1,273.2	48.08	-

Notes:

1. *Actual highway haulage operating cost is $9.93/t applied to milled ore from Gordon only. Indicated $ t is average over total mill feed.*

21.6.2 Basis of Operating Cost

Common to all operating costs estimates are the following assumptions:

- Cost estimates are based on Q2 2017 pricing without allowances for inflation;
- For material sourced in US dollars an exchange rate of $0.75 USD per Canadian dollar was assumed;
- Fuel costs and taxes were established through communications with a Manitoba based supplier. Diesel costs used were $0.7905/L (dyed) and $0.9305/L (undyed, highway). Federal goods and service tax (GST) and Manitoba provincial fuel tax were removed from the dyed fuel price;
- The annual power costs were calculated using a unit price of $0.043/kWh calculated using the Tariff No. 2016-62 from Manitoba Hydro website;
- Labour is assumed to come from mostly Manitoba with sources being locally, Thompson, Flin Flon and Winnipeg;
- Rotations at site will be as follows: hourly personnel, two weeks in / two weeks out, salaried personnel five days in / two days out, or four days in / three days out. Whenever possible a five and two schedule will be applied to local staff; and
- FIFO workers will be housed in an owner supplied accommodation camp managed by a third party.

21.6.2.1 *Basis of Mine Operating Cost*

The mine operating cost for the LLGP was estimated by Q'Pit based on the LOM plan quantities of material movement and the selected equipment requirements estimates.

The mine operating cost estimate has been allocated to the following cost centres:

- Loading;
- Hauling;
- Drilling;
- Blasting;
- Support Equipment;
- Engineering and Management;
- Underground Voids Management for MacLellan;
- Grade Control and Assaying; and
- Miscellaneous

Key considerations and assumptions for the estimate of the open pit mine operating costs include:

- The mine development, operation and maintenance will be using owner personnel and equipment;
- Equipment costs are calculated based on estimated equipment operating time;
- Personnel costs are based on calendar time. Personnel are assigned on the basis of equipment requirements and personnel costs are calculated using the pay scale rates supplied by the owner and a site training period for newly hired personnel;
- The operating costs estimate refers to the commercial production period. The mine operating costs during the pre-production period of two years are to be included with the Project capital costs;
- The blasting area cost estimate was based on the consumption estimates derived for the Project, the unit explosive agent and accessory costs. These unit costs were based on the budget quotes for the Project from explosives manufacturers;
- The diesel costs are calculated from the unit costs for the Project for clear and dyed diesel and the equipment diesel consumption rates. Dyed diesel is for exclusive use by off-highway mining equipment and excludes GST and Manitoba provincial fuel tax. Consumptions rates are based on equipment consumption tables, OEM supplied consumption rates and Q'Pit information;
- Equipment consumables costs are based on OEM supplier quotes and from Q'Pit's database;
- Ground engagement tools and other wear steel consumption rates and costs have been assigned based on information for operations in similar environments;
- An accrual method for parts consumption is also used for spare parts and tire consumption costs. This averages costs through the life of the unit and does not take advantage of the lower cost when equipment is new; and

- The camp accommodation for mine personnel were developed and included with the General and Administrative costs. However, it is noted that, for the purposes of the pit limit design and Mineral Reserve estimates, the mine related camp cost is assigned to the mine operating costs for each site and the highway haulage costs for the Gordon ore.

21.6.2.2 Basis of Process Operating Cost

The following basis was used to determine the LOM process operating costs required for the LLGP in agreement with the cost definition and estimate methodologies outlined in detail below. This basis considers the development of a facility capable of processing 7,000 t/d of ore.

The assumptions made in developing the process operating cost include:

- Production is set at an average of 2.56M t annually;
- Commercial production over 10.4 years of operation;
- Operating costs are calculated based on manpower, power consumption, and process and maintenance consumables;
- Off-site gold refining, insurance and transportation costs are excluded, and are included elsewhere;
- The oxygen plant is owned;
- The sulphur dioxide plant is owned;
- Origin of workforce is Manitoba with 5% commuting from surrounding communities;
- Operating costs and any revenue from production incurred during the pre-production period until commercial production is achieved have been capitalized within Alamos. Owner’s costs;
- Consumables costs are based on data from quotes from relevant suppliers and from similar projects;
- No factor in spare parts was applied to adjust for consumption of less spare parts in early years of operation;
- Grinding media consumption rates have been estimated based on the ore characteristics;
- Reagent consumption rates have been estimated based on the metallurgical test work results; and
- Mobile equipment cost provides for fuel and maintenance, not for purchase or vehicle lease.

21.6.3 Mining Operating Costs

The LOM summary of the estimated mine operating costs for the MacLellan and Gordon open pits by primary area cost centre is shown in Table 21-7. The total operating cost for MacLellan is estimated to average $2.90/t, not including the pre-production period costs. The mine operating cost for Gordon is estimated at $3.07/t.

The costs for MacLellan include the impact of the longer haulage to the tailings management facility during periods of construction.

Table 21-7 – Life of Mine Operating Cost by Area, Excluding Pre-production Costs

Mining Cost by Area	Gordon			MacLellan			Total
	$/t	$M	Percent (%)	$/t	$M	Percent (%)	$M
Drilling	0.41	22.6	13	0.34	54.5	12	77
Blasting	0.57	31.2	19	0.54	85.6	19	116.8
Loading	0.43	23.6	14	0.41	65	14	88.6
Hauling	0.86	46.8	28	0.9	142	31	188.8
Support	0.55	30.2	18	0.41	65.6	14	95.8
Assaying	0.06	3.4	2	0.06	8.8	2	12.3
Voids Management	0	0	0	0.03	5.3	1	5.3
Miscellaneous	0.09	4.7	3	0.09	14.1	3	18.8
Engineering and Management	0.15	8	4	0.14	22.4	5	30.4
Initial Spare Parts Recovery	-0.05	-2.7	-2	-0.02	-3.6	-1	-6.3
Total Mining OPEX	3.07	167.7	100	2.9	459.6	100	627.3

This estimate is based on the consideration that all of the equipment is owned and operated by the mine owner, with a component exchange and rebuild program with the OEM suppliers as well as other parts suppliers in the vicinity of the operation.

The primary equipment operating hours were estimated for each one of the unit operations as outlined earlier in Section 16 of this report by period, on the basis of the mine plan forecasted quantities by pit. The support equipment operating hours were assigned on the basis of experience from past studies.

Several original equipment manufacturer representatives were contacted to provide budgetary quotations for the Project. The accrual method was used for the estimate of the equipment component and wear parts replacement costs. The accrual method may lead to overestimating the early part of the equipment cost cycle, when the equipment is new. This method also leads to an apparent smoothing of the estimated life of mine operating cost. Similarly, the accrual method was used for tires and this calls for the tire costs to be assigned using an hourly cost with the operating costs. As an exception, the cost of the initial set of tires is included as part of the machine acquisition capital cost.

Table 21-8 lists as an example the primary equipment operating hours and costs for Year 2 of operations for MacLellan and Gordon. These costs exclude labour costs. Similar costs are included for the support equipment, with the difference that an average annual usage for each type of machine is assigned. The diesel cost, which forms part of the hourly operating cost of each unit, is based on the diesel consumption rates from consumption tables, OEM information and Q'Pit's database. A factor from 3% to 10% of the diesel consumption was assigned for oil and lube costs, and diesel consumed during idling.

Table 21-8 – Typical Annual Primary Equipment Operating Hours and Costs (Year 2)

Primary	Class	Fleet	Year 2	Operating Costs	Fleet Annual Cost
Equipment Type		Size	h/y	($/h)	($'000s)
Hydraulic Shovel	13.5 m3	2	5,940	399	2,372
FE Loader	11.6 m3	2	3,439	226	778
FE Loader	4.5 m3	2	4,500	77	347
Haul Truck	144 t	12	30,377	197	5,985
Blasthole Drill	6,000 lbf	4	7,770	171	1,332
Blasthole Drill	539 hp	2 (multi-functional)	0	130	0
RC Drill	-		1,164	140	163
UG Voids Delineation	-		649	140	91
Hydraulic Excavator	4.6 m3	1	3,608	90	328
Highway Truck	31.5 t	23	129,881	53	11,396

The estimate of the mine operating costs includes also the following items:

- The primary crusher re-handle at MacLellan. The equipment fleet includes a 4.5 m^3 front end loader for this task;
- The long-term stockpile rehandle costs at MacLellan, which is planned to take place using the primary fleet front end loader and mine trucks;
- The operating cost estimate at Gordon includes the cost of operating a 4.5 m^3 front end loader to support/replace the backhoe used in the transhipment of the material from Gordon to the plant for 1,000 h/y per year; and
- The costs for clean-up materials for the hydraulic excavators and drills, as well as for the rental of small equipment throughout the mine life for projects.

Personnel were allocated by major function for the primary operations of loading hauling and drilling. The drilling area also includes the costs for the additional personnel for voids delineation drilling at MacLellan. Maintenance and mine operations supervision personnel were allocated to the primary unit operations.

21.6.3.1 *Explosives and Blasting Consumable Costs*

Bulk explosives, packaged explosives and blasting accessories costs are based on budget quotations received from two explosives suppliers, on the basis of the estimated quantities for the Project as outlined in Section 16 of this report. The blasting consumables for the Project consist of conventional products; 1 lb boosters, Non-Electric Delay Detonators, detonating cords for tie-in patterns, delays and lead line. Costing for blasting includes use of Blast Movement Technology to enhance selectivity and reduce or minimize mineralization losses.

21.6.3.2 *Drilling Consumable Costs*

Similarly, drilling consumable quotes are based on the budget quotation information received from a major supplier. The selection of the drills governs the drilling consumables, which consist of conventional drill tools such as drill bits, down-the-hole hammers, drill pipe, desk bushings, top thread saver subs and down-the-hole subs.

21.6.3.3 *Grade Control and Assaying Costs*

The assay costs were estimated considering a unit cost of $10.14/assay based on information supplied by the owner from their existing operations in Northern Ontario. The assay requirements by year are based on the following provisions:

- All blast holes in the mineralization zone mined using 5 m operating benches will be assayed;
- For the part of the waste zone mined in 10 m benches, 75% of the blast holes will be assayed with 1 assay per blast hole; and
- RC drilling will be carried out for the delineation of the mineralization on a pattern of 5 m x 10 m for the mineralization zone and 10 m x 20 m in the waste zone and a fixed assay interval of 2.5 m will be utilized. It can be expected that the assay frequency and in particular the RC drilling pattern, and the assay interval will be the subject of additional investigation during subsequent studies and further refinement through operations.

21.6.3.4 *Void Management Costs*

Void management costs include the delineation drilling costs as well as the drilling and explosives requirements for the controlled collapse of the ceiling of stopes, based on the patterns outlined in Section 16 of this report. These costs are site specific to the MacLellan operations.

21.6.3.5 *Diesel Consumption*

LLGP operations are based on an all diesel set of mining equipment and the operating costs estimates are affected by and are sensitive to the diesel cost. The overall diesel consumption is estimated at 0.663 L/t and 0.672 L/t mined for MacLellan and Gordon respectively. Highway haulage diesel consumption is 2.591 L/t hauled.

21.6.3.6 *Highway Haulage, Equipment and Personnel Requirements and Costs*

It is assumed that the highway haulage of the mill feed from Gordon to the primary crusher at MacLellan will be carried out by the owner.

The LOM total operating cost for the highway material loading and hauling is estimated at $9.93/t hauled. This is a separate cost item and, while it uses some mine equipment and facilities for support, it does not form part of the unit mine operating costs for Gordon. The highway haulage cost is included with the process cost for the purposes of the calculation of the cut-off grade for the Gordon ores.

21.6.3.7 *Mine Personnel Rotation*

The work schedule and rotation for the FS assumes two x 12-hour shifts with 14 days on / 14 days off rotation for the operating personnel. Accommodation and personnel transportation are carried under Accommodation within the General and Administrative cost centre.

21.6.3.8 *Mine Personnel Requirements and Costs*

The pay scale for personnel was set by the owner. A site training period allowance for newly hired mine operations personnel equivalent to 1/12 of the annual personnel cost was used.

The management and operations supervision personnel requirements for the LLGP were estimated based on information from similar operations in the Canadian North and the personnel rotation for the Project. The estimate of the mine operating personnel requirements is based on a personnel coverage factor for primary and support equipment of the equipment fleet requirements estimated as outlined in Section 16 of this report. The personnel coverage factor is as follows:

- Primary Loading equipment is covered at 100% of calendar time. This includes the primary hydraulic shovels and front-end loader, the backhoes as well as the ROM rehandle loader;
- Trucks and drills are covered at 90% and 85% respectively to cover the targeted availability. The coverage for the trucks includes all types of units, mine, articulated and highway;
- Support equipment is covered based on a selected number of units for each site and again an 85% target availability;
- Maintenance personnel are assigned using a target ratio of 0.35 maintenance personnel per operations personnel, based on experience from past projects of similar scope and size;
- Personnel estimates include a provision for 6% replacement operators for vacation and absenteeism; and
- Finally, there is a provision for seven contract personnel for the explosives facility operation and the supply of explosive down-the-hole.

Total mine personnel requirements are stable, in the range of 239 to 253 persons from Year 1 to Year 6, while both MacLellan and Gordon are operating and decline gradually thereafter. In addition, a peak of 100 personnel is estimated to be required for the highway haulage operations. The organization charts for the LLGP outlining the peak number of personnel of Mine Engineering, Mining Operations and Mine Maintenance are shown in Figure 21-1, Figure 21-2 and Figure 21-3.

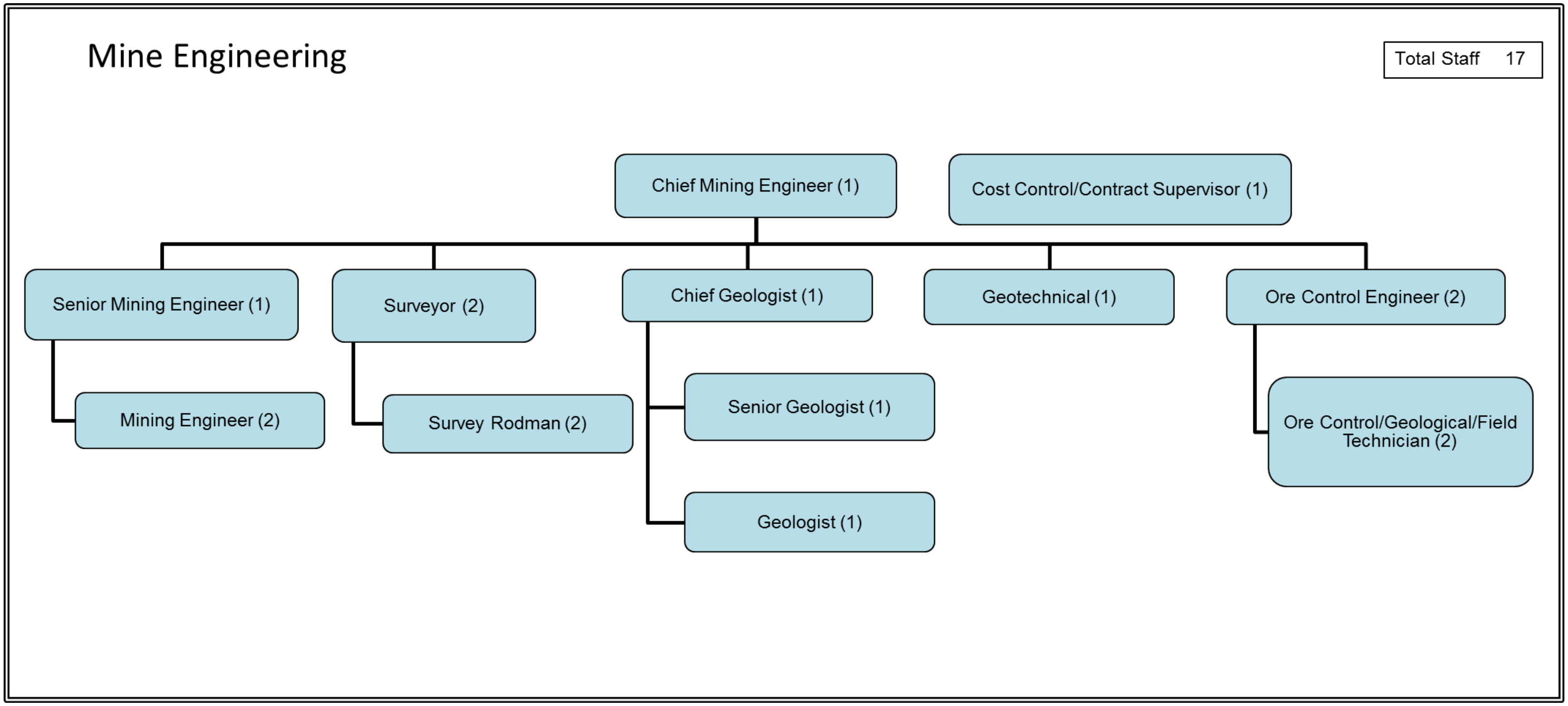


Figure 21-1: Mine Engineering Organization Chart

Source: Q'Pit (2017)

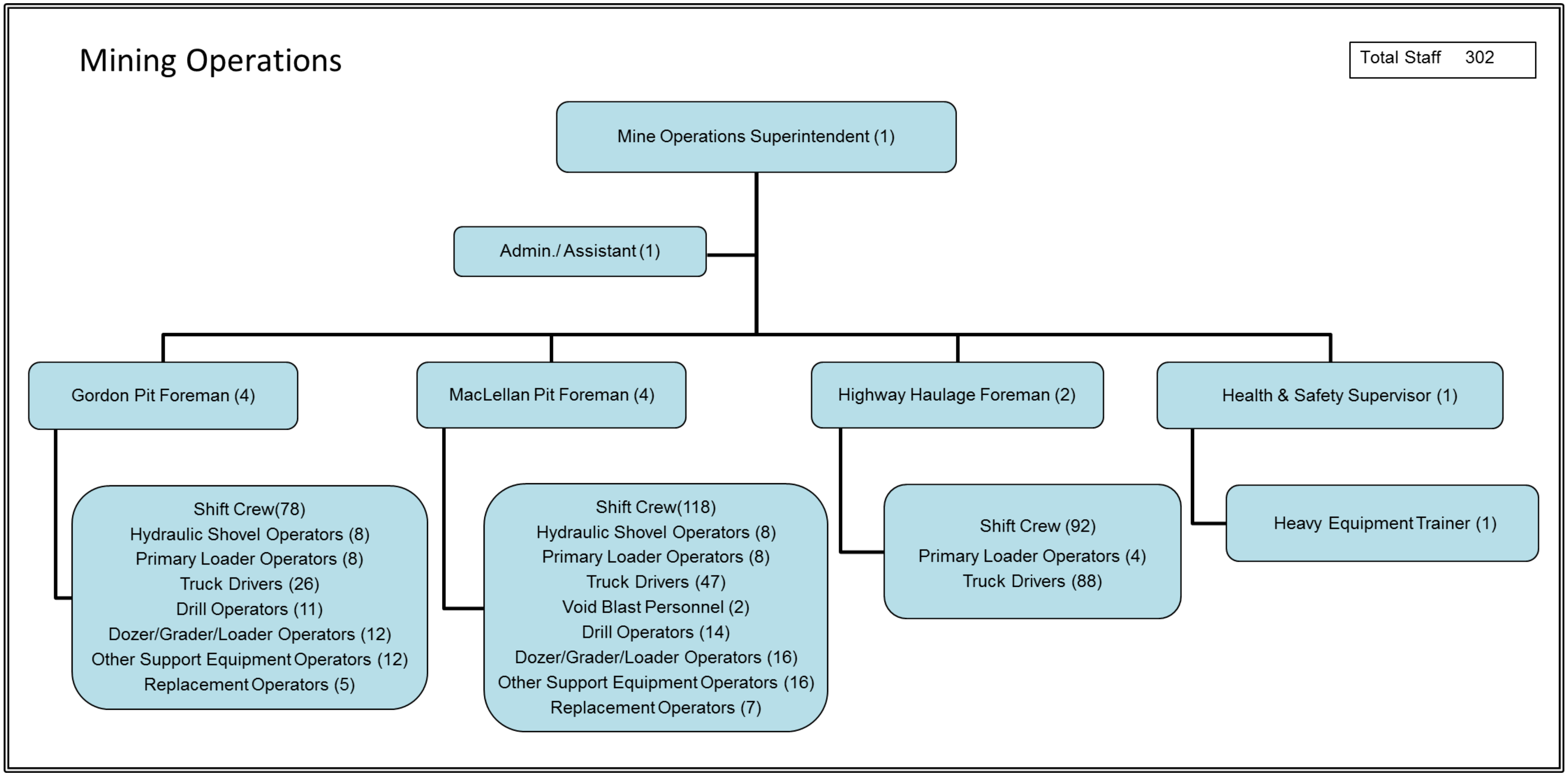


Figure 21-2: Mining Operations Organization Chart

Source: Q'Pit (2017)

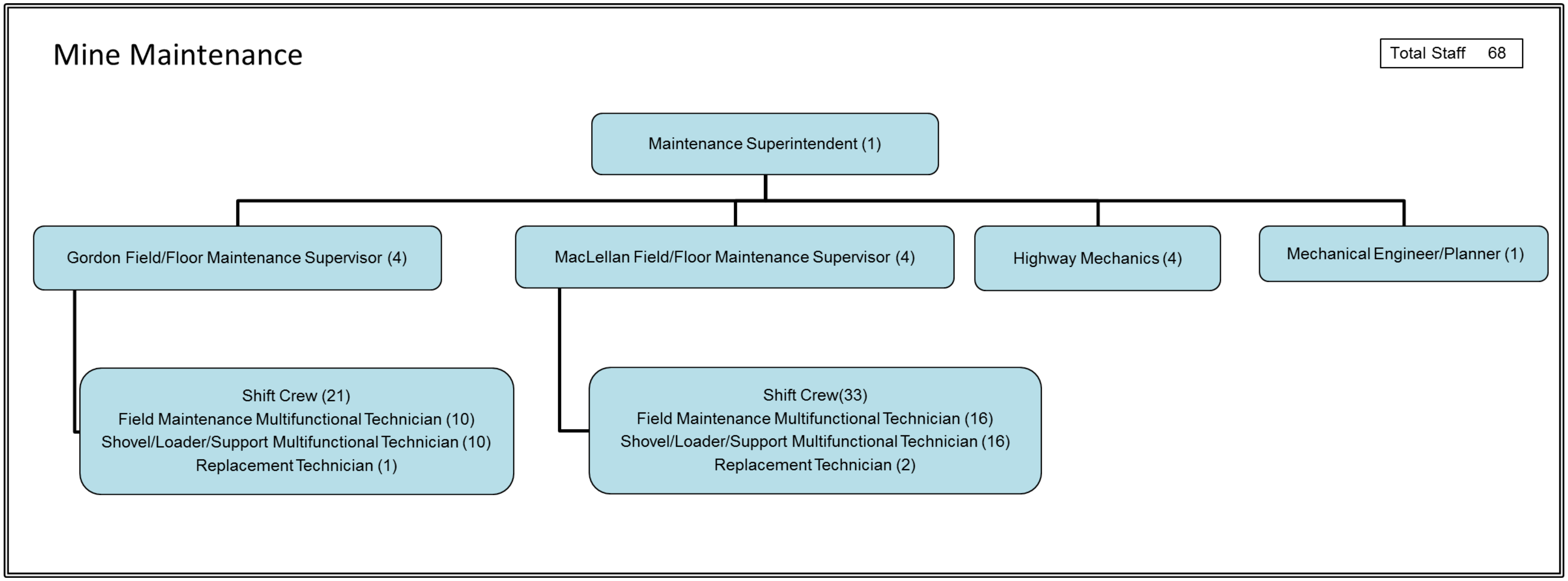


Figure 21-3: Mine Maintenance Organisation Chart

Source Q'Pit (2017)

21.6.4 Process Operating Costs

The LOM process operating cost is $382.8M or $14.46/t milled over 10.4 years. A breakdown of this OPEX value and its unit costs are presented in Table 21-9**.**

Table 21-9 – LOM Process Operating Cost

Cost Centre	$M	$/t milled	% of Total
Labour	119.1	4.5	31
Power	52	1.96	14
Operating Consumables:			
Reagents	85.8	3.24	22
Steel Liners and Ball Media	82.3	3.11	22
Utilities	6.2	0.23	2
Maintenance	26.7	1.01	7
Vehicles and Mobile Equipment	5.5	0.21	1
Laboratory and Assays	4.6	0.17	1
Water Treatment	0.6	0.02	0
Total Process Operating Costs	382.8	14.46	100

21.6.4.1 Labour

Staffing was estimated by benchmarking against similar projects. The labour costs incorporate requirements for plant operation, such as management, metallurgy, operations, maintenance, site services, assay lab and contractor allowance. The total operational labour averages 100 employees and peaks at 112 when including occasional contractors.

The individual personnel are divided into their respective positions, and are classified as either salaried or hourly based. Salaries and wages were provided by Alamos. Moreover, Alamos indicated the specific benefits and bonuses to be allocated. Thus, the rates were estimated as overall rates, including all burden costs, but do not include camp costs (included separately under Accommodation and Transport in the General and Administrative cost centre).

An organizational chart outlining the LLGP labour for the process plant can be seen in Figure 21-4.

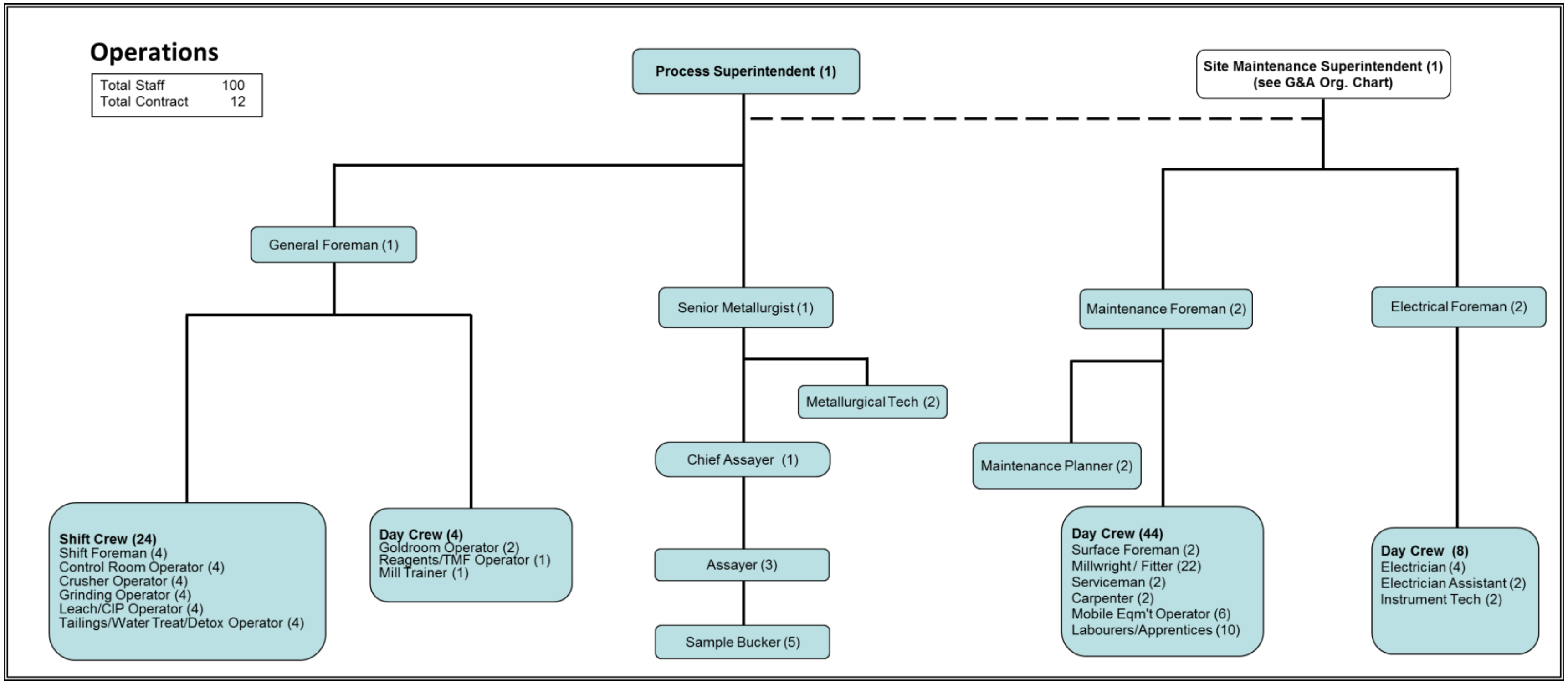


Figure 21-4: Organizational Chart for Overall Process Operations

Source: Ausenco (2017)

21.6.4.2 Power

The processing power draw was based on the average power utilisation of each motor on the equipment list for the process plant and services. Power will be supplied by the Manitoba Hydro grid to service the facilities at the MacLellan site. Diesel fired generators will supply power to the facilities at the Gordon site.

21.6.4.3 Operating Consumables – Reagents and Utilities

These operating consumables are based on reagent consumptions, and propane utilities for heating.

Individual reagent consumption rates were estimated based on the metallurgical test work results, Ausenco's in-house database and experience, industry practice, and peer-reviewed literature. Each reagent cost was obtained through vendor quotations and compared with prices among competing vendors operating in Manitoba. Therefore, the majority of reagent costs are comparable to those obtained in similar gold mining facilities. A detailed description of the reagents required for the process is in Section 17 – Recovery Methods.

Reagents represent approximately 22% of the total process operating cost at $3.24/t milled. Propane utilities represent approximately 2% of the total process operating cost at $0.23/t milled.

21.6.4.4 Operating Consumables – Steel

These operating consumables include liners for the primary and secondary crusher, SAG mill, ball mill and balls media for the mills, and were estimated using:

- Metallurgical testing results (abrasion);
- Ausenco's in-house calculation methods, including simulations; and
- Forecast total power consumption.

Steel consumables represent approximately 22% of the total process operating cost at $3.11/t milled.

21.6.4.5 Maintenance

Annual maintenance consumable costs were calculated based on total installed mechanical CAPEX by area using a weighted average factor from 1 to 5%. The factor was applied to mechanical equipment, plate work and piping. The total maintenance consumables operating cost is $1.01/t milled, or approximately 4.5% of the direct mechanical CAPEX.

21.6.4.6 Light Vehicles and Mobile Equipment

Vehicle costs are based on a scheduled number of light vehicles and mobile equipment, including fuel, maintenance, spares and tires, and annual registration and insurance fees.

21.6.4.7 Laboratory and Assays

Operating costs associated with the laboratory and assaying were estimated according to the anticipated number of assays per day and per year, provided by Alamos. Assay costs include the environmental sampling and assaying. Assay costs associated with processing mine grade control samples or exploration samples are included in the mine OPEX. The laboratory and assays make up approximately 1% of the total process operating cost, and the forecasted requirement will be around 59,000 internal assays per year for the processing plant. Approximately 1,900 samples per year are required for the environmental sampling schedule.

21.6.5 General and Administrative

General and administrative (G&A) costs are expenses not directly related to the production of gold and include expenses not included in mining, processing, external refining, and transportation costs. These costs were developed with input from Alamos based on its existing Canadian operations, and Ausenco in-house data.

A bottom-up approach was used to develop estimates for LLGP's G&A costs over the LOM. These G&A costs were determined for a 10.4-year life of mine with an average cost of $7.10/t milled. These costs were assembled according to the following departmental cost reporting structure:

- Accommodation and Transportation (camps and FIFO cost);
- G&A Personnel (including salaries, wages and business travel);
- Contract Services (including insurance, consultants, sanitation and cleaning, licence fees, freight, equipment rental and legal fees);
- Facilities Management (including vehicles, equipment, and buildings);
- Human Resources (including recruiting, training and community relations);
- Infrastructure Maintenance (including roads maintenance);
- Infrastructure Power (including power, fuel and heat);
- Site Administration, Maintenance and Security (including subscriptions, professional memberships and dues, external training, first aid, office equipment, garbage disposal, bank and payroll fees);
- IT and Telecommunications (including hardware and satellite link);
- Health and Safety (including personal protective equipment); and
- Miscellaneous (including permitting and First Nations impact benefit agreements).

The G&A labour costs were estimated by developing a headcount profile for each department which was then forecast over the life of mine. Labour rates provided by Alamos were applied to develop the total G&A labour cost.

The G&A resources includes 48 employees, 29 salaried and 19 hourly-based. Generally, the salaried staff work 40 hours per week on day shift. Hourly wage-based staff work 12-hour shift per day to support daily 24-hour operations. Figure 21-5 below, shows the G&A organisational chart.

Health and safety equipment, supplies, training and environmental costs were provided by Alamos, as were the IT and telecommunications costs for telecommunication, networking, Internet, computers, radio system and repairs.

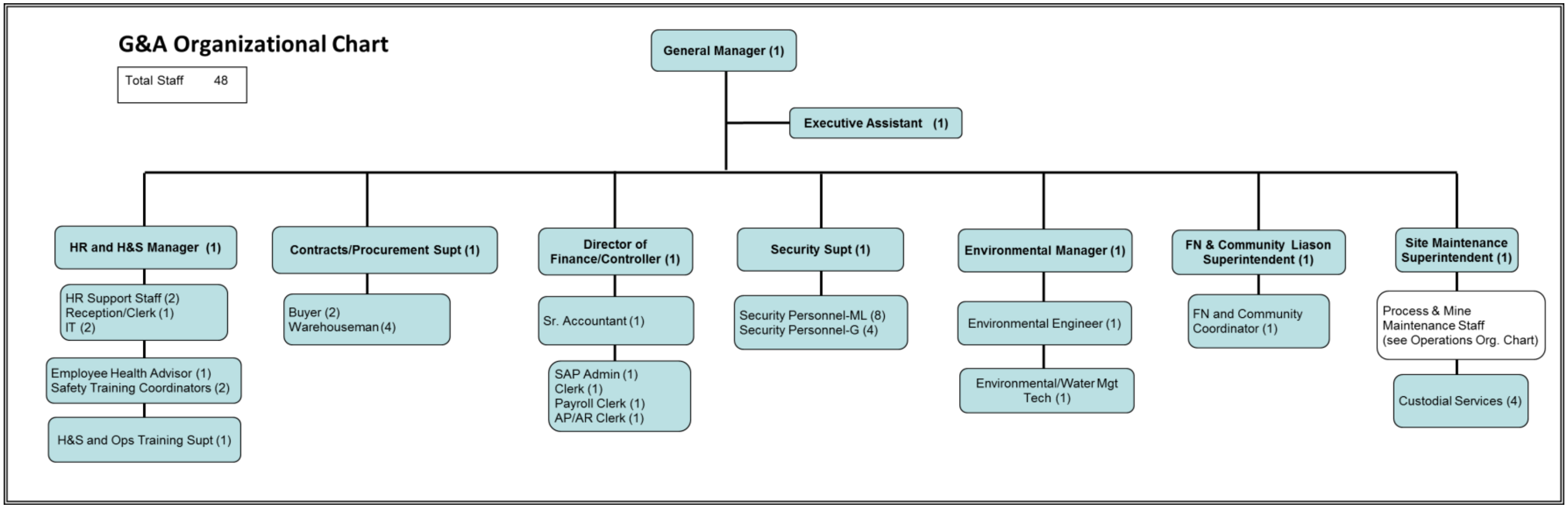


Figure 21-5: G&A Organizational Chart LLGP

Source: Ausenco (2017)

21.6.5.1 Accommodation Camp and Personnel Transportation

The operating cost for the 300-bed production camp was estimated following a cost per occupied bed-day or man-day approach. The total operational camp cost consists of the following costs:

- Accommodation and catering;
- FIFO transportation cost; and
- Local ground transportation.

The overall LOM cost is $90.32/person-day and includes accommodation and catering ($64.81/person-day), FIFO ($20.83/person-day) and local transportation ($4.69/person-day).

The camp sizing during the production phase was based on the estimate of a FTE personnel, which describes the total headcount across all areas to be constantly at site and housed in the camp. The FTE number was calculated on the following basis:

- 2x2 rotation – two out of four weeks = 50% occupancy per person;
- 3x1 rotation – three out of four weeks = 75% occupancy per person (contractor for TMF lift only);
- 4x3 rotation – four out of seven days = 57% occupancy per person; and
- 5x2 rotation – five out of seven days = 71% occupancy per person.

The operation peak bed count is 266 (34 spare beds/89% utilization). This is due to 30 additional people in camp during TMF lift construction (Years 1, 5 and 8). During typical operations, the camp occupancy is about 253 (47 spare beds/84% utilization).

A breakdown summary of LOM G&A costs is shown in Table 21-10.

Table 21-10 – LOM G&A/Accommodations Operating Cost Summary

Department / Area	$M	$/t milled	% of Total
Accommodation and Transportation	81.2	3.07	43
Personnel	53.9	2.04	29
Contract Services	19.1	0.72	10
Assets Operation	2.1	0.08	1
Human Resources	3.3	0.13	2
Infrastructure Maintenance	8.4	0.32	4
Infrastructure Power	2.5	0.09	1
Site Admin, Maintenance and Security	2.5	0.09	1
IT and Telecommunications	2.5	0.10	1
Health and Safety	0.8	0.03	0
Miscellaneous and Permitting	11.7	0.44	6
Total G&A Costs	188.1	7.10	100

22 ECONOMIC ANALYSIS

An engineering economic model was developed to estimate annual cash flows and sensitivities for the LLGP. After-tax estimates were developed to approximate the true investment value.

Sensitivity analyses were performed for variation in metal price, foreign exchange rate, operating costs, capital costs, and discount rates to determine their relative importance as Project value drivers.

The estimates of capital and operating costs have been developed specifically for this Project and are summarized in Section 21 of this report. They are presented in Q2, 2017 Canadian dollars (CAD). The economic analysis has been run with no inflation (constant dollar basis).

22.1 Assumptions

All costs and economic results are reported as CAD$, unless otherwise noted. Table 22-1 outlines the planned LOM tonnage and grade estimates.

Table 22-1 – Life of Mine Plan Summary

Parameters	Unit	Value
Mine Life	Years	10.4
Total Ore	Kt	26,803[1]
Strip Ratio	w:o	7.28
Processing Rate	tpd	7,000
Average Au Head Grade	g/t	1.89
Au Production (Years 1 to 6)	Average koz/y	170
Au Production (Years 1 to 10)	Average koz/y	143

Notes:

1. *Includes ore processed during pre-production period (26,480 kt of ore processed during the production period).*

Other economic factors and assumptions used in the economic analysis include the following:

- US $1,250/oz gold, US $16.00/oz silver and a $0.75 USD/CAD exchange rate were used in the cash flow model;
- Discount rate of 5%;
- Closure cost of $28.1M (US $21.1M);
- No salvage assumed at the end of mine life;
- Working capital outflow of $10.0M (US $7.5M) total in Year -1 and Year 1, offset by $10.0M (US $7.5M) total inflow at the end of the mine life;
- Numbers are presented on a 100% ownership basis and do not include inter-company management fees or financing costs; and

- Exclusion of all pre-development and sunk costs (i.e. exploration and Mineral Resource definition costs, engineering fieldwork and studies costs, environmental baseline studies costs, etc.). However, pre-development and sunk costs are utilized in the tax calculations.

22.2 Revenue and Working Capital

Working capital assumption of $10.0M (US $7.5M) has been accounted for in the economic analysis due to timing difference between cash inflows and cash outflows with respect to operating costs.

Mine revenue is derived from the sale of gold doré into the international marketplace. No contractual arrangements for refining exist at this time. However, the parameters used in the economic analysis are consistent with current industry rates. Gold production and sales are assumed to begin in Q4 Year -1 and continue for almost eleven years. Table 22-2 outlines the assumed sales terms used in the economic analysis. Source: Alamos (2017).

Figure 22-1 illustrates the annual recovered gold and cumulative recovered gold by project year.

Table 22-2 – Sales Terms Used in the Economic Analysis

Assumptions	Unit	Value
Au Payable	%	99.96%
Au Refining and Transportation Charge	$/oz	3.25

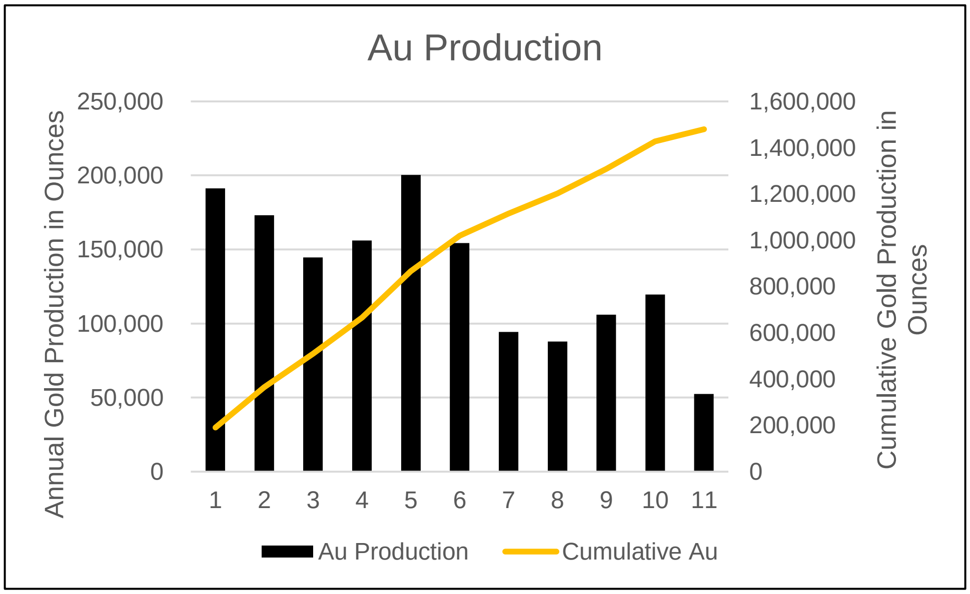


Figure 22-1: Annual and Cumulative Gold Production

Source: Alamos (2017)

22.3 Summary of Operating Costs

Total LOM operating costs, as presented in Table 22-3, amount to $1,273.2M (US $954.9M), including silver by-product credits, royalties and refining and transportation charges. This translates into an average cost of $48.08/t ore processed during the production period. A detailed analysis of the operating costs can be found in Section 21.6 of this report.

Table 22-3 – Summary of Operating Costs

Operating Cost[1]	C$/t Processed	LOM C$M	US $/t Processed	LOM US $M
Mining[2]	$23.69	$627.3	$17.77	$470.5
Haulage[3]	$3.25	$85.9	$2.43	$64.5
Processing	$14.46	$382.8	$10.84	$287.1
G&A	$7.10	$188.1	$5.33	$141.1
Refining and Transport	$0.18	$4.8	$0.14	$3.6
Sub-Total Operating Costs	**$48.68**	**$1,289.0**	**$36.51**	**$966.7**
Silver Credit	($1.01)	($26.7)	($0.75)	($20.0)
Royalties	$0.41	$10.8	$0.31	$8.1
TOTAL Operating Costs	$48.08	$1,273.2	$36.06	$954.9

Notes:

1. *Operating costs exclude working capital*
2. *Average mining cost during the production period is $2.94/t mined (US $2.21/t mined) with a strip ratio of 7.06:1 (7.28:1 including pre-commercial production)*
3. *Haulage costs are reported per total tonne milled. Haulage costs per tonne of Gordon ore hauled average $9.93 (US $7.45)*

22.4 Summary of Capital Costs

The capital costs were used for the economic analysis are set out below. The pre-production period spans 24 months, including EPCM and access road construction. Table 22-4 summarizes the capital costs used in the economic analysis. Detailed information can be found in Section 21 of this report.

Alamos assessed the option of leasing primary, support and miscellaneous equipment required over the LOM, and concluded that this option improves the economics of the Project. The financial model reflects leasing equipment and making principal and interest payments starting in Year -1. Payments made during the construction period are reflected under initial capital, and those made starting in Year 1 and beyond are reflected under sustaining capital.

Table 22-4 – Summary of Capital Costs

Capital Cost	Pre-Production C$M	Production / Sustaining C$M	LOM C$M	Pre-Production US $M	Production / Sustaining US $M	LOM US $M
Mining	$81.7	$137.6	**$219.4**	$61.3	$103.2	**$164.5**
Process Plant	$96.4	$0.0	**$96.4**	$72.3	$0.0	**$72.3**
Utilities and Services	$22.8	$4.8	**$27.6**	$17.1	$3.6	**$20.7**
Onsite Infrastructure	$55.9	$0.0	**$55.9**	$41.9	$0.0	**$41.9**
Offsite Infrastructure	$32.7	$0.0	**$32.7**	$24.5	$0.0	**$24.5**
Tailings Management	$22.8	$23.1	**$45.8**	$17.1	$17.3	**$34.4**
Indirects	$54.6	$3.3	**$57.9**	$40.9	$2.4	**$43.4**
EPCM	$28.9	$0.0	**$28.9**	$21.7	$0.0	**$21.7**
Owner's Cost	$16.3	$0.0	**$16.3**	$12.2	$0.0	**$12.2**
Subtotal	**$412.2**	**$168.8**	**$580.9**	**$309.1**	**$126.6**	**$435.7**
Contingency	$38.4	$0.0	**$38.4**	$28.8	$0.0	**$28.8**
Rehab and Closure	$0.0	$28.1	**$28.1**	$0.0	$21.1	**$21.1**
Total Capital	**$450.6**	**$196.9**	**$647.5**	**$338.0**	**$147.6**	**$485.6**

22.5 Reclamation and Mine Closure

The reclamation and mine closure plan is set out in in Section 21.5.4. The current plan anticipates a cost of $28.1M (US $21.1M) for reclamation and closure, starting with small activities in Year 2 and ramping up in Year 5 as the Gordon deposit approaches the end of its mine life. In Manitoba, reclamation costs are set aside in the form of bonding, or other means, as the liabilities are incurred or as the area to be reclaimed in the future is first disturbed. The bulk of the closure costs and reclamation activity will actually occur after Year 11, after mining and processing has been completed at MacLellan.

22.6 Taxes

The LLGP will be subject to provincial, federal and mining taxes as follows:

- Manitoba Mining Tax: sliding scale with rates between 10% and 17%;
- Manitoba Provincial Income Tax: 12%;
- Federal Income Tax: 15%; and
- Manitoba Provincial Sales Tax: 8%.

The rates above are current as of the date of this report, and are subject to change in the future. Based on these rates and the financial assumptions used in this report, the LLGP is expected to have payable income and mining taxes of $174.2M (US $130.6M) over its 10.4-year life.

Alamos expects to pay the Manitoba PST on all taxable goods or services purchased or rented for its own use, and this includes large vehicles and equipment, SAG mill, ball mill, concentrator, conveyor belts, electrical, HVAC systems and computers. The FS includes an estimate of PST to be paid, with that paid on equipment included in the capital lease payments and the remaining payable PST grouped under indirect costs of initial capital.

22.7 Royalties

The LLGP is subject to a third-party royalty in the first two years of production from the Gordon pit. Total royalty included in the cash flow model is $10.8M (US $8.1M).

22.8 Economic Analysis

The Project is economically viable with an after-tax internal rate of return (IRR) of 12.5% and an after-tax net present value at 5% (NPV$^{5\%}$) of $164.5M (US $123.4M).

Figure 22-2 shows the projected cash flows used in the economic analysis and based on the assumptions in Section 22.1. Table 22-5 shows the detailed results of this evaluation.

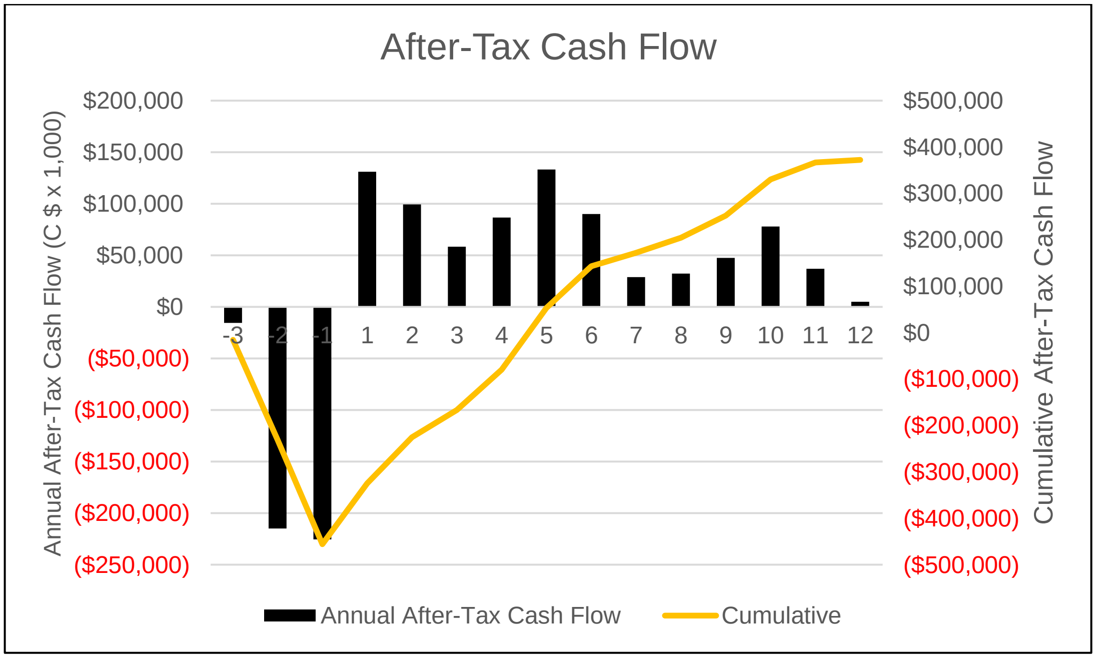


Figure 22-2: Annual and Cumulative After-Tax Cash Flow in C $

Source: Alamos (2017)

Table 22-5 – Summary of Economic Results

Category	Unit	Value (C $)	Value (US $)
Net Revenues	$M	$2,466.8	$1,850.1
Operating Costs[1]	$M	$1,273.2	$954.9
Cash Flow from Operations[2]	$M	$1,019.5	$764.6
Capital Costs, Sustaining Capital and Closure Costs	$M	$647.5	$485.6
Total Cash Cost	$/oz	$860	$645
Mine Site All-In Sustaining Cost	$/oz	$993	$745
Net After-Tax Cash Flow	$M	$372.0	$279.0
After-Tax NPV$^{5\%}$	$M	$164.5	$123.4
After-Tax IRR	%	12.5%	12.5%
After-Tax Payback	Years	4.6	4.6

Notes:

1. *Operating Costs include mining, processing, G&A, royalties, transport and refining costs and silver credit*
2. *Cash Flow from Operations includes changes in working capital*

22.9 Sensitivities

A sensitivity analysis was performed to test project value drivers on the Project's NPV using a 5% discount rate. The results of this analysis are demonstrated in Table 22-6 and Table 22-7, and illustrated in Figure 22-3. The Project proved to be most sensitive to changes in metal price followed by foreign exchange, capital costs and operating costs. A sensitivity analysis of the after-tax results was performed using various gold prices. The results of this analysis are demonstrated in Table 22-8.

Table 22-6 – After-Tax NPV$^{5\%}$ Sensitivity Results

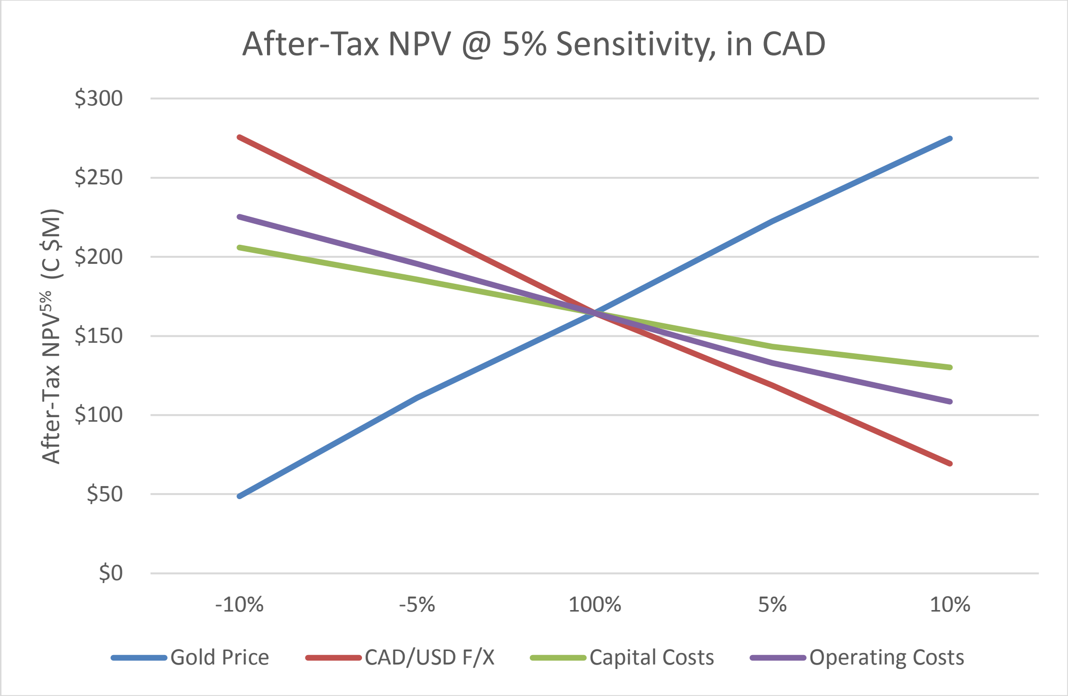
	After-Tax NPV$^{5\%}$, Millions of CAD				
	-10%	-5%	100%	5%	10%
Gold Price	$48.7	$111.0	$164.5	$222.4	$274.8
CAD/USD F/X	$275.6	$220.2	$164.5	$118.8	$69.4
Capital Cost	$205.9	$185.6	$164.5	$143.3	$130.0
Operating Cost	$225.3	$195.6	$164.5	$132.9	$108.6

	After-Tax NPV$^{5\%}$, Millions of USD				
	-10%	-5%	100%	5%	10%
Gold Price	$36.5	$83.2	$123.4	$166.8	$206.1
CAD/USD F/X	$186.0	$156.9	$123.4	$93.5	$57.2
Capital Cost	$154.4	$139.2	$123.4	$107.5	$97.5
Operating Cost	$169.0	$146.7	$123.4	$99.7	$81.4

Table 22-7 – Gold Price Sensitivity on NPV and IRR

Gold Price	After-Tax NPV (C$M)	After-Tax NPV (US $M)	After-Tax IRR (%)
$1,100	$23.3	$17.5	6.1%
$1,200	$123.6	$92.7	10.6%
$1,250	**$164.5**	**$123.4**	**12.5%**
$1,300	$211.1	$158.3	14.5%
$1,400	$297.0	$222.7	18.0%
$1,500	$386.4	$289.8	21.5%

Source: Alamos (2017)



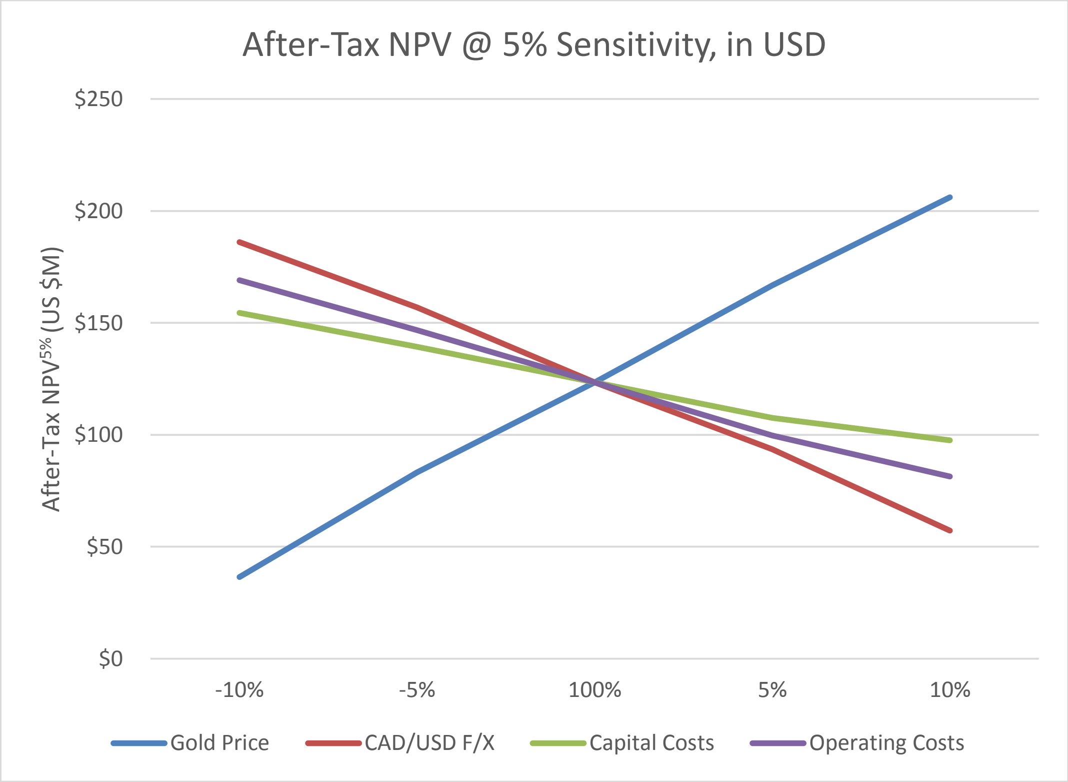


Figure 22-3: After-Tax NPV$^{5\%}$ Sensitivity Results

Source: Alamos (2017)

Table 22-8 – Lynn Lake Project Financial Model Summary in USD

(thousands of USD dollars)	LOM	Y-3	Y-2	Y-1	Y+1	Y+2	Y+3	Y+4	Y+5	Y+6	Y+7	Y+8	Y+9	Y+10	Y+11	Y+12
Assumptions																
Gold Price (US$/oz)	$1,250	$1,250	$1,250	$1,250	$1,250	$1,250	$1,250	$1,250	$1,250	$1,250	$1,250	$1,250	$1,250	$1,250	$1,250	$1,250
Silver Price (US$/oz)	$16.00	$16.00	$16.00	$16.00	$16.00	$16.00	$16.00	$16.00	$16.00	$16.00	$16.00	$16.00	$16.00	$16.00	$16.00	$16.00
CAD to USD	$0.75	$0.75	$0.75	$0.75	$0.75	$0.75	$0.75	$0.75	$0.75	$0.75	$0.75	$0.75	$0.75	$0.75	$0.75	$0.75
Physicals																
Lynn Lake Open-Pit Mining																
Tonnes Mined																
Ore	26,803	0	0	371	2,708	2,878	2,711	2,951	3,647	2,524	1,866	1,813	2,507	2,110	715	0
Overburden	7,604	0	623	1,683	2,155	977	0	0	1,995	170	0	0	0	0	0	0
Waste	187,584	0	899	5,328	15,661	23,194	24,301	21,620	21,173	23,227	21,263	15,197	11,241	3,895	585	0
Total Material Moved	221,991	0	1,523	7,383	20,524	27,049	27,012	24,570	26,816	25,921	23,129	17,010	13,749	6,006	1,300	0
Strip Ratio (Waste+OB:Ore)	7.28	0.00	0.00	18.87	6.58	8.40	8.96	7.33	6.35	9.27	11.39	8.38	4.48	1.85	0.82	0.00
Au Grade Mined	1.89	0.00	0.00	1.69	2.40	2.07	1.85	1.87	2.06	2.05	1.42	1.40	1.43	1.80	2.31	0.00
Ag Grade Mined	2.99	0.00	0.00	2.35	2.03	2.21	1.30	1.52	2.41	4.69	4.72	3.80	3.86	4.47	5.65	0.00
Lynn Lake Milling																
Tonnes Milled	26,803	0	0	322	2,555	2,555	2,555	2,555	2,555	2,555	2,555	2,555	2,555	2,555	930	0
Au Grade	1.89	0.00	0.00	1.59	2.51	2.28	1.91	2.06	2.63	2.04	1.27	1.18	1.42	1.59	1.91	0.00
Au Recovery	92.0%	0.0%	0.0%	91.3%	92.9%	92.4%	92.1%	92.2%	92.6%	92.2%	90.8%	90.7%	91.1%	91.3%	91.7%	0.0%
Au Ounces Produced	**1,495,115**	**0**	**0**	**15,006**	**191,221**	**173,209**	**144,705**	**156,117**	**200,299**	**154,285**	**94,412**	**87,932**	**105,910**	**119,515**	**52,505**	**0**
Ag Grade	2.99	0.00	0.00	2.70	1.97	2.08	1.38	1.42	2.67	4.66	3.53	3.50	3.84	4.17	4.99	0.00
Ag Recovery	49.0%	0.0%	0.0%	49.0%	49.0%	49.0%	49.0%	49.0%	49.0%	49.0%	49.0%	49.0%	49.0%	49.0%	49.0%	0.0%
Ag Ounces Produced	**1,263,276**	**0**	**0**	**13,735**	**79,303**	**83,561**	**55,503**	**57,330**	**107,642**	**187,575**	**142,122**	**140,888**	**154,482**	**168,041**	**73,093**	**0**
Financials																
Total Gold Revenue	**$1,850,137**	**$0**	**$0**	**$0**	**$239,026**	**$216,511**	**$180,881**	**$195,146**	**$250,373**	**$192,857**	**$118,015**	**$109,914**	**$132,388**	**$149,393**	**$65,631**	**$0**
Operating Costs																
Total Mining	$534,938	$0	$0	$0	$59,676	$64,277	$66,953	$66,606	$65,624	$55,263	$51,106	$39,977	$34,818	$21,581	$9,058	$0
Processing	$287,081	$0	$0	$0	$27,508	$27,045	$27,045	$27,492	$27,042	$27,042	$27,263	$27,038	$27,038	$27,263	$15,308	$0
Administration	$141,088	$0	$0	$0	$13,871	$13,798	$13,791	$13,924	$13,905	$13,626	$11,935	$12,315	$11,562	$11,164	$11,197	$0
Total Selling Costs and Credits	($16,357)	$0	$0	$0	($801)	($913)	($534)	($535)	($1,232)	($2,621)	($2,040)	($2,037)	($2,210)	($2,394)	($1,040)	$0
Total Royalty	$8,135	$0	$0	$0	$7,701	$433	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL Cash Costs	**$954,885**	**$0**	**$0**	**$0**	**$107,956**	**$104,640**	**$107,254**	**$107,486**	**$105,339**	**$93,310**	**$88,263**	**$77,292**	**$71,208**	**$57,614**	**$34,523**	**$0**
Working Capital	$0	$0	$0	$3,750	$3,750	$0	$0	$0	$0	$0	$0	$0	$0	$0	($3,750)	($3,750)
Mining Taxes	$24,705	$0	$0	$0	$0	$0	$0	$0	$10,115	$3,457	$0	$0	$2,126	$8,681	$326	$0
Income Taxes	$105,917	$0	$0	$0	$0	$0	$0	$9,049	$29,182	$20,251	$3,582	$5,328	$12,741	$19,638	$6,146	$0
Operating Cash Flow	**$764,630**	**$0**	**$0**	**($3,750)**	**$127,320**	**$111,871**	**$73,627**	**$78,611**	**$105,738**	**$75,838**	**$26,170**	**$27,295**	**$46,313**	**$63,459**	**$28,386**	**$3,750**
Capital																
Sustaining Capital	$147,649	$0	$0	$0	$28,934	$37,302	$29,751	$13,582	$5,841	$8,317	$4,387	$3,146	$10,651	$5,006	$732	$0
Initial Capital	$337,964	$11,723	$161,066	$165,175	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Total Capital	$485,613	$11,723	$161,066	$165,175	$28,934	$37,302	$29,751	$13,582	$5,841	$8,317	$4,387	$3,146	$10,651	$5,006	$732	$0
Free Cash Flow	**$279,016**	**($11,723)**	**($161,066)**	**($168,925)**	**$98,386**	**$74,569**	**$43,876**	**$65,030**	**$99,897**	**$67,521**	**$21,783**	**$24,148**	**$35,662**	**$58,454**	**$27,654**	**$3,750**
After-Tax NPV	**$123,378**															
After-Tax IRR	**12.5%**															
Total Cash Costs per oz	$645	$0	$0	$0	$565	$604	$741	$688	$526	$605	$935	$879	$672	$482	$658	$0
All-In Sustaining Cost per oz	$745	$0	$0	$0	$716	$819	$947	$775	$555	$659	$981	$915	$773	$524	$671	$0

Source: Alamos (2017)

23 ADJACENT PROPERTIES

There are no operating properties of significance adjacent to the LLGP.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 Project Execution and Organization

24.1.1 Summary

The Project Execution Plan (PEP) will address the overall Project (objectives, scope, strategies and roles and responsibilities) and provide a comprehensive plan for the development and implementation of LLGP. The PEP covers the plan for engineering, procurement, construction, startup and commissioning of the Project.

The PEP will be followed to accomplish the following objectives:

- Develop and deliver a Project that meets the needs of the shareholders and the community with zero accident and minimum impact on the environment;
- Design for a safe, reliable, environmentally compliant facility to meet construction, commissioning, and operating requirements;
- Construct the Project based on the efficient and economic use of energy and materials seeking to minimize the Project environmental footprint and reduce costs;
- Delivery of a capital efficient Project in compliance with all environmental requirements;
- On-schedule completion; and
- Meet or improve targeted Feasibility Study capital and operating cost estimates during the Execution Phase to maintain the viability of the Project.

The Project schedule has been developed based on continuation of the integrated Alamos and EPCM team approach used during the FS, with each party contributing in areas of their respective strengths. Alamos will provide the overall leadership to make key Project decisions and manage community, environmental, permitting, local authorities and security whilst EPCM will provide engineering, procurement, management and execution personnel that are experienced in cost effective Project delivery in accordance with both Canadian/Manitoba design standards.

The implementation strategy assumes an EPCM implementation with construction packages and a number of smaller EPC packages where either local contractor or specialist technology suppliers have demonstrated cost benefits to the Project.

The execution strategy is to complete the Project within 24 months of receiving the Environmental and Alamos' corporate approval to proceed.

The execution phase has been split into three sub-phases to maintain the forecast ramp-up of metal production milestone. The following phased staged development to full project execution is recommended:

- Phase 1 – Optimization;
- Phase 2 – Early Works; and
- Phase 3 – Full Project Execution.

The bulk of the initial Project execution effort will be undertaken in the EPCM Toronto office with a gradual transfer of all activities to the site office. A summary of the key activities performed from the three Project office locations is as follows:

- EPCM office
    - EPCM's office will act as the hub during the Project set-up, engineering design and early procurement phase. Overall Project management will commence in the home office and will transition to site as the detail design and procurement phases draw to a conclusion.
- Alamos' office in Lynn Lake
    - Alamos and EPCM will establish a local team in Alamos' current Lynn Lake office to manage all local issues and facilitate mobilization up to the full transition to the site office.
- EPCM's site office at Lynn Lake site
    - Alamos and EPCM will establish an initial site team utilising a site office facility that will be located to the north of the open pit to manage all aspects of the early works program. The site team will expand as construction activities intensify up to the point when the full Project management and construction management team resides on-site. When the process plant bulk earthworks are completed the office will be transferred to the process plant laydown area.

Phase 1 – Optimization

Phase 1 will provide an optimized plant design to take advantage of some of the opportunities identified in the FS. Opportunities identified in the FS created the potential for optimizing the proposed discipline engineering design and reduce CAPEX before proceeding to detail engineering Project execution.

Phase 2 – Early Works

Phase 2 will provide basic engineering for early works procurement of long-lead items. This will expedite the project schedule, allowing the procurement team to expedite up front vendor engineering and place equipment fabrication purchase orders by the completion of permit approvals.

The early works program is essential since it enables:

- Critical path activities to be completed prior to winter, and
- Mining and plant site earthworks to commence immediately after environmental and final Project approval.

The early works program progresses the following key critical path activities:

- Completion of geotechnical and hydrogeological field investigations and subsequent analysis to support mine, TMF and process plant detail design;
- Finalisation of detail site survey;
- Detailed design and contractor selection to enable immediate award and mobilisation of the earthworks contractor(s);
- Design and specification of critical procurement packages in preparation for tendering; and

- Commence additional definition and contractor engagement to explore potential benefits of firm price vertical construction packages and/or a combination of EPC and design and construct packages.

Phase 3 – Full Execution Program

Phase 3 is planned for detailed engineering, equipment delivery, and construction.

The full Project execution is phased under four groups as follows:

Detailed Engineering and Procurement

Engineering and design works:

- Assess risks and opportunities remaining from the Feasibility Study and early works package;
- Advance design in accordance with the Feasibility Study objectives to provide a capital-efficient design that presents Alamos with the optimal capital and operating cost outcome;
- Issue procurement and contract packages;
  - Implement the packaging plan and strategy developed during the early works phase to deliver against the agreed cost, schedule and risk/reward profile agreed for the Project;
  - Continue procurement with associated engineering support to develop, tender, award and manage long lead time, specialist equipment, material, fabrication and site works orders and contracts for and on behalf of Alamos; and
- Refinement of Project delivery planning including formation of Project budget / control estimate and schedule and reporting post the early works phase.

Construction

Construction activities include:

- Site facilities will be established with associated infrastructure to allow the construction activities to be conducted safely, and efficiently;
- Demarcation planning will address access and permitting requirements and Health Safety Environment and Community (HSEC) management protocols between construction activities and any parallel operations activities;
- Pre-planning for construction laydown areas, access/egress requirements from work areas and geotechnical condition assessment to support the construction phase;
- Construction will include earthworks, concrete placement, tank erection, steel erection, plate work, equipment and piping installation, electrical power supply, distribution, instrumentation and control; and
- QA/QC processes and implementation leading to final construction verification testing, punch list completion and handover.

Commissioning

Commissioning activities include:

- The commissioning activities flowing from construction completion will include development of performance testing criteria and a test program and delivery of the C0 (construction verification and validation), C1 (dry commissioning) and C2 (wet commissioning) phases, after which responsibility will be handed over to the Alamos Operations team who will undertake C3 (ore commissioning); and
- EPCM will support Alamos in the ore commissioning, performance testing hand over and operational ramp-up of facilities to achieve target production levels.

Detailed commissioning planning will commence during Detailed Engineering to drive the appropriate completion sequencing from construction and engineering.

Operational Readiness and Ramp-up

Concurrent with the engineering and construction phases, Alamos will develop:

- A spares holding strategy and associated maintenance strategy which will drive the spares purchasing, catalogue and storage; and
- Competency benchmarking for operations personnel required to operate the plant, customized training documentation and operating procedures which align with the intended operating philosophy.

Alamos will take responsibility of the facilities at the commencement of C3 commissioning supported by EPCM to achieve performance tests

Alamos technical and support staff will be integrated into functional roles with in the EPCM execution team to facilitate early learning and transfer of knowledge.

24.1.1.1 Proposed LLGP Implementation Strategy

The FS was based on the premise of implementing the Project on an EPCM basis, maximising the strengths of both Alamos and the EPCM. This approach provides Alamos with the required time, during the early works period, to solicit input from potential financiers and develop the required additional level of Project definition to engage potential secondary EPC or Design and Build contractors to inform the basis to finalize the execution strategy prior to full Project approval.

The implementation strategy assumes an EPCM implementation. The head contract is between the Owner and the EPCM contractor. The EPCM contract then manages horizontal construction packages and a number of smaller secondary packages where either local contractors or specialist technology suppliers have demonstrated cost benefits to the Project.

During the early works phase opportunities will be explored to increase the portion of the Project that could be undertaken by EPC and or Design and Build contractors, including vertical construction packages for areas of the process plant.

The power line will be designed and constructed by Manitoba Hydro and out of the scope of the EPCM contractor.

24.1.1.2 Key Assumptions

The Project execution strategy has been based on the following assumptions:

- Sufficient construction resources are available locally and within the province of Manitoba to undertake the work;
- Site work will be 21 days on / seven days off with ten hours per day shifts;
- The Project cannot proceed until the final environmental permits have been received; and
- The existing roads and bridges will support the delivery of goods and materials to site.

24.1.1.3 Constraints and Limitations

Climatic

Due to the significant issues associated with the ground conditions, bulk earthworks need to be undertaken in the winter months. Construction activities have been sequenced to consider the northern weather conditions and outdoor construction windows.

Ideally, for the economical placement of concrete, all major foundations should be poured outside of the winter months. Concrete can be placed year-round, provided a management plan is executed to manage the temperature of the mixed concrete and the temperature profile of the concrete during the hydration period when the plastic mix solidifies and gains strength. For the purposes of normal economic concrete placement, sequencing of concrete placement should:

- Be limited to March to October; and
- Retain insulated formwork in place for as long as possible.

Material selection and joint design will carefully consider the extreme temperature ranges in the region and associated thermal expansion. Construction activities have been sequenced to consider the following:

Environmental

Construction activities have been sequenced to consider environmental constraints inclusive of non-disturbance during the bird nesting season.

The LLGP will likely require particular attention to water management during construction.

Water to be managed will be both surface and groundwater and issues to be managed include sediment management and water discharge conditions.

Logistics

There are currently two modes of transportation to the Project sites; road via PR 391 and air via the Lynn Lake Airport.

PR 391 is a double lane road parts of which are subject to thaw.

Lynn Lake Airport is open year-round and is serviced by charter flights.

A previously installed rail connection has been removed from the Town.

Permits

Construction and operation of the mine, plant and associated infrastructure may not take place until environmental permits have been received.

24.1.1.4 Procurement and Contracting Strategy

A Procurement and Contracting Plan will be developed in conjunction with Alamos. This will outline all roles and responsibilities, procedures, systems and forms to be used to implement the Project.

Procurement Plan

The Procurement Plan will also outline the expediting processes and vendor communication procedures. The overall strategy will include:

- Early engagement on all major packages, risks and opportunities;
- Vendor selection through exhaustive prequalification process; vendors and contractors will be specifically prequalified unless directed otherwise by ensuring that pricing schedules are a true reflection of the Project scope;
- Verification of engineering scopes of work, to validate definition and to enable comprehensive assessment of contingencies;
- Management of forecast at completion dates on a progressive basis;
- Ensuring purchase orders incorporate payment items against vendor data; and
- Expediting will be undertaken by the EPCM contractor.

Purchase orders for vendor engineering of long-lead items will be placed during Phase 2 to meet the target Project completion date. Upon receipt of permits and full Project sanction, purchase orders for long-lead items will be executed together with the completion of the procurement of all other packages.

Contracting Strategy

The Contracting Plan will detail the construction strategy for the Project. The Contracting Strategy Sub-Plan assumes the proposed breakup of packages. Construction contractors will:

- Provide labor, plant and materials required for the execution of allocated packages of work;
- Supply their own first-aid facilities on-site (emergency medical care may be supported by Alamos first responders pending transfer to Lynn Lake Regional Hospital); and
- Supply all temporary facilities including construction consumables, tools and equipment.

The contracting strategy outline has been developed for the purpose of the initial Project establishment based on:

- Understanding the regional contractual capabilities from a technical, financial and managerial perspective;
- An understanding of the time frame for Project delivery and particularly sequencing of the works; and

- The mitigation of Project risks, particularly in respect to delivery of equipment and materials to site.

The key factors that influence the contracting strategy are:

- A desire to introduce competitive tension on-site where possible by having more than one contractor on-site capable of doing multiple scopes of work. This will be achieved by either having more than one discipline contractor on-site or having at least one general contractor. This will allow good performance to be rewarded by potentially adding scope or poor performance resulting in de-scoping of work; and
- The engagement of Manitoba based contractors has a number of advantages, including:
    - Regional and site familiarity including understanding of the customs and weather;
    - Capacity to readily draw from the local employment pool; and
    - Capacity to mobilise quickly to site to undertake the works.

To increase the attractiveness of the Project to larger contractors it may be necessary to combine contract scopes or consider use of a general contractor (a contractor able to perform the bulk of the major work disciplines). Advantages with general contractors and/or contract rationalisation include:

- Rationalisation of contract interface; and
- Less reliance on the EPCM team in supporting contractor supervision.

24.1.1.5 Logistics Strategy

This section deals with the transport overview for movement of cargo from Europe, Asia and North America to Lynn Lake, Manitoba covering routing information on cargo originating from Canada, the United States, Europe, and China. The summary of the logistics plan is as follows:

- EPCM will engage a third-party logistics contractor;
- Goods will be shipped from North America, Europe and Asia based on INCOTERMS 2010;
- If the delivery terms are other than DDP (Incoterms 2010), a third-party logistics provider will be engaged to transport the equipment to site;
- Delivery terms for purchases within Canada are FCA (Incoterms 2010) Loaded on Transport and delivered to site;
- Delivery terms for purchases outside of Canada other than DDP (Incoterm 2010), the third-party logistics contractor will be engaged to transport the equipment to site;
- Part loads and goods delivered prior to the scheduled installation date on-site will be marshaled in Winnipeg; and
- Oversized/specialized transport loads will be delivered direct to site.

Contractor Logistics Management

In general, contractors performing Services on site will expedite the delivery of their equipment and materials to site via their own, standard processes, with responsibility for off-loading and storage into their own holding areas.

24.1.2 Project Organization

24.1.2.1 Organization and Resourcing

The Project Organization is based on an integrated team approach with active attention to minimising the duplication of roles and activities between EPCM and Alamos.

The organization chart for the Project team is shown in Figure 24-1. The Project Organization Chart is a live document, constantly updated as the Project progresses.

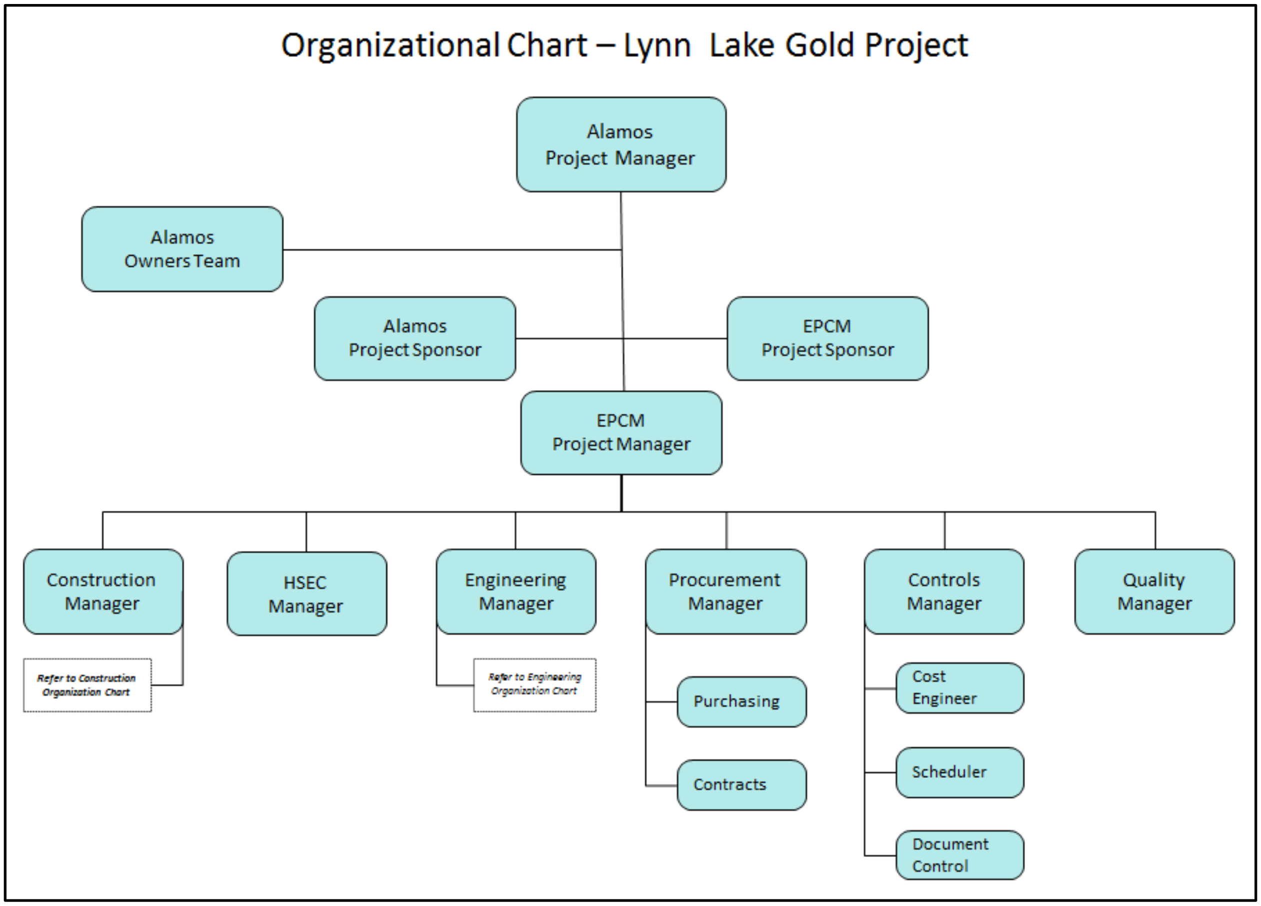


Figure 24-1: Organizational Chart Lynn Lake Gold Project

Source: Ausenco (2017)

The EPCM Project Manager will be responsible for scope, cost and schedule with the Project services, including engineering, Project controls, contracts, construction management and commissioning providing cross Project services to provide a consistent, efficient and holistic outcome for the Project. The Project will be established based on EPCM's project delivery systems and processes. The government liaison, external affairs and environmental services will be provided by Alamos, as well as the overall site security.

24.1.2.2 Alignment Strategy

The Project alignment strategy aims to create shared understanding of the Project vision and strategy to enable the EPCM, Alamos and other internal/external stakeholders to achieve the Project objectives. The overall Project delivery team operates as one team with defined responsibilities, accountabilities and authorities. The team will be established and supported to deliver 'Best for Project' outcomes in line with LLGP Project expectations and critical success factors.

Formal alignment sessions will be scheduled at the commencement of each new phase of the Project. They will be held in the Alamos office to best suit the Project needs. Alignment of the broader team will be achieved by a comprehensive induction program when team members first join the Project and through regular refresher sessions at key times during the Project.

24.1.2.3 Project Governance and Stakeholders

To provide corporate engagement from the various project participants, the Project Sponsors Group will commence its role in Phase 2 of the Project.

24.1.2.4 Project Sponsors' Group

The Project Sponsors' Group will include participants from EPCM and Alamos. The purpose of the Project Sponsors' Group is to enact corporate commitment to the Project through establishing a common purpose in achieving the Project goals, managing events outside of the control of the Project team and supporting the Project in resolution of issues.

The Sponsors will support the combined Alamos-EPCM Project team to protect capital savings and schedule opportunities identified during the Feasibility Study phase to improve the viability of the Project and to help the team to carry these through to the successful delivery of the Project. They will remove barriers to success, provide guidance and direction and facilitate the alignment of our integrated team, providing a culture driven to meet the Project requirement and to celebrate team successes.

24.1.2.5 Owner's Team

The Organization chart of the Owner's team is shown in Figure 24-2.

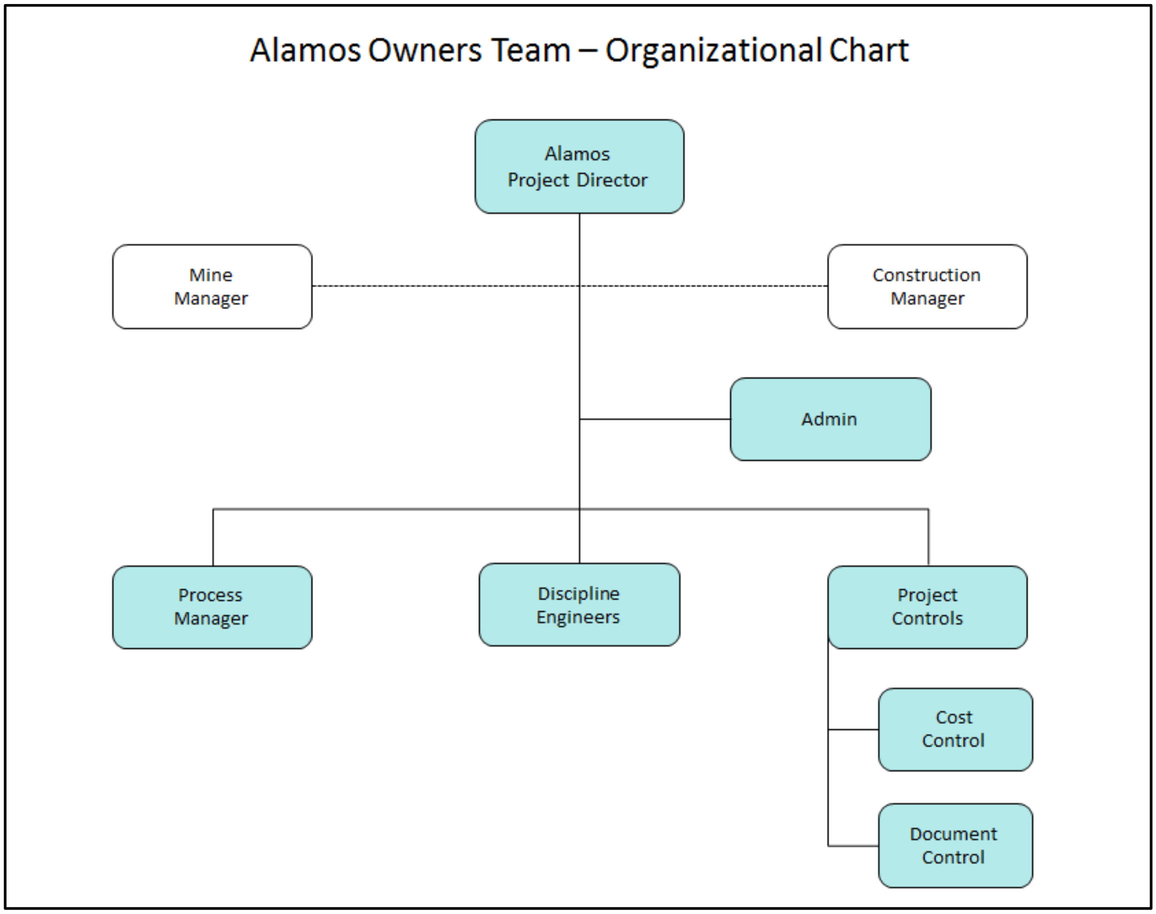


Figure 24-2: Alamos Owner's Team Organizational Chart

Source: Ausenco (2017)

The Owner's Project team is accountable for:

- Engaging Alamos and external stakeholders so that stakeholder requirements and statutory obligations are addressed;
- Managing the mine consultants and mine operations;
- Managing the accommodation strategies;
- Engaging with EPCM in Project risk management;
- Facilitating operations (mining, process, and maintenance) input into design and Project reviews and/or 'safety-in-design' processes;
- Approving the EPCM Project execution plans and sub plans;
- Providing timely response and turn-around to change management requests and Project approvals;
- Providing input into appropriate quality control and inspection processes;
- Managing the interfaces between EPCM and Owner's scope (e.g. mine development); and
- Ensuring smooth transition from the project phase into the operational phase.

During the Commissioning phase the Owner's Project team will:

- Develop operational and maintenance plans for all new plant and equipment;
- Recruit, mobilise and train personnel in the safe operation and maintenance of all plant and equipment and ensuring their availability for the commencement of the pre commissioning activities;
- Develop and implement all of the systems, processes and procedures to operate and support all new facilities during C3 commissioning under guidance from EPCM; and
- Accept and continue to operate the plant and equipment at the end of C3 commissioning.

24.1.2.6 EPCM Team

The EPCM organization will be developed to ensure that the strategies established in the early phases are retained through the full Project execution duration.

As the Owner's Representative, the engineering team will manage the engineering design throughout the implementation phase of the Project for detailed engineering while the procurement team will manage the procurement services.

The Engineering team will focus on delivering quality engineering on time and on budget to meet the needs of the Project.

Full service delivery of the Project will be enabled from the Project site office through mobilization of the construction and Project management team following completion of engineering and other front-end tasks. The construction management team is required to supervise the construction and overall management of the Project on site to ensure work is carried out by the selected contractors in accordance with the issued for construction deliverables prepared for the Project, specifications, standards and all site procedures.

The Engineering and Construction Management Project organization charts are shown in Figure 24-3 and Figure 24-4.

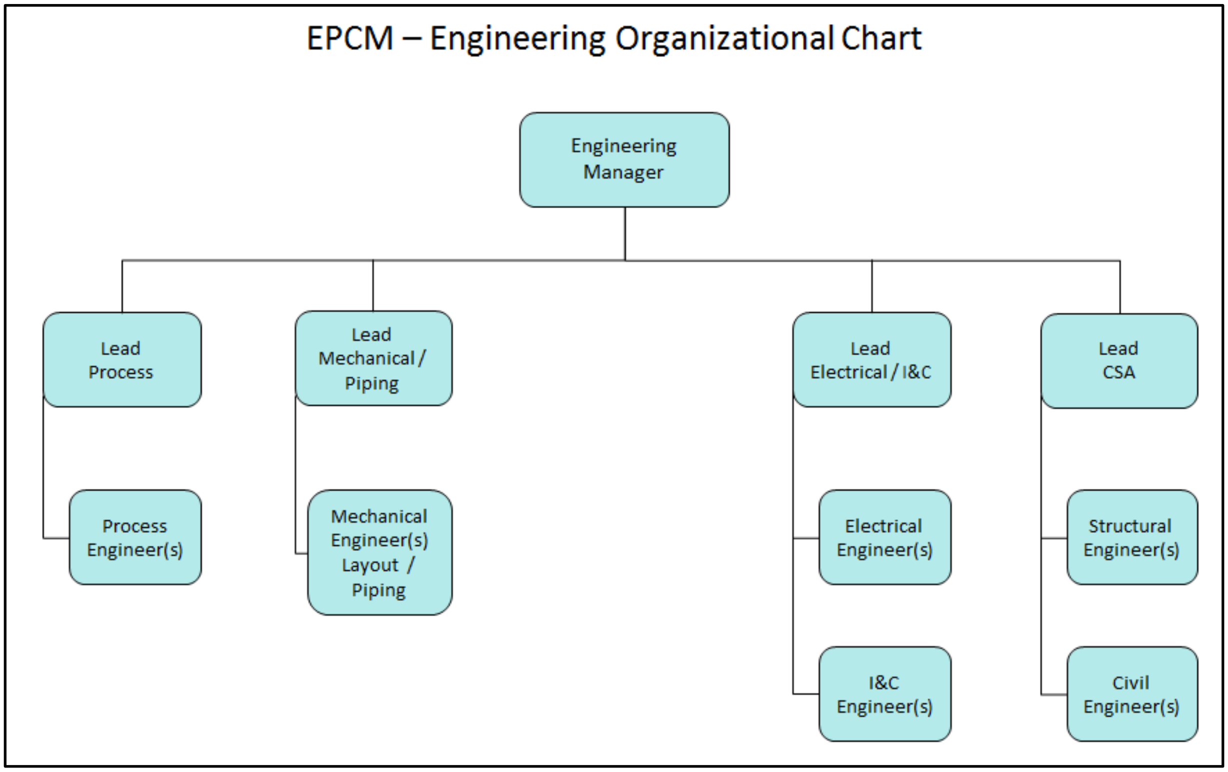


Figure 24-3: EPCM Engineering Organizational Chart

Source: Ausenco (2017)

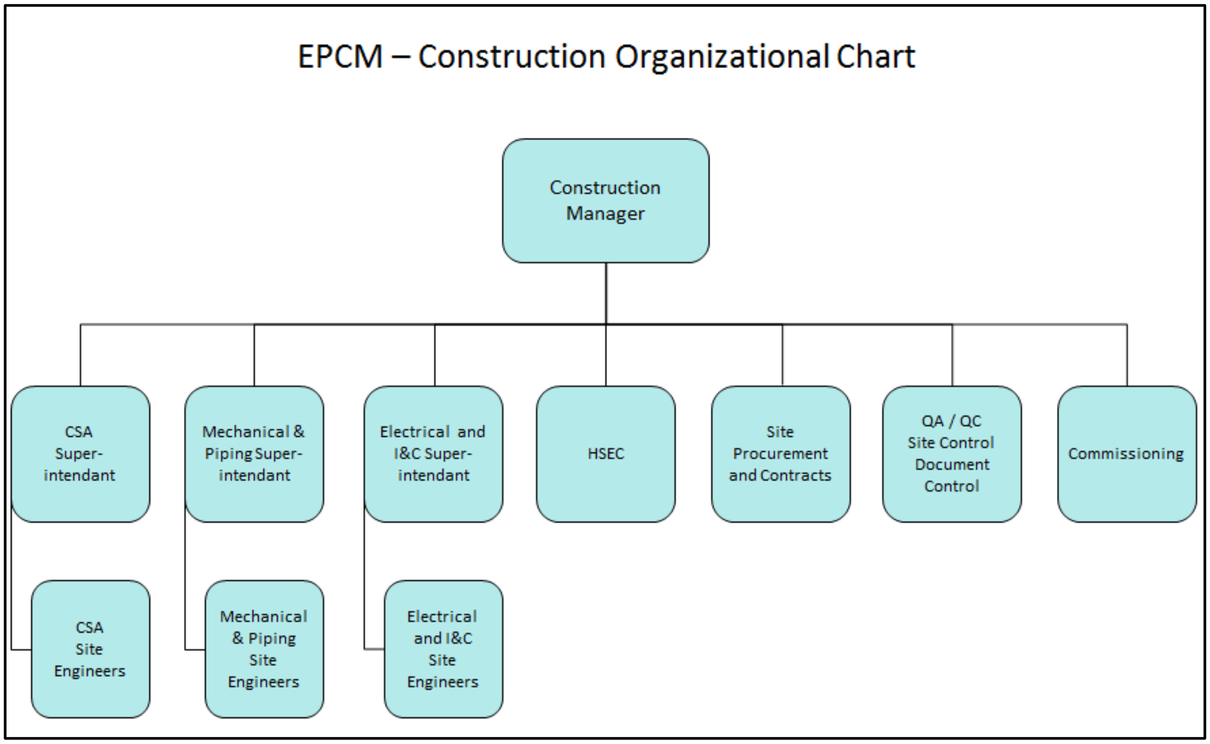


Figure 24-4: EPCM Construction Organizational Chart

Source: Ausenco (2017)

24.1.2.7 Project Management

The EPCM scope of services for the Project will be contracted by Alamos for implementation by an external provider.

The EPCM scope of services includes all engineering, procurement, construction management services, dry, wet and ore commissioning (pre-commissioning and commissioning) and ramp up to full production with associated Project management and Project controls. The EPCM contractor will do what is necessary to deliver the required facilities in a manner which meets budget and schedule, and any additional agreed Key Project Indicators (KPIs).

The strategy for delivery of this project will be based on the following:

- Ensure safety, health and environmental management is at the forefront of the Project execution;
- Hold a Project Kick-off Meeting to review the Project key drivers with the team;
- Meet the key schedule milestones;
- Develop the Project structure/organization detail and plan Project execution strategies sufficiently early to take advantage of optimized vendor and contractor offerings which could minimize risk and/or facilitate schedule;
- Continue to assess Project risks as the Project moves forward through engineering, procurement, and construction phases;
- Initially, the EPCM management team will be based in the EPCM contractor’s home office;
- Process engineering to be supplied by the EPCM contractor with input from Alamos;
- All remaining engineering detailed design to be supplied by the EPCM contractor;
- The use of project engineers to coordinate and take responsibility for the scheduled delivery of packages/systems through engineering, procurement, and interface with construction management and construction;
- Major equipment purchasing and construction contracts will be undertaken by the EPCM contractor. The EPCM contractor will prepare recommendations that will be issued to Alamos, so that the recommendations can be reviewed, and contracts awarded by the EPCM contractor if acceptable;
- Field expediting and coordination of inspection of vendor manufacturing is to be managed by the EPCM contractor;
- Site-based Construction Management team;
- Allowance for site-based QA/QC management included in the construction management site team;
- Allowance for site-based HSEC management included in the construction management site team;
- Allowance for site-based site based Commissioning Management;
- Centralized Project reporting initially from the EPCM contractor’s home office and then transitioning to site; and
- Procurement completed from the EPCM contractor’s home office.

The Project will have a set of plans and procedures that contain strategies and techniques for the proper implementation of the Project. These plans and procedures include the following:

- Project Execution Plan;
- HSEC Plan (developed in conjunction with Alamos);
- Emergency Response Plan (developed in conjunction with Alamos);
- Security Plan (developed in conjunction with Alamos);
- Quality Assurance/Quality Control Plan;
- Project Controls Plan;
- Procurement and Contracting Plan;
- IT and Communications Plan;
- Engineering and Drafting Plan;
- Document Management Plan;
- Construction Management Plan; and
- Commissioning Plan.

All Project plans, with the exception of the Commissioning Plan, will be issued within three months from the Project kick-off meeting. The Commissioning Plan will be prepared and issued at least three months prior to the commencement of commissioning.

Each plan will be supported by procedures. Where possible, the EPCM contractor standard procedures will be used; however, Project-specific procedures and systems may need to be implemented. All procedures will be supported by forms, guidelines and reference documents.

All plans and procedures will undergo regular reviews to ensure their accuracy, relevance and currency by the person responsible for the individual plan or procedure document.

24.1.3 Project Health, Safety and Security

The LLGP Project HSEC Management Plan will align the EPCM HSEC Management Systems with Alamos' specific HSEC Management Systems, Policies, Plans and Procedures for the Project. The HSEC Management Plan and associated contracts, legislation and codes of practice, identify and encompass the standards, working behaviours and safe work practices that will be expected of all LLGP employees including contractors.

The HSEC Management System is comprised of HSEC Management Plans, contractors' HSEC Management Plans, Individual Action Lists, HSEC Critical Safety Requirements and Risk Assessments and the HSEC Information Management System used to record all HSEC data and to track all HSEC actions relating to on and off-site activities for the LLGP.

24.1.3.1 Emergency Response

Emergency response plans will be prepared for the construction phase. These plans will allow for early notification of any incident. Care will be taken to ensure that these plans are integrated with the existing site procedures. The Emergency Response Plan will:

- Describe how the emergency response is initiated and how the emergency teams are activated;

- Specify command, control and communications arrangements;
- Identify the roles and responsibilities of all personnel likely to be at the site of the emergency or involved in the response;
- Include a responsible person nominated as the site Emergency Control Officer;
- Identify emergency resources and personnel; and
- Outline the Project Emergency Response flowchart.

Key health, safety and security issues that require management during the Project execution phase include:

- Personnel safety when in transit as well as on-site. Overall security to be managed by Alamos; and
- Environmental exposure: mitigated by adherence to site safety standards.

24.1.3.2 Alcohol and Drug Policies

No alcohol will be permitted on the LLGP properties. Each contractor will be responsible for implementing a program to make sure there is no illicit drug use. Random drug testing of personnel and critical craft such as cranes and heavy equipment operators will be in place for compliance.

24.1.4 Environment

To manage the environmental impacts during the execution phase an Environmental Aspects and Impacts Register for the Project will be developed. Environmental monitoring for the Project shall be carried out in accordance with the environmental monitoring schedule developed by Alamos. Environmental auditing for the Project shall be carried out in accordance with the auditing schedule developed for the execution phase. Project specific environmental auditing requirements shall be conducted in accordance with the requirements of Alamos and/or the Environmental Impact Assessment.

The Project General Site Induction will include Project-specific environmental requirements. All contractors shall ensure that their employees are trained on environmental aspects relevant to their contract. The aim of environmental training will be to:

- Provide information to enable the workforce to effectively perform their duties with full knowledge of their environmental responsibilities; and
- Introduce the workforce to the LLGP Site Environmental Management Plan.

Key topics to be covered in the environmental training will include:

- Work toward and promote ‘Zero Harm’ to people, the environment and the community;
- Compliance with the law and Alamos’ operating license conditions;
- The framework of the Environmental Management Plan;
- Non-negotiable environmental commitments;
- An outline of the environmental procedures in particular adherence to driving only on approved tracks or roads;

- Individual responsibilities for water management, chemical management, waste management, air quality management, cultural heritage, flora and fauna and noise management; and
- Incident reporting procedures.

Key environmental issues that require management during the Project execution phase and not addressed as a safety issue include:

- Contractors not adhering to site environmental standards: mitigated by pre-qualifying contractors with good environmental records, inclusion of site environmental standards in contracting process, environmental inductions, monitoring of environmental performance by EPCM team; and
- Changes to construction conditions as a result of the environmental approval process.

The Project will support the community relations and local business initiative goals.

24.1.5 Project Controls

The principal aim of Project controls is to assist the Project Manager and Project team in completing the Project within the approved budget and schedule.

The Project scope and schedule form the basis of labor and materials estimates, which in turn provide the capital cost estimate for the Project. The approved project control estimate and master Project schedule are the primary base documents for defining the Project and for measuring and reporting the Project status at any point during Project execution.

A Project Controls and Reporting Plan will be developed that details the systems and processes used to monitor, analyze, control, review and report on the progress of the Project. This plan will also outline the accounting and invoicing functions for the Project, including both the head contract and the works contracts.

The plan will include the following:

- Schedule and human resources management;
- Project reporting;
- Cost control and trend management;
- Procurement and contract management;
- Progress monitoring/earned value for engineering and construction;
- Estimating; and
- Document management.

The plan will include reporting structures to meet Alamos’s requirements, as well as for Project analysis, Project monthly status reports, and internal Project governance reporting.

The management approach is to highlight trends in cost and schedule, thus allowing the management team to be proactive when making decisions regarding changes affecting either the Project cost or schedule.

Throughout the various phases of the Project, key members of the Project controls team will assure continuity with the client, design team, procurement, construction, commissioning and management.

24.1.6 Engineering

24.1.6.1 Overview

The engineering team will deliver the basic and detailed design for the scope of facilities on the LLGP, and provide deliverables and technical support to ensure the effective procurement, construction, commissioning, and operation of the facilities. An Engineering Management Plan will be developed that will detail the strategy, processes and standards for the delivery of the engineering requirements of the Project.

The design for the Project will consider the following:

- Design for safety incorporating Safety in Design methodology and matching it to the Canadian standards, regulations and Alamos HSEC policies;
- Design for the environment with specific regard to the environmental approvals required by Alamos LLGP mines;
- Ongoing consideration of opportunities to improve Project value through ingenuity of design while ensuring that the ultimate operational objectives of the facility are firmly maintained;
- Constructability, operability and maintainability;
- Requirement for the design to be based on proven technology with minimum risk;
- Ease of operation, reliability and maintenance;
- Standardization of equipment with other Alamos operating entities;
- Maximum use of existing information provided by Alamos;
- Alignment of equipment lists with asset numbers to meet Owner requirements;
- Use of work by specialist consultants, where required or as designated by the Owner;
- Risk management, Hazard and Operability Study (HAZOP) and design reviews; and
- Requirements for site engineering, construction and commissioning activities.

The Engineering Management Plan will outline the overall design and revision processes for Project activities. It sets out specification and flow diagram standards, and references all engineering procedures, systems and forms to be used.

Engineering and procurement teams will be co-located in the same office for efficient management of the closely intertwined (engineering and procurement) functions.

Engineering will provide technical advice when required to procurement and quality departments throughout the Project.

24.1.7 Construction

The key objectives of the construction management plan are:

- Deliver the Project to the point of handover status in accordance with the commissioning schedule;

- Provide overall co-ordination of the Project site;
- Provide organisation, planning and management for field activities;
- Supervise and coordinate the work of construction contractors;
- Maintenance of the budget for construction of the plant in accordance with the established budget;
- Provide information for preparation of reports by the project engineers;
- Assist the project engineers and contract administration to minimise Contractor variation claims; and
- Provide office services and administration for field activities.

Particular challenges for construction on this project include:

- Location and conditions;
    - Constructing in a challenging environment;
    - Sedimentation controls given the amount of surface area to be opened up and the significant volumes of ground and subsurface water;
    - Maintaining road surface condition suitable for the hauling of materials; and
    - Availability of laydown areas for storage of construction materials on-site.

24.1.7.1 Site Security

Alamos will have overall responsibility for site security. This includes the establishment of security systems and processes as well as the provision of security personnel and equipment.

EPCM will liaise with the Alamos site security personnel and systems to leverage suitable support and ensure the security protocol and systems align with the Project objectives.

24.1.7.2 Construction Staffing

A preliminary examination of the construction staffing plan has been conducted, based on the Feasibility Study implementation schedule and estimated installation hours. The estimated construction manpower is expected to peak at approximately 250 direct construction workers on-site as shown in Figure 24-5. The construction labour, EPCM and Alamos staff required at site for the construction phase consists of:

- An average of 200 direct construction workers with a peak of 250;
- An average of 20 EPCM construction management staff with a peak of 25;
- An average of eight Alamos office and site team staff with a peak of 12; and
- Operational staff will be progressively joining Alamos ramping up in the final months of pre-production and before operations commence as shown in Figure 24-6.

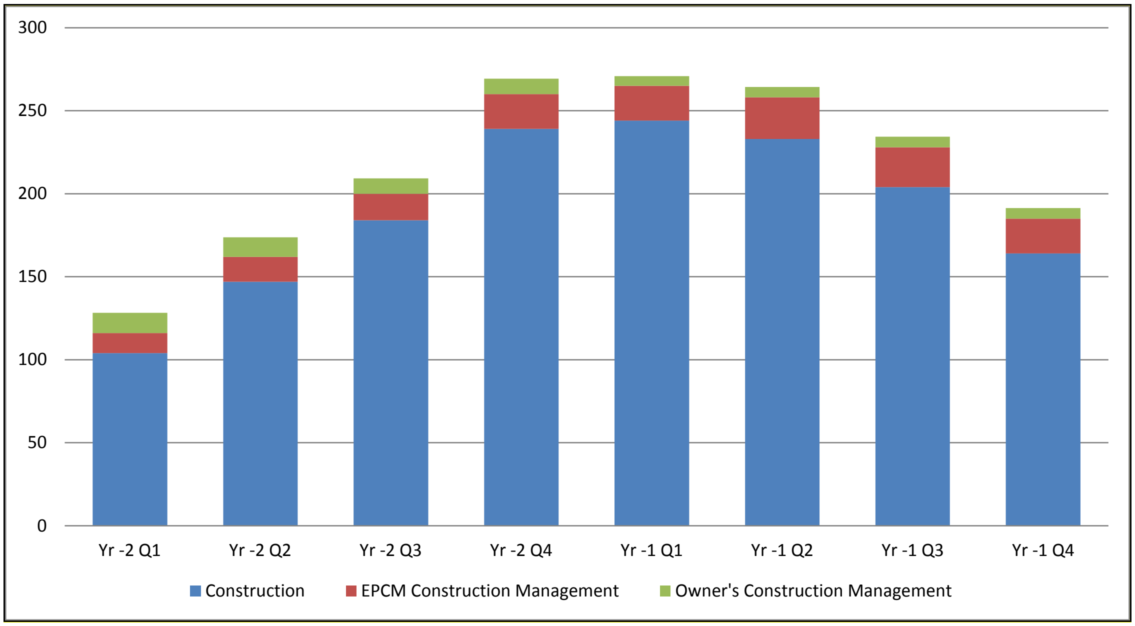


Figure 24-5 : Total Construction Labour

Source : Ausenco (2017)

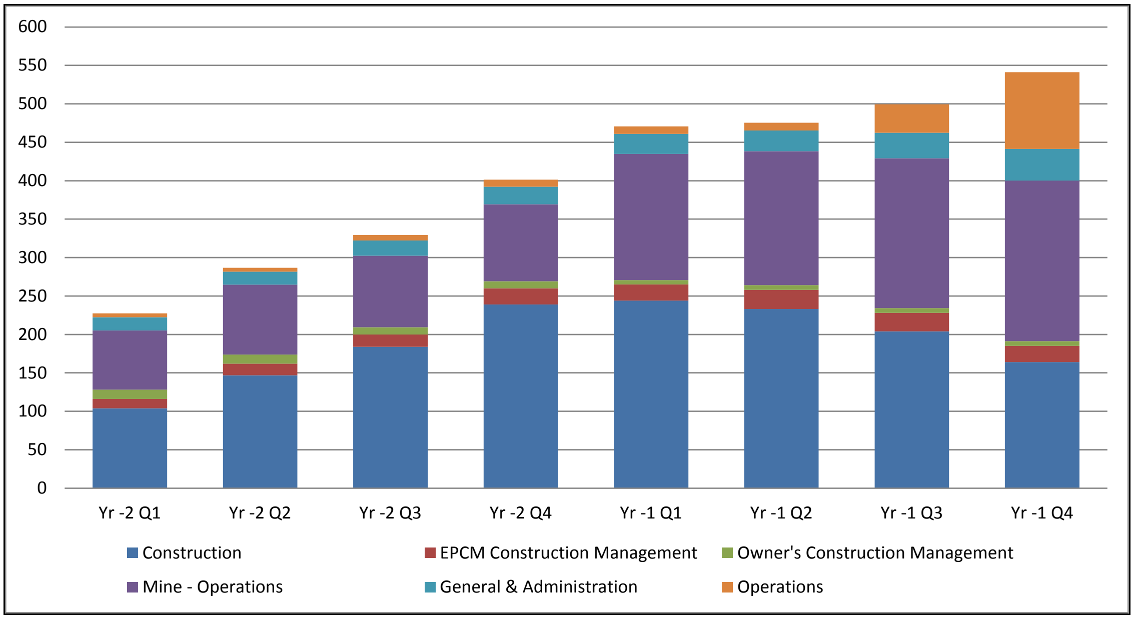


Figure 24-6: Total Labour On-site Including Operations

Source: (Ausenco 2017)

The level of staffing on-site will be subject to consideration of shift rosters, contractor selection and erection methodology, including the extent of remote/off-site pre-assembly and modularized delivery.

24.1.7.3 Contractor Temporary Facilities

The lay down yards will be identified and nominated ahead of contractor's mobilisation to site, as it will be illustrated on the Construction Layout drawing.

Laydown yards shall be sheeted in compacted mine waste and lightly crowned so that precipitation run off occurs to the perimeter.

Contractors will be required to establish their own temporary facilities. Contractors will bring and establish their own offices for the duration of their contract only, with an obligation that these facilities will be removed within one month of the final completion of works.

On completion of its construction activities, each contractor will be responsible for demobilisation of all temporary buildings and facilities and clean-up of the area allocated to it.

24.1.7.4 Mobilization

Immediately upon Phase 2 commencement the Construction Manager will be mobilized in the home office and will commence the planning process. A Project kick-off meeting will be held at the commencement of Phase 2 to define roles and responsibilities and provide the team with a clear understanding of the scope definition for the Project.

24.1.7.5 Construction Management Plan

As part of the early work for the Construction Management team, a Construction Management Plan will be developed. The plan provides the basis for mobilization, construction kick-off, and construction management through to completion of the Project. Key components of this plan include:

- HSEC Plan and site rules;
- Emergency Response Plan;
- Environmental Management Plan;
- Site Human Resources Plan; and
- Site-specific procedures (e.g., Travel Procedure).

Construction management will conform to the practices documented in the Project Quality Plan, Risk Management Plan and in strict accordance with the HSEC Management Plan.

24.1.7.6 Pre-Commissioning

The completion of construction is followed by pre-commissioning, which involves inspection and testing plant components and control systems. Pre-commissioning ensures that the equipment installation is complete, the operating conditions are met, and the equipment is ready to accept feed materials. In general, pre-commissioning testing does not include process material as part of the testing, but where possible may use inert material, such as water, to validate pumping systems, instrumentation and related controls.

The objectives of pre-commissioning are to ensure that:

- Each piece of equipment has been installed correctly and will operate safely and reliably to meet its' designed performance;
- All plant control system control sequences and Interlocks function correctly; and

- The plant as a whole will operate safely and reliably to meet its designed performance.

Pre-commissioning is incorporated within Steps C0 to C2 of the commissioning schedule.

24.1.8 Commissioning and Handover

24.1.8.1 Objectives

Commissioning covers the formal handover and acceptance of process equipment and commissioning modules between the various commissioning stages, from the completion of installation by contractors and suppliers through verification of plant and equipment dry or pre-commissioning by field engineers and design engineers, to final commissioning by the commissioning team.

The objectives of formal commissioning are:

- Ensure zero harm is maintained;
- Ensure commissioning is completed in an orderly manner;
- Ensure that all permits and isolation systems for transition from ‘Plant Dead’ to ‘Plant Live’ are defined and adhered to;
- Define handover, the different parties involved and their responsibilities at each stage; and
- Achieve plant performance acceptance criteria, demonstrated via performance and documents.

24.1.8.2 Philosophy

The commissioning process can be summarized by the time-based sequence of activities summarized below. This model provides the framework for EPCM’s documented commissioning procedures.

- Preparation of Commissioning Strategy;
- Development of Commissioning Plan;
- Preparation of Documentation and Mobilisation;
- C0 Plant and Equipment Verification;
- C1 Dry Commissioning;
- C2 Wet Commissioning; and
- C3 Ore Commissioning, Performance testing and Ramp up.

24.1.8.3 Commissioning Phase C0 – Construction Verification

Review of construction documentation and completion prior to acceptance by the EPCM Contractor’s commissioning team. Typically includes construction, commissioning, client inspection and formal punch listing, with an agreed-upon prioritization and schedule for some of the outstanding works to be completed, prior to proceeding to Phase C1.

This phase entails demobilization from site following the completion of performance trials, completion of punch-list items to Alamos’ satisfaction, and subsequent hand-over of documentation, including as-built drawings.

24.1.8.4 Commissioning Phase C1 – Dry Commissioning

The C1 Phase of commissioning includes bump/rotation tests, valve checks, and other commissioning activities able to be conducted once the equipment subsystems have been energized and prior to introduction of water or ore. Generally, completion of dry commissioning for all the subsystems and equipment in an agreed-upon area of the plant would trigger practical completion.

24.1.8.5 Commissioning Phase C2 – Wet Commissioning

The purpose of this phase is to identify leaks; to understand performance of pumps, agitators and other similar process equipment; and to determine if any manual, automated or sequenced opening and closing or isolation of systems is required. Upon completion of this phase, primary control of the plant typically passes from the EPCM Contractor to the Owner prior to the introduction of ore.

24.1.8.6 Commissioning Phase C3 – Ore Commissioning, Performance Testing and Ramp Up

This phase will include first-fill of reagents and a staged and gradual introduction of slurry into the plant under the EPCM Contractor’s supervision. The aim of this part of the commissioning process is to:

- Introduce operational personnel to the new plant and equipment in a supervised hand-over of operating control;
- Gradual introduction over a relatively short period of time from crushing through to tailings systems; and
- Establish control density, charge levels while slowly increasingly the plant throughput to the design capacity; involves tuning of control loops and other minor adjustments to satisfy the commissioning team that the design requirements and availability can be met.

Performance testing and ramp up will commence once the EPCM Contractor’s commissioning manager is satisfied that the plant has been satisfactorily commissioned. Maintaining the plant operation and feed materials within the throughput design criteria and conducted trials typically depends on the plants ability to maintain sustained levels of production over a fixed duration and recovery.

24.1.8.7 Plant Commercial Production

Once Phase 3 is completed, and plant commercial production is declared.

24.1.9 Risk Management

24.1.9.1 Approach

Risk management activities will be undertaken in line with accepted risk management standards (ISO 31000:2009), and have a focus on practicality and follow through of mitigation strategies.

EPCM Project Risk Management processes are focussed on the identification of risks and opportunities associated with realising maximum Project value. Technical risk management including HAZOPs and Design Review processes are managed by the Engineering discipline.

Independent Peer Reviews – Project Phase

Independent peer reviews will be scheduled to occur periodically during the engineering and construction phases to ensure the Project delivery strategy and actual progress satisfies Alamos' objectives.

24.1.9.2 Audits

Audits will be undertaken during the contractor prequalification process. These will include technical, capacity and financial and inspections of contractor fabrication facilities and where relevant and practicable, previous, similar projects.

The Project Quality Manager will develop and implement an audit program to ensure that the Project effectively implements the Project Quality Plan and applicable Procedural requirements. The plan will have the audit dates populated six months in advance, when Project Schedule data is available.

The audit program not only ensures that the Project Quality Plan is implemented on the Project but also includes audits on contractors and key vendors to enable determinations as to whether they are effectively implementing their respective Quality Plans and Procedures. The Project may conduct its audits independently, or in conjunction with vendor/contractor's scheduled audits.

EPCM will conduct the following audits:

- Project set-up;
- Engineering and design processes and quality of deliverables;
- Procurement and contracts process; and
- Construction and commissioning establishment.

The full audit schedule, including nomination of responsible parties, will be developed during the execution phase of the Project.

Implementation of the vendor surveillance programs will provide further assurance of the effectiveness of contractor's Project Quality Plan and the Individual Quality Plans of key vendors.

24.1.10 Project Schedule

A detailed Project execution schedule has been developed based on continuation of the integrated Alamos and EPCM team approach used during the Feasibility Study, with each consultant contributing in areas of their respective strengths. The preliminary detailed Project execution schedule is shown in Figure 24-7. The overall Project Schedule (Schedule) Alamos will provide the overall leadership to make key Project decisions and manage community, environmental, permitting, local authorities and security whilst EPCM will provide engineering, procurement, management and execution personnel that are experienced in cost effective Project delivery in accordance with both Canadian/Manitoba design standards.

Construction activities have been sequenced to consider the following:

- Northern weather conditions and outdoor construction windows; and
- Environmental constraints inclusive of non-disturbance during the bird nesting season.

The following assumptions have been considered to develop the schedule:

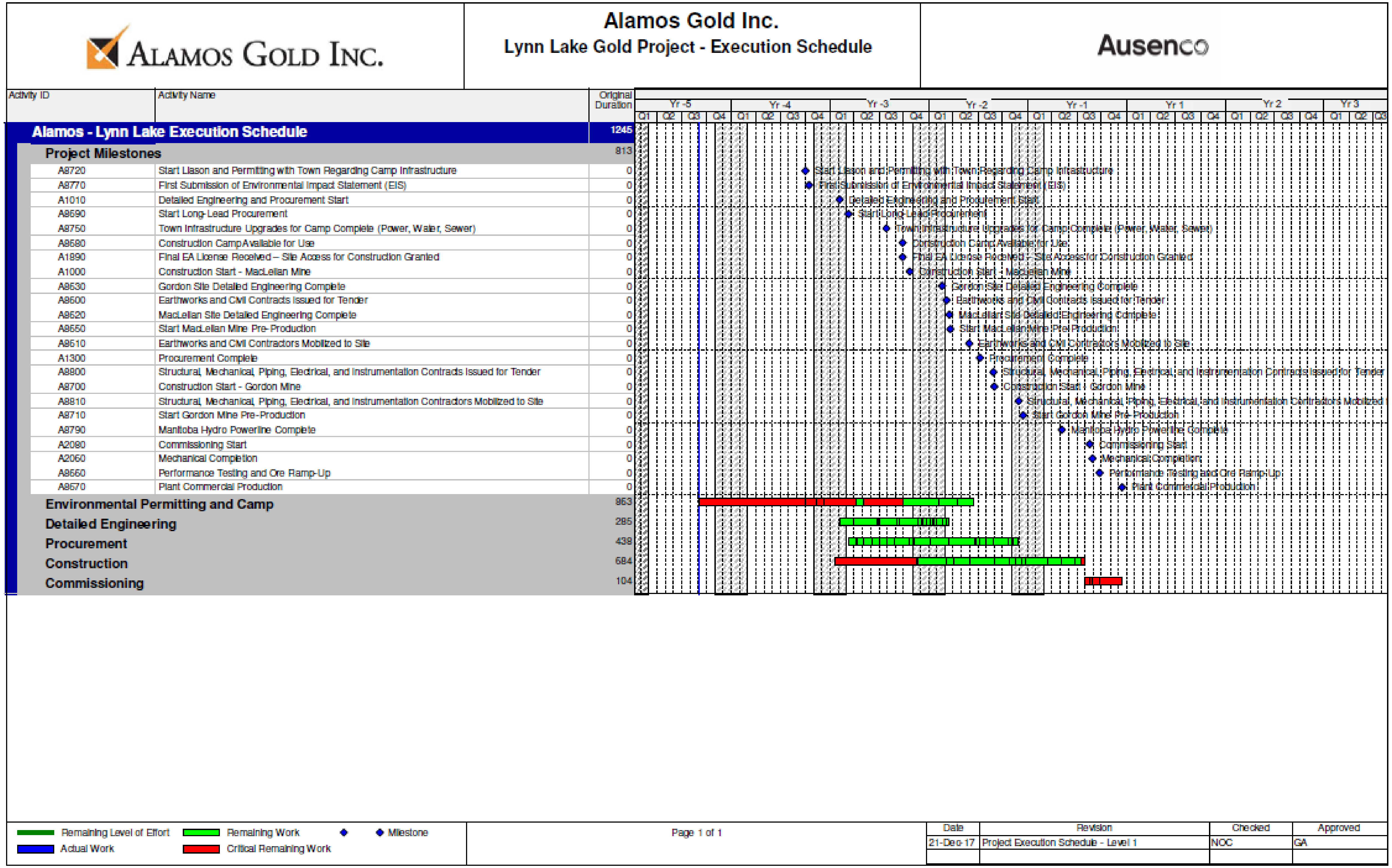
- The working calendars used in the Project schedule represent a five-day work week for engineering, a 21 days on / seven days off schedule with ten-hour work days for construction and a five–day work week for fabrication, delivery and start-up activities;
- Construction will start after receiving the Final Environmental Assessment License so site access for construction is granted;
- Activity durations are estimated based on similar historical projects and the judgment of the Project team;
- Construction work has been assumed to continue throughout the winter months but an attempt to limit concrete etc. to non-winter work will be undertaken;
- The bird nesting season is from late April until late August. No treed areas should be disturbed during this timeframe; any timber clearing should be completed prior to mid-April or start after August. All durations and activities were generated with Project management team experience and from past projects;
- The schedule is logically driven with as few constraints as possible; and
- The schedule follows all normal Critical Path Method best practices and works to limit lag whenever possible.

To meet the targeted hand-over in Q3 Year -1, permitting activities need to be underway in Q3 of Year -5 followed by early works engineering starting in Q1 Year -3 to secure procurement of long-lead equipment prior to full Project sanction. The critical path flows through the completion of the environmental permitting process, which leads to the ability to mobilize to site and begin the bulk earthworks. Once the Project receives full sanction, the longest path runs through the issuing of the Bulk Earthworks Contract then the construction activities of the process building and milling area. The critical path follows the earthworks, concrete installation, then structural steel and mechanical installation, finally followed by the electrical and instrumentation connections of the process areas. Following the construction activities the critical path continues through the commissioning activities into ramp up and full production.

To reach the schedule completion the discussions with the town government regarding the camp need to begin in Q4 Year -4. The design and procurement need to begin Q3 Year -4 to be ready for use by Q3 Year -3 for construction. Additionally, the permitting and design of the power line by Manitoba Hydro needs to begin in Q1 Year -3 to be complete by Q2 Year -1.

The current schedule is based on the receipt of vendor engineering of the long-lead items in the Q3 Year -3 and award of purchase orders for all vendor fabrication in Q3 Year -3 upon completion of permits. The fabrication and delivery times are based on information received from vendors during the Feasibility Study. Procurement is aligned with the required at site (RAS) dates and construction activities to meet the schedule requirements. To meet the planned Project completion target, the construction contracts will be tendered before the completion of detailed design.

The pre-production period which includes detailed engineering, procurement, construction and commissioning is planned for 30 months, with a 23 month construction period. The current schedule then has a three month ramp-up period from Mechanical Completion to Full Commercial Production.



Alamos Gold Inc.

Lynn Lake Gold Project - Execution Schedule

Activity ID	Activity Name	Original Duration
	Alamos - Lynn Lake Execution Schedule	1245
	Project Milestones	813
A8720	Start Liason and Permitting with Town Regarding Camp Infrastructure	0
A8770	First Submission of Environmental Impact Statement (EIS)	0
A1010	Detailed Engineering and Procurement Start	0
A8690	Start Long-Lead Procurement	0
A8750	Town Infrastructure Upgrades for Camp Complete (Power, Water, Sewer)	0
A8680	Construction Camp Available for Use	0
A1890	Final EA License Received – Site Access for Construction Granted	0
A1000	Construction Start - MacLellan Mine	0
A8630	Gordon Site Detailed Engineering Complete	0
A8600	Earthworks and Civil Contracts Issued for Tender	0
A8620	MacLellan Site Detailed Engineering Complete	0
A8650	Start MacLellan Mine Pre-Production	0
A8610	Earthworks and Civil Contractors Mobilized to Site	0
A1300	Procurement Complete	0
A8800	Structural, Mechanical, Piping, Electrical, and Instrumentation Contracts Issued for Tender	0
A8700	Construction Start - Gordon Mine	0
A8810	Structural, Mechanical, Piping, Electrical, and Instrumentation Contractors Mobilized to Site	0
A8710	Start Gordon Mine Pre-Production	0
A8790	Manitoba Hydro Powerline Complete	0
A2080	Commissioning Start	0
A2060	Mechanical Completion	0
A8660	Performance Testing and Ore Ramp-Up	0
A8670	Plant Commercial Production	0
	Environmental Permitting and Camp	863
	Detailed Engineering	285
	Procurement	438
	Construction	684
	Commissioning	104

🟩 Remaining Level of Effort 🟩 Remaining Work ◆ ◆ Milestone
🟦 Actual Work 🟥 Critical Remaining Work

Page 1 of 1

Date	Revision	Checked	Approved
21-Dec-17	Project Execution Schedule - Level 1	NOC	GA

Figure 24-7: Lynn Lake Gold Project Execution Schedule

Source: Ausenco (2017)

25 INTERPRETATION AND CONCLUSIONS

25.1 Summary

The completion of this Report confirms the technical feasibility and economic viability of the Project. The LLGP is composed of two properties MacLellan and Gordon, based on a conventional open pit mining with a centralized site processing plant facilities and tailings management facility. The processing plant, located at MacLellan, has an expected nominal processing throughput of 7,000 t/d with an estimated 10.4-year production life.

The main conclusions by area are detailed below.

25.2 Geology and Mineral Resource Estimate

Alamos personnel reviewed and audited the historical exploration data available for the LLGP as well as the exploration methodologies adopted to generate this data. Exploration work is professionally managed and procedures are adopted that meet accepted industry best practices. The author is of the opinion that the exploration data are sufficiently reliable to interpret with confidence the boundaries of the gold mineralization and support evaluation and classification of mineral resources in accordance with generally accepted CIM Estimation of Mineral Resource and Mineral Reserve Best Practices Guidelines and CIM Definition Standards for Mineral Resources and Mineral Reserves.

The drilling database includes information from 1,975 drill holes (1,490 at MacLellan and 485 at Gordon) comprising 299,030m of drilling, and includes 173 new drill holes (32,131m) drilled by Alamos which have been added since the previous mineral resource estimate was conducted in 2014. The mineral resource statement effective December 01, 2017 is provided in Table 25-1.

Table 25-1 – Consolidated LLGP Mineral Resource Statement, December 01, 2017

MacLellan

Category	kTonnes	Au Grade (g/t)	Ag Grade (g/t)	Au koz	Ag koz
Measured	2,110	1.86	5.34	126	362
Indicated	2,236	1.24	4.24	89	305
Total Measured and Indicated	***4,347***	***1.54***	***4.77***	***215***	***667***
Inferred	750	1.62	2.80	39	67

Gordon

Category	kTonnes	Au Grade (g/t)	Ag Grade (g/t)	Au koz	Ag koz
Measured	9	1.72	n/a	0.47	n/a
Indicated	451	1.96	n/a	28	n/a
Total Measured and Indicated	***460***	***1.95***	***n/a***	***29***	***n/a***
Inferred	615	1.30	n/a	26	n/a

MacLellan Underground

Category	kTonnes	Au Grade (g/t)	Ag Grade (g/t)	Au koz	Ag koz
Measured	0	0.00	0.00	0	0
Indicated	1,161	4.37	6.23	163	233
Total Measured and Indicated	***1,161***	***4.37***	***6.23***	***163***	***233***
Inferred	116	3.82	3.43	14	13

Total

Category	kTonnes	Au Grade (g/t)	Ag Grade (g/t)	Au koz	Ag koz
Measured	2,119	1.86	5.31	127	362
Indicated	3,848	2.27	4.34	281	537
Total Measured and Indicated	***5,967***	***2.12***	***4.69***	***407***	***900***
Inferred	1,481	1.66	1.69	79	80

Notes:

- *The Mineral Resources are reported at an assumed gold price of US $1,400/ounce, and an assumed silver price of US $22.00/ounce;*
- *The Mineral Resource estimate was completed by Mr. Jeffrey Volk, CPG, FAusIMM, Director of Reserves and Resources for Alamos Gold Inc.;*
- *Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resources estimated will be converted into Mineral Reserves;*
- *Open pit Mineral Resources are stated as contained within potentially economically open pit above a 0.42 g/t AuEq cut-off for MacLellan and 0.62 g/t Au for Gordon, and includes external dilution at zero grade outside the 0.50 g/t Au solid;*
- *Mineral Resources for the MacLellan Underground are stated above a 2.0 g/t Au cut-off. MacLellan Underground block grades are undiluted;*
- *Totals may not add due to rounding;*
- *Contained Au and Ag ounces are in-situ and do not include metallurgical recovery losses; and*
- *Mineral Resources are exclusive of Mineral Reserves.*

The QP considers the mineral resource model documented herein to be sufficiently reliable to support engineering and design studies to evaluate the viability of the LLGP at a feasibility level.

25.3 Mining Methods and Reserves

The development of the MacLellan and Gordon deposits is planned with the use of open pit mining methods. The summary of Mineral Reserves is contained in Table 25-2. The Mineral Reserve estimate is based on cut-off grades of 0.69 Au g/t for Gordon and 0.47 Equivalent Au g/t for MacLellan, calculated on the basis of the design parameters of the Project, which include an Au price of US $1,250/Au oz and an exchange rate of USD/CAD of 0.75. The current geological model estimation methodology inherently introduces dilution of 15% and 13% for MacLellan and Gordon respectively in the estimate of the block, no additional dilution or mining recovery factors were applied.

Table 25-2 – LLLGP Mineral Reserve Statement, December 01, 2017

	Class	Tonnage (Mt)	Au Grade (g/t)	Ag Grade (g/t)	Au Oz Contained (x1000)	Ag Oz Contained (x1000)
Gordon	Proven	2.31	2.82	n/a	210	n/a
	Probable	6.41	2.27	n/a	468	n/a
	Total Proven and Probable	8.72	2.42	n/a	678	n/a
MacLellan	Proven	9.55	1.91	5.01	586	1,539
	Probable	8.53	1.32	3.79	361	1,039
	Total Proven and Probable	18.08	1.63	4.43	947	2,578
Lynn Lake Gold Project	Proven	11.86	2.09	4.03	796	1,539
	Probable	14.94	1.73	2.16	829	1,039
Total Proven and Probable		26.80	1.89	2.99	1,625	2,578

Notes:

- *Mineral Reserves reported are in agreement with the CIM Definition Standards for Mineral Resources and Mineral Reserves*
- *The Mineral Reserve is estimated using metal prices of US $1,250/Au oz and US $15.00/Ag oz.*
- *Totals may not add up due to rounding.*
- *The estimates were carried out using cut-off grades of 0.69 Au g/t for Gordon and 0.47 Equivalent Au g/t for MacLellan and a metallurgical Au recovery of 89-94% for Gordon and 91-92% for MacLellan.*
- *The design parameters applicable are detailed in Section 15 of this report.*
- *The estimate of the Mineral Reserves was carried out under the supervision of Efthymios Koniaris, PhD., P.Eng., of Q'Pit Inc.*

The concurrent operation of Gordon and MacLellan is planned, which permits the mill feed build up to the plant nominal capacity of 2.56 Mt/y from the start-up of operations.

The open pit mine development is based on conventional proven operating methodologies and equipment. It will benefit from the availability and will contribute in the further development of experienced and knowledgeable local and regional workforce.

Both sites will benefit from the compact design and in particular from the short haulage times due to the location of the waste dumps.

The Project benefits from the early development of Gordon due to its higher grades.

The operation in two mine sites concurrently, and highway haulage is one of the unique characteristics of the Project. Highway haulage requires intensive use of equipment and personnel. This intensity may be reduced if larger capacity highway haulage can be utilized.

The operation in two sites also provides with the opportunity to realize synergies in the use of personnel and capital resources.

25.4 Processing

Test work carried out on representative samples of the deposits indicates that the MacLellan and Gordon mineralization can be processed with cyanide leaching and carbon adsorption. Leached slurry is amenable to cyanide detoxification using SO_2/air technology.

The proposed Lynn Lake process plant design will be based on leach/CIP technology, which will consist of crushing, grinding, thickening, pre-aeration and leaching, CIP, cyanide detoxification, carbon elution and regeneration, and gold smelting.

The LLGP is also a past producer with established production using the same process flow sheet as proposed.

25.5 Infrastructure

Both sites have existing access roads connecting to PR 391. A new 2.5 km long road will be built to access the selected MacLellan process plant site.

The overall water management concept is to divert non-contact water to reduce the amount of contact water to be managed. Contact water will be stored in the TMF or collected in settlement ponds, and will not require treatment beyond solids settling before being discharged to the environment. Gordon pit lakes will be dewatered during construction, there is no requirement for treatment prior to discharge to the environment. During operation, MacLellan pit dewatering will be by convention diesel driven sump pumps. Gordon will utilize a combination of pumping wells around the pit perimeter to intercept groundwater and conventional in pit diesel driven sump pumps. MacLellan makeup water will be via a fresh water pumping station located on the Keewatin River. Gordon fresh water will be supplied via submersible pumps from Gordon Lake.

The TMF will receive water from the mine, mill tailings, and precipitation runoff from the mill site, transportation corridors, and waste stockpiles. There is no discharge of water from the TMF to the environment. A decant tower will be located on the crest of the internal dyke to collect water for reclaim. Tails pumping to, and reclaim water from the TMF occurs year-round. The initial construction sequence for the TMF starter dams is anticipated to last about 18 months. Future dam raises projected in Years 2, 6, and 9 will required approximately 8-month construction windows each.

Average power demand at MacLellan is 13.8 MW, and will be supplied by Manitoba Hydro Line 6, which requires upgrades to operate at 138 kV. Incoming 138 kV will be stepped down at the MacLellan main power station by two 15/20 MVA transformers and distributed at 6.9 kV. A single VFD will be used to start SAG and ball mill motors to minimize voltage drop. One MW of emergency power will be provided by a diesel generator. Average power demand at Gordon is 200 kW, and will be supplied by two 300 kW diesel generators in duty/standby configuration. Power distribution will be at 4.16 kV.

A dedicated commercial satellite network will be employed to handle the data and Internet needs. The satellite link at MacLellan will communicate via microwave links to Gordon site and to the Alamos office in the Town of Lynn Lake. Operations will use radio communication.

MacLellan will have two fuel distribution stations. One station will service the mining fleet with three high flow pumps supplying dyed diesel. The other will have one gasoline pump for the light vehicle fleet and two high flow diesel pumps for the road haul trucks supplying un-dyed diesel. Gordon will have one fuel distribution station which will service the mining fleet with two high flow pumps supplying dyed diesel. Propane will be used for heating purpose and maintenance shop requirements at MacLellan and Gordon.

The MacLellan site will have several process and infrastructure buildings, including primary crusher building, process plant building, sulphur burning plant, oxygen plant, truck wash building, administration office, mine truck shop/warehouse, assay lab and powder magazine. The Gordon site will have few structures, limited to truck shop, truck wash building, and administration office/gate house.

The construction and operations camp will be located within the Town of Lynn Lake limits and will be used continuously throughout the construction, pre-production and operations phases. Camp infrastructure will be tied into existing Town facilities. The camp facility will be comprised of a 300-bed purchased camp, plus 100-bed leased camp for the two-year pre-production period.

25.6 Environmental Considerations

The process of environmental licensing and permitting of the LLGP is well understood and a preliminary schedule outlining the critical steps has been developed and integrated into the Project execution schedule. The obtaining of environmental approvals is a critical path for the Project and no construction activities can commence until the required licenses, permits and other authorizations are obtained.

Considerable environmental baseline information has been obtained and compiled for the sites and surrounding areas covering aspects of the atmospheric, water resources and geochemistry, terrestrial, aquatic, and human and socio-economic environment. Based on this information, there are no environmental aspects that appear to be limiting to the Project’s development, as the Project design has considered appropriate environmental mitigation measures, including avoidance of sensitive environmental areas.

It is anticipated that the results of the EIS, including implementing identified mitigation measures, will prevent the Project causing significant adverse environmental effects, including effects from accidents and malfunctions, effects of the environment on the Project, and cumulative effects. Regulatory comments on the EIS will define future conditions of approval which could require refinements to Project components or additional mitigation measures to be implemented.

Alamos continues to work with Indigenous communities and other stakeholders to understand potential effects of the Project on traditional land uses and activities.

25.7 Capital and Operating Cost

The CAPEX was developed according to AACE Class 3 guidelines for a Feasibility Study Estimate with a -10% to +15% accuracy. The initial capital cost of the Project, including processing, initial mine equipment lease payments and pre-production activities, infrastructure, spares and other direct and indirect costs is $450.6M (US $338.0M). Sustaining capital cost is $196.9M (US $147.6M), and includes MacLellan and Gordon primary and support equipment lease payments, spare parts, additional TMF lifts, water management and closure costs. Total Project capital cost including initial and sustaining capital is $647.5M (US $485.6M).

Overall operating cost for mining ($26.94/t milled, $713.3M LOM), processing ($14.46/t milled, $382.8M LOM), G&A/Accommodation ($7.10/t milled, $188.1M LOM) and external refining ($0.18/t milled, $4.8M LOM) is $48.68/t milled or $1,289.0M LOM. After adjusting for royalties and silver credits, the expected operating cost is $48.08/t milled or $1,273.2M LOM (US $954.9).

25.8 Risks

Risk identification and mitigation was ongoing throughout the FS, and will continue through value/detailed engineering, construction, operations and closure. Risks were identified and qualitatively ranked in the LLGP Risk Register. As the Project moves from feasibility into the execution phase, it will be necessary to update the Project risk register.

25.8.1 Mining

The estimate of the Mineral Reserves is subject to the risks related to the geological model, the mine plan and dilution and mining recovery during operations. The risk due to dilution and mining recovery in particular is addressed by the provision of both RC and blasthole assaying as well as the selection of the type and size of the primary loading units.

The startup schedule and startup equipment and personnel deployment is one of the main areas of risk and opportunity which will benefit by further studies and schedule definition and detailing.

Additional studies and information collected during the early stages of the operation on rock properties affecting the estimates of the drill and blast requirements as well as the wear steel and ground engagement tools consumption can be used to mitigate the related technical risks. The rate of the Project production buildup also permits adjustments with the realized experience with respect to the number of deployed production drills, drill hole diameter as well as the drill patterns and powder factors.

It is considered that the risk related to the management of the operations around the underground stopes and voids has been appropriately addressed at this stage of the Project by including in the feasibility study cost estimates additional equipment and personnel as well as the costing of a modified operating environment in the vicinity of these openings. Two additional owner personnel are included for the drilling and blasting operations in the vicinity of voids. A ground penetrating radar as well as a hydraulic hammer attachment for the control of the oversize boulders expected while operating in the vicinity of the underground voids. The estimates for the drilling and blasting requirements considered the delineation drilling as well as adjusted drill diameter and patterns for blasting in the immediate vicinity of openings.

The underground openings in MacLellan will be encountered three years after startup of mining operations and one year after startup of processing operations. It is expected that a site standard operating procedure will be established for MacLellan during its first year of operation, which may use additional information as required.

25.8.2 Environmental Assessment

There are several external factors that could contribute to the risk of a delayed environmental approval process. Once Alamos has submitted the EIS, the government may, ‘stop the clock’ with one or more rounds of information requests (IRs). Regulators will determine if consultation/engagement efforts have been sufficient. Capacity constraints within the regulatory agencies and/or any changes made to government policies and expectations around environment act legislation generally and/or Indigenous engagement specifically could also affect environmental approval timelines. In an attempt to minimize or avoid any approval delays, Alamos, in conjunction with Stantec, will regularly liaise with regulators through the environmental assessment preparation and EIS review. Engagement and consultation is ongoing with Indigenous communities. The extent of required Traditional Knowledge and Traditional Land and Resource Use studies has not been fully defined at this time.

25.8.3 Tailings Management Facility

The following list provides the risks associated with the TMF during construction and operations as has been carried forward in the current LLGP Risk Register.

- Inadequate characterization of the TMF foundation conditions could lead to increased construction material requirements and costs;
- Water management issues, associated with both the quantity and quality of the inflows to the TMF, could result in excess water stored in the TMF that would require treatment and discharge to the environment to maintain dam containment. This would require the implementation of a water treatment during operations to reduce stored water volume in the TMF;
- Damage to the dam liner due to improper construction or installation could result in excess seepage, overwhelming the downstream sumps and causing uncontrolled discharge to the environment and incurring additional costs for environmental rehabilitation and the implementation of additional controls; and
- A failure of the tailings dam would result in the uncontrolled release of water and/or tailings into the environment, resulting in a mill shutdown and significant costs for reconstruction and environmental rehabilitation.

The above risks have been currently classified as low, as it is recognized that contingency planning and the implementation of engineering and quality controls during the design, construction, and operational phases will be implemented to mitigate these risks.

25.8.4 Water Management

High groundwater seepage into the Gordon pit is expected. The planned dewatering wells will reduce this risk; however, consideration for installation of dewatering wells must be included in the construction schedule.

25.8.5 Process

Test work showed that extraction of gold from high-arsenic grade MacLellan ores (nominally > 0.3% As compared to the test work average of 0.1% As) was highly variable (55%-90%). Lower gold extraction occurred in two of the 34 ore-grade and low-grade tests. It is assumed that the occurrence of these lower-recovery ores is not significant in terms of the overall reserve tonnage at MacLellan and that if these occur, a suitable blending or stockpiling strategy will be developed to manage these.

25.8.6 Infrastructure

Cost and schedule impact of MH upgrades required to operate Line 6 at 138kV. Preliminary discussions with MH have indicated it will take up to 36 months to design, purchase and install the equipment necessary for the Project once an Interconnection Facilities Study is completed and signed agreements are in place. The timeline includes an Environmental Assessment and submission of an Environmental Impact Statement required to obtain a license to construct the 5km 138kV transmission line from the 138kV tap at Lynn Lake to MacLellan. Substantial power will be required for the process plant commissioning and start-up.

Increased heavy traffic during construction and operations will accelerate degradation of existing bridges and road infrastructure on PR 391 between the Town of Lynn Lake, MacLellan and Gordon sites.

Cost and schedule impact of infrastructure upgrades required in the Town of Lynn Lake to facilitate the construction of the accommodation camp and services.

25.8.7 Staffing and Transport

Work force availability in the Town of Lynn Lake will be marginal. The majority of skilled management and labour will be FIFO from larger communities in Manitoba. Development of the local employee base will be an important mitigation measure for staff attraction and retention during operations.

25.9 Opportunities

25.9.1 Mining

- Examine the option of a marginally deeper, and larger Mineral Reserve excavation at MacLellan;
- Examine options for alternate process rates;
- Explore option for subcontracting preproduction, mining, and highway haulage from MacLellan to Gordon; and
- Investigate the use of larger capacity trucks for the highway haulage between Gordon and MacLellan to reduce fleet size, trip count and personnel.

25.9.2 Environmental Assessment

- Optimize the Gordon diversion ditch to take advantage of the natural topography, reducing the length and associated earth work quantities.

25.9.3 Tailings Management Facility

- Investigate relocating the TMF to the west to take advantage of natural topography and shortened tails and reclaim pipelines;
- Investigate utilizing borrow source material within the TMF limits to increase capacity and reduce the environmental foot print;
- Optimize the internal rock dyke and reclaim pumping system; and

- Consider the feasibility / stability of centre-line raising of the TMF dams to save on dam material quantities.

25.9.4 Process

- Consider moving the process plant site to the overburden stockpile planned location to eliminate the new access road and minimize the amount of earthworks, clearing and grubbing. This could allow the commencement of plant site construction earlier in the schedule as the area is easily accessible. The relocation of the plant site would eliminate the need to place material in the East Pond; and
- Consider a CIL circuit instead of the leach/CIP circuit which will reduce the number of tanks. Locating the CIL tanks outside would lead to a decrease in the process building size.

25.9.5 Infrastructure

- Review the pit dewatering strategy at Gordon with potential to reduce pumped water volume. Consider the removal of the dewatering wells outside of the pit, replacing these with additional in-pit sump capacity and an optimized settling pond;
- Review the option to utilize the existing 69kV MH Line 6 without the upgrade and supplement with on-site generation;
- Review building construction type and size for truck shops, truck washes, warehouse, ore stockpile cover and administration buildings to refine the CAPEX according to the planned mine life; and
- Update the foundation design to incorporate new geotechnical findings, using rock anchors for anchoring foundations to minimize the estimated concrete volume.

25.9.6 Capital Cost

- Consider the possibility of bundling the purchase of major equipment with a single vendor where applicable to obtain favorable pricing to improve Project economics; and
- Investigate build and own contracting strategy (over-the-fence) for oxygen and sulphur burning plants to reduce initial CAPEX.

25.9.7 Operating Cost

- Investigate bulk contracts for supply of reagents from a single source; and
- Investigate combined purchasing with Alamos' other Canadian operations.

26 RECOMMENDATIONS

26.1 LLGP Project Recommendations

The following activities should be in place in the next phase of the Project development:

- Submission and approval of the EIA/ EA;
- Liaise with Lynn Lake officials with regards to the camp and associated infrastructure;
- Prepare a camp package to go for tender;
- Secure all required environmental and construction permits; and
- Manage and mitigate key risks and pursuing the opportunities to improve Project economics including Value Engineering.

A list of specific recommendations has been developed per area as shown in the sub-sections below.

26.1.1 Geology and Mineral Resource Estimate

Previous work by Carlisle has identified a sizable underground resource which has not been included in this study. Additional drilling is required to infill some gaps in the drilling data with the potential to evaluate the underground resources, and test the depth extensions of the gold mineralization.

It is also recommended to conduct additional geological and mineralogical studies aimed at improving the understanding of the geological and structural setting of the deposit and revised 3D modeling to assist in future mine planning and refinement of the ARD model.

26.1.2 Open Pit Mining

The work carried out to date on the LLGP has been done in accordance to industry standards, and with applicable risks related to the open pit mining and the estimate of the Mineral Reserves identified. This includes the pit limit optimizations, mine design and mine equipment selection.

The in-situ material value and cut-off grade models are expected to be updated and refined with each iteration of the mine plan. The Mineral Reserves were estimated using cut-off grades of 0.69 g/t Au for Gordon and 0.47 g/t Equivalent Au for MacLellan. The cut-off grades were calculated based on the design parameters active at the time of the work. It is recommended that a refinement of the cut-off grade model be undertaken, based on the results of the FS.

Similarly, it is recommended that the MacLellan pit limit be examined with taking into account the FS estimates of the mining costs, as well as the constraints, if any, of the tailings dam and waste storage capacities. The pit limit design of MacLellan was carried out based on the design parameters at the outset of the FS, which proved conservative. The FS design parameters and estimated mining costs may support a marginally larger pit limit and Mineral Reserve for MacLellan.

26.1.3 Metallurgy and Process Plant

The following metallurgical work is recommended for the next phase of engineering:

- Ongoing program to confirm competency and hardness characteristics in which suitable samples are selected based on the most recent mine plan and which represent the initial two years of operation and up to about 50 m depth; and
- Conduct additional grinding simulations if ore comminution characteristics from the ongoing program are significantly different to those used in the basis for design.

The following process plant options should be considered for the next phase of engineering:

- Review capital and operating costs for risk mitigation of throughput compared to the base case 2C-SAB circuit for the following options:
    - Reduction in capital cost if provision in layout for pebble crusher is removed;
    - Additional capital cost incurred with installation of a pebble crusher circuit;
    - Reduction in capital cost with alternative 3CB circuit; and
    - Operability of each circuit option with respect to dust, materials handling, cold climate.

26.1.4 Infrastructure

A power study to confirm capacity of Manitoba Hydro Line 6 and evaluate power options must be a high priority as this will be a key item that could drive the Project schedule.

26.1.5 Tailings Management

The current level of study for the LLGP is feasibility. This will be followed by detailed design and then the preparation of construction documents and specifications. This is an evolutionary process as the level of study and design progresses.

For the TMF, the following tasks should be considered for the next stage of engineering:

- Further development of the waste rock management plan with consideration of materials to be used for construction of the start-up dams that are non-acid generating and non-metal leaching;
- Further evaluate the potential borrow sources for dam construction in terms of quantity available and suitability;
- Finalize the tailings dam design;
- Complete a comprehensive closure plan; and
- Appropriate regulatory agencies must be consulted and relevant permits and approvals shall be acquired.

26.1.6 Environment

The environmental recommendations are as follows:

- Environmental assessment, other environmental permitting and planning, mitigation, management and follow-up monitoring plans, and associated consultation/ engagement activities should proceed with the objective of gaining environmental approval for the Project in line with the overall Project schedule;
- Geochemical characterization should continue as the processing and mining plans are detailed, with modification to the mineral waste management plan as appropriate; and
- Estimated closure and reclamation costs should be reassessed in the next phase of development as more detailed engineering designs become available.

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28 QUALIFIED PERSONS CERTIFICATES

CERTIFICATE OF QUALIFIED PERSON

I, Leonard Paul Staples, as a co-author of the Technical Report, do hereby certify that:

I am VP and Global Practice Lead at Ausenco, 855 Homer Street, Vancouver, British Columbia, Canada.

This certificate applies to the technical report entitled, "Technical Report on the Feasibility Study for the Lynn Lake Gold Project" with an effective date of December 01, 2017 (the "Technical Report").

My qualifications and relevant experiences are that:

1. I graduated with a Bachelor of Science degree in Materials and Metallurgical Engineering from Queens University in 1993.
2. I am a registered Professional Engineer of New Brunswick (membership number 4832).
3. I have worked as a Metallurgist and process engineer continuously for a total of 23 years.
4. I have read the definition of Qualified Person set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a Qualified Person for the purposes of NI 43-101.
5. I have experience in process operation, design and management from over 15 similar studies or projects including the 80 Mt/y Grasberg complex in Indonesia (1998-2003), the 20 Mt/y Lumwana Project in Zambia (2005-2007), the 26 Mt/y Constancia Project in Peru (2010-2015) and the 38 Mt/y Dumont feasibility study in Canada (2010-2016).
6. I have visited the Property on September 26, 2017 and September 27, 2017.
7. I am the co-author of this report and responsible for section 2, 3, 18, except 18.3 and 18.14, 21 except 21.3 and 21.6, and 24 as well as the related portions of the Summary, Interpretations and Conclusions, and Recommendations and accept professional responsibility for those sections of this technical report.
8. I am independent of the issuer.
9. I have not had prior involvement with the property that is the subject of the Technical Report.
10. I have read National Instrument 43-101 and those parts of the Technical Report for which I am responsible have been prepared in compliance with that instrument.
11. As of the date of the certificate, to the best of my knowledge, information and belief, the Technical Report contains all material scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: January 25, 2018

Signature: [signed and sealed]

Name: Leonard Paul Staples

CERTIFICATE OF QUALIFIED PERSON

I, Edward James McLean, QP for the metallurgy and processing sections, do hereby certify that:

I am Manager Minerals Consulting, Ausenco Solutions Canada Inc., 365 Bay St, Toronto, Ontario.

This certificate applies to the technical report entitled, “Technical Report on the Feasibility Study for the Lynn Lake Gold Project” with an effective date of December 01, 2017 (the “Technical Report”).

My qualifications and relevant experiences are that:

1. B.Sc. (Metallurgy), Uni. of Queensland (1975), FAusIMM, with 40 years experience in relevant minerals processing applications, metallurgical testing, process design, commissioning and operation.
2. I have read the definition of Qualified Person set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a Qualified Person for the purposes of NI 43-101.
3. I have visited the Property on September 26, 2017 and September 27, 2017.
4. I am the co-author of this report and responsible for sections 13 and 17 as well as the related portions of the the Summary, Interpretations and Conclusions, and Recommendations and accept professional responsibility for those sections of this technical report.
5. I am independent of the issuer.
6. I have not had prior involvement with the property that is the subject of the Technical Report.
7. I have read National Instrument 43-101 and those parts of the Technical Report for which I am responsible have been prepared in compliance with that instrument.
8. As of the date of the certificate, to the best of my knowledge, information and belief, the Technical Report contains all material scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: January 25, 2018

Signature: [signed and sealed]

Name: EJ McLean

CERTIFICATE OF QUALIFIED PERSON

I, Jeffrey Volk, residing at 163 Homestead Rd, Bailey, CO, USA, do hereby certify that:

I am employed as Director – Reserves and Resources with Alamos Gold Inc. (Alamos) with an office at 181 Bay St, Suite 3910, Toronto, Ontario, Canada.

This certificate applies to the technical report entitled, "Technical Report on the Feasibility Study for the Lynn Lake Gold Project" with an effective date of December 01, 2017 (the "Technical Report").

My qualifications and relevant experiences are that:

1. I graduated with a Master of Science degree in Structural Geology from the Washington State University in 1986. In addition, I have obtained a Bachelor of Arts degree in geology from the University of Vermont in 1983. I have over 27 years of operational and consulting experience and continuous employment in the minerals industry, specifically in mineral resource estimation, production geology, feasibility studies and economic evaluations. I am knowledgeable in all aspects of public reserve/resource disclosure and compliance. I have completed resource modeling, due diligence, acquisition and evaluations assignments for precious and base metals, platinum group metals, laterite and uranium in Russia and the Former Soviet Union, Australia, Africa, Peru, Philippines, Mexico, Chile and North America gained with Barrick Gold, SRK and Alamos Gold;
2. I am a fellow of the Society of Economic Geologists and a Certified Professional Geologist and member of the American Institute of Professional Geologists (AIPG #CPG-10835). I am also a Fellow and Member of the Australian Institute of Mining and Metallurgy (FAusIMM #304113);
3. I have personally inspected the subject project several times during the period 2014-2016, most recently during the period May 14-17, 2016
4. I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of National Instrument 43-101 and this technical report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1;
5. I, as a qualified person, am not independent of the issuer as defined in Section 1.5 of National Instrument 43-101. I am employed by Alamos Gold Inc., which is a producing issuer;
6. I am the co-author of this report and responsible for sections 6, 7, 8, 9, 10, 11, 12 and 14 as well as the related portions of the the Summary, Interpretations and Conclusions, and Recommendations and accept professional responsibility for those sections of this technical report;
7. As an employee of the issuer I have been involved with the property continuously since 14 November 2014;
8. I have read National Instrument 43-101 and confirm that this technical report has been prepared in compliance therewith;
9. As of the date of the certificate, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated: January 25, 2018

Signature: [signed and sealed]

Name: Jeffrey Volk

CERTIFICATE OF QUALIFIED PERSON

I, Paolo Toscano, residing at 2369 Gamble Rd. Oakville, ON, Canada, do hereby certify that:

I am employed as Director – Projects with Alamos Gold Inc. (Alamos) with an office at 181 Bay St, Suite 3910, Toronto, Ontario, Canada.

This certificate applies to the technical report entitled, “Technical Report on the Feasibility Study for the Lynn Lake Gold Project” with an effective date of December 01, 2017 (the “Technical Report”).

My qualifications and relevant experiences are that:

1. I graduated with a Master of Applied Science degree in Material Science and Engineering from the University of Toronto in 2001. In addition, I have obtained a Bachelor of Engineering degree in Metallurgy from McGill University in 1995. I have over 22 years of operational and consulting experience and continuous employment in the minerals industry, specifically in metallurgy, process design, scoping, pre-feasibility, and feasibility studies, EPCM design and study management.
2. I am registered as a Professional Engineer Ontario, P. Eng #: 90478231.
3. I have personally inspected the subject project several times during the period 2016 – 2017, most recently on October 11, 2017 and October 12, 2017.
4. I have read the definition of “qualified person” set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of National Instrument 43-101 and this technical report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.
5. I, as a qualified person, am not independent of the issuer as defined in Section 1.5 of National Instrument 43-101. I am employed by Alamos Gold Inc., which is a producing issuer.
6. I am the co-author of this report and responsible for sections 4, 5, 19, 22, and 23 as well as the related portions of the the Summary, Interpretations and Conclusions, and Recommendations and accept professional responsibility for those sections of this technical report.
7. As an employee of the issuer I have been involved with the property continuously since 14 March 2016;
8. I have read National Instrument 43-101 and confirm that this technical report has been prepared in compliance therewith;
9. As of the date of the certificate, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated: January 25, 2018

Signature: [signed and sealed]

Name: Paolo Toscano

CERTIFICATE OF QUALIFIED PERSON

I, Adwoa Cobbina, MASc, PEng, QP for the Gordon and MacLellan Site Water Management, do hereby certify that:

I am a water resources engineer for Golder Associates Ltd. 6925 Century Avenue, Suite #100, Mississauga, Ontario, Canada L5N 7K2.

This certificate applies to the technical report entitled, “Technical Report on the Feasibility Study for the Lynn Lake Gold Project” with an effective date of December 01, 2017 (the “Technical Report”).

My qualifications and relevant experiences are that:

1. I am a professional engineer with a BSc in Biosystems Engineering and an MASc in Civil Engineering. I am accredited in Ontario as a member in good standing of the Association of Professional Engineers of Ontario. I have 10 years of experience specialising in surface water management and flood management primarily for mining projects.
2. I have read the definition of Qualified Person set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a Qualified Person for the purposes of NI 43-101.
3. I have not visited the Property.
4. I am the co-author of this report and responsible for sections 18.3.2, 18.3.3, 18.3.10 and 18.3.12 as well as the related portions of the the Summary, Interpretations and Conclusions, and Recommendations and accept professional responsibility for those sections of this technical report.
5. I am independent of the issuer.
6. I have not had prior involvement with the property that is the subject of the Technical Report.
7. I have read National Instrument 43-101 and those parts of the Technical Report for which I am responsible have been prepared in compliance with that instrument.
8. As of the date of the certificate, to the best of my knowledge, information and belief, the Technical Report contains all material scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: January 25, 2018

Signature: [signed and sealed]

Name: Adwoa Cobbina

CERTIFICATE OF QUALIFIED PERSON

I, Karen Besemann, PGeo, QP for the Hydrogeological Conditions for the MacLellan and Gordon sites, do hereby certify that:

I am a hydrogeologist at Golder Associates Ltd. 33 Mackenzie St., Sudbury, Ontario, Canada, P3C 4Y1.

This certificate applies to the technical report entitled, “Technical Report on the Feasibility Study for the Lynn Lake Gold Project” with an effective date of December 01, 2017 (the “Technical Report”).

My qualifications and relevant experiences are that:

1. I am a professional geoscientist with a B.Sc. in Geology from the University of Ottawa. I am a accredited in Ontario as a member in good standing of the Association of Professional Geoscientists of Ontario. I have 18 years of experience specialising in hydrogeological investigations, baseline studies and Environmental Assessments for mining projects.

9. I have read the definition of Qualified Person set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a Qualified Person for the purposes of NI 43-101.

10. I have not visited the Property.

11. I am the co-author of this report and responsible for sections 18.3.1 and accept professional responsibility for that section of this technical report.

12. I am independent of the issuer.

13. I have not had prior involvement with the property that is the subject of the Technical Report.

14. I have read National Instrument 43-101 and those parts of the Technical Report for which I am responsible have been prepared in compliance with that instrument.

15. As of the date of the certificate, to the best of my knowledge, information and belief, the Technical Report contains all material scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: January 25, 2018

Signature: [signed and sealed]

Name: Karen Besemann

CERTIFICATE OF QUALIFIED PERSON

I, Luiz Castro, Ph.D., PEng, QP for the geotechnical rock investigation and open pit design for the MacLellan and Gordon sites at the MacLellan site, do hereby certify that:

I hold the poistion of Principal and Senior Rock Mechanics Engineer at Golder Associates Ltd. 6925 Century Avenue, Suite #100, Mississauga, Ontario, Canada L5N 7K2.

This certificate applies to the technical report entitled, “Technical Report on the Feasibility Study for the Lynn Lake Gold Project” with an effective date of December 01, 2017 (the “Technical Report”).

My qualifications and relevant experiences are that:

1. I am a professional engineer with a Ph.D. (Rock Mechanics) from the University of Toronto. I am an accredited in Ontario as a member in good standing of the Association of Professional Engineers of Ontario (membership number 90517921). I have more than 30 years of experience specialising in geotechnical site investigations and pit slope design for Civil Engineering and Mining projects.
2. I have read the definition of Qualified Person set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a Qualified Person for the purposes of NI 43-101.
3. I have visited the Property on June 25, 2015 to June 27, 2015.
4. I am the co-author of this report and responsible for section 16.2 and accept professional responsibility for that section of this technical report.
5. I am independent of the issuer.
6. I have not had prior involvement with the property that is the subject of the Technical Report.
7. I have read National Instrument 43-101 and those parts of the Technical Report for which I am responsible have been prepared in compliance with that instrument.
8. As of the date of the certificate, to the best of my knowledge, information and belief, the Technical Report contains all material scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: January 25, 2018

Signature: [signed and sealed]

Name: Luiz Castro

CERTIFICATE OF QUALIFIED PERSON

I, Rui Couto, MASc, PEng, QP for the Subsurface Conditions for the MacLellan and Gordon sites and the Tailings Management Facility Design at the MacLellan site, do hereby certify that:

I am a geotechnical engineer for Golder Associates Ltd. 6925 Century Avenue, Suite #100, Mississauga, Ontario, Canada L5N 7K2.

This certificate applies to the technical report entitled, “Technical Report on the Feasibility Study for the Lynn Lake Gold Project” with an effective date of December 01, 2017 (the “Technical Report”).

My qualifications and relevant experiences are that:

1. I am a professional engineer with a BASc and MASc in Civil Engineering from the University of Toronto. I am an accredited in Ontario as a member in good standing of the Association of Professional Engineers of Ontario. I have 12 years of experience specialising in geotechnical site investigations and tailings facility design for mining projects.
2. I have read the definition of Qualified Person set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a Qualified Person for the purposes of NI 43-101.
3. I have visited the Property on September 26, 2016 and September 27, 2016.
4. I am the co-author of this report and responsible for sections 18.3.14, 18.3.15, 18.14 and 21.3.3 as well as the related portions of the the Summary, Interpretations and Conclusions, and Recommendations and accept professional responsibility for those sections of this technical report.
5. I am independent of the issuer.
6. I have not had prior involvement with the property that is the subject of the Technical Report.
7. I have read National Instrument 43-101 and those parts of the Technical Report for which I am responsible have been prepared in compliance with that instrument.
8. As of the date of the certificate, to the best of my knowledge, information and belief, the Technical Report contains all material scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: January 25, 2018

Signature: [signed and sealed]

Name: Rui Couto

CERTIFICATE OF QUALIFIED PERSON

I, Efthymios Koniaris, PhD., P.Eng., as a co-author of the Technical Report, do hereby certify that:

I am currently employed as the President in the Engineering firm Q'Pit Inc.: 819 Blackburn Mews, Kingston, Ontario, Canada, K7P 2N6.

This certificate applies to the technical report entitled, "Technical Report on the Feasibility Study for the Lynn Lake Gold Project" with an effective date of 30 December 01, 2017 (the "Technical Report").

My qualifications and relevant experiences are that:

1. I am a graduate of Queen's University at Kingston, with a PhD granted in 1991 and an MSc granted in 1983. I completed my undergraduate studies at the Technical University of Athens, Greece, with a degree in Mining and Metallurgical Engineering granted in 1980. I am a member in good standing of the Professional Engineers Ontario (Consulting Engineer Designation #100046258), the Technical Chamber of Greece (#31180) and the Canadian Institute of Mining. I have been working in the Mining industry continuously since my graduation. The primary areas of my work include open pit mine design and mine planning, open pit limit design, phase design and open pit mine planning, open pit reserve estimates, open pit mine equipment type and size selection, estimates of capital and operating costs.
2. I have read the definition of Qualified Person set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a Qualified Person for the purposes of NI 43-101.
3. I visited the project sites on May 31, 2017 and June 01, 2017.
4. I am the co-author of this report and responsible for sections 15, 16 except 16.2, 21.3.1, 21.3.6 and 21.6.2.1 as well as the related portions of the the Summary, Interpretations and Conclusions, and Recommendations and accept professional responsibility for those sections of this technical report.
5. I am independent of the issuer.
6. I have not had prior involvement with the property that is the subject of the Technical Report.
7. I have read National Instrument 43-101 and those parts of the Technical Report for which I am responsible have been prepared in compliance with that instrument.
8. As of the date of the certificate, to the best of my knowledge, information and belief, the Technical Report contains all material scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: January 25, 2018

Signature: [signed and sealed]

Name: Efthymios Koniaris

CERTIFICATE OF QUALIFIED PERSON

I, Karen Mathers, as a co-author of the Technical Report, do hereby certify that:

I am a Senior Project Manager and Principal for Stantec Consulting Ltd. (Stantec) with an office at 500-311 Portage Avenue, Winnipeg, Manitoba, R3B 2B9.

This certificate applies to the technical report entitled, “Technical Report on the Feasibility Study for the Lynn Lake Gold Project” with an effective date of December 01, 2017 (the “Technical Report”).

My qualifications and relevant experiences are that:

1. I am a graduate of the University of Manitoba with an Honours B.Sc. and an M.Sc. in Geological Sciences in 1996 and 1999, respectively.
2. I am a Professional Geoscientist registered with Engineers Geoscientists Manitoba (P.Geo. FGC, member no. 22692) and have practiced my profession continuously since 2001 and have worked for Stantec over this entire time.
3. I have over 19 years’ experience in environmental consulting and assessing environmental effects related to project developments.
4. I have read the definition of Qualified Person set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a Qualified Person for the purposes of NI 43-101.
5. I have visited the Property multiple times; most recently on October 11, 2017 and October 12, 2017.
6. I am the co-author of this report and responsible for sections 18.3.11, 18.3.13, 18.14.7, and 20 as well as the related portions of the the Summary, Interpretations and Conclusions, and Recommendations and accept professional responsibility for those sections of this technical report.
7. I am independent of the issuer.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I have read National Instrument 43-101 and those parts of the Technical Report for which I am responsible have been prepared in compliance with that instrument.
10. As of the date of the certificate, to the best of my knowledge, information and belief, the Technical Report contains all material scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: January 25, 2018

Signature: [signed and sealed]

Name: Karen Mathers