Pre-Feasibility Study
Mine Arnaud Inc.
Sept-Iles Deposit, Québec
Final Report

Respectfully submitted to:
Mine Arnaud Inc.

Effective Date:
July 24, 2013

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DATE AND SIGNATURE PAGE

The effective date of the Technical Report NI 43-101 on the Arnaud Project, Sept-Iles, Quebec, Canada is July 24, 2013.

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1. Summary

1.1 Executive Summary

This report presents a pre-feasibility study (PFS) of the Arnaud Deposit. The primary change since the previous feasibility study report is that significant drilling was undertaken to upgrade the resources category from inferred to measured and indicated. The economic analysis was reinitiated based on the updated resources retaining most of the economic assumptions used in the previous study. Further Metallurgical testing was also undertaken in order to verify the feasibility of the process flowsheet.

1.2 The Property

Mine Arnaud's property is located in the Québec North Shore region about 650 km East of Québec City and 900 km from Montréal. The closest cities are Sept-Îles (15 km to the East with a population of 27,623) and Port-Cartier (40 km to the West). The property comprises 218 contiguous claims covering an area of 5,558.05ha of which 2,963ha is Crown Land. The property is 100% held by Mine Arnaud which is, in turn, a subsidiary of Investissement Québec (IQ) therefore, the project is owned at 62% by IQ and 38% by Yara (eventual buyer of the concentrate to be produced). The property hosts an apatite deposit comprising significant phosphate resources. The Property has excellent access to infrastructure and work force due to its proximity to Sept-Iles. The property is limited to the Southeast by the St-Laurence River and the Bay of Sept-Îles and, to the North by an important Hydro-Québec corridor. Arnaud Railway (operated by Wabush Mine - a subsidiary of Cliff Natural Resources), which is connecting Arnaud Junction and Pointe-Noire, also runs through the property in a general East-West direction. The site was visited by Claude Duplessis Eng., and Jonathan Gagné Eng, MBA, in October of 2012.

1.3 Deposit

Discovery of the deposit is fairly recent (1992) and was eventually staked and explored by "Société Québecoise d'exploration Minière" or SOQUEM; which is a state owned entity. The deposit is hosted in the Sept-Îles Anorthosite Complex (SAC): a large, layered, unmetamorphosed, mafic intrusive suite of Cambrian age. The funnel shaped intrusion displays concentric layering. Mineralization occurs above the Critical Zone in what is termed the Mine Series Stratigraphy. At the base of the deposit massive olivine-ilmenite-magnetite-apatite rocks occur in bands up to several meters thick in gabbro (Nelsonite Layer). Stratigraphically above the Nelsonite horizon, ilmenite, magnetite, and apatite are disseminated, throughout the host gabbros in varying quantities, in three principal layers referred to as the Railroad, Upper and California layers. The layers have shallow dips (-20° to -40°) to the southeast.

Chlorine (Cl), a contaminant in the process of apatite concentrate, is found in the host rock and in the ore composing the Sept-Îles deposit. This particularity is a major concern for Mine Arnaud due to the fact that the Cl content of the produced concentrate cannot exceed 0.14% (YARA specifications). To this date the relationship between the Cl in the head assays and in the concentrate are not perfectly understood. Recent work on the subject enabled SGS Geostat to create a predictive model based on a modal and statistical analysis of in-situ feed K_2O% from metallurgical testwork done by COREM.

1.4 Drilling and Resource Estimation

Completion of 189 drill holes for 23,995m in 2012 and 2013 resulted in a significant increase of measured+indicated resources from 105Mt at 5.32% P_2O_5 to 482Mt at 4.18% P_2O_5. Part of this tonnage increase is at the expense of inferred resources which were reduced from 157.4Mt at 4.66% P_2O_5 to 42.8Mt at 3.52% P_2O_5.

The mineral resource estimate has been calculated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definitions Standards for mineral resources in concordance with National Instrument 43-101 – Standards of Disclosure for Mineral Projects. Mineral resources which are not mineral reserves do not have demonstrated economic viability. Inferred mineral resources are exclusive of the Measured and Indicated resources. For additional information on assumptions used to estimate the resources declared herein please see section 14 of this report.

The resources were estimated using kriging interpolation of blocks within Genesis© software. Each mineralized layer was domained separately, and variography was completed independently. On the total 2,859,094 blocks, 2,537,166 blocks have been interpolated for the P_2O_5% value and the density value. Cl was interpolated for 2,771,847 blocks; most missing values for Cl are from 2010 SOQUEM DDH campaign and would need to be re-assayed in order to be interpolated in the next resource estimation. In order to proceed to pit optimization and resource reporting, certain metallurgical parameters had to be imported in the block model and calculated from interpolated data. In particular, Weight Recovery (Wrec) was estimated through the formulas defining a linear correlation with P_2O_5%. The pit shell from the optimization was used to limit the extent of the mineral resources at depth. The mineral resources are stated at two different cut off grades, depending on the zone they belong to. The general cut off grade is 1.76% P_2O_5 except for the blocks inside the Nelsonite envelope where a cut off grade of 1.91% P_2O_5 was used. Due to the low grade nature and high Cl content, the blocks belonging to the California zone and the blocks above the California zone were subtracted from the model and considered as waste material.

1.5 Mineral Resource Estimate

The open pit optimization was conducted on the Sept-Îles deposit to limit the mineral resources at depth, under the NI 43-101 requirement of "reasonable prospect of economic extraction" regarding mineral resources. The resulting calculated mineral resource is:

Table 1-1: Mineral Resources

Category	Cut Off (P2O5%)	Zones	Tonnage (Mt)	Average P2O5 (%)	Average WRec (%)	Average P2O5 Conc (%)	Average Cl Conc (%)
Measured	1.91	Nelsonite	38.48	5.91	13.62	38.98	0.082
Measured	1.76	Others	332.39	3.95	9.25	38.25	0.124
Measured		**TOTAL**	**370.87**	**4.16**	**9.70**	**38.69**	**0.106**
Indicated	1.91	Nelsonite	9.39	6.22	14.31	39.06	0.087
Indicated	1.76	Others	101.48	4.06	9.48	38.27	0.127
Indicated		**TOTAL**	**110.87**	**4.24**	**9.89**	**38.75**	**0.113**
Inferred	1.91	Nelsonite	-	-	-	-	-
Inferred	1.76	Others	42.76	3.52	8.29	38.11	0.158
Inferred		**TOTAL**	**42.76**	**3.52**	**8.29**	**38.11**	**0.158**

Notes: -The mineral resource estimate has been calculated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definitions Standards for mineral resources in concordance with National Instrument 43-101, Standards of Disclosure for Mineral Projects.

-Mineral resources which are not mineral reserves do not have demonstrated economic viability.

-Inferred mineral resources are exclusive of the Measured and Indicated resources.

-Resources are constrained by Pit Shell and under the bottom surface of the California zone.

-Effective date 10-07-13.

-Others are referring to the material categorize as Combine and Surrounding mineralization.

The resources stated in the present report have to be treated carefully when reporting and using for further economical analysis. Assumption regarding the WRec and Rec were made in order to proceed with pit optimization. The statistical relations used in order to determine block WRec and Rec still need to be verified by metallurgical testing. The following Mineral Resources do not represent Mineral Reserves because they have not yet demonstrated economic viability.

1.6 Metallurgical Testing and Mineral Processing

Because all previous testwork made at COREM and SGS Lakefield prior to 2012 were done on samples that most likely were not representative of the future mill ore feed, Mine Arnaud requested that more tests and a pilot plant run be done on a representative sample consisting of the following ore blend proportion : 36.1% Upper, 44.5% Railroad and 19.4% Nelsonite. Unfortunately, the last pilot plant run completed in September-October 2012 at COREM returned ambiguous results and flotation tests made at SGS Lakefield in March 2013 failed to attain acceptable apatite concentrate grades and recoveries. Following COREM's recommendation, and at SGS's request, Mine Arnaud demanded that five more lock cycle tests (LCT's) be done at COREM, to provide evidence for the feasibility of the process. At the suggestion of SGS Geostat, the LCT's were to be witnessed by its metallurgical engineer representative. COREM' projects T1405 and T1518 served as the core of chapters 13 and 17 of this pre-feasibility study.

The first LCT was done on a non magnetite sample that had already been tested in September 2012 (COREM's Project T1405) on a seasoned ore blend comprising 36.1% Upper, 44.5% Railroad and 19.4% Nelsonite to prove that with the same ore blend, employing the same recipe, the results are highly reproducible. The ore for these LCT`s came from a weathered surface sample that was specifically retrieved for the COREM's September-October 2012 pilot plant trial. At the request of SGS Geostat, the ore for the other four LCT`s was prepared by SGS Lakefield from the same ore blend but coming from drill core rejects and selected according to the ore grades. Originally there was supposed to be only four LCT's representing low, medium and high grade samples. Eventually, it was decided to add a fifth sample of medium grade material also prepared by SGS Lakefield. This fifth LCT became necessary because at that time, it was taken for granted that all medium grade ores were similar and the same reagent recipe could be used. The decision to add a fifth sample was in response to the difference in the metallurgical results between a medium grade weathered surface sample and a medium grade non altered sample, and the very different metallurgical response between the first two LCT's, supposedly both made on medium grade samples utilizing the same reagent recipe.

Contrarily to the four first LCT's, where the magnetite separation was done with a hand magnet in a flotation cell, the magnetite separation for this last LCT was done utilizing the pilot plant laboratory double drum low intensity magnetic separator. This procedure was chosen mainly due to the lack of time for the test but also because both COREM and SGS Geostat metallurgists agreed that this procedure was more reproducible and representative of a commercial mill operation.

Table 1-2: Summary of LCT's Results

Date	Feed	WRec	Rec.	Concentrate				
	% P2O5	%	%	% P2O5	% Fe+Al	% Mg	Ca/P	% Cl
09/14/2012	4.71	10.82	90.5	39.46	1.26	0.31	2.13	0.127
06/04/2013	4.69	10.75	89.9	39.26	1.08	0.27	2.07	0.133
06/05/2013	5.56	13.61	90.5	36.96	2.23	0.72	2.22	0.100
06/06/2013	7.26	16.29	87.5	39.02	1.25	0.40	2.21	0.081
06/11/2013	3.51	8.03	88.7	38.83	1.30	0.45	2.21	0.097
06/12/2013	4.18	9.85	94.1	39.97	0.76	0.27	2.21	0.087

More than probably due to the absence of a quick iron and titanium analyser, this last series of tests partly failed to prove beyond the shadow of a doubt that the process is viable and reliable. SGS Geostat is nevertheless confident that if the ore is specifically mined based on its low chlorine content, in a well monitored mill using the right flotation machines, with ROM and concentrate blending possibilities, a concentrate grade of at least 39% P_2O_5 will be achieved along with an apatite recovery in the 90% range while meeting all of Yara' specifications

1.7 Mining and Mineral Reserve

Taking into account the geometry and the depth of the mineralized zone, only open-pit mining method has been considered in this study. The near surface resources will be mined by a large open pit, which will have 28 years of production following three years of construction and pre-production period. Mining operations will be conducted by the Project operator. Surface mining will follow the standard practice of an open-pit operation; with conventional drill and blast, load and haul cycle using a drill/truck/shovel mining fleet. The overburden and waste rock material will be hauled to the overburden and waste disposals areas near the pit. The mineralization will be drilled, blasted and loaded by hydraulic excavators and delivered by large mining trucks to the gyratory crusher or stockpiles near the crushing plant.

A life-of-mine scenario was developed over 365 days per year. The mine plan is calling for a yearly full capacity production of 11,201,120 tonnes processed per year, reached at year 4, and an average stripping ratio of 0.76 over 28 years of mining operation, plus 2.5 years of stockpile processing. The following graph is summarizing the production and the waste removal.

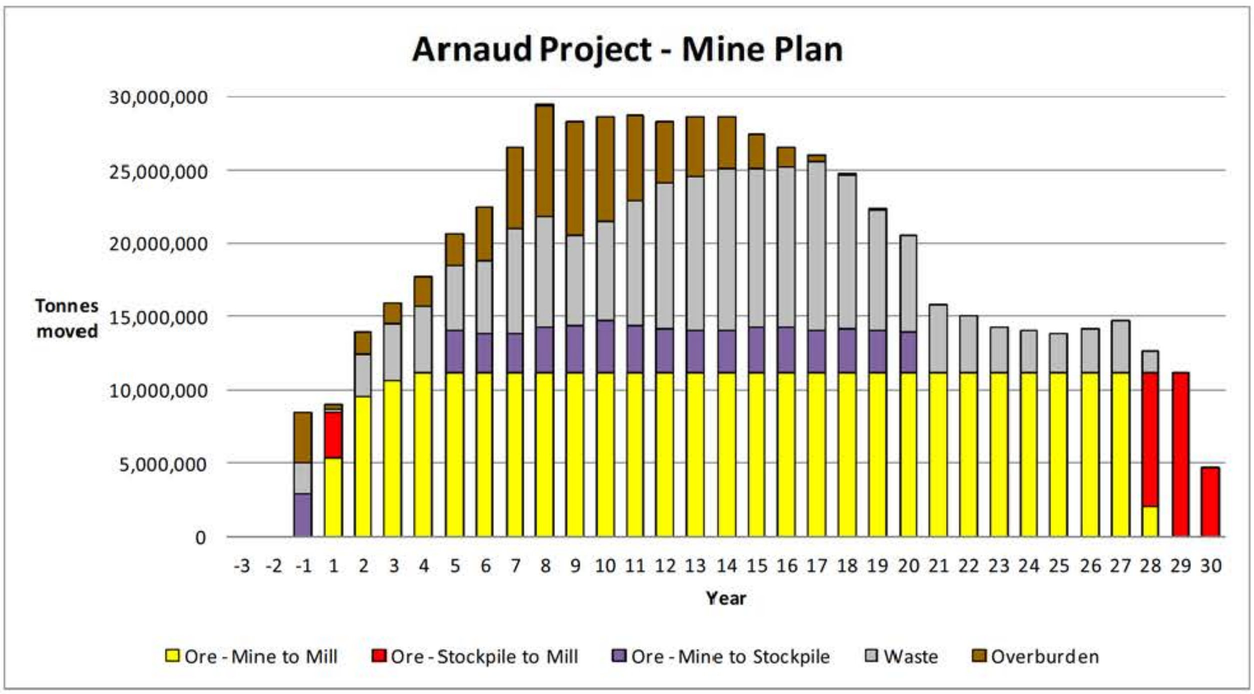

Figure 1-1: Developed Mine Plan

The reserves derived from the detailed pit design and the mine plans have been estimated in accordance with the definitions and guidelines adopted by the Canadian Institute of Mining, Metallurgy, and Petroleum (CIM Standards on Mineral Resources and Reserves). The reserves are based entirely on measured and indicated resources and were converted as probable and proven reserves respectively. The detailed mineral reserve estimate is shown in following Table.

Table 1-3: Mine Arnaud Project Reserves (presented as mill feed)

Material Type		Cut-off (%P2O5)	Tonnes*	Grade* (%P2O5)	Grade (%Wrec)	Concentrate tonnes	Chlorine** (%Cl)
Ore (Probable Reserves)	Combine	1.76%	54 400 000	4.67%	10.85%	5 900 000	0.112%
	Surrounding material	1.76%	1 900 000	2.39%	5.75%	110 000	0.136%
	Nelsonite	1.91%	8 100 000	5.79%	13.34%	1 080 000	0.081%
	Total		**64 400 000**	**4.74%**	**11.01%**	**7 090 000**	**0.108%**
Ore (Proven Reserves)	Combine	1.76%	227 000 000	4.26%	9.93%	22 540 000	0.110%
	Surrounding material	1.76%	7 000 000	2.50%	6.00%	420 000	0.109%
	Nelsonite	1.91%	26 000 000	5.62%	12.97%	3 370 000	0.073%
	Total		**260 000 000**	**4.34%**	**10.13%**	**26 330 000**	**0.105%**
Ore (Total Reserves)	Combine	1.76%	281 400 000	4.34%	10.11%	28 440 000	0.111%
	Surrounding material	1.76%	8 900 000	2.47%	5.95%	530 000	0.114%
	Nelsonite	1.91%	34 100 000	5.66%	13.06%	4 450 000	0.075%
	Total		**324 400 000**	**4.42%**	**10.30%**	**33 420 000**	**0.106%**

Note: **This reserve includes 2% dilution and 98% mining recovery for Railroad, Upper and Surrounding material ore types and a 10% dilution and 90% mining recovery for the Nelsonite ore type. Surrounding material is referring to mineralized material between zones but excluding the sterilized material as defined in Figure 16-2.**Chlorine grade estimation in concentrate, see section 12.4.*

1.8 Capital and Operating Costs

Table 1-4: Capital expenditures (Capex)

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Next Table presents a summary of total estimated operating costs used to develop the cash flow Table. Operating costs were estimated based on the average over the life of the mine.

Table 1-5: Operating costs (Opex)

Operating cost	unit	$*

*Based on a concentrate production of 1.2 Mt per year

**Mining cost of first mining bench

1.9 Economic Analysis

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Based on the mine plan presented in Section 16, a cash flow forecast had been prepared. A summary of the base case results is given in next Table.

Table 1-6: Project Cash Flow Summary

Items	Units	Value

*After start of production

Table 1-7: Project Sensitivity Results

Parameters	Units	-30%	-20%	-10%	0%	+10%	+20%	+30%

1.10 Conclusion, Interpretation and Recommendations

Following the last study realized on Mine Arnaud deposit, numerous progresses have been made. Drilling significantly increased the level of confidence of the mineral resources and it is now possible to estimate the quality of the produced concentrate using different models. Increasing geological knowledge will enable to limit unpredicted variation of topics like ore grade, concentrate produced and mining sequence.

SGS recommend to Mine Arnaud to move forward to the next step in the development of its phosphate project with completion of the recommendation for additional works.

Recommendations for further development of the Sept-îles deposit are made in section 26 of the report and can be summarized, with cost estimates, in the following table.

Table 1-8: Future Work Cost Estimate

	Item	Amount	Source
Geology	Condemnation drilling	$1,400,000	*Mine Arnaud , SGS Canada In. &Ccontractors*
	Define structures in pit	$10,000	*SGS Canada Inc.*
	Cl prediction tool	$15,000	*SGS Canada Inc.*
	Re-assay Cl on hole 1166-10-83	$2,000	*Commercial lab*
	Solidify Wrec	$20,000	*SGS Canada Inc.*
	Magnetite distribution	NA	*Mine Arnaud in house project*
	Pit Slope and OVB stability	$325,000	*SGS Canada Inc. & Golminds*
	OVB Instrumentation	$175,000	*SGS Canada Inc. & Golminds*
Processing	In pit blending parameters	NA	*Mine Arnaud in house project*
	45 tonnes pilot plant	$ 1,000,000	*SGS Canada Inc.*
	Lock Cycle Test on blends	$40,000	*SGS Canada Inc.*
	Mineralogy and liberation size sutdy	$100,000	*SGS Canada Inc.*
	Filter press test	$10,000	*SGS Canada Inc.*
	Kiln vs flash dryer	NA	*Will be evaluated in EPCM*
	SPI and BWI caracterization	$250,000	*SGS Canada Inc.*
Environmental	Investigation of propane burner	$15,000	*SGS Canada Inc.*
Mining & Economical	Optimise design & planning	NA	*Mine Arnaud in house project*
	Review equipment fleet	NA	*Will be evaluated in EPCM*
	Confirm wall stability of Ovb	NA	*Mine Arnaud in house project with consultant*
Feasibility	Technical Report	$ 3,300,000	
Sub-total		**$6,700,000**	
Contingencies	Contingencies 25%	$1,700,000	
TOTAL		**$8,400,000**	

2. Introduction

The pre-feasibility (PFS) is for the developpement of the Sept-îles deposit, operated by Mine Arnaud Inc. located near Sept- Îles, Quebec. This PFS has been based solely on the development of the Sept-îles deposit. Mine Arnaud Inc. commissioned the engineering consulting group SGS Canada Inc. – Geostat ("SGS Geostat") to perform this Study. This Report was prepared at the request of Mr. François Biron, Project Director of Mine Arnaud, a Canadian private company, with its head office situated at:

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2.1 Terms of Reference

The authors for SGS are Mr. Claude Duplessis, Eng., Mr. Jonathan Gagné, Eng., MBA, Mr. Gilbert Rousseau, Eng., Mr. Gaston Gagnon, Eng., and Mr. Jean-Philippe Paiement, M.Sc., P.Geo. This Technical Report was prepared in compliance with the NI 43-101 format and regulations.

Information in this report is based on a critical review of the documents and information provided by the personnel of Mine Arnaud, in particular Mr. Hugo Latulippe, Eng. and Mr. Bruno Perron, Eng. A complete list of the reports available to the authors is found in the References section of this report.

2.2 The Technical Report

This technical report was prepared to support the disclosure of mineral resources and mineral reserves for the Arnaud property ("Property" or "Project"). The report describes the basis and methodology used for modeling and estimating the Arnaud apatite deposit located on the property from drill holes completed by Mine Arnaud during previous exploration programs (including 2012 and 2013 campaigns). The report also presents a review of the history, geology, sample preparation, analysis, data verification of the project and recommendations for future work. Following the updated mineral resource, SGS Geostat completed an update of the Project economic analysis and disclosure of new minerals reserves. This updated analysis was conducted using a combination of parameters derived from the previous Feasibility Study and established directly by SGS.

The reader must be advised that the content of this Technical Report is based on the previously published Feasibility Study (FS) prepared by Roche-Ausenco. Some sections remain the same and new information has been added in their respective sections.

2.3 Disclaimer

It should be understood that the mineral resources which are not mineral reserves do not have demonstrated economic viability. The mineral resources and reserves presented in this Technical Report are estimates based on available sampling and on assumptions and parameters available to the author. The comments in this Technical Report reflect the author's best judgment in light of the information available. During the mineral resource estimation process and the economical modeling, various assumptions were made. These assumptions were used in order to calculate the modeling cut-off grades and resources/reserves cut-off grades following the "reasonable prospect for economic extraction" stated in the NI 43-101 regulations and to develop the financial model. The reserves derived from the detailed pit design have been estimated in accordance with the definitions and guidelines adopted by the Canadian Institute of Mining, Metallurgy, and Petroleum (CIM Standards on Mineral Resources and Reserves). The reserves are based entirely on measured and indicated resources and were converted as probable and proven reserves respectively. They are also limited to the final pit limit and above the marginal cut-off grade.

Moreover as per sections taken and used from the previous study prepared by Roche-Ausenso, SGS has reviewed and presumed these to be accurate and usable for the purpose of this report. This does not wave the responsibility of Roche-Ausenco for the work performed in their study.

2.4 Site Visit

A site visit was conducted by Claude Duplessis, Eng. and Jonathan Gagné, Eng., MBA at the Sept-Îles project site in October 2012. Jean-Philippe Paiement, M.Sc., P.Geo., and Floran Faiello visited the Chibougamau core storage location in November 2012. The site visits enabled the authors to examine the core storage facilities, the area corresponding to the Sept-Îles deposit and get familiar with the region. During the site visit, the authors collected independent control samples of the drill core pulp reject for chemical testing plus coarse core reject for metallurgical tests.

2.5 Abbreviations and Terms of Reference

Abbreviation	Description
$	Dollar
%	Percentage
"	Inch
¢/kWh	Cents per kilowatt hour
°	Angular degree
°C	Degree Celsius
µm	Micron
2SD	Two standard deviations

Abbreviation	Description
3D	Three dimensions
3DL	Three times the detection limit
3SD	Three standard deviations
AADT	Annual average daily traffic
ABA	Acid Base Accounting
ACQ	Québec Construction Collective Agreement
AGP	Acid Generation Potential
AI	Abrasion index
ALS	ALS Minerals
Am3/hr AMSL	Actual cubic meter per hour Above Mean Sea Level
ANFO	Explosives based on ammonium nitrate and diesel
ANP	Acid Neutralization Potential
AREMA	American Railway Engineering and Maintenance-of-Way Association
ATM	Atmosphere
ATV	All-terrain vehicles
AWWA	American Water Works Association
BAPE	Bureau d'audiences publiques sur l'environnement
BH	Borehole
BPL	Bone Phosphate Lime
BV	Best value
BWI	Bond ball mill work index
C.O.G.	Internal cut-off grade
C:V	Capacity of the accumulation area divided by the volumeof the dikes
CAD	Canadian dollar
CANMET	Canada Centre for Mineral and Energy Technology
CAPEX	Capital expenditure (Capital Cost Estimate)
Cat6	Category 6 UTP Cable
CCME	Canadian Council of Ministers of the Environment
CCTV	Closed circuit television
CDE	Canadian Development Expenses
CEAEQ	Centre d'expertise en analyse environnementale du Québec
CEE	Canadian Exploration Expenses
CEPA	Canadian Environmental Protection Act
CFM	Cubic feet per minute
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CLM	Consolidated Thompson Iron Mines
CN	Canadian National
COREM	Centre de recherche minérale

Abbreviation	Description
CRM	Certified reference material
CTEU-9	Water Leaching Test
CW	Crest width
CWI	Crusher work index
d	Day
d/a	Day per year
dBA	Decibel
DC	Direct current
DDH	Diamond drill hole
DFO	Department of Fisheries and Oceans, Canada
Dia DL	Diameter Detection Limit
DRASTIC	A Standardized System for Evaluating Ground Water Pollution Potential Using Hydrogeologic Settings
E	East
EBITDA	Earnings before interest, depreciation, and amortization
EBS	Environmental Baseline Study
EFG	Eriez Minerals Flotation Group
EHS	Environmental health and safety
EMP	Environmental management plan
EPCM	Engineering, Procurement and Construction Management
EQA	Environment Quality Act
ESIA	Environmental and Social Impact Assessment
FCC	Fertilizer and Chemical Consultancy Ltd.
FOB	Freight on board
FS	Feasibility Study
ft	Feet
g	Gram
G&A	General and administration
g/l	Gram per liter
g/t	Gram per metric ton
GEA	GEA Barr-Rosin
Genivar	GENIVAR Société en Commandite
Geostat GHG	SGS Geostat Green House Gas
GIIP	Good international industry practice
GPS	Global positioning system
GTK	Geological Survey of Finland
h	Hour
h/d	Hour per day
ha	Hectare

Abbreviation	Description
HCM	hydrogeologic conceptual model
HDPE	High Density Polyethylene
HG	Head Grade
HP	Horsepower
HPGR	High Pressure Grinding Rolls
HQ	Hydro-Québec
I/O	Input/Output
IBA	Important Bird Area
ICP	Inductively Coupled Plasma
ID2	Inverse distance squared
in	Inch
ing.	Member of the "Ordre des ingénieurs du Québec"
IOC	Iron Ore Company of Canada
IP	Internet Protocol
IQ	Investissement Québec
IRM	Internal reference material
IRR	Internal rate of return
IT	Information Technology
JK	Julius Kruttschnitt
JVA	Joint Venture Agreement
kg	Kilogram
kg/h	Kilogram per hour
kg/kWh	Kilogram per kilowatt hour
kg/t	Kilogram per metric ton
km	Kilometer
km/h	Kilometer per hour
kPa	Kilopascal
kV	kilovolt
kW	Kilowatt
kWh	Killowatt hour
kWh/t	Kilowatt hour per metric ton
kWh/y	Kilowatt hour per year
l	Liter
l/s	litres per second
LGOS	Low-grade ore stockpile
LIMS	Low-Intensity Magnetic Separation
LOI	Loss of ignition
LOM	Life-of-mine

Abbreviation	Description
LRS	Liquid Resistance Starter
m	Meter
m/s	Meter per second
m2	Square meter
m^3	Cubic meter
m3/h	Cubic meter per hour
Ma	million years
masl	Meters Above Sea Level
Mbps MC	Megabits Per Second Moisture Content
MCC	Motor control centre
MDDEP	Ministère du Développement durable, de l'Environnement et des Parcs
Met-Chem	Met-Chem Canada inc.
METSO	METSO Corporation
Mg	Milligram
min	Minute
mm Mo mps	Millimeter Month Metre per second
MRC	Regional municipalities
MRNF	Ministère des Ressources Naturelles et de la Faune
MRNQ	Ministère des Ressources Naturelles du Québec
MSDEP	Ministry of Sustainable Development, Environment andParks
MSDS	Material safety data sheets
Mtpa	Million tonnes per annum
Mtpy	Million tonnes per year
MVA	Megavolt-ampere
MW	Megawatt
N	North
N/mm2 NAPS	Newton per square millemeter National Air Pollution Surveillance Program
NEL	Nelsonite
NFPA	National Fire Protection Association
NI	National Instrument
NN	Nearest Neighbour
Nox	Nnitrogen oxides
NPV	Net present value
NQ	Core diameter for drill holes of 47.6 mm
NW	Northwest
O/F	Overflow
OB OER	Overburden Environmental discharge objectives

Abbreviation	Description
OK	Ordinary Kriging
OPEX	Operational Expenses
P&ID	Process and instrumentation diagram
P1	Site in the vicinity of the 3685, Highway 138
P2	Highway 138, at the entrance to the mine site
P3	Site in front of the rotary intersection, Arnaud Street
P_{80}	Particle size that is 80% finer in the overflow
PFS	Pre-feasibility study
ppm	Part per million
PRQEP	Bill modifying the Regulation respecting the quality of drinking water
psi psig	Pounds per Square Inch Pounds per Square Inch Gauge
Q1	Quarter 1
Q2	Quarter 2
Q3	Quarter 3
Q4	Quarter 4
QA/QC	Quality assurance/Quality control
QEMSCAN	Integrated automated mineralogy and petrographysolution
QNS & L	Québec North Shore and Labrador
QNSL	Québec North Shore and Labrador
Rec	Recovery
Roche-Ausenco	Roche-Ausenco Joint Venture
ROM	Run-of-mine
RPA	Roscoe Postle Associates
RPM	Revolutions per Minute
RQD	Rock Quality Determination
RQEP	Regulation respecting the quality of drinking water
RWI	Bond Rod Mill Work Index
s	Second
S.I.H.	hydrogeologic information system
S2	Bulk sample - Nelsonite Ore type
S3	Bulk sample - Upper Ore type
S4	Bulk sample - California Ore type
SAB	SAG/Ball Mill circuit
SAC	Sept-Îles Anorthositic Complex
SAG	Semi-Autogenous Grinding Mill
SARM	South African Reference Materials
SE	Southeast
SEM SER	Scanning Electron Microscope Slip Energy Recovery

Abbreviation	Description
SG	Specific Gravity
SGA	Studiengesellschaft für Eisenerz-Aufbereitung
SGS SIL	SGS Lakefield Research Limited Safety Integrity Level
SIMDUT	Système d'information des matières dangereuses utilisées au travail
SMC	Abbreviated Drop Tests
SMS SNF	Surface Mineral Substances SNF Canada Ltd
SPLP	Synthetic Precipitation Leaching Procedure
SRR	Bulk sample - Railroad Ore type
t	Dry metric tons
t/d	Dry metric tons per day
t/h	Dry metric tons per hour
t/h/m2	Dry metric tons per hour per square meter
t/m2/h	Dry metric tons per square meter per hour
t/m3	Dry metric tons per cubic meter
t/y	Dry metric tons per year
ta	Abrasion breakage
TCLP	Toxicity Characteristic Leachate Procedure
TDMA	Time division multiple access
TJCM	Table Jamésienne de Concertation Minière
TLs	Threshold limits
TMF	Tailings management facility
tpd	Tonnes per day
tph	Tonnes per hour
tph/m2	Tonnes per hour per meter square
tpy	Tonnes per year
TSS	Total suspended solids
U/F	Underflow
UPS	Uninterruptable power supply
USD	US dollar
USEPA	United States Environmental Protection Agency
usgpm	usgallon per minute
UTM	Universal Transverse Mercator
V	Volt
W	West
w/w	Weight (% of total solution/mixture weight)
WHIMS	Wet High-Intensity Magnetic Separation
Whittle	Whittle® 4X software package
WRD	Waste rock dump
wt%	Percent weight

Abbreviation	Description
WW82	Wheat starch
XRF	X-Ray Fluorescence (analytical method)
y	Year
ZCI	Lower Coronitic Zone
ZCR	Critical zone, located at the contact between the LayeredSeries and the Transitional Series
ZCY	Cyclic zone
Zec	Zone d'exploitation contrôlée
Zec Matimek	Association de chasse et pêche Sept-Ilienne Inc
ZGA	Gabbro zone
ZGT	Gabbro-Troctolite Zone
ZTI	Lower transitional zone
ZTP	Troctolite Porphyry Zone
ZTS	Upper Transitional Zone

3. Reliance on Other Expert

SGS prepared this Report using reports and documents as noted in Section 27 "References" of this Report. The current Report has been written by the main authors (Mr. Claude Duplessis, Eng., Mr. Jonathan Gagné, Eng., MBA, Mr. Gilbert Rousseau, Eng., Mr. Gaston Gagnon, Eng. and Mr. Jean-Philippe Paiement, M.Sc., P.Geo), who are responsible for sections 1 to 27 and its content, with the exception of sections 19 (Market Studies and Contracts) and 20 (Environmental Studies, Permitting and Social or Community Impact).

Additional drilling at the request of SGS to fill a wedge and footwall contact has been performed in 2013 by Innovexplo, an independent geological service company based in Val d'Or. SGS QP's did not visit the site during the drilling and relies on the service company for the accuracy of the information provided.

Section 11 has been in part reproduced from the feasibility study (FS) and was prepared by Roche-Ausenco and RPA, SGS QP's have reviewed the work done and are satisfied with the conclusions, interpretations and recommendations stated in this given section.

Section 19 relies on Yara's letter of intent confirming its interest in buying Mine Arnaud apatite concentrate at the specified price used in the previous feasibility study. This letter of intent is presented in the Appendices section of this report. Section 20 has been reproduced from the feasibility study (FS) and was prepared by Roche-Ausenco which remains responsible for their work. SGS QP's have reviewed the work done in the feasibility study and are satisfied that the information presented is accurate and sufficiently detailed for the current technical report and disclosure.

This document presents an opinion based on professional judgment and reasonable care. The conclusions are consistent with the level of detail included in this study and based on the information available at the time of writing.

The authors endorse the professional liabilities of their assertions only within these aforementioned limits. These assertions were provided only with the intent to estimate reserves and the economic viability of the Project, to provide Arnaud with a strategy in developing the project and as a prerequisite to justify the funding needed to complete a feasibility study.

4. Property Description and Location

The property is located in the Québec North Shore region about 650 km East of Québec City and 900 km from Montréal. The closest cities are Sept-Îles (15 km to the East) and Port-Cartier (40 km to the West). The deposit is partly covered by the St-Laurence River in the Central North part of the Bay of Sept-Îles. It is easily accessible through Highway 138, which runs approximatively one kilometre to the south of the deposit limit along the Bay of Sept-Îles shoreline. Highway 138 is a provincial road linking the communities of the Québec North Shore along the St-Lawrence River to the rest of Québec province. The immediate vicinities of the Project are scarcely populated and mainly concentrated along Highway 138 with relatively limited local traffic. From Highway 138, the site is accessible via bush trails which run more or less north-south. The Project is located within the boundaries of Sept-Îles and is zoned as agricultural land, mainly used for the harvesting of firewood.

The property is held 100% by Mine Arnaud which is, in turn, a wholly owned subsidiary of Investissement Québec (IQ). The project is owned at 62% by IQ and 38% by Yara.

The property overlaps National Topographic Series (NTS) Map references 22J/1, 22J/2, 22J/7 and 22J/8. The approximate geographic centre of the property is at longitude 66° 31' 38" west and latitude 50° 16' 13" north. Universal Transverse Mercator (UTM) coordinates for the Project centre utilizing projection North American Datum (NAD) 83, Zone 19 are approximately 676,203 m East and 5,571,606 m north.

The property is comprised of 218 contiguous claims covering an area of 5,558 ha.

Exploration on the property is a joint venture between IQ and Yara. SGS Geostat verified the property title and mineral rights on the Ministère des Ressources Naturelles et de la Faune's (MRNF) web site. The 218 claims, as registered with the MNRF, are 100% owned by Mine Arnaud and are in good standing. Expiration dates range from July 2014 to August 2015.

The area underlying the property is Crown Land (2,963 Ha). Mine Arnaud has the first right to acquire the surface rights to the property by taking the property to mining lease status. Under Québec Mining Legislation, the owner of the mining rights can make use of the timber on the leased property by paying a nominal fee if such timber is deemed to be of commercial value. Mine Arnaud holds surface rights to the site of the former California racetrack (SOQUEM, 2011).

Mine Arnaud reports that there are no outstanding royalties, back-in rights or payments on the Project. Certain claims, designated in APPENDIX 1 – Claim List, are limited by electrical transmission lines, mining facilities and an exclusive lease for Surface Mineral Substances (SMS). Other than those listed in APPENDIX 1 – Claim List, there are no other encumbrances on the Project.

Other mining restrictions exist at the Southeastern edge of the property block. Since 2011, the beach area of the property is characterized as Leisure territory, where mineral exploration is prohibited (Figure 4-3).

Figure 4-1: Project Location

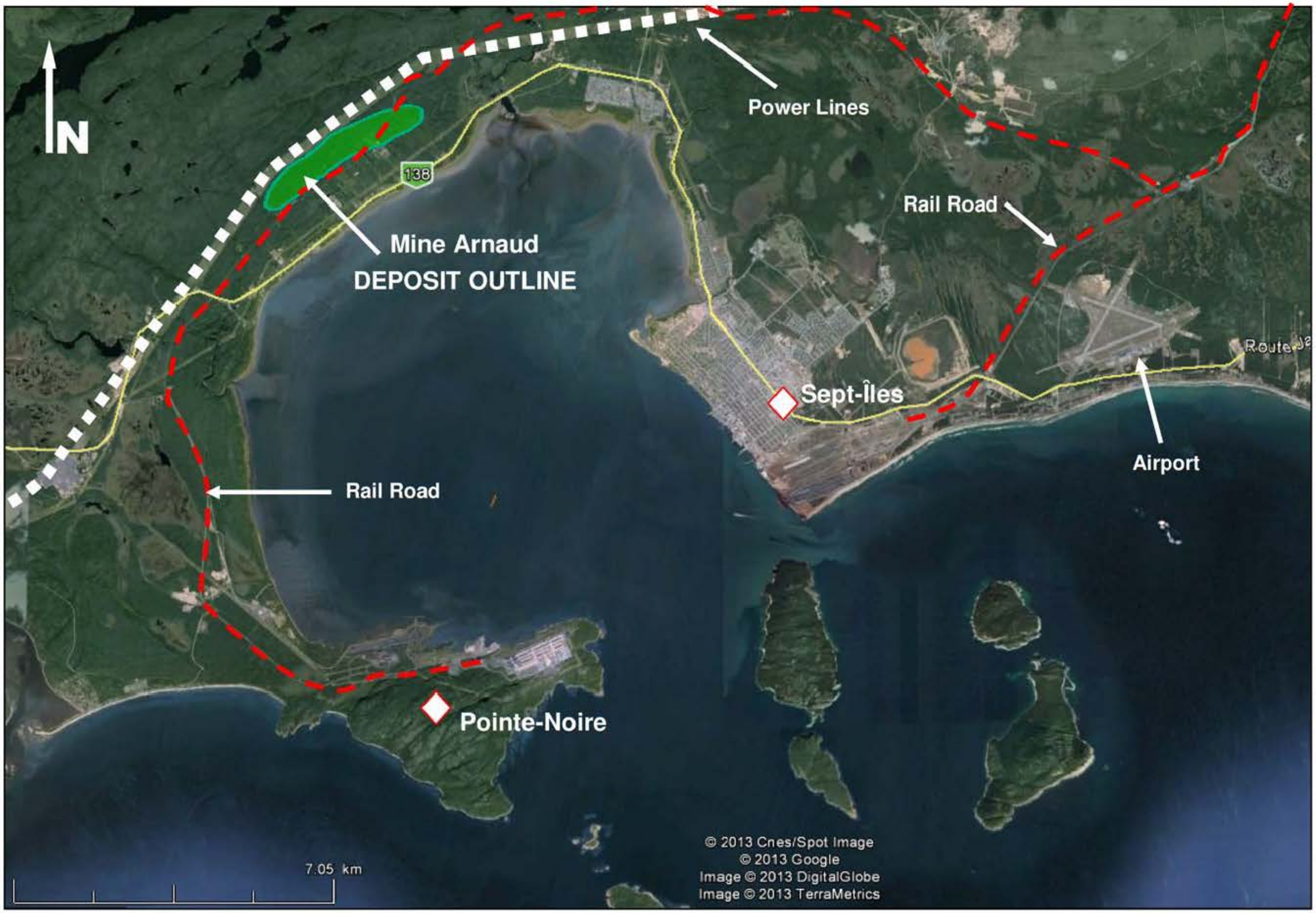


Figure 4-2: Project Location Map

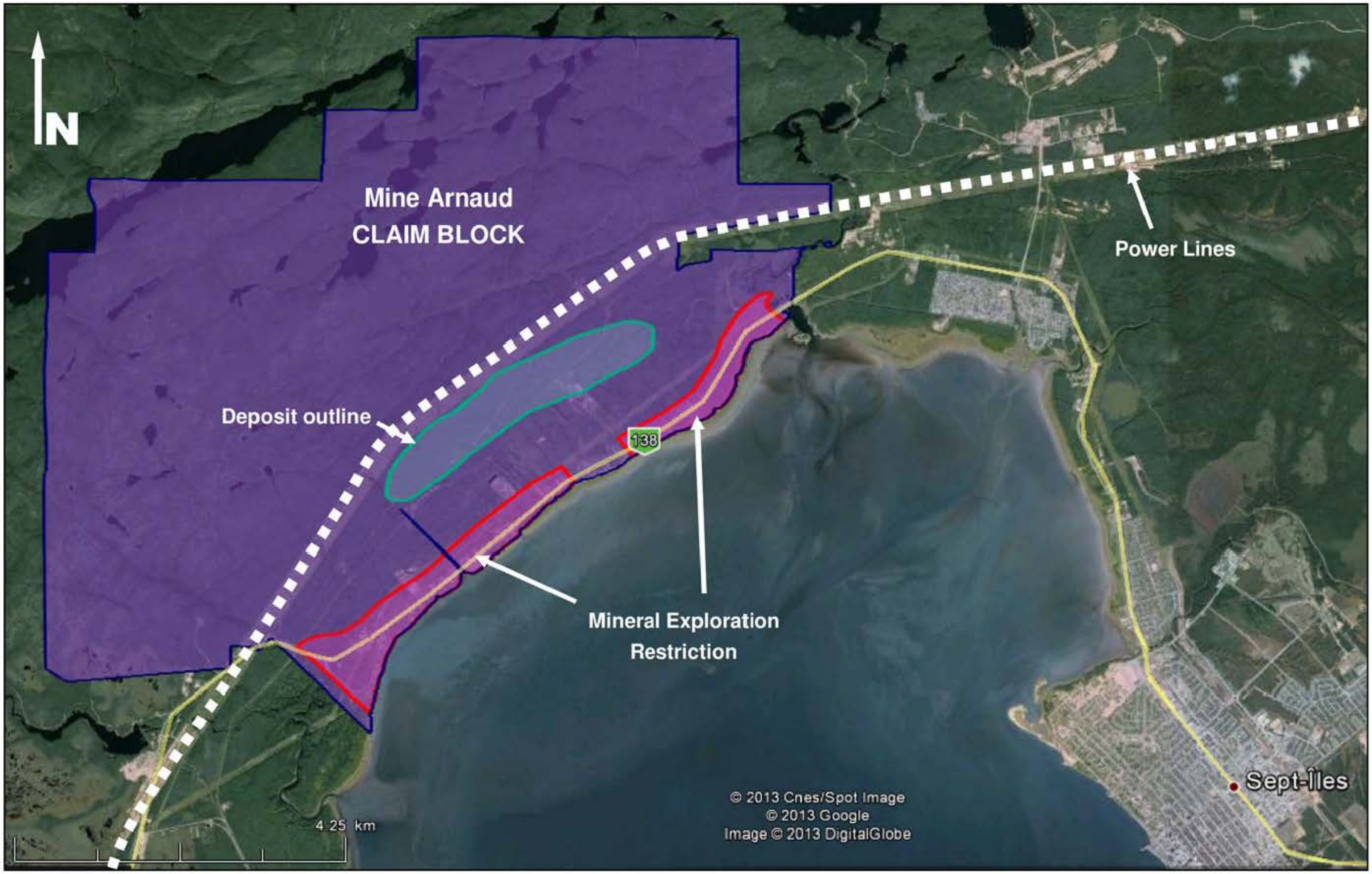


Figure 4-3: Claim block and mineral exploration restrictions map

5. Access, Climate, Local Resources, Infrastructure and Physiography

The information contained in this section was derived from the Feasibility Study (prepared by Roche-Ausenco) dated of February 2012 and was updated by SGS Canada Inc. on July 4th 2013.

5.1 Access

The property is located about 15 km west of Sept-Îles and is easily accessible through Highway 138, which runs approximatively one kilometer to the south of the deposit limit along the Bay of Sept-Îles shoreline. Highway 138 is a provincial road linking the different communities along the St-Lawrence River's North shore. The 138 Highway also connects this region to Southern Quebec through Québec city. From Highway 138, the mine site can be access via bush trails which run more or less northwest-southeast.

Otherwise, the regional road network is not very dense and is mostly limited to a secondary road linking Highway 138 to Pointe Noire Area. This road to Pointe-Noire is the access point to the port facilities (Quai de la Relance), but also to major industrial sites (Aluminerie Alouette, Wabush Mines and Consolidated Thompson).

The north part of the deposit is limited by a large and important Hydro-Québec corridor (Figure 4-3) which already contains three 735 kV power lines coming from Churchill Falls. A fourth line (from La Romaine Hydro project) will be constructed in 2013 to the south of the already existing ones. In addition, a 161 kV line runs in parallel of the 735 kV lines and a second 161 kV line is located to the south of the property between Highway 138 and the deposit limit.

Arnaud Railway (operated by Wabush Mine-a subsidiary of Cliff Natural Resources), which is connecting Arnaud Junction and Pointe-Noire, also runs through the deposit in an East-West direction (Figure 4-2). Rail connection to North American network is available through a railcar ferry terminal located at Pointe-Noire. The ferry-rail allows transfer of railcars from the Québec North Shore network and is operated by Canadian National (CN). However, through the Québec North Shore and Labrador (QNSL) and Arnaud rail carriers, access to Sept-Îles and Labrador City is easily available.

5.2 Local Resources and Infrastructure

Sept-Îles is a major regional center of the Québec North Shore. It is part of the MRC des Sept-Rivières which covers about 32,000 km^2. The city of Sept-Îles is the regional administrative centre for government services of the administrative region. Sept-Îles has a long history of mining-related activity, and has well-developed infrastructure to support mining projects. It is connected to major iron ore mines in Labrador via the Québec North Shore and Labrador Railways. The city also hosts a deep sea port built for shipping iron concentrates and pellets (Pointe-Noire) year-round. The Alouette Aluminum refinery, the fifth largest in the world, operates near the Pointe-Noire port (Figure 4-2).

As the entry point for goods and services bound for the iron ore mines, Sept-Îles has access to heavy industrial equipment and skilled labor force. According to the 2006 census, the population of Sept-Îles was 27,623, including the neighboring Galix, Clarke City, Moisie and the native communities of Uashat and Maliotenam. Census data indicates approximately 15% of Sept-Îles' total population completed an apprenticeship or trade school education and an additional 23% held college diplomas or university degrees (Statistics Canada). The area has a robust and affordable electrical power system built and maintained by Hydro-Québec.

5.3 Physiography

The Mine Arnaud Project is entirely located within the Canadian Shield. The bedrock is mainly composed of crystalline rocks (granite, anorthosite, migmatite, quartzite and syenite) attached to the Grenville geological province. The southern part of the deposit drains into the Bay of Sept-Îles via a series of more or less northwest-southeast parallel streams. The western portion of the site drains into Clet Creek, which flows into the Bay of Sept-Îles. The northern part of the site is located in the watershed of an unnamed stream flowing more or less in an east-west direction into Rivières des Rapides, approximately 1 km north of the Bay of Sept-Îles and about 3 km (as the crow flies) downstream from the water intake of the Sept-Îles municipality.

The landscape is composed of small hills which started in the Bay of Sept-Îles at an elevation of about 5 m to culminate at around 130 m immediately to the north-west of the site.

5.4 Climate

The coastal plain of Sept-Îles region has a maritime climate influenced by the proximity of the St. Lawrence Gulf. Relatively warm in the winter and cold during the summer, the waters of the Gulf reduce seasonal and daily thermal amplitudes. The area is also characterized by a high frequency of fog reducing visibility.

The normal and extreme precipitation levels related to rain and snow are provided in Table 5-1. On average, it falls annually 757 mm of rain (65%) and 412 cm of snow (35%), for an equivalent total precipitation of 1,156 mm of water. The average annual evaporation rate from lakes located in the Project area is approximately 405 mm. The annual rate of evapo-transpiration, which is the total amount of water transferred from the soil to the atmosphere by evaporation at the ground level as well as transpiration from plants, is 450 mm. However, there is no evapo-transpiration from November to April. The annual net precipitation (total precipitation less evapo-transpiration) in this area for a zone covered by forest is 706 mm per year.

5.5 Temperature

Minimum, maximum and average temperatures for each month are provided in Table 5-1. The average annual temperature is 0.8°C. The monthly average temperature is less than or equal to 0°C from October to April. The coldest and the warmest months are January and July with average temperatures of -15.3°C and 15.3°C.

5.6 Winds

The dominant wind direction is North from November to March and East from April to October (Table 5-1). On an annual basis, the winds from the East are the most frequent and blow at an average speed of just below 20 km/h (Figure 5-1). Maximum wind speed measured at Sept-Îles occurred in December 1960 and reached 101 km/h, while wind gusts of 161 km/h were recorded in February 1958. The most violent winds gusts are generally coming from the Northwest and the East.

East winds are more frequent in the spring, summer and autumn. Summer marks the arrival of a quieter period. Winds then blow with less intensity and, even if the East winds are still predominant, they can come from all directions, whereas the periods with no wind represent approximately 10% of the time. During summer (June-August), winds from the East are the most frequent and strong. With the fall arrival, winds direction changes gradually to blow more within the first and the fourth quadrant.

In winter time, the winds are predominantly of N and NNE, but they also often come from the NW, the WNW, and the W. NW, N and WNW winds are frequent and relatively strong. Although less frequent, E winds are the strongest (4.4% of the time over 20.5 km/h).

5.7 Vegetation

The study area is located in the moss spruce-stand domain. In this area, the forest cover is clearly dominated by black spruce stands, sometimes associated with balsam fir. Some hardwoods (white birch, quaking aspen and balsam poplar) grow in this area, while the underbrush is covered by feather mosses and ericaceous shrubs and is characterized by the quasi-absence of herbs. At the mine site, the dominant stands are black spruce and fir stands.

Forest dynamics is primarily governed by the spruce and forest fires which play a significant role. As the region receives an average of 1,156 mm of precipitation per year (including 65% as rain), the risk of water erosion can be important, namely in zones subject to deforestation.

Table 5-1: Climate Normal and Average (1971-2000) – Sept-Îles Airport

	Jan	Feb	Mar	Apr	May	Jun	July	Aug	Sept	Oct	Nov	Dec	YEAR
TEMPERATURE:													
Daily Average (°C)	-15.3	-13.4	-7.1	0	5.9	11.7	15.3	14.2	9.3	3.4	-3.1	-11.3	**0.8**
Daily Maximum (°C)	-9.8	-7.8	-2.1	3.8	10.3	16.4	19.6	18.8	13.6	7.4	0.7	-6.5	**5.4**
Daily Minimum (°C)	-20.9	-19.0	-12.1	-3.8	1.5	7	10.9	9.6	4.8	-0.6	-7	-16.1	**-3.8**
PRECIPITATION													
Rainfall (mm)	9.3	10.9	26	61	83.1	99.3	99.8	91.1	113.2	97.5	48.3	18	**757.4**
Snowfall (cm)	87.3	59.7	64.7	37.5	9.1	0	0	0	0	7.9	49	96.9	**412**
Precipitation (mm)	87.4	67.2	88.8	102.8	94	99.3	99.8	91.1	113.2	106.5	97.7	108.1	**1156**
Average snow depth (cm)	56	68	66	40	5	0	0	0	0	0	5	32	**23**
WINDS													
Speed (km/h)	16	15.4	17	16.7	14.9	13.9	12.4	12	13.2	14.1	15.2	15.8	**14.7**
Most Frequent Direction	N	N	N	E	E	E	E	E	E	E	N	N	**E**
Max. Hourly Speed (km/h)	97	90	80	93	83	89	64	68	80	80	89	101	**-**
Max. Gust Speed (km/h)	161	161	121	124	121	129	103	113	154	122	130	159	**-**
HUMIDITY													
Average Vapor Pressure (kPa)	0.2	0.2	0.3	0.5	0.7	1.0	1.3	1.3	1	0.6	0.4	0.2	**0.6**
Average Relative Humidity 0600LST(%)	67.1	66.7	70.2	72.8	73.0	74.8	80.1	82.9	84.8	82.0	77.9	72.2	**75.4**
Average Relative Humidity 1500LST(%)	63.4	60.3	64.0	68.0	66.5	67.3	70.8	70.1	71.3	69.0	68.9	68.1	**67.3**
PRESSURE													
Average Station Pressure (kPa)	100.5	100.6	100.6	100.6	100.7	100.5	100.4	100.6	100.7	100.8	100.6	100.6	**100.6**
Average Sea Level Pressure (kPa)	101.2	101.3	101.3	101.3	101.4	101.2	101.1	101.3	101.4	101.5	101.3	101.3	**101.3**
VISIBILITY (Hours with)													
< 1km	23.9	13.2	21.3	26.3	20.0	23.5	26.0	24.5	22.1	16.9	18.0	27.7	**263.1**
1 to 9 km	110.3	84.4	97.9	103.6	77.6	70.7	79.2	69.4	76.2	69.7	86.6	115.9	**1041.6**
> 9 km	609.8	579.7	624.7	590.1	646.5	625.8	638.8	650.2	621.7	657.5	615.4	600.4	**7460.7**

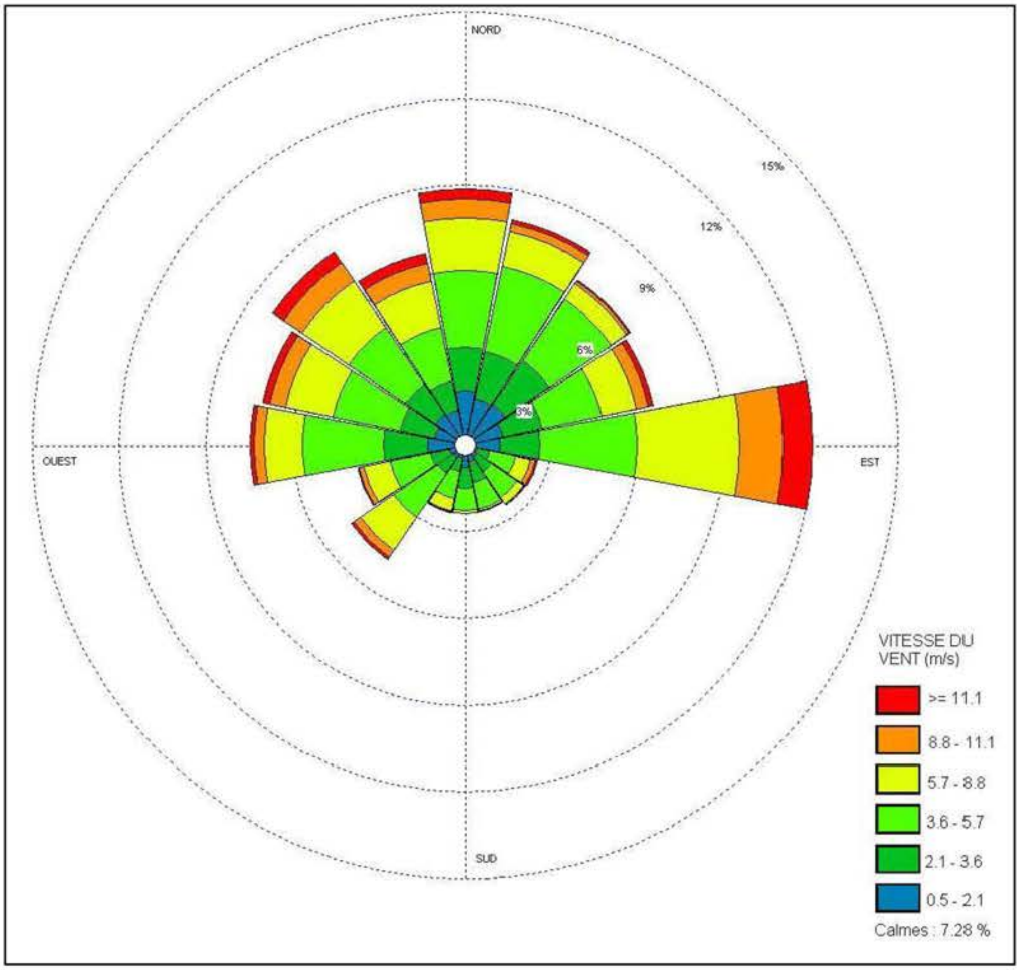


Figure 5-1: Wind Rose – Sept-Îles Airport Meteorological Station – Annual Average
Source: Environnement Canada - Archives nationales d'information et de données climatologiques (données 1953-2005). Données consultées le 20 janvier 2011. Fichiers météorologiques canadiens pour l'énergie et le génie (FMCEG).

5.8 Soil

The landscape and unconsolidated deposits of Québec's North Shore are highly affected by the last glaciations. Sept-Îles area is mainly covered by marine-glacial deposits (according to Klassen and Al. 1992). This material is characterized by many undifferentiated layers of gravel, sand and clay. This can be mainly observed at low altitudes and near the coast. Moving upland (north), the landmark is characterized by a thin layer of till over the bedrock and some moraine. Moraines comprise a mix of boulder, sand, silt and clay. On the top of those layers, there is usually a thin layer of the glacial outwash mainly composed of sand. AMEC and Journeaux & associates showed via geotechnical campaigns that all of the above described features are present on the project's coverage.

The northern portion of the property is covered with a thin layer of till. Boreholes (BH-1 and BH-2), carried out by AMEC, show a layer of sandy gravel with boulders of about 1-2 meters thick. This part of the property is located at higher elevation, and the bedrock is covered by a thin layer of material upwards of the 80 meter elevation. However, peat deposits were found below the 75 meter elevation as revealed by Journeaux and Associates geotechnical investigation. Peat deposits are 5 meters thick and correspond to depressions filled with clay after the retreat of the sea.

Moving forward, the region is covered by a glacial-marine deposit as the elevation decreases going toward Saint-Laurence estuary. During the glacier's retreat, the region was submerged by the Goldthwait Sea. Marine clay was deposited under the 80 meter elevation. A thick clay deposit was observed south of the property. As showed in the boreholes BH-9 and BH-10 realized by AMEC, the clay deposit can be thicker than 25 m. Overlying the clay deposit, a 2-4 meters layer of sand can be observed. This layer was deposited after the retreat of the sea. This layer corresponds to the outwash delta and to the sand beaches.

6. History

The information contained in this section was derived from the Feasibility Study (prepared by Roche-Ausenco) dated of February 2012 and updated by SGS Canada Inc. on July 5th 2013.

Apatite, magnetite and Ilmenite mineralization at the Sept-Îles deposit was recognized in early 1992 by a SOQUEM exploration team and the property was subsequently staked by SOQUEM. Several studies were completed between 1992 and 2005, including geological mapping, drilling programs, resource definition drilling and estimation, metallurgical testing, open pit mine prefeasibility study and environmental impact assessment study (1996), feasibility study (2002) and an updated feasibility study (2005).

The Sept-Îles property was acquired through the staking of open ground that had not seen any modern mineral exploration, development work or significant mineral production. Oxide-iron rich rocks near the present property were the target of an unknown amount of iron and titanium production in the early twentieth century from the Molson, or Chutes du Cran-de-Fer Mine, located at the Cran-de-Fer falls on the River des Rapides.

SOQUEM first explored portions of the Sept-Îles complex for magmatic sulphide deposits in 1977. One rock sample, taken near Clet Creek (ruisseau Clet), assayed 10.8% P_2O_5.

A two phase exploration program was undertaken in 1995 and 1996 that followed up on detailed mapping work done by Jules Cimon of the Ministère des Ressources Naturelles du Québec (MRNQ) in 1994. A 45-hole diamond drill program was conducted and was followed by a mapping and sampling program. In 1995, SOQUEM and Norsk Hydro entered in a joint venture agreement (JVA) to fund a Feasibility Study that would be based on the production of apatite, Ilmenite, and potentially magnetite as a secondary product. A Pre-Report feasibility study was started in 1996 and based on bench-scale test work conducted on drill core. A 145 t bulk sample was extracted from the drill-tested area in 1996 that returned values of 6.6% P_2O_5 and 9.2% TiO_2 (Genivar, 2008) and was used for pilot plant scale test work at Lakefield Research Laboratories Inc. (Met-Chem, 2002).

A feasibility study was conducted in 1997 by the Consortium Met-Chem Pellemon based on a production rate of 600,000t apatite and 425,000t Ilmenite concentrate per year, utilizing a 100% flotation-based processing flowsheet.

In 2002, Met-Chem completed an updated feasibility study incorporating the results of a pilot plant study based on a revised annual production rate of 600,000t of apatite concentrate and 243,000t of Ilmenite concentrate per year. The lower production rate was in response to market conditions at the time of the study. A Market Study was produced by SNC-Lavalin Ltd. (SNC-Lavalin) in 2005 (Genivar, 2008).

Field work in 2008 consisted exclusively of diamond drilling and is discussed below. Also in 2008, an updated, NI 43-101 compliant estimation of the Mineral Resources, using a block model, was conducted by Genivar Inc. based on geological interpretations done by SOQUEM geologists using data from surface exposures and diamond drilling. The estimated mineralized zones covered an area of approximately 2.6 km in length, 600 m in width and up to 350 m in depth below surface.

Exploration work in 2010, 2011, 2012 and 2013 comprised diamond drilling programs, resources estimations, further metallurgical testing and Feasibility Study in 2012 by Roche-Ausenco. The drilling programs were conducted by SOQUEM in from 2010 to 2011 and then in 2012 Axor conducted resource definition dirlling followed by further drilling conducted by InnovExplo.

A Feasibility Study and Capital Cost estimate was completed by Roche-Ausenco in 2012, based on Roscoe Postle Associates (RPA) Mineral Resources estimation of 2012. At the time the Mineral Resources comprised: 105Mt Indicated and Measured at an average grade of 5.32% P_2O_5 and an additional 157Mt inferred at an average grade of 4.66% P_2O_5.

7. Geological Setting and Mineralization

The information contained in this section was derived and modified (June 2013 by SGS Canada Inc.) from the Feasibility Study (prepared by Roche-Ausenco) dated of February 2012.

7.1 Regional Geology

The St. Lawrence valley formed approximately 560 My ago accompanied by volcanic and magmatic activity that resulted in the emplacement of carbonatites along the Saguenay and Ottawa grabens, and Anorthosite Complex, including the Sept-Îles Anorthosite Complex (SAC), a large, layered, unmetamorphosed, mafic intrusive suite of Cambrian age rocks. The SAC is located along the north shore of the St. Lawrence River within the Grenville Province of the Canadian Shield. The SAC has a funnel shape with a diameter of about 80 km with its center located approximately 35 km south-southeast of the municipality of Sept-Îles. Only 5% of the complex outcrops on the Sept-Îles archipelago and west of the municipality of Sept-Îles. More than 85% of the complex is underwater (Figure 7-1).

The rocks have been interpreted to form an orderly suite of successive members of gabbroic to granitic rocks with major units commonly layered. Within the gabbroic rocks some layers are enriched in apatite, ilmenite and magnetite (Genivar 2008). The stratigraphic sequence for the six kilometers thick portion of the SAC, where the project is located, has been studied and presented in a poster session by Jules Cimon wherein the rocks have been grouped into four main sequences (the Lower Series, Layered Series, Transitional Series, and Upper Series). The phosphate-bearing units are located at the base of the Critical Zone (ZCR), which is located at the contact between the Layered Series and the Transitional Series (Figure 7-2). The geological sequences are listed in Table 7-1.

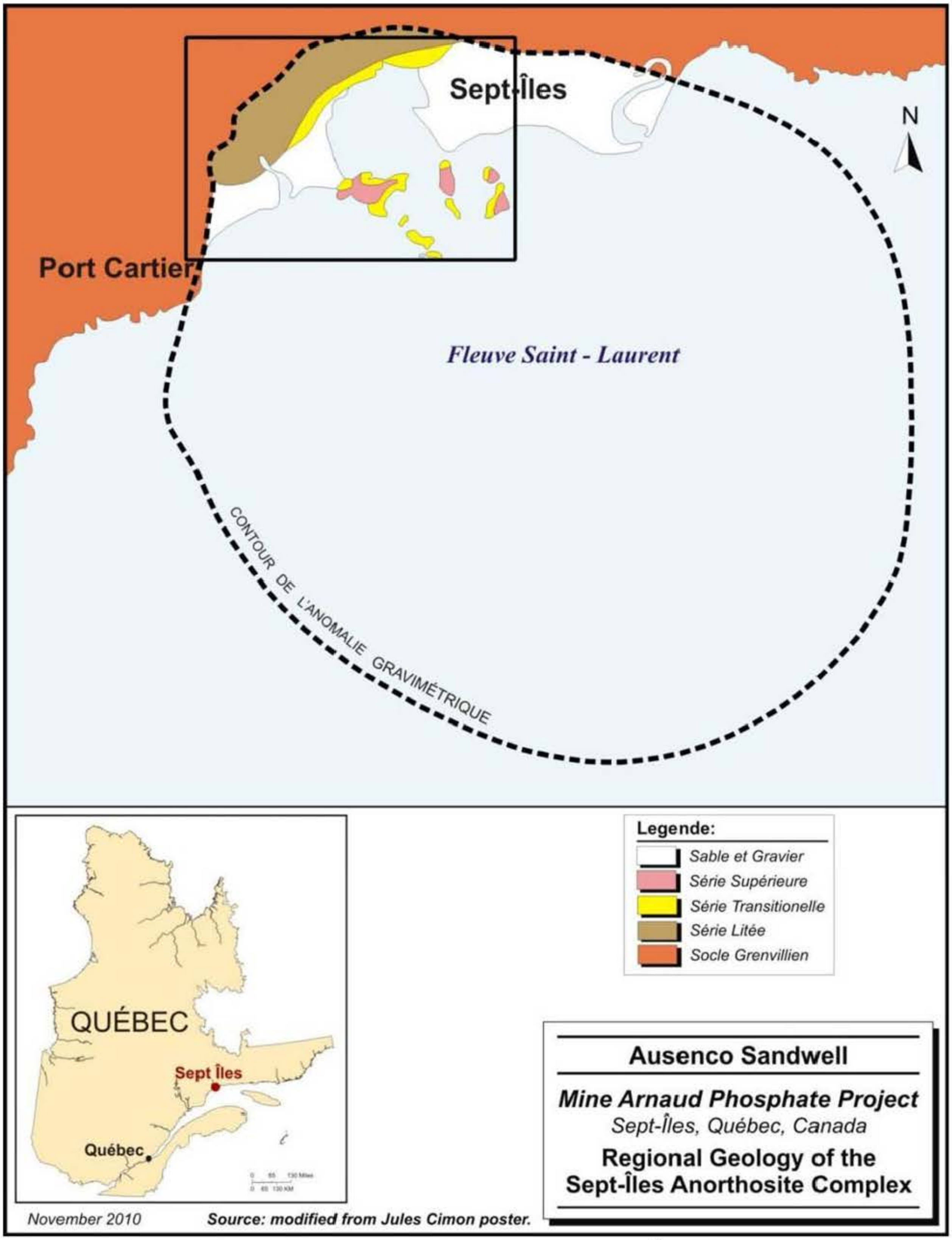


Figure 7-1: Outline of the Gravimetric Anomaly Associated with the Sept-Îles Anorthosite Complex

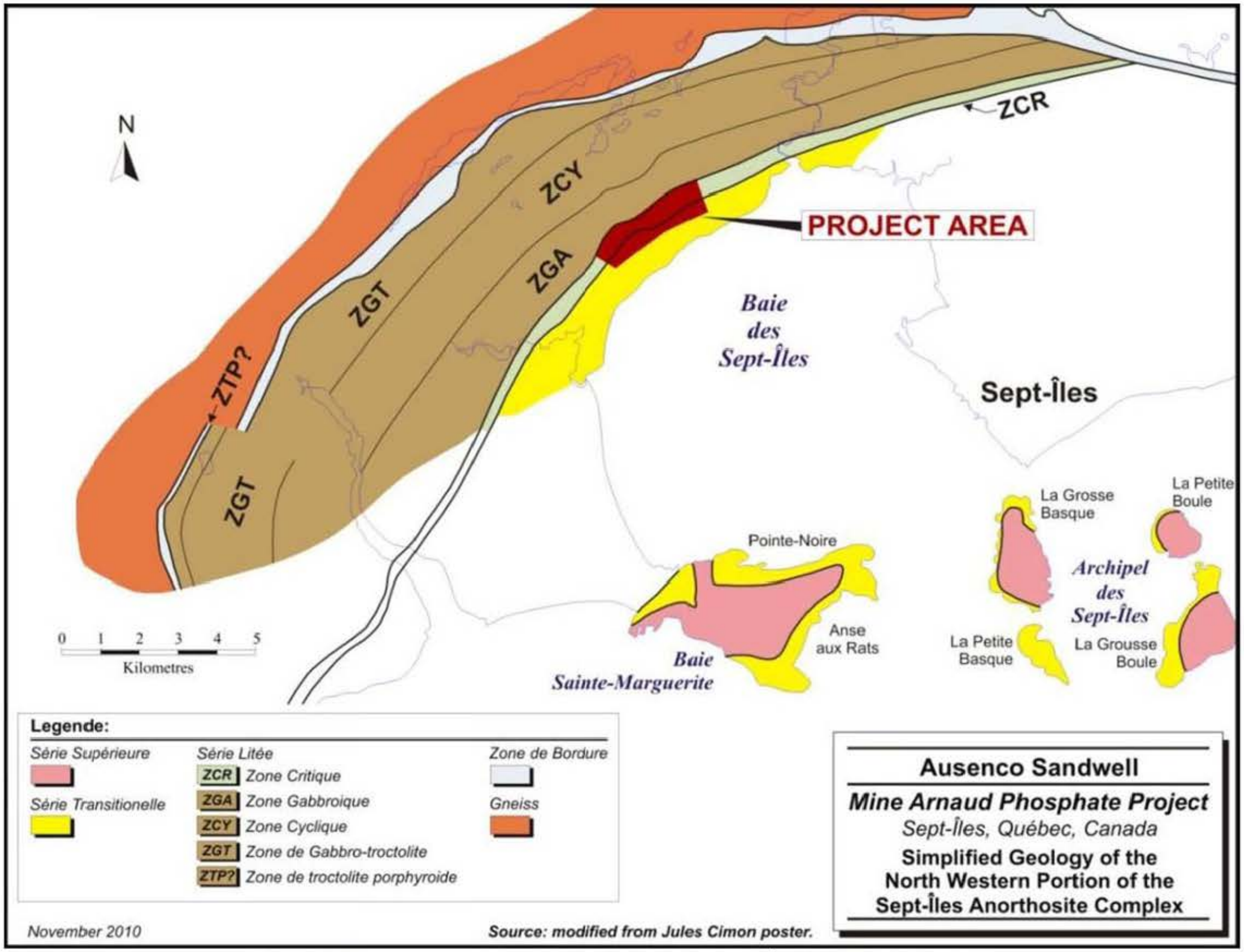


Figure 7-2: Simplified Geology of the N-E Sector of the Sept-Îles Anorthosite Complex

Table 7-1: Stratigraphic Sequence of the Sept-Îles Complex

Mine Arnaud Inc. – Arnaud Mine Project		
Series	**Zones**	**Sub-zones**
Upper Series		
Transitional Series	Upper Transitional Zone (ZTS)	
	Unknown	
	Lower Transitional Zone (ZTI)	
Layered Series	Critical Zone (ZCR)	Microtroctolite
		Gabbro-Nelsonite
		Nelsonite
		Magnetite
	Gabbro Zone (ZGA)	
	Cyclic Zone (ZCY)	Cycle D
		Cycle C
		Cycle B
		Cycle A
	Gabbro-Troctolite Zone (ZGT)	
	Troctolite Porphyry Zone (ZTP)	
Lower Series	Lower Coronitic Zone (ZCI)	
	Unknown	

Source: Derived from SOQUEM, 2011

7.2 Local Geology

The four distinct series that make up SAC are, from the base up, the Lower Series, the Layered Series, the Transitional Series, and the Upper Series. The Lower Series is composed of massive, unbedded leucogabbro and leucotroctolite that is overlain by the Layered Series, which is described in more detail below. The Transitional Series is composed of gabbro and coarse-grained anorthosite that display a band of high magnetic relief that may correspond to stratiform units rich in iron oxide. The Upper Series is differentiated from the other series by the high quantities of quartz and feldspar minerals and is characterised by monzogabbros, monzosyenites, syenites and diorites. Of note is the presence of thin, decimetre- to meter-scale, blocks of white anorthosite that are found in all series in the complex in variable proportions.

The Layered Series has, at its base, a zone of troctolite with beds of olivine leucogabbro and leucotroctolite containing large, centimetre-scale, crystals of plagioclase giving it a porphyritic texture.

Gabbro-troctolite comprises the next zone and contains, in addition to olivine gabbro and troctolite, anorthosite with magnetite rich layers. Units of dunite and magnetite wherlite (peridotite) are also present and are the only ultramafic rocks observed to date in the SAC. The cyclical zone consists of repetitious layers of troctolites and olivine gabbro which are, in turn, overlain by the gabbro zone which is homogenous and composed of a gabbro-magnetite-ilmenite assemblage. The ZCR lies atop the Layered Series and is characterised by a succession of centimetre- to meter-size beds of magnetite ± titano-magnetite and ilmenite and contains up to 2% pyrrhotite locally. This zone ranges up to 30 m to 40 m in width. Above the basal magnetite layer is a layer of Nelsonite (a rock type comprised primarily of magnetite, ilmenite and apatite) with, locally, up to 10% P2O5 and 18% TiO2 in the mineralized beds. The Nelsonite unit varies in thickness from less than one meter to more than ten meters. The contact between basal magnetite and the Nelsonite can either be sharp or can also be gradational over a ten to twenty centimetre width and is an important marker for the onset of phosphate mineralization as no apatite has been observed below this contact except in xenolithic Nelsonite fragments located close to the contact. Above this Nelsonite unit, lies a thick (150 m) succession of stratiform troctolite-iron oxide-apatite layers which represents the bulk of the deposit. Stratigraphically above the ZCR is a 30 m to 50 m thick zone of microtroctolite that is intruded by fine to medium grained, massive olivine leucogabbros that reflect the multiphase nature of the ZCR (SOQUEM, 2011).

The present description of the stratigraphy and mineralization summarizes the understanding of the stratigraphic relationships of the Project using the information available as of June 2011. This understanding will likely change and evolve over time as new information becomes available.

Following review of drill cores during site visits carried out at the Sept-Îles Project site and at the SOQUEM core shack located in Chibougamau, Québec, a schematic stratigraphic column has been constructed from a collaborative effort between SOQUEM and RPA geological staff (Figure 7-3). The intent of this image is to provide a common point of reference for discussions between the various disciplines and different groups. As well, this image will form the basis for construction of the domain models in support of the Mineral Resource estimate for the Project.

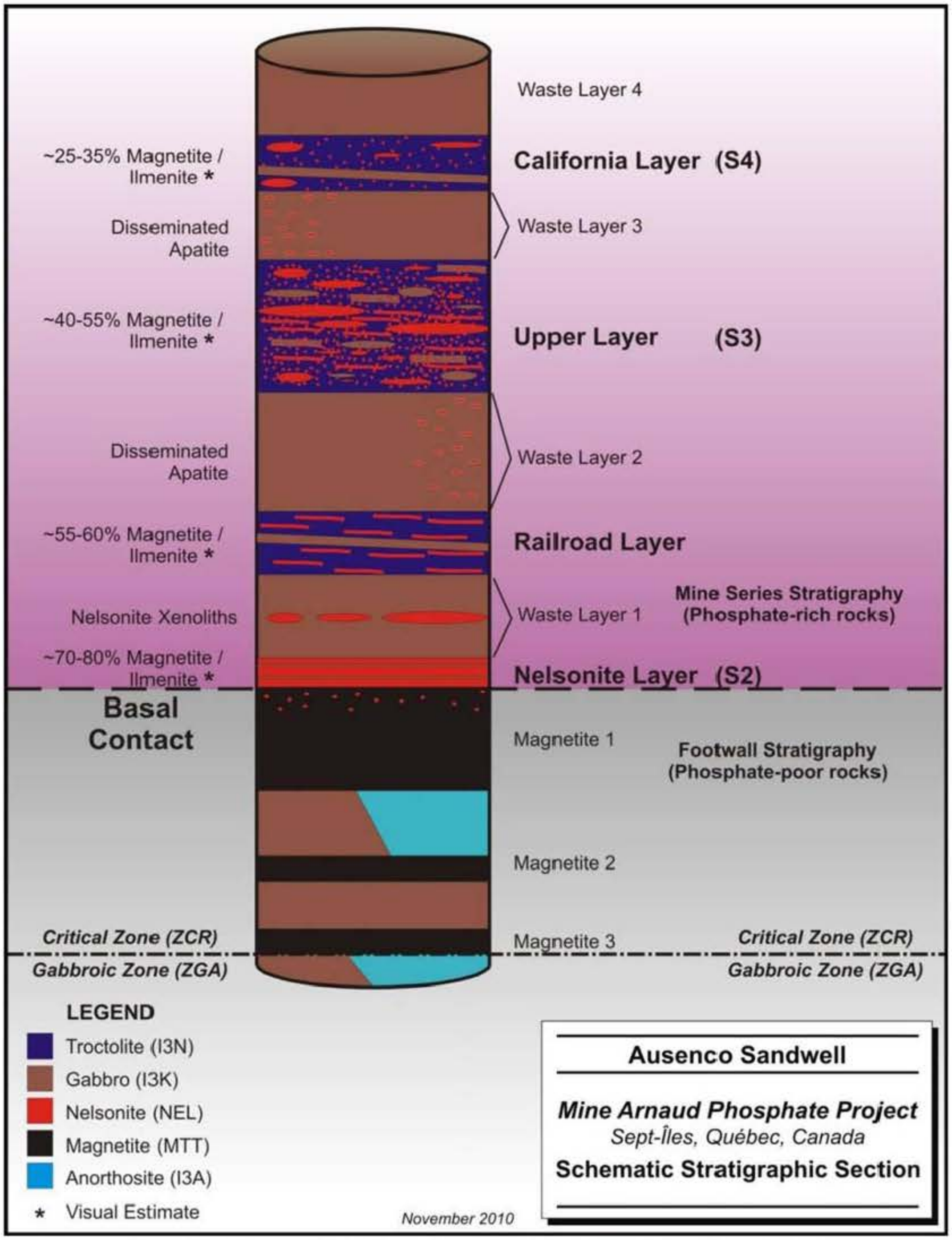


Figure 7-3: Schematic Stratigraphic Section, Mine Arnaud Phosphate Project

7.3 Mineralization

The Project hosts three types of mineralization. 1) The first is hosted in the stratigraphic footwall to the deposit (i.e. below the footwall contact of the Nelsonite unit) and is composed of numerous oxide-rich zones hosting bands of stratiform, nearly-massive magnetite plus ilmenite that are hosted by gabbros. These zones are poor in apatite mineralization. Surface magnetic patterns suggest these rocks have significant lateral extent and thicknesses, up to 50 m. Within the deposit, iron-titanium-phosphorous mineralization takes two forms. 2) At the base of the deposit massive olivine-ilmenite-magnetite-apatite rocks occur in bands up to several meters thick in gabbro (Nelsonite Layer). 3) Stratigraphically above the Nelsonite horizon, ilmenite, magnetite, and apatite are disseminated, throughout the host gabbros in varying quantities, in three principal layers referred to as the Railroad, Upper and California layers.

Apatite concentrations are linked to the magmatic differentiations which enhances concentration within residual magma otherwise present as trace elements in the initial magmatic fluid. High phosphate grades are associated with coarse to very coarse grain sizes and high concentrations of iron and titanium oxides. Low phosphate grades are generally associated with fine to very fine grain size and low concentrations of iron and titanium oxides (Met-Chem, 2002).

8. Deposit Types

The information contained in this section was derived and modified (June 2013 by SGS Canada Inc.) from the Feasibility Study (prepared by Roche-Ausenco) dated of February 2012.

The different phases of drilling are summarized below and helped refine the geological, structural, and geochemical knowledge of the Sept-Îles deposit.

The 2010 drilling identified exploration potential to the northeast. A sub-vertical fault system crosses the mineralized body without significant displacement and two other longitudinal, sub-vertical faults have displacements that vary from nil up to 50 meters and mimic a horst-and-graben type structure. These structures are visible in the magnetic gradient. Since the apatite encountered appears to be associated with magnetism, and the magnetic anomaly continues to the northeast, there is potential for more mineralization in that direction and will be the target of subsequent exploration efforts. Indeed, four widely-spaced drill holes have been completed along the northeastern strike extension and have intersected the phosphate-bearing stratigraphy.

Different stratigraphic levels of mineralization are found and can be described as follows.

8.1 Footwall Layer

The base of the phosphate-bearing sequence is defined as the lower contact of the Nelsonite unit, which forms the top of a sequence of essentially barren massive magnetite layers (Figure 8-1) that are interlayered with units of gabbro and/or anorthosite composition (Figure 7-3). In relative terms, these units contain very low concentrations of apatite and form the footwall units for mining.

8.2 Nelsonite Layer

By definition, the Nelsonite unit comprises rocks that contain greater than 90% magnetite, Ilmenite, titano-magnetite and apatite. The proportion of the various components can vary from place-to-place and can be the dominant component in any one locale. The remainder of the unit is composed of various ferromagnesian and silicate minerals such as plagioclase, olivine and pyroxene. Texturally, this unit can easily be recognized in drill core by the observation of massive layers of the three black iron oxide minerals (magnetite, Ilmenite, titano-magnetite) that are located at the stratigraphic top of the layered magnetite sequence. Little visual difference is observed between the footwall magnetite layers and the Nelsonite, apart from the appearance of white to light-grey coloured apatite. Narrow dikes of gabbroic composition can be seen to cross-cut this unit on occasion and are typically either barren or contain low concentrations of phosphate (Figure 8-3).

8.3 Railroad Layer

The next phosphate-bearing layer that is typically encountered has been referred to as the Railroad Layer. Texturally, this unit is recognized by its stratigraphic position relative to the layered magnetite/Nelsonite sequence and by the development of a weakly to moderately well developed bedded texture. The quantities of black iron-oxide minerals (magnetite, Ilmenite and titano-magnetite) are less than can be seen in the Nelsonite Layer (visually estimated to range from 55% to 60% magnetite/Ilmenite) and the mode of occurrence changes from massive layered to narrower, less continuous layers. As with the Nelsonite Layer, apatite shows a strong association with the discontinuous iron-oxide layers, but can also be observed as disseminated grains in the intervening, relatively silicate mineral-rich material. As observed with the Nelsonite Layer, narrow dikes of gabbroic composition can be seen to cross-cut this unit on occasion and are typically either barren or contain low concentrations of phosphate (Figure 8-4).

8.4 Upper Layer

The next phosphate-bearing layer in the sequence is currently referred to as the Upper Layer, and this unit contains the greatest degree of diversity. Texturally, this unit is recognized by its thickness, the presence of black iron-oxide minerals, the massive texture, the presence of discontinuous fragments (xenoliths), and layers of Nelsonite. The concentration of the iron-oxide minerals in general can be seen to be less than either the Railroad or the Nelsonite Layers, and their contents are visually estimated to range from 40% to 55% magnetite/Ilmenite. The apatite grains can be seen to reside in association with the Nelsonite xenoliths, and as disseminated grains throughout the unit. The assay information that is available as of March 2011 suggests that the Upper Layer can often be portioned into three sub-layers by the phosphate content. A central, slightly higher grade, sub-layer can often be seen to be enveloped by lower grade units above and below. This central unit can be recognized by the presence of weakly developed, narrow, discontinuous layers of Nelsonite.

Few cross-cutting dikes are observed within this unit, however numerous xenoliths of gabbroic composition can often be observed to be present at all levels. In general terms, the phosphate grade within this unit will decrease, as the quantity of these barren/low-grade xenoliths increases.

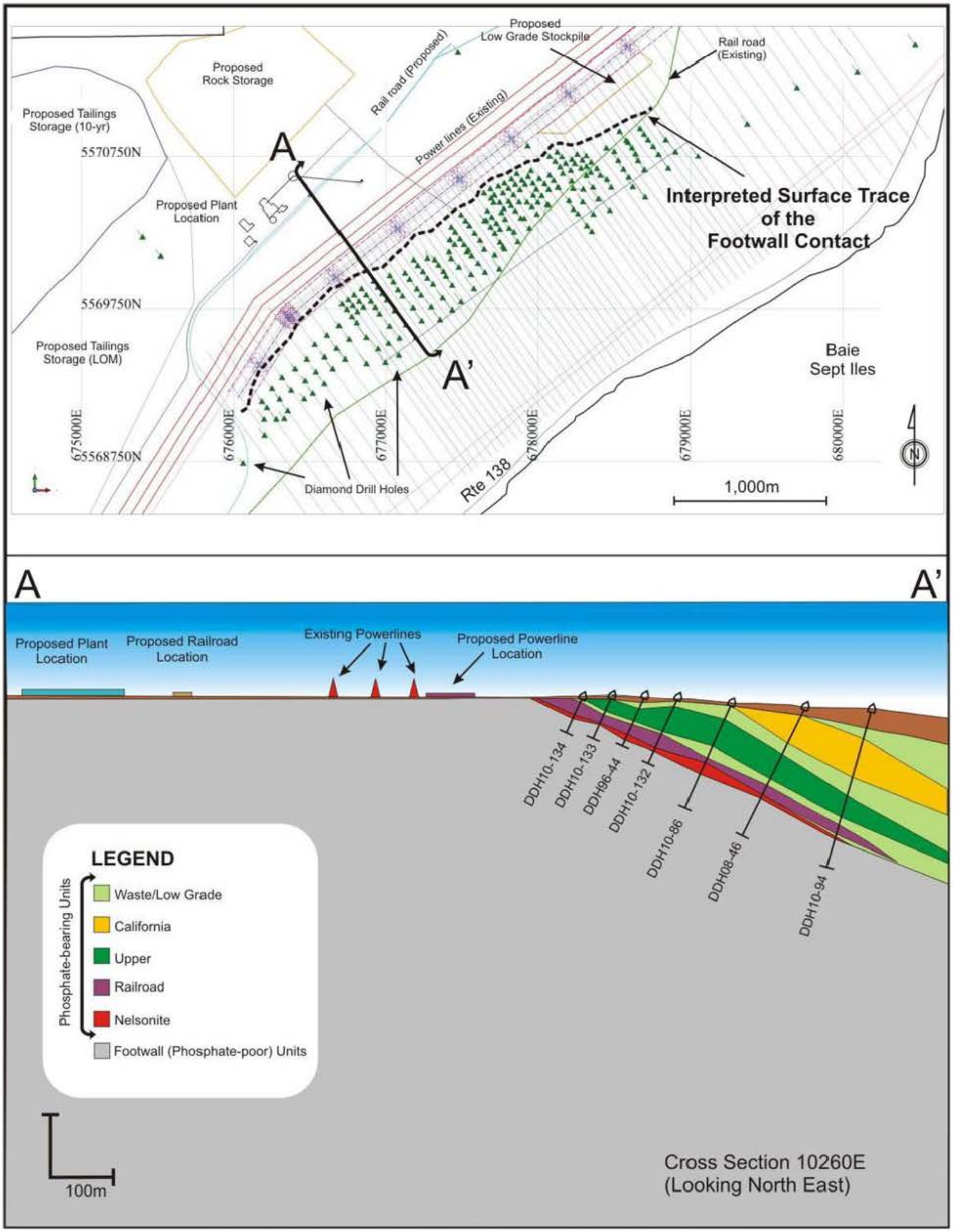


Figure 8-1: Generalized Cross Section 10260E (Looking Northeast)

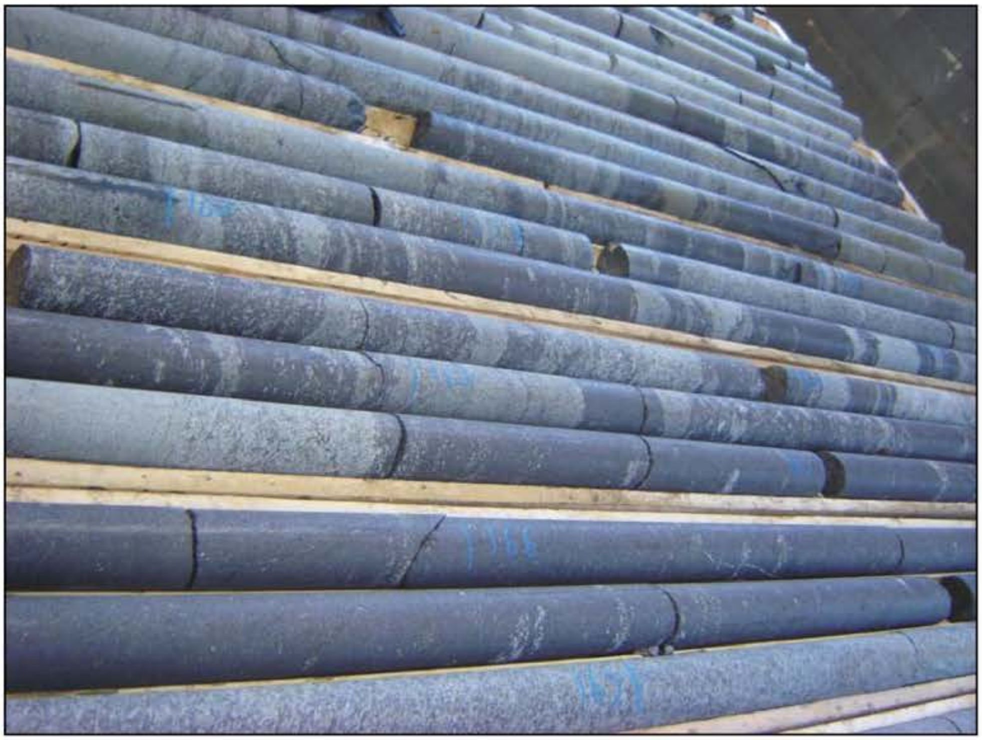


Figure 8-2: View of the Footwall Massive Magnetite Units

Figure 8-3: View of the Nelsonite in DDH1166-10-85
(129.0 m to 129.5 m Assayed 5.88% P2O5)

Figure 8-4: View of the Railroad unit in DDH1166-10-85
(103.0 m to 106.0 m Assayed 10.16% P2O5, 106.0 m to 108.0 m Assayed 8.28% P2O5)

8.5 California Layer

This layer forms the upper limit of the phosphate-bearing sequence as presently understood. It has not been intersected as many times as the three other layers, as it is located relatively high up in the stratigraphy and is only intersected in drill holes that are targeting to test the Nelsonite layer at greater depths. This layer is characterized by the relatively low concentration of iron-oxide minerals (visually estimated to range from 25% to 35% magnetite/Ilmenite). As with the Upper Layer, the apatite can occur in association with xenoliths of Nelsonite and as disseminated grains that are distributed throughout the silicate/ferromagnesian mineral-rich matrix. Narrow dikes of gabbroic composition can be seen to cross-cut this unit on occasion and are typically either barren or contain low concentrations of phosphate.

8.6 Waste Layer

Spaced between these four units are layers dominated by units of gabbroic composition. For the most part, these layers often are barren of phosphate mineralization, or contain only low values. However, the assay information available as of March 2011 show that the phosphate content of these units can vary, and at times are seen to contain potentially economic quantities of phosphate, depending upon the cut-off grade that is applied. The apatite in these units is observed to vary in its form of occurrence in each of the waste layers. The apatite is commonly observed, in general, to be associated with xenoliths of Nelsonite in Waste Layer 1, while the apatite is more often seen to occur as disseminated grains in Waste Layers 2 and 3.

In addition, a barren or low-grade gabbro layer was observed to exist between the footwall contact of the Nelsonite Layer and the underlying footwall stratigraphy. This lower unit was observed to be present only on occasion and was not able to be correlated from hole-to-hole or from section-to-section.

9. Exploration

The information contained in this section was derived and modified (June 2013 by SGS Canada Inc.) from the Feasibility Study (prepared by Roche-Ausenco) dated of February 2012.

9.1 Exploration History

The Sept-Îles property was acquired through the staking of open ground that had not seen any modern mineral exploration, development work or significant mineral production. Oxide-iron rich rocks near the present property were the target of an unknown amount of iron and titanium production in the early twentieth century from the Molson, or Chutes du Cran-de-Fer, Mine located at the Cran-de-Fer falls on the River des Rapides.

In 1953 and 1954 the northern portion of the SAC was explored for iron and titanium by Hollinger (Québec) Ltd. and the Iron Ore Company of Canada. Geological and geophysical work was followed by a diamond drilling program of three holes in the area of Hall River.

A minor drill program comprising two drill holes of 12.2 m (40 ft.), date and diameter unknown, was conducted by M. Dugas near River des Rapides. The holes intersected the magnetite unit which underlies the current deposit.

9.2 SOQUEM Exploration

SOQUEM first explored portions of the Sept-Îles complex for magmatic sulphide deposits in 1977. One rock sample, taken near Clet Creek (ruisseau Clet), assayed 10.8% P_2O_5.

A two phase exploration program was undertaken in 1995 and 1996 that followed up on detailed mapping work done by Jules Cimon of the Ministère des Ressources Naturelles du Québec (MRNQ) in 1994. A 45-hole diamond drill program was conducted and was followed by a mapping and sampling program which yielded 237 samples that were assayed for apatite using gravimetric analysis. Nelsonite samples returned values that ranged from 5.54% to 13.08% P_2O_5 while apatite-rich samples of olivine gabbro and troctolite graded 3.25% to 8.35% P_2O_5. Minor amounts of apatite were also encountered in the basal magnetite unit. In addition, four magnetic profiles were done over the centre of the property.

In 1995, SOQUEM and Norsk Hydro entered in a joint venture agreement (JVA) to fund a feasibility study that would be based on the production of apatite, Ilmenite, and potentially magnetite as a secondary product. A pre-feasibility study was started in 1996 and based on bench-scale test work conducted on drill core. A 145 t bulk sample was extracted from the drill-tested area in 1996 that returned values of 6.6% P_2O_5 and 9.2% TiO_2 (Genivar, 2008) and was used for pilot plant scale test work at Lakefield Research Laboratories Inc. (Met-Chem, 2002).

In 1997, 36 of 45 drill holes were entered into a computer database and used to estimate resources. The work was done by Systèmes GEOSTAT International (GEOSTAT) and formed the basis of a new geological model. This model was used for mining analysis, optimization, design and planning (Genivar, 2008). The estimated Mineral Resources preceded the adoption of NI 43-101, are considered historic in nature, and should not be relied upon.

A feasibility study was conducted in 1997 by the Consortium Met-Chem Pellemon based on a production rate of 600,000t apatite and 425,000t Ilmenite concentrate per year, with no magnetite production, utilizing a 100% flotation-based processing flow sheet. Beneficiation entirely by flotation was deemed too expensive, so high intensity magnetic separation of apatite and Ilmenite with gravity separation of Ilmenite was proposed. Laboratory and pilot plant scale testing was done and received sufficient encouragement that SOQUEM revised previous operating and capital costs in a study done by Met-Chem Canada Inc. (Met-Chem) in 1999 based on a production rate of 1.0 Mt of apatite and 400,000t of Ilmenite concentrates per year. SOQUEM became the sole owner of the property when Norsk Hydro restructured and terminated the JVA (Met-Chem, 2002).

In 2002, Met-Chem completed an updated feasibility study incorporating the results of a pilot plant study based on a revised annual production rate of 600,000t of apatite concentrate and 243,000t of Ilmenite concentrate. The lower production rate was in response to market conditions at the time of the study. A market study was produced by SNC-Lavalin Ltd. (SNC-Lavalin) in 2005 (Genivar, 2008).

As a component of the feasibility study, Met-Chem produced a Mineral Resource estimate for the Project. Using five meter composites and a 500 m search radius, resources were interpolated into the 10 m by 10 m by 8 m block model using inverse distance to the power of two (ID2) weighting. Resources were classified as Inferred and were quoted at 196 Mt grading 6.6% P_2O_5 and 8.81% TiO_2 undiluted with no cut-off grade specified (Met-Chem, 2002). Due to the layered nature of the deposit, with zones (ore) and inter-zones (waste), dilution is inherent in the mining and grades are sensitive to block size and bench height. Met-Chem analysed the bench effect on dilution and recommended the selection of an eight meter bench height. Using the recommended bench height and a cut-off grade of 4.5% P_2O_5 and 6.7% TiO_2, diluted Mineral Resources were estimated to be 185.6 Mt at 6.2% P_2O_5 and 8.4% TiO_2 (Met-Chem, 2002).

In 2004, Norsk Hydro sold its rights and obligations to the Project to its subsidiary, Yara. In 2007, SOQUEM transferred the Project's mineral rights to Mine Arnaud. Under the terms of the agreement, SOQUEM retains the management of the mining titles subject to an agreement which is renewable every two years (SOQUEM, 2011).

Field work in 2008 consisted exclusively of diamond drilling and is discussed below.

Also in 2008, an updated, NI 43-101 compliant estimation of Mineral Resources, using a block model, was conducted by Genivar Inc. (Genivar) based on geological interpretations done by SOQUEM geologists using data from surface exposures and diamond drilling. The estimated mineralized zones covered an area of approximately 2.6 km in length, 600 m in width and up to 350 m in depth below surface. Drill hole spacing was on the order of 100 m by 150 m. In the centre of the mineralized zones, where the thickest, high-grade intersections were encountered, the drilling density was 100 m by 100 m. A total of 1,963 analyzed intervals from 47 drill holes were used in the estimate. Four main zones were identified, down from the historic seven, and calculated, along with lower-grade interzones, with a lateral search radius of 200 m. No grade capping was done on assays and one meter composites were calculated. Block dimensions were 20 m by 10 m by 5 m. Grades were interpolated into the block model using ID2 weighting with Gemcom GEMS software. Using a 4% P_2O_5 cut-off grade, Indicated Resources were calculated to be 148 Mt at 6.2% P_2O_5, 8.7% TiO_2, 30.7% Fe_2O_3 and Inferred Resources of 86 Mt at 6.3% P_2O_5, 9.1% TiO_2 and 31.2% Fe_2O_3.

9.3 Mine Arnaud Exploration

Drilling resumed in 2010 to support a feasibility-level study of the Sept-Îles property. Holes were designed on 100 m by 100 m drill centers as per Genivar's 2008 recommendations. In the second phase of the program, completed in 2011, the goal was to increase the drilling density to a 50 m by 100 m grid for the portion of the deposit that is proposed to be operated in the first ten years. A drill spacing reduction, to 50 m by 50 m, was also proposed in areas that were suspected to be affected by the presence of faults, as determined from interpretation of the magnetic signatures and the topographic lineaments. As a result of this work, mineral resources were estimated using the ISD method for a total of 105,296,442 tons of measured and indicated at an average grade of 5.32%P_2O_5 and 157,365,106 tons of inferred at an average grade of 4.66% P_2O_5. The cut-off grade used to report the mineral resources was 2.6% P_2O_5. No mineral reserves were identified in this Feasibility Study due to the insufficient drilling grid.

Further drilling was conducted in 2012 in order to tighten the grid in some areas of the deposit.

There has been no production at the Project to date.

10. Drilling

The information contained in this section was derived and modified (June 2013 by SGS Canada Inc.) from the Feasibility Study (prepared by Roche-Ausenco) dated of February 2012.

10.1 Drilling Procedures

Drill holes were planned (azimuth, dip, length) by geologists on vertical cross-sections and on vertical longitudinal sections. Historically, drill collars were spotted on the field lines with the use of surveying equipment. Usually, two front sights, identified with wood pickets, were used to align the drill rig. For phase one of the 2010 program, holes were located in the field using a Garmin GPSMAP 60Cx instrument. After the drilling was completed, the collars were surveyed by Group Cadorette of Sept-Îles, Québec. In addition to the phase one holes, Group Cadorette surveyed the 2008 collars. For phase two, drill hole collars were located by Roussy Michaud, a survey company based in Sept-Îles, Québec, using a differential global positioning system (DGPS) (SOQUEM, 2011).

The core diameter for all drill holes completed in 2010 was NQ (47.6 mm). Hole deviations (azimuth and dip) were measured with Reflex EZ-Shot borehole survey instruments approximately every 50 m. These instruments provide accuracy better than ±1°. RPA noted that the presence of magnetite stratigraphy may affect the accuracy of the down-hole instrument readings. RPA stated that, due to relatively shallow depths of the drill holes, the current spacing of the drill hole collars, and the homogenous nature of the mineralized body, these potential deviations will have little effect on the Mineral Resource estimate.

Once retrieved from core barrel the core was placed in sequential order in marked and prepared core boxes labelled with the hole number. Each run, usually three meters, is identified by a wood block on which the depth of the hole was marked. At the end of each shift, core boxes were bound and transported by the drill foreman to the core logging facility where the boxes were opened by SOQUEM personnel (Met-Chem, 2002). For the 2010 program, drill core was shipped to the SOQUEM warehouse in Chibougamau, Québec by Porlier Express Sept-Îles. Logging and sampling was done after field work was completed. In 2011 (phase two), drill core was transported from the work site to Chibougamau by Transport Thibodeau where it was logged and sampled upon reception (SOQUEM, 2011). In 2012 and 2013, the holes were logged at a facility near site in Sept-Îles.

Core was measured and logged for lithology, structure, texture, and alteration by SOQUEM staff. Rock Quality Determination data (RQD) and specific gravity (SG) measurements were also recorded. SOQUEM also reports that core photographs were taken for all the drill holes. SOQUEM geologists selected samples for SG determinations based on visually estimated iron and titanium content, high or low, to make the results as representative as possible (Met-Chem, 2002).

10.2 Previous SOQUEM Drilling

SOQUEM's initial drill program began in 1995 and consisted of 12 NQ holes for an aggregate depth of 2,065.5m that yielded a total of 662 assays. Drilling in 1996 consisted of 33 holes for a total depth of 3,257.7 m and 997 assays. In 2008, an additional 15 holes were drilled in the western part of the property for an aggregate depth of 2,678.6 m with 729 samples taken.

Drill core had been archived in Sept-Îles where some of it had been used for metallurgical testing and the development of a mineral processing flow sheet. SOQUEM retrieved the remaining core from the 1995 and 1996 drill programs and transported it to Québec City in 2008. Core from the 2008 drilling is store at SOQUEM's facility in Chibougamau, Québec.

10.3 2010 Drill Programs

Drilling resumed in 2010 to support a feasibility-level study of the Sept-Îles property. Holes were designed on 100 m by 100 m drill centers as per Genivar's 2008 recommendations. In the second phase of the program, completed in 2011, the goal was to increase the drilling density to a 50 m by 100 m grid for the portion of the deposit that is proposed to be operated in the first ten years. A drill spacing reduction, to 50 m by 50 m, was also proposed in areas that were suspected to be affected by the presence of faults, as determined from interpretation of the magnetic signatures and the topographic lineaments. Drilling on the western extension of the mineralized body was also proposed at a 100 m by 100 m spacing. Most of the drilling was done on an azimuth of 325° and an inclination of -65°. Holes that were not drilled to these specifications were changed because of local ground conditions at the proposed collar site. In the cases where the drill holes had to be moved from their original proposed locations, the hole azimuths and dips were altered accordingly so the same target was intercepted. A total of 181 holes were cored in this program, totaling 21,277.6 m in depth, with 8,867 samples taken.

The SOQUEM technical team conducted the work. A total of 38 holes totaling 7,758.5 m in depth were drilled during the first phase of the program starting in May, 2010. Holes were terminated when the ZCR lithology was traversed and the holes had entered the Gabbro Zone (ZGA) unit. A total of 3,078 samples were collected during this phase of drilling and 164 Certified Reference Materials (CRMs) and blanks were inserted into the sample stream in phase one (SOQUEM, 2011).

The second phase of the program started in October, 2010 and comprised 143 holes with an aggregate depth of 13,519.1 m. At total of 5,789 samples were collected during this phase of drilling, including ten for platinum group elements, and 408 control samples (CRM and blanks) were inserted. All of the core from these two drilling programs was transported to Chibougamau and is stored at the SOQUEM core storage facility there.

10.4 2012 Drill Programs

In the winter of 2012, a drilling campaign was undertaken by AXOR on behalf of Mine Arnaud with the goal of increase the level of confidence in the mineral resources. The drilling campaign primarily focused on the lateral; extension of the deposit and the possible eastern. The drilling campaign followed the recommendation of RPA (2011) following the resources estimation (RPA 2011).

The drilling was done using "NQ" size drill rods and positions were surveyed using a "REFLEX EZ-Shot" instrument. Holes were implanted using a surveying company and DGPS instrument, once the drilling was completed, the collar position was re-surveyed for final positioning.

A total of 180 holes were drilled using 3 drilling rigs between February 2012 and April 23rd 2012. The total length of the campaign is 22,958 meters over which a total of 9,280 samples were sent to ALS laboratories in Val d'Or, Québec Canada for P_2O_5% analysis. A total of 7,405 XRF analyses were done for chlorine content and 6,798 pycnometer measurements for density. During the sampling process, 608 control samples were inserted in the sampling chain for a total of 190 blanks, 220 standards and 198 duplicates.

10.5 2013 Drill Programs

In 2013, a 9 holes drilling program was conducted by Mine Arnaud, under InnovExplo management. The total length of the drilling campaign is 1,041 meters and focused primarily on increasing the level of confidence of the mineral resources in the "wedge" area and northern limit of the deposit (Figure 10-1). The drilling campaign added 414 new assay results to the database. Assays were done by ALS Chemex in Val d'Or, Québec and results were reported for the whole rock composition and Cl content. Assays were conducted using XRF instrumentations.

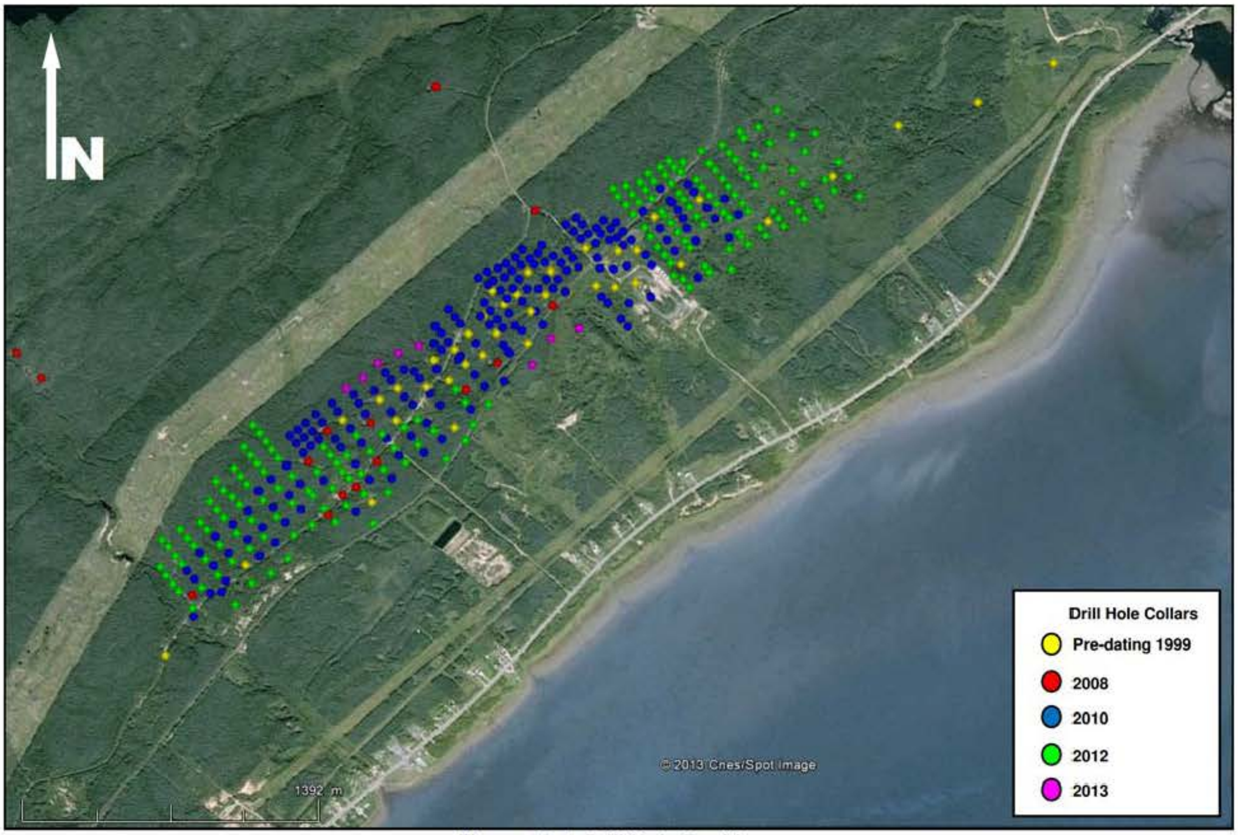


Figure 10-1: Drill Hole Positions

11. Sample preparation, Analyses and Security

The information contained in this section was derived and modified (June 2013 by SGS Canada Inc.) from the Feasibility Study (prepared by Roche-Ausenco) dated of February 2012.

Security was ensured through the restricting access to the drill core and ensuring that transport was done by SOQUEM/AXOR/InnoxExplo employees and its contractors. The nature and distribution of the mineralization, and the size of the mineralized body, are such that attempts at tampering, through the introduction of high-grade material, are not practical.

Over the years the sample selection, preparation and shipping was handled by different parties. From 1995 to 2010, SOQUEM operated the exploration program. In 2012, AXOR became operator and in 2013 InnovExplo conducted the sampling, preparation and shipping.

In 1995 and 1996, analysis of the core was done by Chimitec Ltée. (Chimitec) of Val d'Or, Québec. In 2008, samples were sent to ALS Chemex (ALS, formerly Chimitec) an ISO/IEC 17025:2005 accredited and ISO 9001:2008 certified laboratory. The sample preparation for the 2010 drilling programs was carried out by Table Jamésienne de Concertation Minière (TJCM) located in Chibougamau, Québec. In 2012 and 2013, samples were sent to ALS in Val d'Or, Québec Canada.

11.1 Sampling Method and Approach

Assay samples were determined by geologists and designated by marks on the core at the start and end of each interval. Samples started and terminated at major geological boundaries. The large scale of the deposit and the disseminated nature of the mineralization allow for sample intervals to be up to three meters in length with a minimum length of one meter.

Once delineated, core was halved longitudinally with a core saw by technical personnel. One half of the core sample was placed into a uniquely numbered sample bag with a matching tag. The remaining half-core was returned to the core box for reference and later sampling. The reference core boxes were sealed with a wooden cover, bound with bailing wire and archived. Presently archive core boxes are stored either in Chibougamau in SOQUEM core library or in Sept-Îles. All mineralized core from the deposit area was systematically sampled and assayed. In some homogeneous units, where the logging geologist has judged the grade to be low, only representative intervals of core were selectively sampled (Genivar, 2008). The drill holes completed in the 2010, 2012 and 2013 drilling programs were sampled in their entirety.

Core recovery on the Project, based on RQD measurements, is greater than 90%. Most drill holes were oriented roughly perpendicular to the layering of the mineralized body so intersections are representative of the true width of the deposit (Genivar, 2008).

11.1.1 Sample Preparation and Analysis (1995 to 1996)

Core samples, typically six to twelve kilograms in weight, were received and crushed in their entirety. This crushed material was sub-sampled (one kilogram) and the split was pulverized to -200 mesh and a second sub-sampled taken for analysis. Pulp and reject duplicates were archived by SOQUEM in Québec City, Québec. The main minerals of economic interest were apatite, Ilmenite and magnetite. In the summer of 2013, all the core and rejects were transported from Québec to Sept-Iles.

Initially, assaying was done by gravimetric methods for apatite and later analyzed for other minerals, including apatite, using Inductively Coupled Plasma (ICP) methods. This allowed for the comparison of the apatite results as a supplemental check to the laboratory's internal Quality Assurance/Quality Control (QA/QC) protocols. Genivar plotted the results of 291 pairs from the 1995 drill program and 296 pairs from the 1996 work, and found good agreement between the two assaying methods for both programs.

Chimitec's internal laboratory QA/QC program called for the analysis of a Certified Reference Mateiral (CRM) and the duplicate analysis of a number of samples. Approximately 20% of the samples submitted in 2005 were reassayed using the same gravimetric method on a second split from the original pulp. In 1996, this number declined to 10% of the samples. Genivar plotted 254 assay pairs and the results showed good correlation. For other elements, approximately 4% of samples were repeated using the whole rock ICP method.

In 1996, Chimitec modified its analytical routine and standards for ICP whole-rock analysis which resulted in significant differences for some elements, such as sodium, but not the minerals of interest, i.e., apatite, Ilmenite and magnetite.

11.1.2 Sample Preparation and Analysis (2010)

For both phases of the program, samples were prepared by Table Jamésienne de Concertation Minière (TJCM) and shipped to ALS Minerals (ALS) in Vancouver. To increase productivity some samples were prepared by ALS group laboratories in Sudbury, Val d'Or or Thunder Bay. Sample preparation protocols followed those of previous campaigns.

A total of 8,857 samples were analyzed for major compounds (SiO_2, Al_2O_3, Fe_2O_3, CaO, Na_2O, K_2O, Cr_2O_3, TiO_2, MnO, P_2O_5, SrO, and BaO) and ten elements including (Au, Pt, Pd, Cu, Ni and S) by X-Ray Fluorescence Spectrometry (XRF). In addition, 608 samples were sent to Australia to be analyzed, by XRF, for chlorine.

For XRF analysis, a 0.66 g of pulverized sample is heated with a mixture of 12:22 lithium metaborate to lithium tetraborate which includes an oxidizing agent (lithium nitrate). Once melted, the mixture is poured into a mold place and allowed to cool. The resulting disc is then analyzed using XRF. A typical analysis of iron ore includes a determination of loss of ignition (LOI) at 1,000°C usually measured with a thermo-gravimetric analyzer or, alternatively, measured manually (SOQUEM, 2011).

An additional 572 samples were analyzed as part of SOQUEM's QA/QC program. A total of 191 blanks (one result was discarded), 188 CRM and 192 duplicates comprised the control samples that were introduced into the sample stream.

11.1.3 Sample Preparation and Analysis (2012-2013)

Due to time limitations, samples were prepared in two different locations. Before March 29, 2012, the samples were prepared (dried, crushed, pulverized and riffle split) by the TJCM in Chibougamau, Québec. QAQC samples were inserted in the sample sequence by the TJCM and shipped for analysis to ALS. After March 29, 2012, QAQC material was inserted and full half core were sent to ALS for preparation and analysis.

Analysis at ALS was done using the ME-XRF 06 protocol which reports the following elements: Si, Ti, Al, Fe, Mg, Ca, Mn, Na, K, P, Cr, Ba and Sr. Chlorine was analyzed using the ME-XRF 21u protocol.

11.2 Magnetic Susceptibility and Density Measurements

From the Feasibility Study (prepared by Roche-Ausenco) dated of February 2012.

A MPP-EM2S probe was used during phase one of the 2010 drill program (Holes 1166-10-61 to 1166-10-98) to test magnetic susceptibility of rock in each hole at three meter intervals.

Density measurements for 381 samples were conducted by TJCM between 2010 and 2012 using the following procedures. Samples are received and logged in with date, time, number of samples and the name of the receiving technician. Individual samples were dried or, for concurrent samples, clearly marked samples within core boxes are measured sequentially. The scale used for the measurements was a Mettler Toledo, SB 16001 that was turned on five minutes before the first measurement to ensure its stability. The scale was calibrated daily, and as needed during the shift, using a 4,000 g certified weight. To validate each series of measurements a standard (quartz crystal) was inserted every 20 samples on the sample number ending in 20. A duplicate was also inserted every 20 samples at sample intervals ending in 10, 30, 50 etc. The water bath was changed daily and its temperature was noted regularly to ensure consistency. Samples were cleaned, as required, then its weighed in air was recorded. The sample was then placed within a steel basket that was suspended below the scale and completely submerged in water. The net weight (weight of the sample and basket in water minus the weight of the basket in water) was recorded. A calculation is made to give the relative density of the sample using the following formula:

$$Relative\ Density = \frac{Weight\ of\ sample\ in\ air}{Weight\ of\ sample\ in\ air - Net\ weight\ of\ sample\ in\ water}$$

In 2012, 414 density measurements were also done by AXOR using the CÉGEP de Sept-Îles laboratory and a similar procedure.

11.3 Quality Assurance and Quality Control Review

An independent QA/QC program was not initially in place at Mine Arnaud. In an effort to address this issue SOQUEM established a program to produce independent Certified Reference Material (CRM), from existing reference core, for use in the 2010 and subsequent drill campaigns. Details of the program and results are given below. For results that predate the production of the independent CRMs, RPA (2012) has examined the internal laboratory QA/QC data from ALS. In addition to the minerals of economic interest, (i.e., apatite, ilmenite and magnetite), internal laboratory QA/QC was also conducted for chlorine since this element can potentially impact the marketability of the phosphate product. The results from these analyses are presented separately.

11.3.1 ALS Internal Duplicates and Certified Reference Materials

From the Feasibility Study (prepared by Roche-Ausenco) dated of February 2012.

An independent QA/QC program, consisting of the routine introduction of independent CRMs and certified blanks into the sample stream and second laboratory assaying of duplicate samples, was not available for review. In 2012, RPA has used, instead, the internal QA/QC data provided by ALS for the materials of economic interest (apatite, ilmenite, and magnetite) to assess the appropriateness of the data for use in an estimation of Mineral Resources.

As part of the ALS protocols for XRF analysis, two internal CRMs, one blank and one pulp duplicate are inserted for every batch of 39 samples thus accounting for 10.3% of the overall assay results. RPA has already reviewed these data and results are discussed by drill campaign.

11.3.1.1 2008 Drill Program

From the Feasibility Study (prepared by Roche-Ausenco) dated of February 2012.

A total of 26 duplicates were inserted as part of the 2008 drill program. RPA plotted the original assays against the duplicate assays, for the minerals of economic interest, on scatter plots and examined the results. For apatite, ilmenite, and magnetite the correlation between originals and duplicates was good. RPA also plotted these results on relative difference (Thompson-Howarth) charts and inspected them for any indication of bias. No bias was observed in the data.

A total of 32 blanks were introduced into the sample stream in 2008 with the results for ilmenite, apatite and magnetite all returning values below the laboratory detection limit indicating no cross-contamination between samples.

Five internal CRMs were used by ALS in 2008 as a check for precision and accuracy of results. Of these, only one, STSD-4, was used in sufficient quantity to yield statistically significant results. RPA noted that this CRM was not certified but provisional. RPA compiled and plotted the results for apatite, ilmenite and magnetite from 54 analyses. No variability data was given for accuracy assessment so a factor of ± 10% of the best value (BV), provided by the manufacturer of the CRM, was used as the threshold limits (TLs).

For apatite, three failures were encountered where results exceed TL for both precision and accuracy. RPA noted that mean of the 54 determinations exceed the BV for the (provisional) CRM, by approximately 20%, indicating a potential high bias for the apatite assays.

For ilmenite, all results plotted within a very narrow range and all were within the TLs. The mean of the samples were, however, lower than the BV for ilmenite by a minor amount. RPA did not consider this difference to be significant.

For magnetite, the assay results also plotted within a narrow range within the TLs. The mean of the 54 assays was also slightly above the BV but, as with ilmenite, RPA did not consider the difference to be significant.

11.3.1.2 **2010 Drill Program**

From the Feasibility Study (prepared by Roche-Ausenco) dated of February 2012.

For the 2010 drill program a total of 298 lab duplicate samples were taken. As with the 2008 data, RPA rewieved the results for the three economic minerals on scatter and Thompson-Howarth plots. The scatter plots indicated that pulp reproducibility is very good for all three minerals and the relative difference plots do not indicate any bias in the results.

A total of 302 blanks were plotted and assessed for indication of cross-contamination during sample preparation. A result that exceeded three times the detection limit (3DL) for the mineral being analyzed is considered a failure. For the three principal minerals, all results were below the 3DL threshold but it is noted that 78 results did exceed the detection limit for apatite (0.001% $_{P2O5}$).

Two CRM were used by ALS for the 2010 program, SCH-1 and STSD-4. Results for each CRM were plotted on graphs in general chronological order and inspected to confirm if precision was within an acceptable range (precision charts) for the three minerals of economic interest. For the precision charts, a failure was considered to be one assay value greater than three standard deviations (3SD) from the assayed mean or two consecutive results greater than two standard deviations (2SD) from the assayed mean.

For SCH-1 a total of 261 results were plotted for apatite, ilmenite, and magnetite and, overall, result were good with only minor failures noted. One result exceeded 3SD for apatite and eleven sets of consecutive samples, 31 samples in total or 11.9%, exceeded 2SD for ilmenite. It is noted that all failures for ilmenite were above 2SD from the assayed mean but, in real terms, the amount of the exceedance was small (0.003% TiO_2).

A total of 293 results were also plotted for STSD-4 and evaluated for precision. Minor failures were noted, with eight samples plotting below 3SD for apatite, two results returning greater than 3SD for magnetite, and two results above 3SD from the assayed mean for ilmenite.

For accuracy charts, a failure was considered to be one assay value greater than 3SD from the nominated BV or two consecutive results greater than 2SD from the BV. Results were given for the major element (cat ion) of the minerals so those values were converted by RPA to the oxide value for reporting consistency.

For SCH-1, accuracy results were good with no failures for apatite. Results for ilmenite showed 50 samples returning values greater than 3SD from the mean for a failure rate of 19.2%. The exceedance, as seen with the Precision Chart, was minor (0.003% TiO_2). For magnetite, the mean of the 261 results was below 3SD from the BV provided by the manufacturer indicating the iron content of the samples could potentially be underestimated. RPA has recommended that these data be reexamined by SOQUEM and samples be re-assayed where appropriate.

For CRM STSD-4, also used in the 2008 program, no variability data was given for accuracy assessment so a factor of ± 10% of the BV was used for the TLs. RPA again noted that this standard is provisional and is not certified. Assay results for apatite plotted on the upper TL indicate the modest potential for apatite grades to be overstated. RPA recommended that these results be reexamined by SOQUEM and, where appropriate, be re-assayed. Accuracy charts for ilmenite indicated that the assay results plot within an acceptable range but all results plot below the BV of the CRM and suggested the assay values for ilmenite have the potential to be understated. Results for magnetite are good with the majority of the results close to the BV.

RPA noted that no re-assaying was done based on QA/QC failures and recommends that CRM results be examined in a timely manner. When failures are identified, RPA recommended that failed samples, along with a reasonable number of "shoulder" samples, be re-analyzed. If the results continue to be outside acceptable tolerances, then the entire batch should be re-analyzed.

11.3.1.3 **2012 Drill Program**

Laboratory Duplicates

For the 2012 drill program a total of 218 lab duplicate samples were taken. At the time of this report access to 210 laboratory duplicates results are available. A total of 8 lab duplicates are from holes newer than 1166-13-189 which is the cut-off for SGS's validated database. Two (2) of the sample duplicates appeared to be composites of two samples and were removed from the population for simplicity. A total of 208 sample duplicates were utilized by SGS for statistical analysis. SGS plotted the data on scatter plots, Thompson-Howarth plots, and for selected components HARD plots. SGS primarily investigated correlation between P_2O_5, TiO_2, Fe_2O_3 the major minerals of interest and for SiO_2 the major rock forming mineral. For P2O5 (summary statisticsTable 11-1) there is less than 1% difference between the means of two populations with slightly higher average in the originals. The difference in the minimums is about 7% however it appears to be approaching the detection limit. The coefficient of correlation R = 0.9867 (and its square R2= 0.9735) indicating high degree of correlation. The t-Test and sign test neither indicate any bias. A histogram of the percent difference from the original sample has a normal distribution, indicating low bias. The paired plot indicates that the bulk of the data is within ±20% of the 1:1 line. There are 6 points that are obviously outside of the threshold limit, in fact there are more at the lower end but they are not particularly concerning. There are 5 samples that have failed criteria; they all have paired mean values less than 0.5.

Table 11-1: Summary Statistics P_2O_5 duplicates

	Summary Statistics (univariate)		
	Original P_2O_5	Duplicate P_2O_5	Relative Difference
Count	208	208	0.00%
Min	0.01	0.02	7.14%
Max	11.91	12.12	1.77%
μ	2.90	2.89	-0.61%
median	1.24	1.24	0.00%
skewness	1.17	1.14	-3.00%
σ	3.30	3.25	-1.63%
kurtosis	0.12	0.07	-39.50%
range	11.90	12.11	1.77%
variance	10.89	10.53	-3.24%

Bivariate Statistics	
$\Sigma(\mu_1 \cdot \mu_2)$	2187.31
count	208
Covariance	10.57
R	0.9867
R^2	0.9735

SIGN TEST	
Σ [+]	99
Min	0.43
Max	0.57
Result	**0.47**
Count	208

Table 11-2: Summary Statistics TiO2 duplicates

	Summary Statistics (univariate)		
	Original TiO_2	Duplicate TiO_2	Relative Difference
Count	208	208	0.00%
Min	0.95	0.95	0.00%
Max	26.26	26.66	1.52%
μ	8.03	8.02	-0.13%
median	5.73	5.82	1.48%
skewness	1.23	1.25	1.90%
σ	6.08	6.09	0.12%
kurtosis	0.56	0.68	22.09%
range	25.31	25.71	1.58%
variance	36.96	37.05	0.25%

Bivariate Statistics	
$\Sigma(\mu_1 \cdot \mu_2)$	7485.50
count	208
Covariance	36.16
R	0.9772
R^2	0.9548

SIGN TEST	
Σ [+]	102
Min	0.43
Max	0.57
Result	**0.49**
Count	208

Table 11-3: Summary Statistics TiO_2 duplicates

	IN	OUT	#IN	#OUT
±10%	84.6%	15.4%	176	32
±20%	92.8%	7.2%	193	15
±50%	99.0%	1.0%	206	2
±100%	99.0%	1.0%	206	2

$$\%RD = \frac{\text{Duplicate Value} - \text{Original Value}}{\text{Original Value}}$$

A histogram of the percent difference in TiO_2 from the original sample has a normal distribution, indicating low bias. The paired plot indicates that the bulk of the data is within ±20% of the 1:1 line. From the paired plot, there are 3 points that are obviously outside of the threshold limit, in fact there are 15 samples total (Table 11-3) outside the threshold limit, most of which are less than 10% TiO_2.

For Fe_2O_3 (summary statistics Table 11-4) there is a less than 1% difference between the means of two populations, with a slightly lower average in the original values. The minimum and maximum values have reasonable absolute and relative differences, with both relative differences being less than 2%. The coefficient of correlation R = 0.9617 (and its square R^2= 0.9249) indicates high degree of correlation. The t-Test and sign test neither indicate any bias. The sign test indicates that approximately 53% of the duplicates are larger than original.

Table 11-4: Summary Statistics Fe_2O_3 duplicates

	Summary Statistics (univariate)		
	Original orig	Duplicate dup	Relative Difference
Count	208	208	0.00%
Min	5.42	5.33	-1.66%
Max	68.11	68.00	-0.16%
µ	28.89	29.08	0.64%
median	24.69	24.57	-0.51%
skewness	1.11	1.14	2.53%
σ	13.99	14.27	1.95%
kurtosis	0.63	0.68	7.82%
range	62.69	62.67	-0.03%
variance	195.80	203.53	3.95%

Bivariate Statistics	
$\Sigma(\mu_1 \cdot \mu_2)$	39742.15
count	208
Covariance	191.99
R	0.9617
R^2	0.9249

SIGN TEST	
Σ [+]	110
Min	0.43
Max	0.57
Result	**0.53**
Count	208

A histogram of the percent difference from the original sample has a normal distribution with a high centre, indicating low bias. The paired plot indicates that the bulk of the data is within ±20% of the 1:1 line. From the paired plot there are 3 points all along original ≈ 20% are obviously outside of the threshold limit. Table 11-5 details the distribution of duplicates; only 5 samples are outside the ±20% lines out of 208 samples for Fe_2O_3.

Table 11-5: Sample proportions within relative ranks – Fe_2O_3

	IN	OUT	#IN	#OUT
±10%	92.3%	7.7%	192	16
±20%	97.6%	2.4%	203	5
±50%	99.0%	1.0%	206	2
±100%	99.5%	0.5%	207	1

$$\%RD = \frac{\text{Duplicate Value-Original Value}}{\text{Original Value}}$$

For SiO_2 (summary statistics) there is a less than 0.1% difference between the means of two populations. There is a large difference in the minimum with a -17.7% difference from the original. This is surprising with the minimum content being above 1% silica. There are similar maximums. The coefficient of correlation R = 0.9707 (and its square R^2= 0.9423) indicate high degree of correlation. The t-Test and sign test neither indicate any bias. The sign test indicates that approximately 52% of the duplicates are larger than original.

Table 11-6: Summary Statistics SiO_2 duplicates

	Summary Statistics (univariate)		
	Original orig	Duplicate dup	Relative Difference
Count	208	208	0.00%
Min	3.06	2.52	-17.65%
Max	55.08	54.69	-0.71%
μ	31.51	31.54	0.08%
median	36.54	36.03	-1.40%
skewness	-0.71	-0.72	0.91%
σ	13.90	13.88	-0.12%
kurtosis	-0.79	-0.76	-3.42%
range	52.02	52.17	0.29%
variance	193.16	192.69	-0.25%

Bivariate Statistics	
$\Sigma(\mu_1 \cdot \mu_2)$	38766.91
count	208
Covariance	187.28
R	0.9707
R^2	0.9423

SIGN TEST	
Σ [+]	108
Min	0.43
Max	0.57
Result	**0.52**
Count	208

A histogram of the percent difference from the original sample has a normal distribution with a high centre, indicating low bias. The paired plot indicates that the bulk of the data is within ±20% of the 1:1 line. From the paired plot there are 3 points that are obviously outside the threshold and 2 further points just outside the lines. Table 11-7 details the distribution of duplicates; a total of 15 samples are outside the ±20% lines out of 208 samples for SiO_2. Of the 15 outliers 9 (60%) are in the 20-30% bracket, or near the ±20% threshold.

Table 11-7: Sample proportions within relative ranks – SiO_2

	IN	OUT	#IN	#OUT
±10%	86.1%	13.9%	179	29
±20%	92.8%	7.2%	193	15
±50%	99.0%	1.0%	206	2
±100%	99.5%	0.5%	207	1

$$\%RD = \frac{\text{Duplicate Value} - \text{Original Value}}{\text{Original Value}}$$

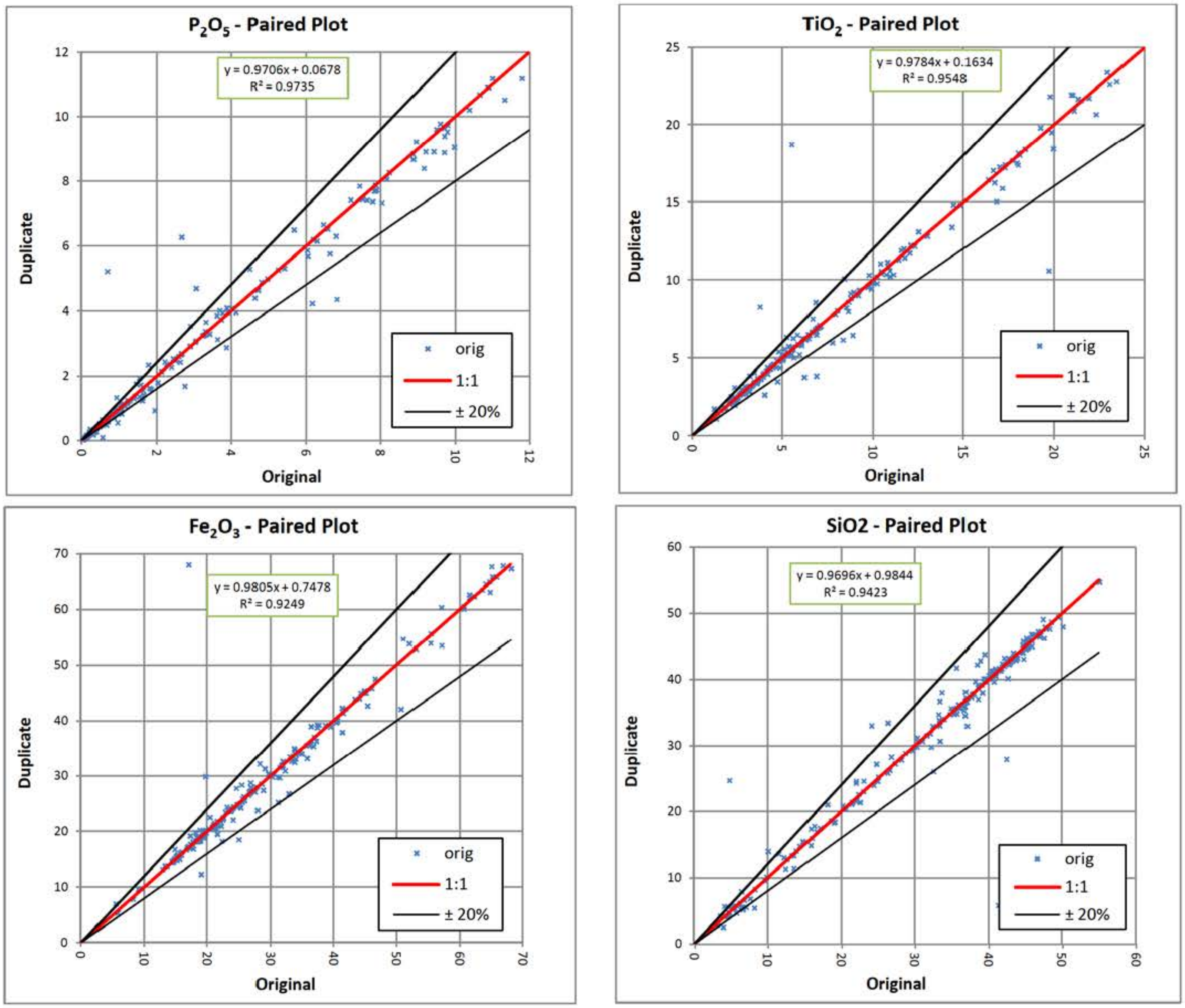


Figure 11-1: Scatterplots for Duplicate analyses

Laboratory Standards

Before assessing the standards we made a graphic representation (Figure 11-2) to check potential mislabels. The graph has 242 plotted points in total from 3 different standards. It appears that there could be 5 potential samples not belonging to a particular population. 3 of which appear to fit nicely into another standard population, there are potential candidates for mislabels. Arrows have been drawn in Figure 11-2 to highlight potential errors.

It appears that samples 2318, 2616 and 2158 could be mislabeled standards; samples 5257 and 1659 have another error, potentially sample sequence, and should be investigated. Due to the small number of errors, SGS consideres that the data is adequate for use in the resources estimation phase of the work.

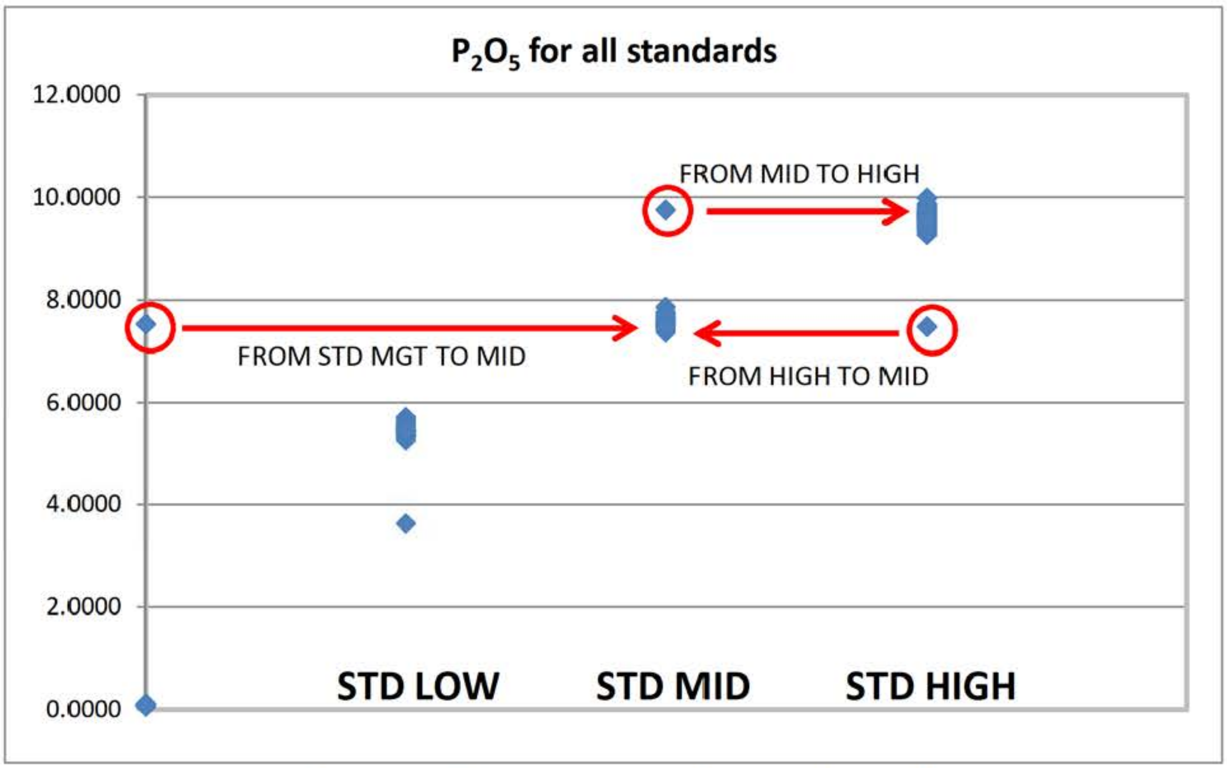


Figure 11-2: Comparison of P2O5 values grouped by standard type – All Standards

11.3.2 Quality Assurance/Quality Control for Chlorine

11.3.2.1 Chlorine Duplicates

From the Feasibility Study (prepared by Roche-Ausenco) dated of February 2012.

RPA analyzed duplicates for chlorine assay results from 323 samples. These samples were taken from the same pulp duplicates at ALS and the results were received by Mine Arnaud between September, 2010 and March, 2011. RPA plotted these duplicate results and found very good correlation between the original and the duplicate assays as shown in Figure 11-3. RPA noted that check assay duplicates were not done at an independent laboratory and recommends that this practice be adopted in the future, which was done in 2012-2013. The duplicate results were also plotted on Thompson-Howarth plots and examined for evidence of bias. RPA found no evidence of bias for the chlorine assays.

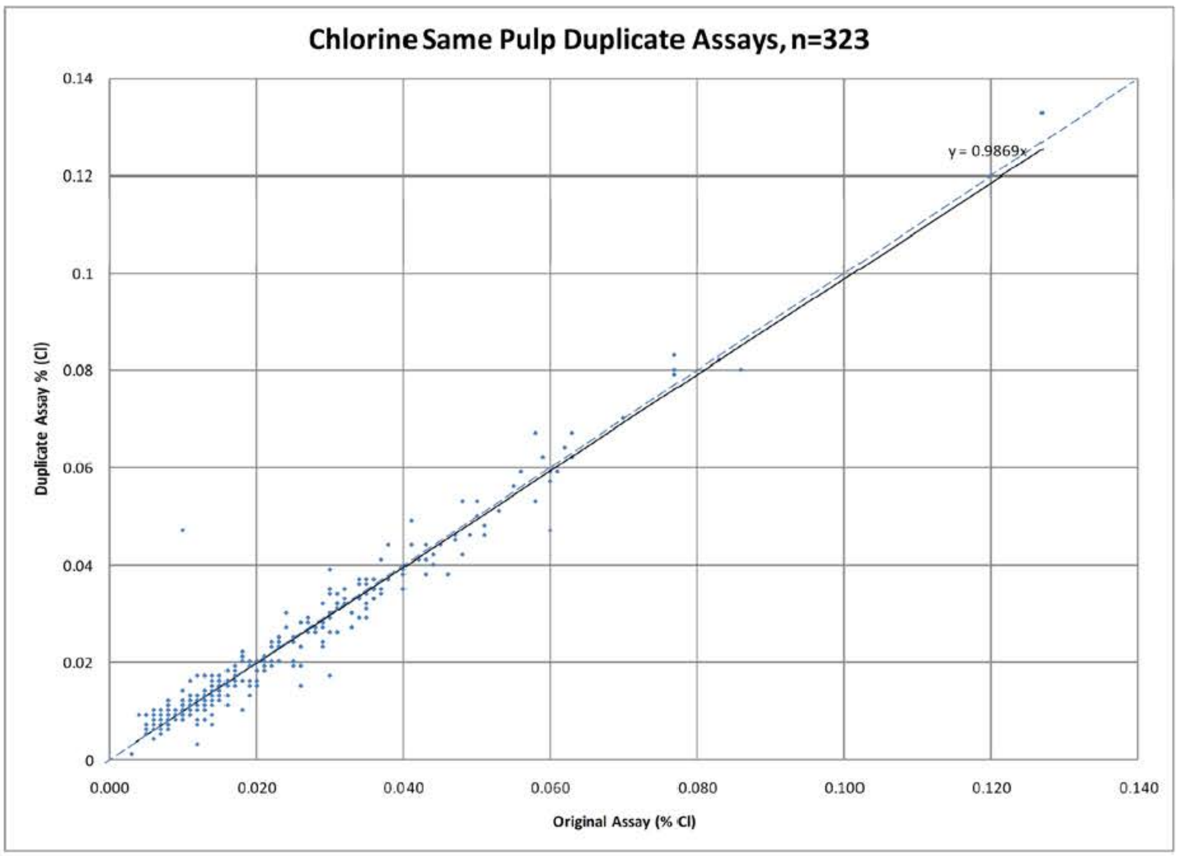


Figure 11-3: Results of Same Pulp Duplicates for Chlorine at ALS

Chlorine Certified Reference Materials

From the Feasibility Study (prepared by Roche-Ausenco) dated of February 2012.

No external CRM were introduced into the sample stream to independently evaluate the chlorine results. ALS, however, used four commercial CRMs (SARM-3, SARM-11, SARM-39 and SARM-45) for internally evaluating the performance of the assay results. The four CRM are products of the South African Reference Materials (SARM) program but only SARM-3 lists a certified value for chlorine analysis and its certificate does not include confidence limits or the standard deviation values for the assays used to establish the CRM. Threshold limits of ± 10% of the BV was assigned by RPA and used to assess accuracy for SARM-3 and only precision charts were plotted for the other CRM.

RPA plotted the assays in general chronological order, and inspected the results to confirm if precision was within an acceptable range. The failure criteria were the same as those described above, i.e., two consecutive samples outside of 2SD from the assayed mean or one sample outside of 3SD from the assayed mean.

For all CRM the results were found to be acceptable with no systematic bias observed. Graphical results from SARM-3 showed only one failure where one determination plotted below the acceptable TL out of 32 assays and the certified mean (0.12% Cl) was identical to the precision mean calculated from the data. RPA noted, however, that the sample result that failed on the precision chart plotted within the acceptable range for the accuracy chart. All results plotted within an acceptable range for SARM-11 and SARM-39. RPA noted that SARM-45 displayed the most scatter but only one result plotted above the upper TL and three plotted below the lower TL out of 197 assays.

Based on RPA's opinion the chlorine assay results from the internal laboratory CRM employed by ALS show no sign of systematic bias and results plotted within acceptable ranges.

Chlorine Blanks

From the Feasibility Study (prepared by Roche-Ausenco) dated of February 2012.

RPA noted that no external blank samples were inserted into the sample stream to independently confirm that cross-contamination of samples did not occur but ALS, as part of its QA/QC protocols, did insert its own blanks. RPA plotted the results of these internal checks. The 231 results show no systematic contamination.

Seventeen of the 231 blanks, or 7.4%, of the samples exceeded three times the DL. RPA notes this number is quite high and recommended that a program of external blank insertion be adopted.

11.3.3 SOQUEM Internal Reference Material (2010 Drilling)

From the Feasibility Study (prepared by Roche-Ausenco) dated of February 2012.

In 2009, as part of its QA/QC protocol, SOQUEM established a program to produce Internal Reference Materials (IRMs) that represent a range of grades. The IRMs were produced from non-weathered reference drill core that had been previously split and analysed. Specific intervals were chosen based on previous apatite, Ilmenite and magnetite results. The halved core was halved again with one half (quarter core) retained for reference and the remaining half sent to the laboratory, TJCM in Chibougamau, Québec, for compositing to achieve the target grades.

Once handling loss was considered, the amount of material needed was calculated and mixing proportions were determined to cover the range of concentrations required for each IRM. More material was collected than necessary to ensure sufficient availability.

Ideally, the sample chosen had assayed values similar to the target specification of the IRM composite. A nominal maximum value of 10% to 11% P_2O_5 was set along with a nominal minimum value of 3% to 4% TiO_2 for samples comprising the Nelsonite composite. The high grade (Nelsonite) and the low grade (gabbro) were subject to the same preparation protocol. Each were crushed and pulverized to established specifications, as detailed previously. This process yielded approximately 12 kg of Nelsonite and five kilograms of gabbro respectively. The samples used, and their respective grades, are tabulated in Table 11-8.

Table 11-8: Samples Comprising IRM Composite

Mine Arnaud Inc. – Arnaud Mine Project						
Drill Hole	**Length (m)**	**Unit**	**Sample No.**	**P_2O_5 (%)**	**TiO_2 (%)**	**Fe_2O_3 (%)**
1166-08-46	3	Gabbro	503716	0.069	3.41	19.75
1166-08-47	3	Gabbro	503789	0.056	3.91	19.28
Total	6			0.0625	3.66	19.515

Mine Arnaud Inc. – Arnaud Mine Project						
Drill Hole	**Length (m)**	**Unit**	**Sample No.**	**P_2O_5 (%)**	**TiO_2 (%)**	**Fe_2O_3 (%)**
1166-08-47	2.9	Nelsonite	503776	11.377	19.13	44.01
1166-08-48	1.4	Nelsonite	503848	10.591	18.88	43.77
1166-08-49	0.6	Nelsonite	503927	10.102	18.77	42.88
1166-08-49	1.4	Nelsonite	503928	10.765	19.35	45.73
1166-08-53	1.7	Nelsonite	804540	10.857	18.78	42.71
1166-08-55	2.1	Nelsonite	804632	10.498	18.98	45.04
Total	10.1			10.837	19.01	44.14

Weighted averages were calculated for the Nelsonite and gabbro pulp composites. Composite grades were used, in turn, to calculate the proportions of each required to achieve the grade specifications for each of the proposed IRM as shown in Table 11-9.

Homogenized, pulverized gabbro was spread out upon a thoroughly cleaned flat work surface. The gabbro pulp was then divided and measured into trays, by weight, based on calculated proportions for each IRM. The trays were designated (i.e., NEL-50, NEL-60, etc.). The Nelsonite pulp was homogenized and divided by the same methods. Nelsonite pulp was added to the specified trays of gabbro pulp in the calculated proportions necessary to achieve the IRM target grades. The mixture from each tray was then placed on glossy paper, homogenized again, and then split into 50 g sub-samples and packaged. Three packets of NEL-100, ten packets of NEL-0, and 50 packets of NEL-50 to NEL-95 comprise the 313 IRMs that were produced by this process.

Table 11-9: Internal Reference Materials and Composite Proportions

Mine Arnaud Inc. – Arnaud Mine Project				
IRM	**Proportion NEL (%)**	**Mass NEL (g)**	**Mass Gabbro (g)**	**No. of CRM**
NEL-100	100	150	0	3
NEL-95	95	2,375	125	50
NEL-90	90	2,250	250	50
NEL-80	80	2,000	500	50
NEL-70	70	1,750	750	50
NEL-60	60	1,500	1,000	50
NEL-50	50	1,250	1,250	50
NEL-0	0	0	500	10
TOTAL		11,275	4,375	313
Estimated		12,000	5,000	

One sample from each of the IRM were sent to three independent ISO 9001 certificate and ISO/IEC 17025 accredited, laboratories and analysed using XRF. The three laboratories were ALS, SGS Mineral Services, Lakefield (SGS) and COREM. Care was taken to choose laboratories whose detection limits, equipment sizing and minimum sample size requirements are compatible with the proposed IRM specifications. The results of these analyses, along with the weighted average calculations from the original samples and the assayed mean from the three laboratories, are presented in Table 11-11 to Table 11-12.

Table 11-10: XRF P2O5 Results For SOQUEM IRM from Nominated Laboratories

IRM	Calculated Grade (% P_2O_5)	ALS (% P_2O_5)	SGS (% P_2O_5)	COREM (% P_2O_5)	Assayed Mean (% P_2O_5)
NEL-100	10.837	10.703	11.000	11.000	10.901
NEL-95	10.299	10.295	10.400	10.400	10.365
NEL-90	9.760	9.846	9.740	9.800	9.795
NEL-80	8.682	8.700	8.750	8.740	8.730
NEL-70	7.605	7.555	7.690	7.600	7.615
NEL-60	6.527	6.383	6.730	6.510	6.541
NEL-50	5.450	5.316	5.480	5.530	5.442
NEL-0	0.063	0.064	0.060	0.070	0.065

Table 11-11: XRF TiO2 Results for SOQUEM IRM from Nominated Laboratories

IRM	Calculated Grade (% TiO_2)	ALS (% TiO_2)	SGS (% TiO_2)	COREM (% TiO_2)	Assayed Mean (% TiO_2)
NEL-100	19.014	18.560	18.300	18.300	18.387
NEL-95	18.247	18.020	17.400	17.600	17.673
NEL-90	17.479	17.400	16.700	16.700	16.933
NEL-80	15.944	15.840	15.200	15.200	15.413
NEL-70	14.408	14.240	13.800	13.700	13.913
NEL-60	12.873	12.630	12.400	12.300	12.443
NEL-50	11.337	11.160	10.800	10.900	10.953
NEL-0	3.660	3.640	3.550	3.590	3.593

Table 11-12: Fe2O3 Results for SOQUEM IRM from Nominated Laboratories

IRM	Calculated Grade (% Fe_2O_3)	ALS (% Fe_2O_3)	SGS (% Fe_2O_3)	COREM (% Fe_2O_3)	Assayed Mean (% Fe_2O_3)
NEL-100	44.143	43.800	44.900	45.400	44.700
NEL-95	42.912	42.980	43.500	43.800	43.427
NEL-90	41.681	41.500	42.400	42.300	42.067
NEL-80	39.218	39.020	39.800	39.700	39.507
NEL-70	36.755	36.890	37.400	37.200	37.163
NEL-60	34.292	34.210	35.200	35.000	34.803
NEL-50	31.829	31.770	32.500	32.600	32.290
NEL-0	19.515	19.620	20.200	20.300	20.040

SOQUEM plotted the results from each laboratory against the assayed mean and made the following observations:

- The results from the three laboratories showed good correlation for apatite with ALS showing the most dispersion, albeit very low;
- For ilmenite, the results from ALS were 3% higher than those from SGS and COREM but the dispersion was minimal;
- For magnetite, ALS returned values 2% lower than SGS and COREM but, again, the dispersion was low.

SOQUEM concluded that the internal IRM were acceptable for apatite analysis by XRF, but recommended that these IRM only be used for ilmenite and iron oxide analysis in situations where the relative uncertainties (i.e., 3% and 2%, respectively) have negligible consequence. In the case of Sept-Îles, further testing of ilmenite concentrations was recommended. SOQUEM also concluded that it was not possible, given the limited number of samples, to assess if one laboratory performed better or worse than the others.

11.3.4 **2010 Independent Quality Assurance/Quality Control Results**

From the Feasibility Study (prepared by Roche-Ausenco) dated of February 2012.

A total of 571 control samples comprising 191 blanks (after one sample was discarded), 188 IRM and 192 duplicates were introduced into the sample stream during the two phases that made up the 2010 drill campaign. Blank material was made up of commercially purchased (ornamental) dolomite that produced results less than 0.1% for both apatite and Ilmenite. One blank was inserted for every 50 samples. The IRMs were also inserted into the sample stream at a rate of one per 50 samples. Duplicates were made up of a second split of the reference drill core (quarter core) and were also inserted at a rate of one per 50 samples with care taken to ensure that non-consecutive sample numbers were used.

Assay results from the blank samples were plotted and inspected by RPA. One sample was discarded due to an obvious error, leaving 191 valid samples of which, a total of 63 exceeded 3DL for apatite, 62 exceeded 3DL for ilmenite, and 19 exceeded 3DL for magnetite. SGS agrees with RPA's opinion that considers this failure rate to be high but observed no systematic failure. RPA recommended that these failures be investigated and that the preparation laboratories be alerted to the possibility of cross-sample contamination.

The 192 field duplicate (quarter core) assay results were plotted on a scatter diagram against original determinations and found to have very good correlation for all three minerals of economic interest. These data were also plotted on Thompson-Howarth plots. The relative difference plot for apatite showed more scatter than those for Ilmenite and magnetite but the data, in the RPA's opinion, is free of bias.

RPA inspected the results from the respective IRMs. RPA notes that IRM NEL-0 and NEL-100 were not used during the program under review. Since there was not sufficient analysis of each of the IRM by the three laboratories to establish the variability for each, RPA assigned a value of 10% above and below the mean as the TLs for accuracy. RPA found that all of the 188 assays were within this range with the exception of one NEL 50 IRM that was below the lower TL for both magnetite and ilmenite.

To assess precision, the same failure criteria was used, i.e., one sample outside of 3SD from the assayed mean for all samples or two consecutive assays outside of 2SD from the mean. A total of eight of the 188 IRM assays, or 4.3%, plotted outside of these ranges. Only one IRM was above the upper threshold, NEL 50 for apatite, while the other seven failed determinations were below the lower TLs. There was no observed pattern and IRMs NEL 60, NEL 90 and NEL 95 were free of failures. Three individual assays plotted below lower TL for NEL 80 for apatite, Ilmenite, and magnetite samples while two, for Ilmenite and apatite, plotted below the lower TL for NEL 70. Two other samples failed from NEL 50 plotting below the lower TL for magnetite and Ilmenite.

In the opinion of RPA the majority of the IRMs performed reasonably well but RPA recommended additional round robin testing to be done to increase confidence in the BVs and help establish confidence limits.

11.3.5 2012 Independent Quality Assurance/Quality Control Results

During the 2012 drilling campaign, AXOR inserted 6 quality control samples every 100 samples, comprising 2 IRMs, 2 blanks and 2 core duplicates. A total of 608 samples are available for QA/QC analysis, with 1910 blanks, 220 IRMs and 198 duplicates.

AXOR inserted 4 different IRMS that were prepared by the TJCM with historical core from Mine Arnaud project. When comparing the IRMs results with the expected value and 2SD and 3SD, we can observe that the samples from the batches analyzed on April 12th 2012 and May 17th 2012 show a great variability with IRM values over and under 3SD value. No specific reason was found explaining the results, and the samples contained in these 2 batches were not re-analyzed.

Duplicates, corresponding to quarter cores, were plotted in scatter plot and show a R^2 = 0.9905 with no major discrepancies from the 1:1 ratio line. Duplicates have no significant errors or bias.

Blanks material was prepared with quartzite from the Charlevoix region but was not analyzed for P_2O_5, hence is not a certified blank. When plotted in chronological order, we can observed that 75% of the samples have a P_2O_5 value <0.05%. The remaining values show results between 0.05%P_2O_5 and 0.18%P_2O_5. The origin of this error cannot necessarily be attributed to contamination during preparation process since the blank may contain P_2O_5. The error noted for blanks is still too small to pose problems in the assay results for resources estimation purposes.

11.3.6 2013 Independent Quality Assurance/Quality Control Results

Information about the 2013 sampling program was not available at the moment of writing this report.

11.4 Summary and Conclusions

Internal QA/QC results from ALS indicate good correlation for same pulp duplicates for the three principal minerals of economic interest for the 2008, 2010 and 2012 drill programs. All values derived from the insertion of blanks into the sample stream by ALS were within acceptable ranges. Three assay values exceeded the upper TLs for both precision and accuracy for the only statistically significant internal CRM used for the 2008 drill program. Failures were also encountered in the 2010 and 2012 drill programs but these were not systematic. Most of these failures are not, in RPA's and SGS opinion, significant and will have no material impact on the estimation of Mineral Resources. The poor performance, with respect to precision, of the magnetite CRM, however, should be investigated with ALS. The introduction of the SOQUEM IRMs, and the performance of the assays with respect to the magnetite, lends greater confidence to the integrity of the magnetite results.

For the analysis of chlorine, RPA found good correlation between the original and duplicate assays that were done as part of the laboratory's internal QA/QC program. A total of 7.4% of the chlorine blanks introduced by ALS exceeded 3DL. In RPA's opinion and in agreement with SGS, external blanks should be inserted into the sample stream by Mine Arnaud and assay results monitored on a regular basis. In terms of the introduction of CRMs into the sample stream, all results were found to be acceptable for both precision (reporductibility of a result) and accuracy (most representative value).

The introduction of SOQUEM IRMs that were independent of the laboratory began in 2010. Based on its own analysis of the IRMs, SOQUEM concluded they were acceptable for apatite analysis by XRF but recommended caution when employing them for Ilmenite and magnetite. RPA and SGS has reviewed the results from SOQUEM's insertion of the IRM into the sample stream over both phases of the 2010 drill program and 2012 drill program and found that, overall, the results were acceptable but recommends further analyses to increase confidence in the nominated values and establish confidence limits. Duplicate results from field (quarter core) duplicate samples showed very good correlation between original and duplicate assays. The results from the insertion of blanks, however, showed a high failure rate but the failures appear random and not systematic.

RPA and SGS noted that no re-assaying was done based on QA/QC failures and recommends that IRM results be examined in a timely manner. When failures are identified, SGS recommends that failed samples, along with a reasonable number of "shoulder" samples, be reanalysed. If the results continue to be outside acceptable tolerances, then the entire batch should be reanalysed.

In SGS's opinion, the Sept-Îles Project will benefit from an independent QA/QC program but the analysis done on the historical and 2010 drill programs is sufficient to make the current data acceptable for the estimation of Mineral Resources.

12. Data Verification

A site visit was conducted by Claude Duplessis, Eng. and Jonathan Gagné, Eng., MBA at the Sept-Îles project location on October 3rd 2012. Jean-Philippe Paiement, M.Sc. P.Geo. and Floran Faiello visited the Chibougamau core storage location in November 2012. The site enabled the authors to visit the core storing facilities, the deposit area corresponding to the Sept-Îles deposit and get familiar with the region. During the site visit, the authors proceeded to take independent control samples of the pulp reject for chemical testing and coarse core reject composites for Metallurgical test work validation.

The data verification was done on four major points: 1) Validation of the database and relations between each table (collars, deviations, lithologies and assays); 2) Independent control sampling on pulp rejects for P_2O_5; 3) Independent Metallurgical test work on core coarse rejects composite and 4) Chlorine behavior from feed material to concentrate.

12.1 Database Validation

The database transferred to SGS Geostat for resources estimation purposes was created by Mines Arnaud using Geotic©. SGS Geostat proceeded to exported the database to Geobase© for validations and corrections. The database contains 430 holes, 14,252 survey measurements, 21,059 assays and 7,881 lithologies.

No major issues were found in the database during automated validation process conducted by SGS using Access and Genesis©. Deviation a 0m depth were also removed from the deviation table and transferred to collar orientation columns in the collar table, which is databse structure specification for Genesis©.

As part of data verification, RPA conducted spot checking of the drill hole database which still stands for SGS validation process. Approximately 10% of the drill holes that intersected the mineralized domain models at Mine Arnaud were selected for validation on a semi-random basis. In all, a total of 24 drill holes were selected for examination, 19 of which were of relatively recent vintage (completed in the 2008 and 2010 drilling programs). Due to the fact that SOQUEM utilize a fully integrated information management system, the locations of the drill hole collars provided in the drill logs of these 19 holes were compared against the original survey information. As well, the phosphate assays contained in the digital database supplied were compared against the information presented in the original laboratory certificates and was accomplished by accessing the original data directly in the ALS Chemex database.

The information contained in the paper drill logs for the remaining five drill holes (completed in 1995 and 1996) was compared against the information contained in the digital database. A number of variations were noted between the phosphate assays contained in the drill hole database and the paper logs. In general, the phosphate assays in the database were lower than those stated in the paper drill logs. Investigation with the SOQUEM team reveals that the source of the discrepancy is due to the fact that the phosphate values were determined by multiple assay methods during those two drilling campaigns. As such the digital drill hole database used a different set of assay values so as to ensure that the digital database used assays that were determined by the same analytical method.

In addition, it was noted that the collar coordinates differed between those presented in the paper drill logs and those contained in the digital database. It was determined that these differences were as a result of a change in the coordinate system from the Modified Transverse Mercator system originally used for the 1995 and 1996 drilling campaigns to the Universal Transverse Mercator system for all subsequent drilling programs.

A set of independent samples whose purpose would have been the independent determination of the presence of the material of interest (phosphate) was not taken during the site visits, as a number of independent tests have been conducted during the completion of metallurgical testwork during 2008 and more recently in 2010 and 2011. This, in conjunction with the physical observation of the presence of apatite in the rock and drill core, leads RPA to the conclusion that the presence of phosphate mineralization has been determined by independent means.

As a result of its data validation efforts, RPA believes that the drill hole data representing the phosphate mineralization intersected by drilling at the Mine Arnaud deposit is appropriate for use in the preparation of Mineral Resource estimates.

12.2 Independent Control Sampling

During the November 2012 site visit by Jean-Philippe Paiement, M.Sc. P.Geo and Floran Faiello, series of independent controls samples were selected on given holes. The pulps for the choosen samples were sent to Accurassay Laboratories in Thunder Bay, Ontario Canada. The samples were assayed using XRF and results for P_2O_5 were reported to SGS.

A total of 464 samples were analyzed and compared to original assay results. The sample pairs were plotted on a scattergram and the correlation between the two set returned a R2 = 0.9913 which shows a strong correlation between the original and duplicate values (Figure 12-1). A slight difference is observed in the mean between the two set of data, with 4.28% P_2O_5 for the originals and 4.29% P_2O_5 for the duplicates. Sign test shows a negligible but existing bias between original results and duplicates where 56% of the pairs have a lower original results compared to the duplicate. The percentage of difference between the two sample set is 1.3% (Figure 12-1) and is considered negligible, added to the fact that the original are 1.3% lower than the duplicates.

It is the opinion of SGS that the P_2O_5 assay values show a strong enough reliability to be used in the process of mineral resources estimation.

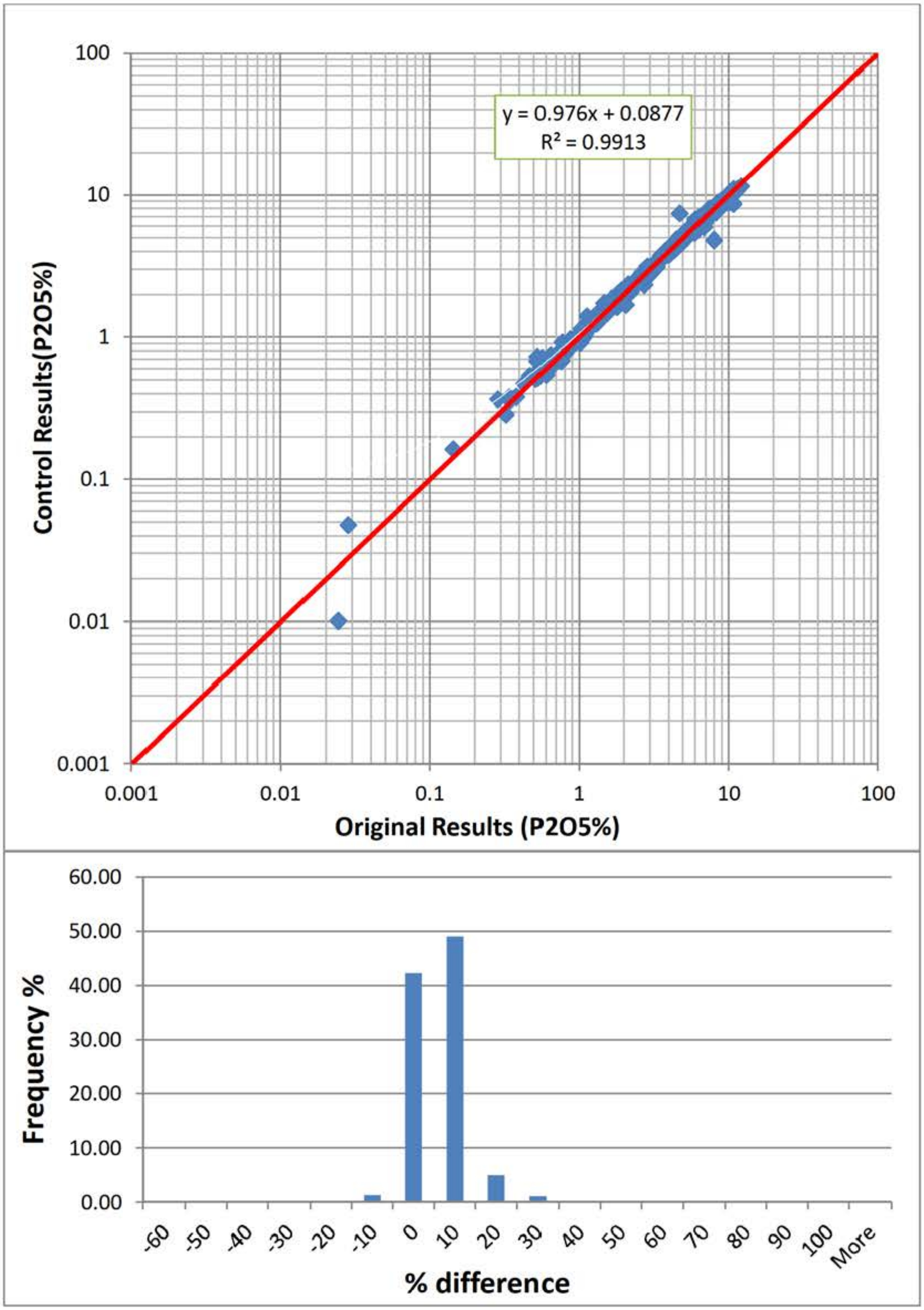


Figure 12-1: Statistical comparison between original assays and independent control samples

12.3 Independent Metallurgical Testing

During the November 2012 site visit by Jean-Philippe Paiement, M.Sc. P.Geo and Floran Faiello, series of independent metallurgical composites were taken from drill core sample rejects at the Chibougamau and Sept-Îles storage sites. A list of the composites is available in Table 12-1. Initially all composites were to be tested at SGS Canada Inc. Lakefield Laboratories (SGS Lakefield) in order to validate the metallurgical parameters. During the course of the project, the scope of these tests was changed and only three different composites were tested by SGS Lakefield.

Because SGS Lakefield failed to attain acceptable concentrate grades and recoveries, it was decided to do, under the supervision of SGS Geostat's metallurgist, four more lock cycle tests (LCT) at COREM and accept the results as an undisputable proof of the feasibility of the process. These tests are discussed in Section 13.4 of this report.

Table 12-1: List of composites used by SGS for Metallurgical testing

COMPOSITE #	Hole Name	From	To	Ave P_2O_5%
Composite 1	1166-10-120	2.65	21.87	4.21
Composite 2	1166-10-120	59.48	74.45	3.81
Composite 3	1166-10-120	79.79	92.10	3.66
Composite 4	1166-08-49	25.86	87.72	3.42
Composite 5	1166-08-49	120.06	142.11	5.23
Composite 6	1166-08-49	151.41	164.18	5.37
Composite 7	1166-10-195	7.12	40.20	5.72
Composite 8	1166-10-195	52.05	58.57	4.78
Composite 9	1166-10-195	65.06	72.57	8.88
Composite 10	1166-10-73	6.7	52.93	3.65
Composite 11	1166-10-73	58.00	80.14	6.11
Composite 12	1166-10-73	88.02	97.97	7.09
Composite 13	1166-10-161	12.24	82.21	5.18
Composite 14	1166-10-161	88.39	110.68	6.27
Composite 15	1166-10-161	120.14	124.37	7.12
Composite 16	1166-10-175	10.99	17.36	4.68
Composite 17	1166-10-175	24.56	65.83	5.81
Composite 18	1166-10-175	65.84	72.06	4.05
Composite 19	1166-12-16	11.47	34.23	6.21
Composite 20	1166-12-16	42.50	46.18	6.62
Composite 21	1166-12-16	49.39	51.90	6.73
Composite 22	1166-12-12	18.81	95.76	4.47
Composite 23	1166-12-12	103.23	135.23	5.94
Composite 24	1166-12-12	143.94	151.52	7.51
Composite 25	1166-12-108	33.77	86.06	3.72
Composite 26	1166-12-108	96.74	125.09	5.69
Composite 27	1166-12-108	125.09	133.73	5.74
Composite 28	1166-12-34	8.04	62.23	2.99
Composite 29	1166-12-34	95.71	113.02	3.74
Composite 30	1166-12-34	118.87	136.02	5.72
Composite 31	1166-12-132	19.55	78.43	3.49
Composite 32	1166-12-132	109.74	131.58	5.84
Composite 33	1166-12-132	139.75	153.32	2.82

12.4 Chlorine Behavior Study

Following Mine Arnaud's and Yara demands, SGS Geostat conducted work in order to establish a model for the prediction of chlorine in the concentrate. The models are based on geology, mineralogy, geochemistry and testwork form COREM (T-1224 RIDER1, T-1405 and T-1518).

The object of this particular study is part of a separate report and is included in the APPENDIX 3: CHLORINE REPORT of this present report.

In order to predict the chlorine behavior in the deposit and the chlorine content of the concentrate, three models were generated. The first two models are based on modal and statistical analysis and use the different mineralized zones to prediction the chlorine content. In these models, the chlorine content is derived from the predicted chloro-phlogopite and chloro-apatite content of the concentrate which uses the recovery model for each mineral. These two models are considered optimistic (low Cl values for the concentrate) regarding the chlorine content of concentrate. The third model uses a single statistical analysis correlating the K_2O content of the feed material and the Cl content of the concentrate produced during the metallurgical testwork (T-1224 and T-1518). This third model is considered pessimistic (higher Cl values of the concentrate) regarding the Cl content of the concentrate.

$$Cl\%\ concentrate = 0.2608 \times K_2O\%\ feed + 0.0503$$

The third model is used in this report in order to report the Cl content values for the concentrate. Following Yara's restriction, the Cl limit of the concentrate cannot be higher than 0.14% in order to be saleable. A letter from Yara stating the concentrate characteristics is found in APPENDIX 2.

13. Mineral Processing and Metallurgical Testing

13.1 Introduction[1]

Over the years, many metallurgical testworks and other mineralurgical studies were carried out on the Arnaud's apatite ore.

Mineralization of the Sept-Îles apatite, magnetite and ilmenite deposit was recognised early in 1992 by SOQUEM. Several metallurgical testworks were completed in the years 1992-2005, leading to a feasibility study in 2002 followed by an updated feasibility study in 2005. In 2007-2008, Mine Arnaud completed a work program leading to a scoping level study based on previous test works while incorporating some new concepts. In 2009, Mines Arnaud completed a corrosion study and metallurgical tests for the development of a new process flowsheet.

In the light of the increase of the demand for apatite associated with a substantial price increase, Mine Arnaud decided in early 2010 to re-actualize the Feasibility Study by conducting a new drilling campaign and additional metallurgical pilot plant testworks.

A more modern flowsheet for the recovery of apatite and ilmenite was developed by GTK in Finland in 2009. SGS Lakefield was directed to conduct a pilot plant test program to confirm the feasibility of producing a high-grade apatite concentrate and to provide engineering data for the processing facility design.

Preliminary testing at SGS was unable to replicate the process performance for recovering ilmenite and it was decided to conduct pilot plant testing for apatite beneficiation only. Apatite beneficiation was accomplished using mechanical flotation cells and a suite of reagents selected initially by GTK and partially adapted by SGS. Once the SGS test program was completed, it was decided that additional work needed to be performed in order to answer some remaining uncertainties, reduce the environmental impact related to the use of the selected suite of reagents, and reduce the operating cost. At the same time, the choice of using mechanical flotation cells was questioned considering that the trend in apatite beneficiation is based on the use of flotation columns. Consequently and since pilot plant runs are the only way to simulate a real commercial mill operation, COREM was directed to perform additional laboratory and pilot plant testwork to confirm the process flowsheet using column flotation cells.

For a good part, the results of this additional laboratory testwork (Corem' projects T1224 and T1242) served as the basis of the Roche-Ausenco Feasibility Study.

Apatite flotation in column cells was tested at COREM at the beginning of the fourth quarter of 2010. In the mean time, COREM introduced a suite of reagents they had developed during the 1997 initial feasibility study of this particular deposit. This suite of reagents is simpler and more environmentally friendly than the one previously used at SGS Lakefield. The pilot plant tests at COREM were conducted

[1] This introduction is for a great part an excerpt from the Roche-Ausenco Feasibility Study (February, 2012).

with two different blends of ore types. While most of the work was conducted with a blend of weathered ore (surface samples) consisting of 80% Upper (S3) and 20% Nelsonite (S2), some important work was also conducted on a blend of 20% Nelsonite (S2), 25% Upper (S3) and 55% of the Railroad (SRR) ore types. This later ore blend was considered, at the time, as the most representative for its proportion of the various ore types to be sent to the mill. The pilot plant test results showed that for these blends, a final concentrate grade of ≥39.0% P2O5 could be produced with a P2O5 recovery of over 88%, while meeting the specifications of Yara (Mine Arnaud's principal buyer for the apatite concentrate) for impurity content. Towards the end of the Study, the mine planning showed that the proportion of the various ore types would be different that the ones used during the pilot plant testing and the proportion of the various ore types will vary during the course of the mine life. Nevertheless, in its report Roche-Ausenco considered that these variations should not have dramatic impacts on the Project. Roche-Ausenco also recommended that it would be preferable to perform some additional laboratory work prior to the exploitation of the mine.

Table 13-1: Comparative results obtained at the pilot Plant for the first and second phase (2011)

Pilot Plant	Sample*	Raw Feed	Final Concentrate			
Date - Hour	Type	P2O5 %	Wt Rec%	P2O5 %	P2O5 Rec %	
March 4 – 13h45	S2	4.85	12.28	36.87	93.38	
March 4 – 14h30	S2	4.79	12.29	36.16	92.75	
March 7 – 11h15	S2	4.03	8.68	37.85	81.24	
March 7 - 16h00	SRR/S2/S3	7.04	18.49	40.14	93.14	Different from R-A[2]
March 8	SRR/S2/S3	7.43	17.03	37.59	86.62	Day average
March 9	SRR/S2/S3	7.66	7.91	41.24	42.61	Day average
March 10	SRR/S2/S3	7.80	6.93	41.12	36.51	Day average
March 11	SRR/S2/S3	7.70	6.47	41.22	34.60	Day average
March 14	SRR/S2/S3	7.74	11.53	38.59	62.25	Day average
April 29 – 13h30	S2/S3	5.95	10.97	41.01	75.61	
April 29 – 15h15	S2/S3	6.24	11.03	40.94	72.34	
May 3 – 14h20	S2/S3	5.89	12.92	40.50	88.83	
May 3 – 15h30	S2/S3	5.92	13.05	40.69	89.70	
May 3 – 16h30	S2/S3	6.05	13.12	40.40	87.64	
May 5 – 14h45	S2/S3	5.97	13.16	40.75	89.84	
May 5 – 16h20	S2/S3	6.12	13.15	40,12	85.30	
May 6 - 11h05	S2/S3	5.76	11.75	41.78	85.30	Different from R-A
May 6 – 12h50	S2/S3	5.67	11.90	39.90	83.73	
May 6 – 14h15	S2/S3	5.74	12.44	39.74	86.09	
May 12 - 10h30	S2/S3	6.12	13.40	40.59	88.90	Different from R-A
May 12 - 13h00	S2/S3	6.05	13.33	39.99	88.14	
May 12 – 17h30	S2/S3	6.32	12.96	40.96	84.03	

* S3 = Upper

S2 = Nelsonite

SRR = Rail Road

S2/S3 = 20% Nelsonite, 80% Upper

SRR/S2/S3 = 55% Rail Road, 25% Upper, 20% Nelsonite

[2] Data taken directly from Corem's report (T1242) – do not match the Roche-Ausenco's Feasibility Study since Roche Ausenco had only preliminary results.

Even if Roche-Ausenco concluded that for the blends tested, a final concentrate grade of ≥39.0% P_2O_5 along with a P_2O_5 recovery of over 88% could be produced while meeting the specifications of Yara for impurity content, too many unknowns remained, especially in regards to the reproducibility of the results at the pilot plant level and to a lesser extent the most probable ore proportion being fed to the mill.

Seeing this and before committing itself to a capital expenditure of some 800 M$, Mine Arnaud following recommendations from both Corem and Roche-Ausenco and advised by SGS Geostat, decided to add to the Roche-Ausenco Feasibility Study by reworking the mining block model and also by doing more metallurgical testworks on individual type of ore and on the most probable mill feed blend.

Following the Roche-Ausenco Feasibility Study many more laboratory testworks were carried out both at the bench scale and pilot plant levels.

13.2 Validation of Pilot Plant Flowsheet

A meeting held at the end of June 2012 confirmed that Mine Arnaud and its partners wanted to go forward with pilot plant testing at COREM. This decision was taken to raise the level of confidence in the process flowsheet as well as the representativity of the ore samples (ratio of the 3 ore units). Also, some concerns were raised regarding the blending of the ore zones during the previous pilot plant testing (2011).

To do so, Mine Arnaud arranged to send to Corem

13 040 kg of Upper ore

16 040 kg of Rail Road ore

12 870 kg of Nelsonite ore

13.2.1 Ore (blend) Chemical and Physical Characterization

At the request of Mine Arnaud, Corem blended the received ores to a very specific proportion that will, most likely, represent the mill ore feed for at least the first ten years of operation. Consequently the blended composite was made according to the following ratio of 36.1% Upper, 44.5% Rail Road and 19.4% Nelsonite by weight.

Upper/Railroad/Nelsonite Blend Chemical Analysis

Table 13-2: Chemical analysis for selected elements

SiO_2 %	Al_2O_3 %	Fe_2O_3 %	MgO %	CaO %	Na_2O %	K_2O %	TiO_2 %	MnO %	P_2O_5 %	Cl ppm
31.15	9.66	26.68	5.93	11.38	1.86	0.33	7.54	0.31	4.75	280

Crusher Work Index

Table 13.3 gives the results of the low energy impact test as well as the specific gravity of the ore blend.

Table 13-3: Results of the low energy impact test on the Upper/Railroad/Nelsonite blend sample

	CWI	S.G
	kWh/t	kg/t
Composite	10.1	3.40

Bond Ball Mill Work Index and Apatite Liberation Fineness

Table 13.4 presents the results of the average Bond ball mill Work Index and the liberation fineness for the Upper/Railroad/Nelsonite blend sample. Individual Bond ball mill indexes were also conducted during the preparation of the blend sample. The Upper/Railroad/Nelsonite blend sample showed a W.I. of 12.5kWh/t.

Microscopic examination confirmed that no matter the type or grade of the ore, at a P_{80} of around 125 µm, practically all apatite particles are free from the gangue. Therefore all composite samples were ground to +/- 125 µm prior to flotation.

Table 13-4: Results of the Bond ball mill Work Index and liberation fineness

SAMPLE ZONE	SAMPLE % PROPORTION	FEED P_{80} (µm)	PRODUCT P_{80} (µm)	BALL MILL W.I (KWh/t)
Upper	36.1	2180.0	124.6	13.1
Rail Road	44.5	2249.0	122.4	12.6
Nelsonite	19.4	2222.0	126.5	11.7
Total/Average	100.0	2217.0	124.5	12.5

Magnetic Separation

Prior to the lock cycle test and pilot plant runs the blend underwent a grinding and a magnetic separation stage to remove the titanomagnetite minerals.

Ore grinding proceeded by steps in order to account for the production capacity of each circuit. The non-magnetic product containing the apatite represented the apatite flotation feed. About 1.2% of the apatite was lost in the magnetic product which represented about 20% of the initial feed weight. The removal of the magnetic product from the raw ore feed enhanced the flotation feed apatite grade by about 1.08%, from 4.75% to 5.78% P_2O_5.

13.3 Flotation

In September and October 2012, following the recommendations from the previous pilot plant runs and at the request of Mine Arnaud, Corem tested two more column flotation flowsheets on the ore blend as set in paragraph 13.2.1 above[3]. Mine Arnaud requested to validate the final flowsheet by operating the column flotation circuit, on a continuous basis for a minimum of four days (96 hours), with the optimized conditions determined in the previous work and to be confirmed via a laboratory locked-cycle flotation test (LCT). Even if it is not particularly evident, the metallurgical results demonstrated that the pilot plant flowsheet with one rougher, one scavenger one cleaner and one cleaner/scavenger was the most promising.

Table 13-5: Lock cycle test on an ore blend comprising 36.1% Upper, 44.5% Rail Road and 19.4% Nelsonite Sept 14, 2012 From Corem's Report T1405

	WRec %	P_2O_5 Rec %	P_2O_5 %	Al %	Fe_t %	Mg %	Ca %	K_2O	Cl %
Fresh Feed (Calc.)	100.0	100.0	4.71	**4.80**	19.87	2.17	7.95	0.30	0.030
Magnetic Conc.	20.11	1.7	0.39	1.79	53.85	1.07	0.63	0.06	0.017
Flot. Feed (Calc.)	79.89	98.3	5.80	5.56	11.32	4.24	9.79	0.36	0.034
Final Conc.	10.82	90.5	39.46	0.64	0.64	0.31	36.59	0.12	0.127

[3] 36.1% Upper, 44.5% Rail Road, 19.4% Nelsonite

Table 13-6: Comparative results obtained in pilot plant (flotation columns) on same ore blend as for above LCT - October 2012 (includes magnetic separation) From Corem's Report T1405

	01/10/12	01/10/12	02/10/12	02/10/12	03/10/12	03/10/12	03/10/12	04/10/12	04/10/12
TYPICAL	DAY	NIGHT	DAY	NIGHT	DAY-AM	DAY-PM	DAY-N	DAY	NIGHT
P_2O_5 Feed %	4.73	4.46	4.59	4.88	4.62	4.41	4.66	4.57	4.71
P_2O_5 Conc. %	39.24	32.03	37.60	37.66	38.76	38.88	38.27	41.35	41.33
P_2O_5 Rec %	89.50	22.60	63.40	52.30	69.60	86.80	83.40	79.60	69.80
Cl Conc %	0.13	0.12	0.11	0.13	0.14	0.14	0.14	0.14	0.14
Fe+Al Conc %	1.59	3.92	1.88	1.67	1.55	1.90	1.64	0.57	0.49
Mg Conc %	0.29	0.57	0.43	0.43	0.41	0.46	0.46	0.22	0.18

Mine Arnaud requested that the apatite concentrate met the following specifications:

1. P > 17.0% (39% P_2O_5)
2. Ca/P < 2.2
3. Fe + Al Soluble in HNO_3 < 1.0%
4. Mg soluble in HNO_3 < 0.3%
5. Cl < 0.1%
6. Whole rock analyzed by XRF
7. Chlorine analyzed by UV Spec analysis

The above 2012 LCT and pilot plant testwork were not supervised by the QP responsible for the Mineral Processing and Metallurgical Testwork section of this report. As such, the results were not independently verified, but are believed to be of sound quality.

13.4 Additional Flotation Tests (Corem Project T-1518 - June 2013)

Due to the fact that both pilot plant runs returned ambiguous results (see above) and flotation tests made at SGS Lakefield failed to attain acceptable concentrate grades and recoveries (section 12.3), it was decided to conduct, under the supervision of SGS Geostat's metallurgist, four more lock cycle tests (LCT) at COREM and accept the results as an undisputable proof of the feasibility of the process.

These lock cycle tests were to be carried out on a sample composite that was previously experimented (see Table 13.5 above) and on three more composites of different grades but always of same ore blend. The composites were prepared by SGS Lakefield and came from fresh ore (core samples) from distinct locations in the deposit. SGS Geostat is of the opinion that since the composites prepared by SGS Lakefield came from crushed rejects from drill core samples, they are more representative of the actual variability of the deposit's mineralization. The composites that were used for the 2012 LCT and pilot plant were solely from blasted rocks at the surface of the ore body.

13.4.1 Flowsheet

The same flowsheet was used for these lock cycle tests as the one used in September 2012 (Corem Project 1405).

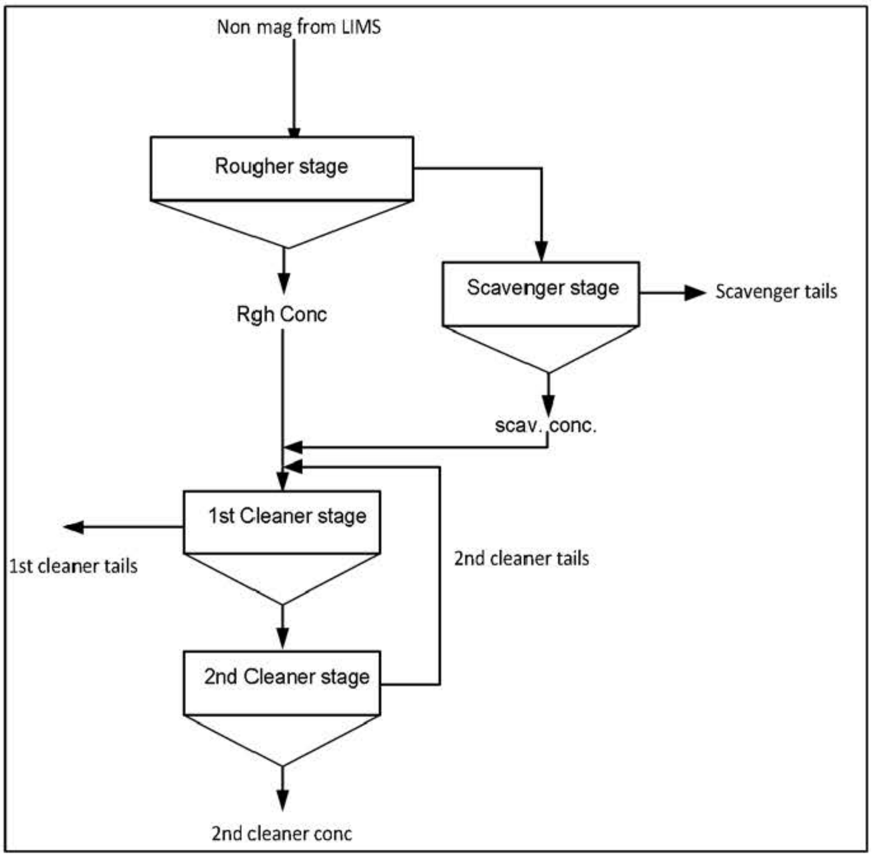


Figure 13-1: LCT's flowsheet

13.4.2 Results of Additional Flotation Tests

The first LCT was mainly to demonstrate that for the same ore characteristics, by applying the very same recipe and the same operation conditions, results are reproducible.

Table 13-7: Repeat of lock cycle test of September 14, 2012 (June 04, 2013)

	WRec %	P_2O_5 Rec %	P_2O_5 %	Al %	Fe_t %	Mg %	Ca %	K_2O	Cl %
Fresh Feed (Calc.)	100.0	100.0	4.69	**5.18**	19.30	3.51	8.09	0.32	0.028
Magnetic Conc.	20.12	1.70	0.39	1.79	53.85	1.07	0.63	0.06	0.017
Flot. Feed (Calc.)	79.88	98.3	5.77	6.04	10.60	4.12	9.97	0.38	0.031
Final Conc.	10.75	89.9	39.26	0.40	0..68	0.27	37.57	0.06	0.133

Table 13-8: Lock cycle test on "Medium Grade" composite (June 05, 2013)

	WRec %	P_2O_5 Rec %	P_2O_5 %	Al %	Fe_t %	Mg %	Ca %	K_2O	Cl %
Fresh Feed (Calc.)	100.0	100.0	5.56	4.30	20.02	4.37	8.42	0.19	0.04
Magnetic Conc.	22.12	2.50	0.64	1.66	49.45	1.43	0.94	0.04	0.06
Flot. Feed (Calc.)	77.88	97.50	6.96	5.05	11.66	5.44	10.55	0.23	0.03
Final Conc.	13.61	90.5	36.96	0.63	1.60	0.76	35.80	0.10	0.10

Since the reagents addition used for this test is the same as for the previous one, it is clear that both samples do not come from the same population. As explained above the composite used for this test (Table 13.8) comes from drill core samples whereas the composite used for the repeat of the lock cycle test of September 2012 (Table 13.7) came from a bulk sample retrieved from the surface of the deposit.

Obviously this last test lacked of iron depressant which resulted in a relatively good recovery at the expense of the grade of the concentrate.

Table 13-9: Lock cycle test on "High Grade" composite (June 06, 2013)

	WRec %	P_2O_5 Rec %	P_2O_5 %	Al %	Fe_t %	Mg %	Ca %	K_2O	Cl %
Fresh Feed (Calc.)	100.0	100.0	7.26	3.24	24.70	3.26	9.30	0.11	0.031
Magnetic Conc.	34.65	3.10	0.64	1.66	49.45	1.43	0.94	0.040	0.020
Flot. Feed (Calc.)	65.35	96.90	10.77	4.08	11.58	4.24	13.74	0.15	0.036
Final Conc.	16.29	87.5	39.02	0.29	0.96	0.39	37.65	0.04	0.081

Table 13-10: Lock cycle test on "Low Grade" composite (June 11, 2013)

	WRec %	P_2O_5 Rec %	P_2O_5 %	Al %	Fe_t%	Mg %	Ca %	K_2O	Cl %
Fresh Feed (Calc.)	100.0	100.0	3.51	5.71	18.82	4.49	7.52	0.17	0.024
Magnetic Conc.	23.37	2.90	0.43	2.16	48.12	1.59	1.14	0.05	0.020
Flot. Feed (Calc.)	76.63	97.10	4..45	6.79	9.89	5.13	9.47	0.21	0.025
Final Conc.	8.03	88.70	38.83	0.36	1.01	0.45	37.52	0.03	0.097

Based on the previous results when it became clear that there was no easy and single recipe for the flotation reagents addition and that they had to be closely monitored to the mineralogy of the ore, Corem took great care to adjust the flotation reagents to the apparent amount of iron and titanium oxides in both the rougher and primary cleaner concentrates of cycles one and two of the next LCT.

Table 13-11: Lock cycle test on "Medium-Low Grade" composite (June 12, 2013)

	WRec %	P_2O_5 Rec %	P_2O_5 %	Al %	Fe_t %	Mg %	Ca %	K_2O	Cl %
Fresh Feed (Calc.)	100.0	100.0	4..18	5.58	19.31	3.70	8.02	0.16	0.019
Magnetic Conc.	18.33	0.40	0.09	2.04	54.52	1.10	0.41	0.04	0.000
Flot. Feed (Calc.)	81.67	99.6	5.10	6.37	11.40	4.29	9.73	0.19	0.024
Final Conc.	9.85	94.1	39.97	0.25	0.51	0.27	38.44	0.03	0.087

13.5 Analysis Results Validation

Upon reception of the concentrate assay results, SGS requested that the samples be analyzed in a third party laboratory, in order to ensure accuracy of the data. Flottation concentrate from the four LCT's and feed material were sent to SGS Laboratory in Lakefield to be re-assayed. Results for P_2O_5%, K_2O% and Cl% were reported and compared with the results from COREM.

No systematic biais were found and comparison of the results on scatter plots give a strong correlation between the result from COREM and SGS (P_2O_5% R^2=0.9994, K_2O% R^2=0.973 and Cl% R^2=0.9816). Analytical results for the concentrate given by COREM are considered acceptable by SGS.

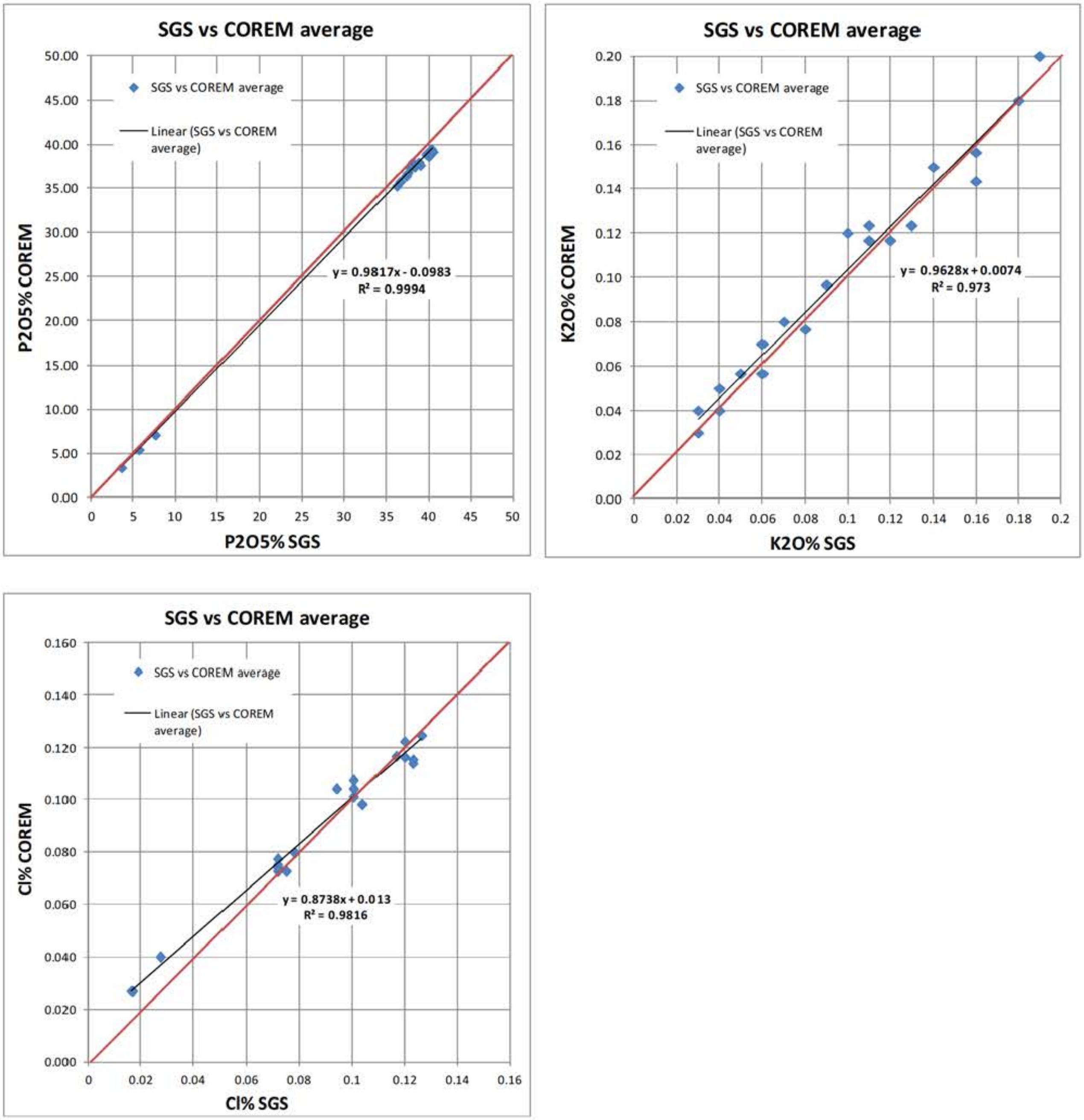


Figure 13-2: Comparison scatter plot of COREM and SGS analytical results

13.6 Conclusion

Because this last series of tests did not prove beyond any doubt that it is possible to obtain +39% P_2O_5 at the final concentrate and much less on a continuous basis, more tests will have to be done. Presently, the only way to assess, at the laboratory level, if the apatite concentrate is clean enough is by visual examination of a minute concentrate sample under a binocular. If not, more reagents are added to the subsequent flotation stage. Wherever the next series of tests will be made, a quick device analyser (at least for iron and titanium) should be used.

It is clear that there is no such thing as a unique "Recipe" for the flotation of the apatite and some "fine tuning" will eventually have to be done since no real reagents optimization was ever asked or tried on the Arnaud's ore. In a commercial mill, ore mineralogy and grade will have to be closely monitored for the reagents addition via a good in-line analyser.

Finally, as a general rule SGS Geostat never extrapolates beyond what was obtained at the laboratory or pilot plant level. However, in this case we are confident that if the ore is specifically mined based on its low chlorine content, in a well monitored mill using the right flotation machines, with ROM and concentrate blending possibilities, a concentrate grade of at least 39% P_2O_5 will be achieved along with an apatite recovery in the 90% range while meeting all of Yara' specifications.

14. Mineral Resource Estimates

The mineral resource estimation work for Mine Arnaud's Sept-Îles project was conducted by Jean-Philippe Paiement, M.Sc., P.Geo., under the supervision of Claude Duplessis, Eng. The modeling, geostatistics and grade interpolation of block model was done using Genesis© software developed by SGS Geostat.

14.1 Database

The database used for the resources estimation was transmitted to SGS Geostat by Mine Arnaud on April 5, 2013 in Geotic© format. The database comprised 430 drill holes (Figure 14-1) with entries for:

1. Down Hole Survey (n = 14,252);
2. Assays (n = 21,059);
3. Lithologies (n = 7,881).

The database was validated upon importation in Genesis, which enable to correct minor discrepancies between the table entries, in the surveys and lithologies. Most of the errors were caused by survey and/or lithologies entries going past the hole length as it was entered in the collar table. These errors have been corrected.

Vertical sections have been generated, oriented N329° in order to respect and follow the drilling pattern, in which every section follows a drilling section. In general, the sections have a 50m spacing between them (Figure 14-2).

A topographic surface was transferred to SGS Geostat by Mine Arnaud. The surface was processed and normalized in order to correct the distortion in the edges (Figure 14-3). A surface representing the contact between overburden and fresh rock was also generated. This section was modeled on each section.

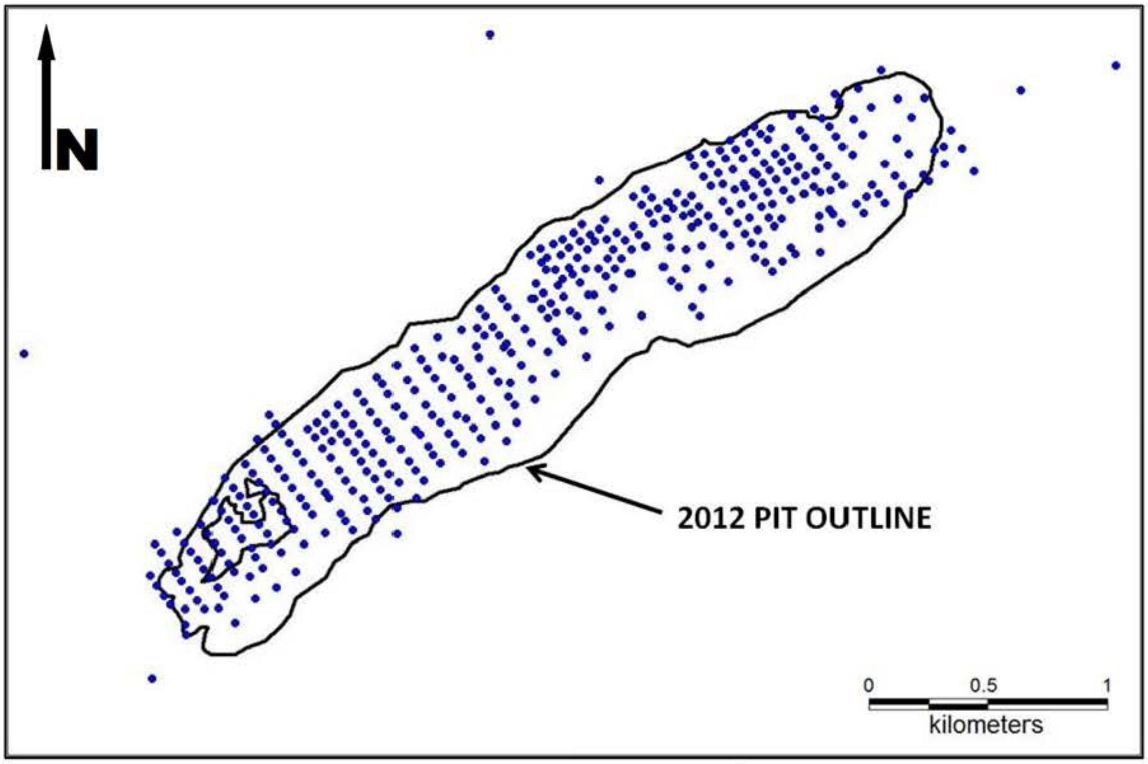


Figure 14-1: Drill holes collars positioning relative to the 2012 pit outline

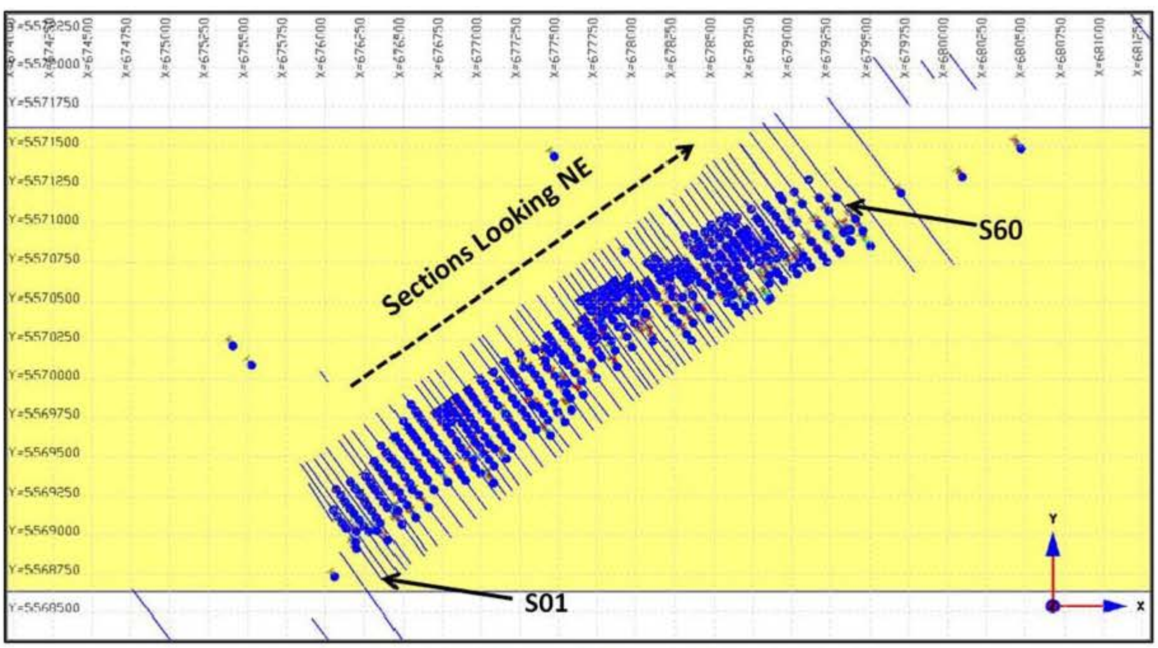


Figure 14-2: Plan view showing the trace of each vertical sections

SGS

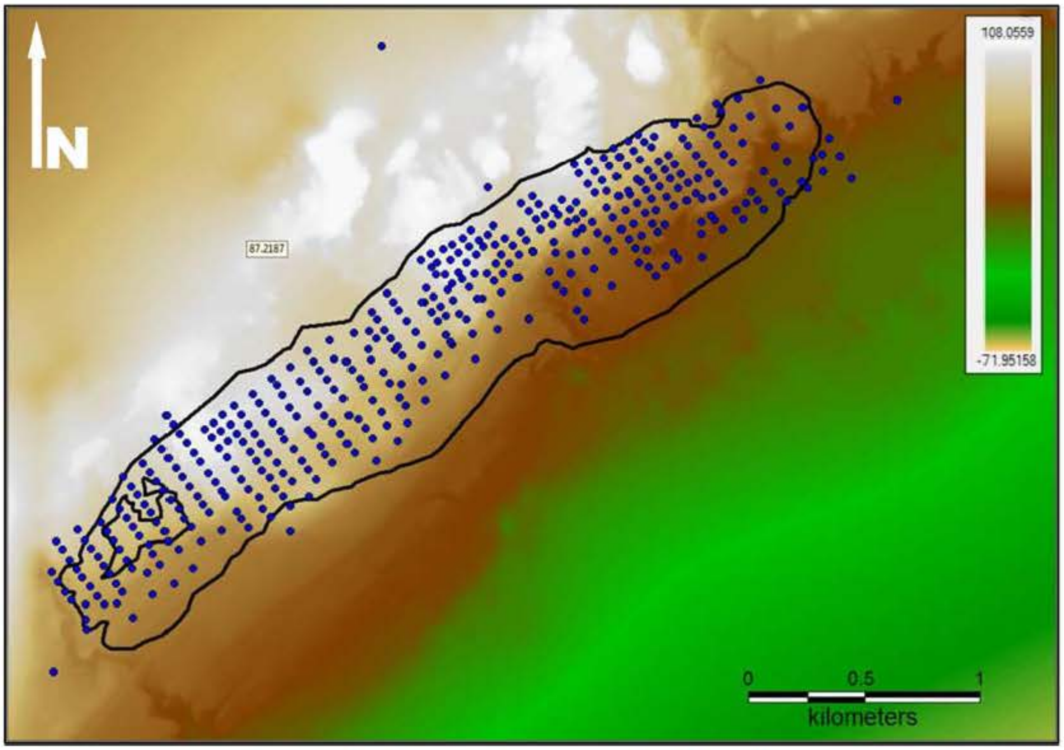


Figure 14-3: Topographic surface and 2012 pit outline

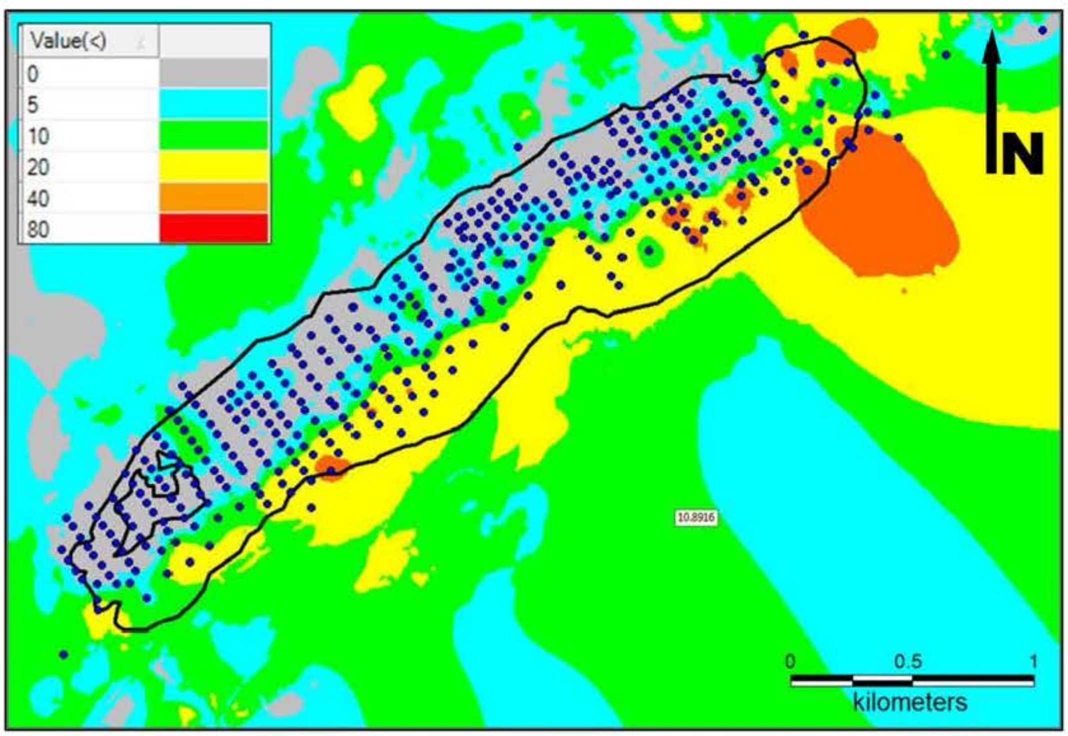


Figure 14-4: Overburden thickness grid relative to the 2012 pit outline

14.2 Geological and Structural Model

Lithological envelops were transmitted to SGS Geostat in DXF format (Figure 14-6). These envelops served as a base for the section interpretation of the mineralized solids. In total, 4 envelops were modeled in the past: 1) Nelsonite; 2) RailRoad; 3) Upper and 4) California.

These envelops were based on geological information gathered in drill holes, to this date, no surface mapping exist for the project. The different zones correspond to distinct apatite bearing lithologies in the rock formations (Figure 14-5). The zones actually correspond to cyclic enrichment in Phosphorous during fraction crystallization of the Sept-Îles igneous complex.

Different faults have been identified in the drilling data. These faults are easily interpreted in 3D particularly when looking at the disturbances in the geological envelops, where some blocks clearly appear as displaced. Two major fault types were identify: 1) faults sub-parallel to the igneous horizons with a shallow dip and N065° orientation and 2) sub-vertical faults cross-cutting the deposits with an orientation of N125°.

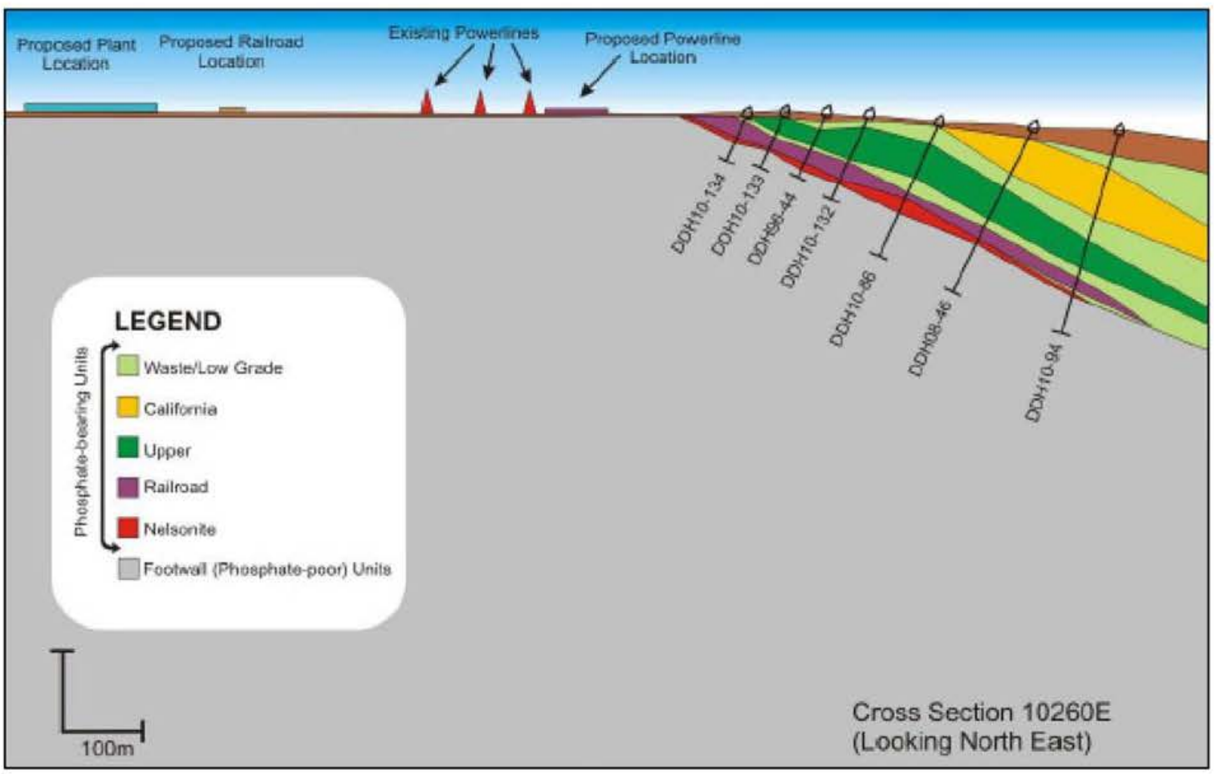


Figure 14-5: Schematic geological cross section (from Mine Arnaud Feasibility Study, 2011)

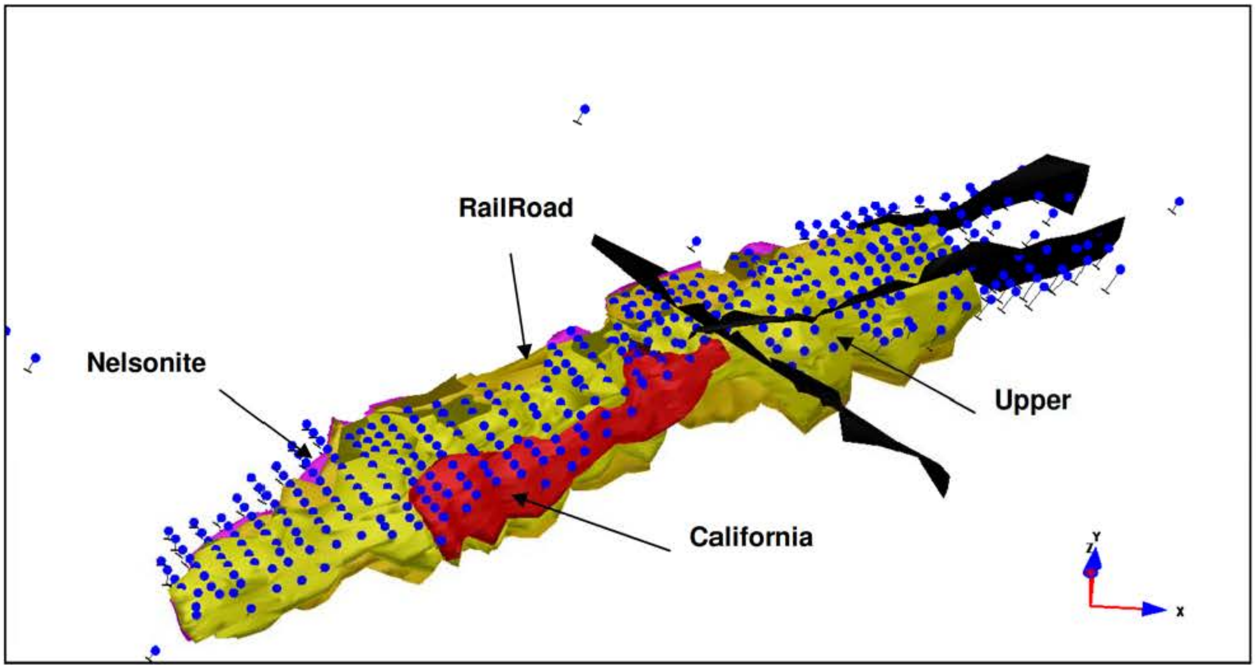


Figure 14-6: 3D view of the geological envelops provided by Mine Arnaud

14.3 Mineralized Intervals and Mineralized Solids

Mineralized intervals, corresponding to average grade of combined assays, were generated following the limits of the geological envelops. These intervals were then adjusted to respect a modeling minimal grade of 2.0% P_2O_5. The modeling minimal grade was established using the mean value of all the assays and the resources cut-off grade of the past reports (Figure 14-7). In the event that a single hole, in the middle of a geological envelop was lower that the minimal modeling grade, the hole was still integrated in the solids and is considered as internal waste.

Based in the mineralized intervals and geological envelops, solids were digitized on each sections. The solids are extrapolated to 50-100m towards the SE from the last point of intersection and to the topographic surface in the NW portion (Figure 14-8). The solid apexes were snapped to the mineralized intervals. Initially the 4 geological zones were modeled as independent mineralized solids. During this process, the author observed that the RailRoad and Upper zones were not consistently different and when looking at the geostatistic it became more evident that they were part of a single mineralized zone (Figure 14-9). The Upper and RailRoad zones were therefore combined together to form the Combine zone (Figure 14-10).

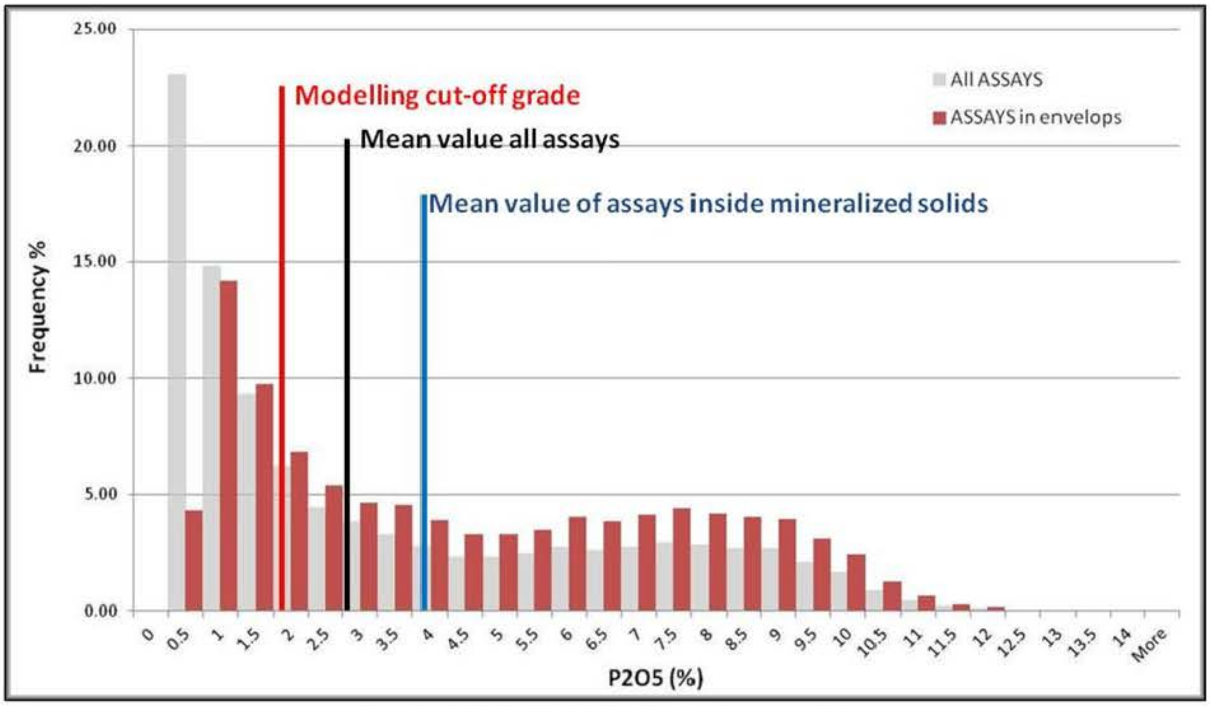


Figure 14-7: Statistic distribution of the assay data

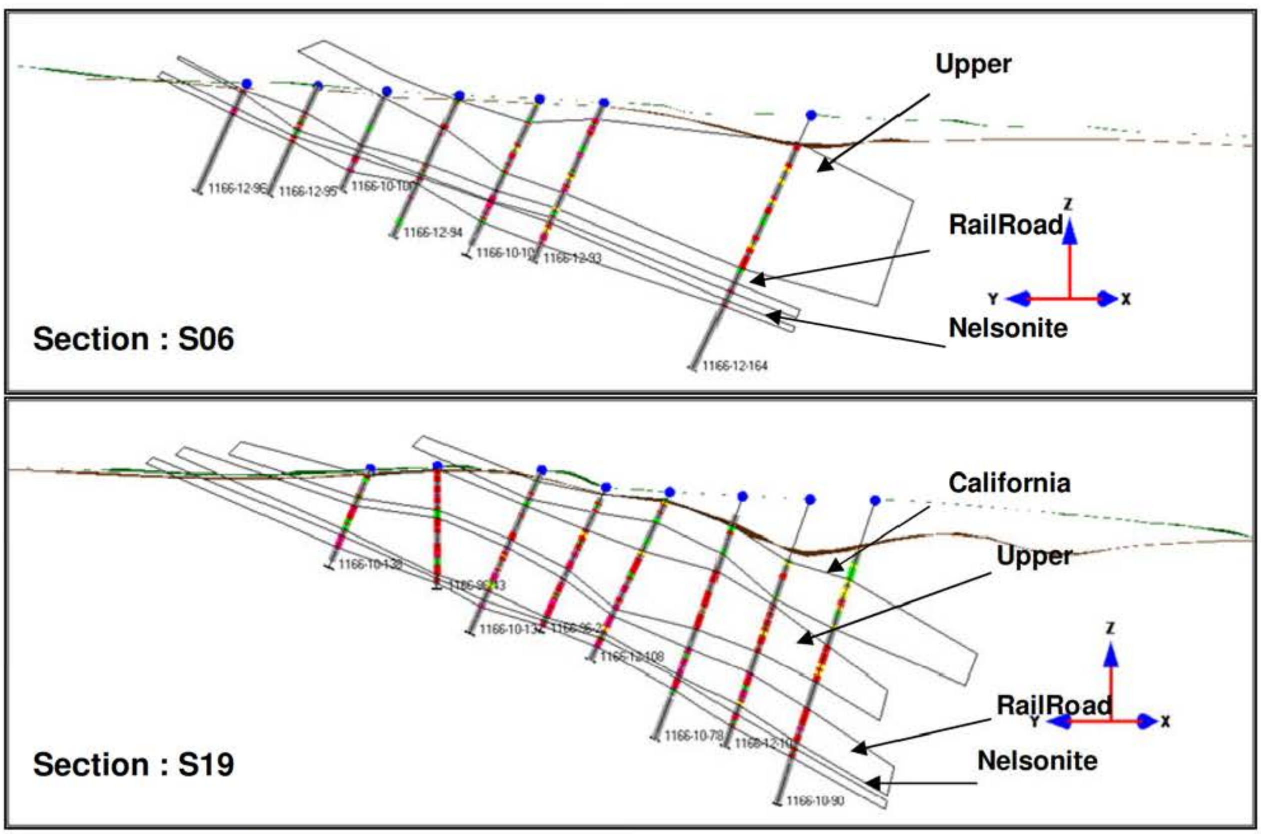


Figure 14-8: Examples of solid interpretation on sections (looking northeast)

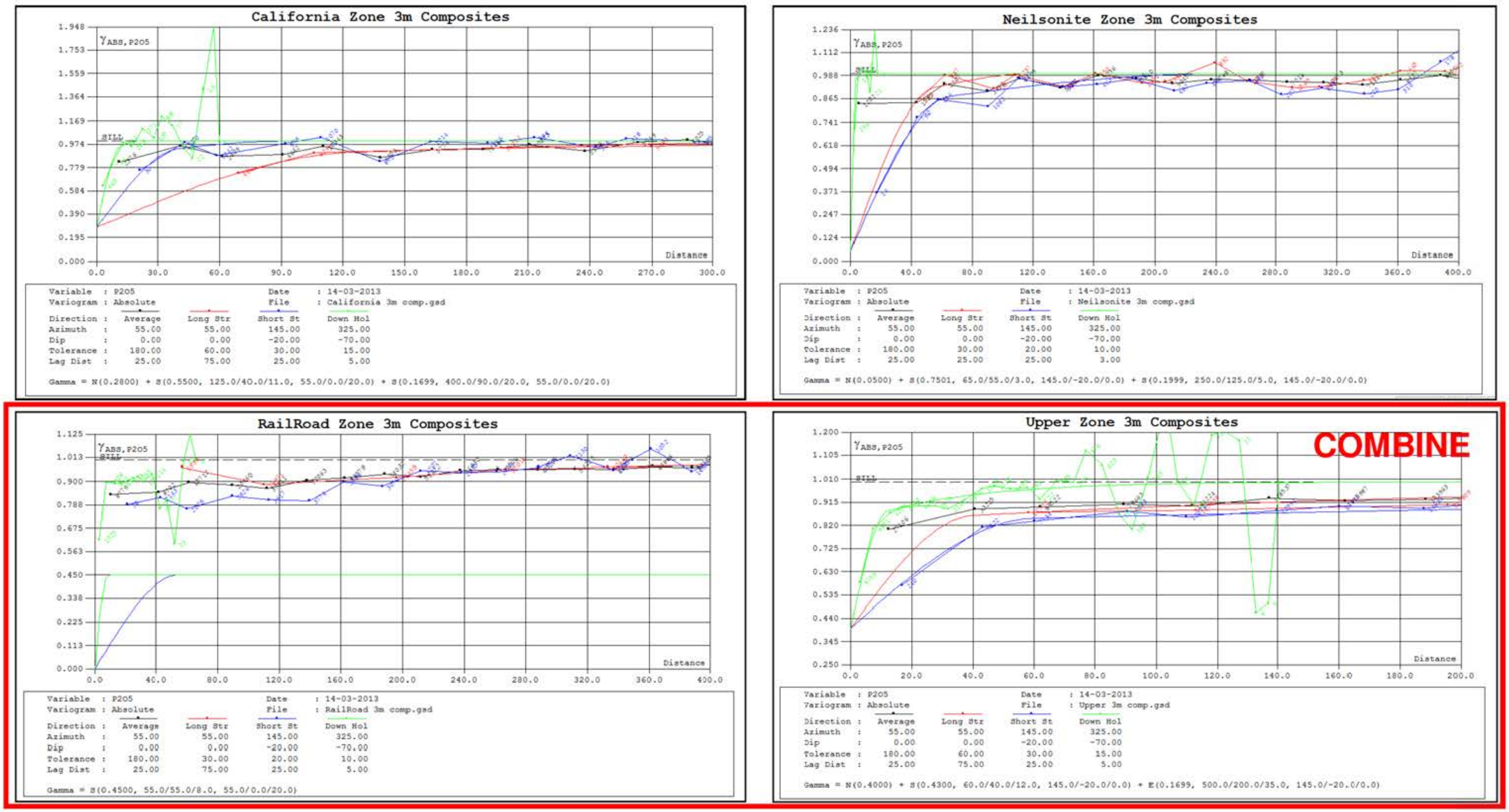


Figure 14-9: Variograms for the 4 initial zones

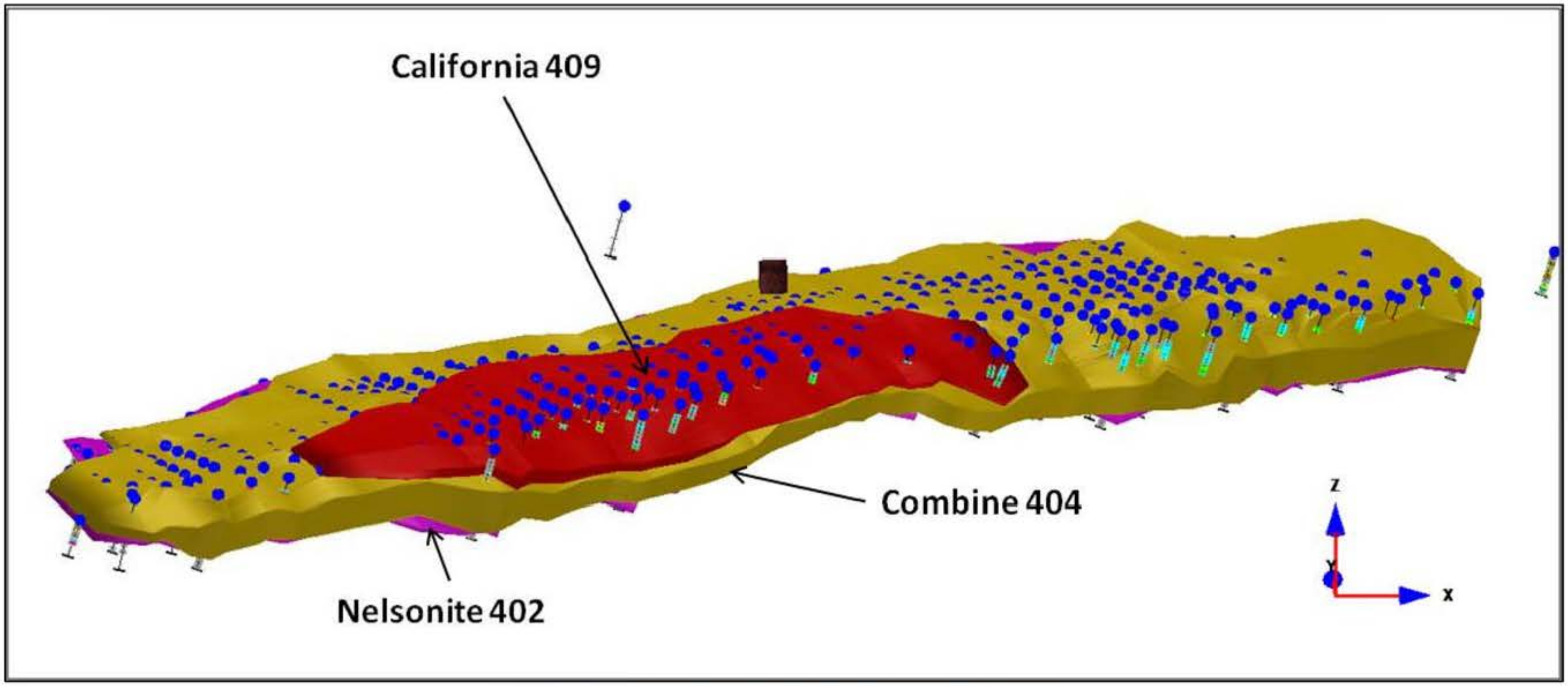


Figure 14-10: 3 mineralized envelops for resources estimation

14.4 Compositing of Assays

The assays present inside the limits of the mineralized intervals were re-divided in equal length composites of 3m. These composites are used to interpolate to the block values. Different sets of composites were generated for the 3 mineralized solids and also for the waste between the mineralized zones. Hence, 4 different and independent sets of composites will be used in the estimation of the resources.

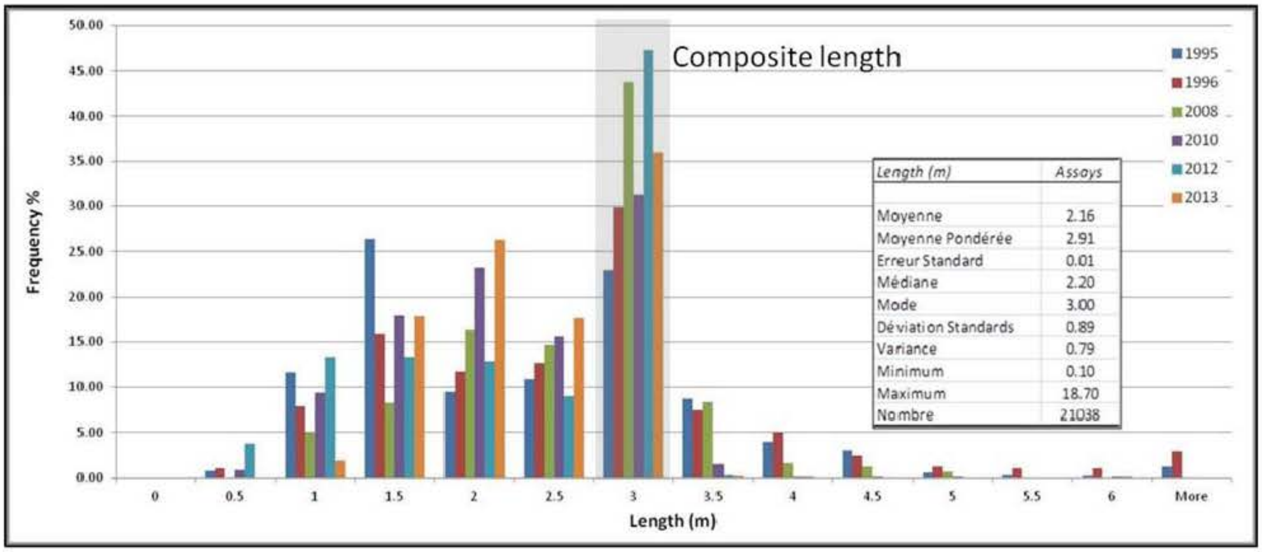


Figure 14-11: Assays length statistics

14.5 Geostatistics and Variography

In order to interpolate the different blocks in the three zones, the composites where independently analyzed using standard statistic tools and variography. These steps enable to validate the compositing process and mineralized solids generation. The mathematical models derived from the Variograms will be used to interpolate the blocks using Kriging. The exercise was performed for the critical elements of the block model: 1) P_2O_5; 2) Cl and 3) K_2O.

14.5.1 P_2O_5 Variable

The histograms for all of the 3 zones show a bi-modal distribution caused by the presence of waste intervals in the mineralized solids (Figure 14-12). These waste intervals were included in the solids in order to respect the continuity of the zones and geology. The composites for the California and Combine zones follow the distribution of the assays quite closely with a normalization of high and low values. The Nelsonite zone does not show the same relations due to the fact that it is less thick and that the waste samples are not represented in single composites but instead they dilute the composites, hence the single mode normal distribution of the composite data (Figure 14-12). Nelsonite has the highest P_2O_5 value and it decreases in Combine and then California.

Variograms were then generated for each of the composite sets. Even if the overall orientations of the zones are quite similar, some differences were observed in the spatial distribution and continuity of the P_2O_5 values. The nugget effect varies from 13% in the Nelsonite to 41% in the Combine zone. The California zone (Figure 14-12) shows the best continuity along the long axis of the deposits (azimuth = N055°; max range = 400m) whereas the Combine (Figure 14-12) and Nelsonite (Figure 14-12) zones shows the best continuity along the dip direction of the zone (N145°; max ranges of 500m and 250m). The variography models (Table 14-1) for each of the 3 zones were used in the kriging interpolation process and the ranges also help in the determination of the search ellipsoids and resources classification process.

14.5.2 Cl Variable

The histograms for California and Combine zones show a single mode slightly skewed distribution (Figure 14-13), whereas the Nelsonite distribution is bi-modal just like the P_2O_5 distribution observed before. All three distributions have a mode at 0.10% Cl but Nelsonite also presents a mode at 0.30% Cl. The Nelsonite bi-modal distribution reflects closely the Cl in apatite relation observed in the chlorine study. In California and Combine, the high values could also be linked to the high presence of phlogopite, a K_2O bearing mineral, see below. The highest average Cl is observed in the Nelsonite zone and decreases towards to stratigraphic top in Combine and then California. This follows the P_2O_5 behaviors observed above.

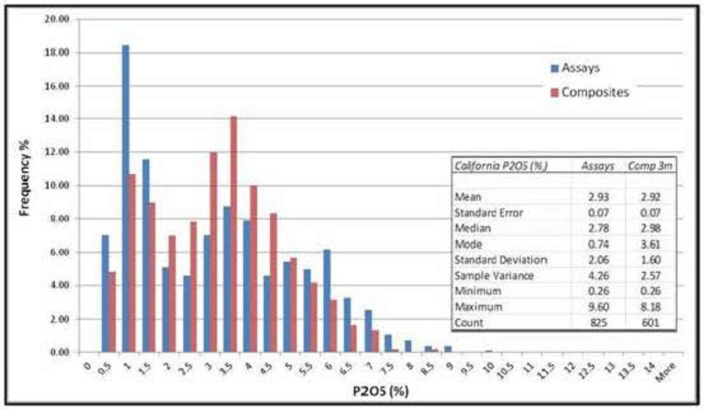
The relatively good correlation between Cl and P_2O_5 (Figure 14-15) with R^2 ranging from 0.41 to 0.72 enables the authors to use the same base variograms for Cl as for P_2O_5. The variograms modeling was then adjusted when possible to better reflect the Cl behavior, but anisotropy and ranges where kept close to the P_2O_5 values (Table 14-1). Variography modeling was successfully done for both the California and Combine zones, but not for Nelsonite. Hence, the P_2O_5 model was used for this one. The nugget effect is relatively low for the Nelsonite (13%) and California (1%) zones and reaches 41% in the Combine Zone. First spherical component ranges are very similar to P_2O_5 and are lower for the second component in the Combine Zone.

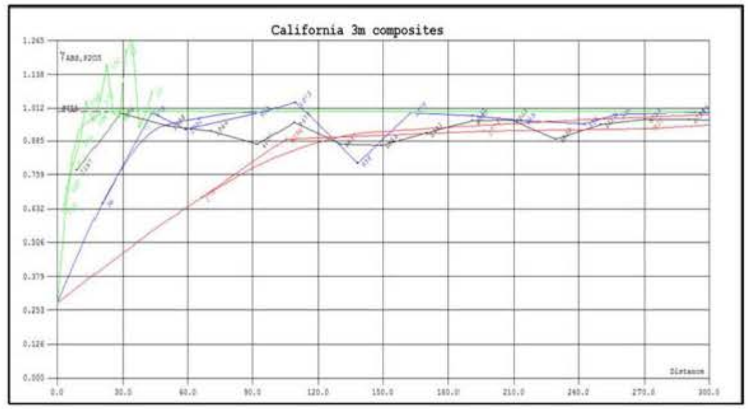
14.5.3 K_2O Variable

The histograms for California show a normal distribution (Figure 14-14), whereas the Combine and Nelsonite distribution are single mode slightly skewed distributions. The observed relation between K_2O and the zones is the inverse of what is observed for P_2O_5 and Cl. The K_2O average increases from Nelsonite to Combine and then in the California Zone (Figure 14-14), California has the highest average K_2O value.

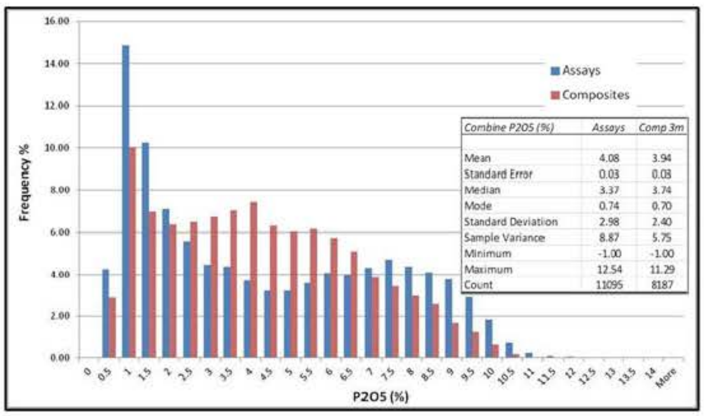
The relatively good correlation between K_2O and P_2O_5 enables the possibility of using the same base variograms for K_2O and P_2O_5. The variograms modeling was then adjusted when possible to better reflect the Cl behavior, but anisotropy and ranges where kept close to the P_2O_5 values (Table 14-1). It was possible to adjust the variography models for the Combine zone only. The nugget effect is 10% and spatial continuities are comparable to P_2O_5 models. For the California and Nelsonite, the modeled variogram of P_2O_5 were used for K_2O interpolation (Table 14-1).

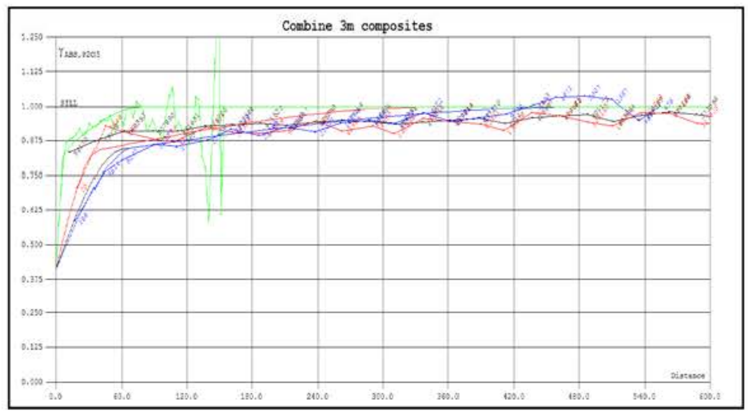
CALIFORNIA ZONE



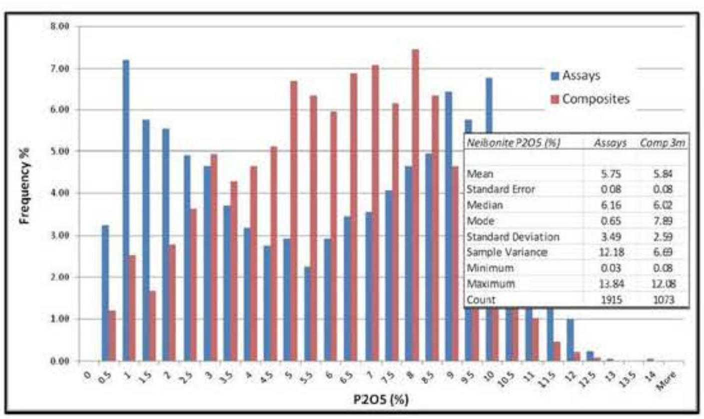


COMBINE ZONE





NELSONITE ZONE



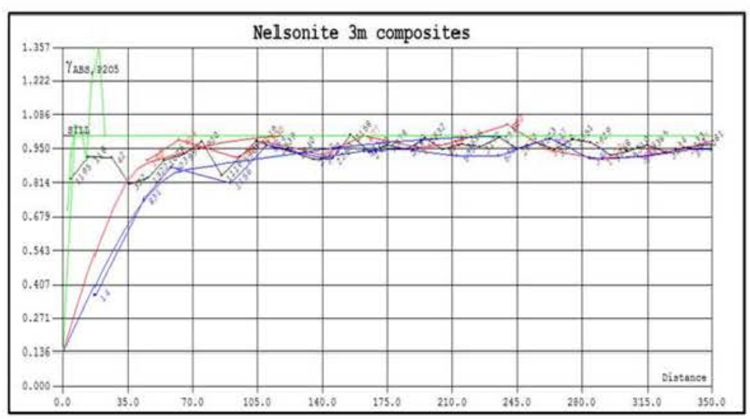
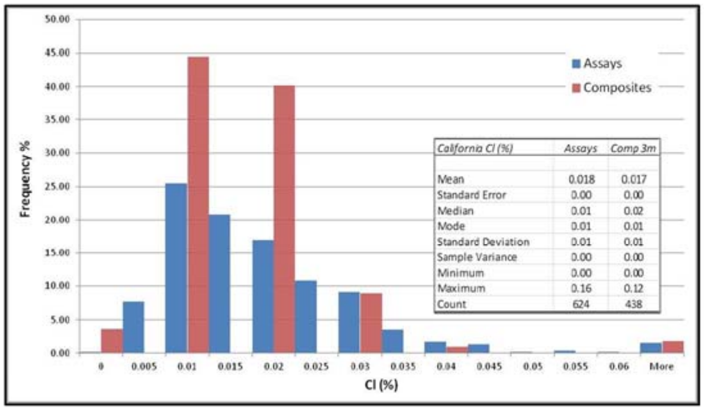
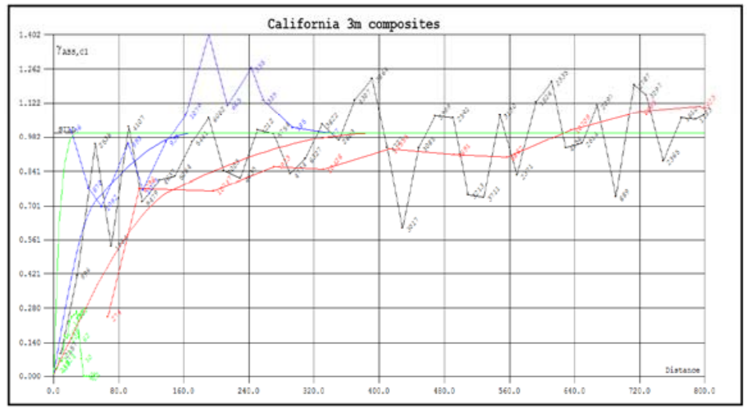


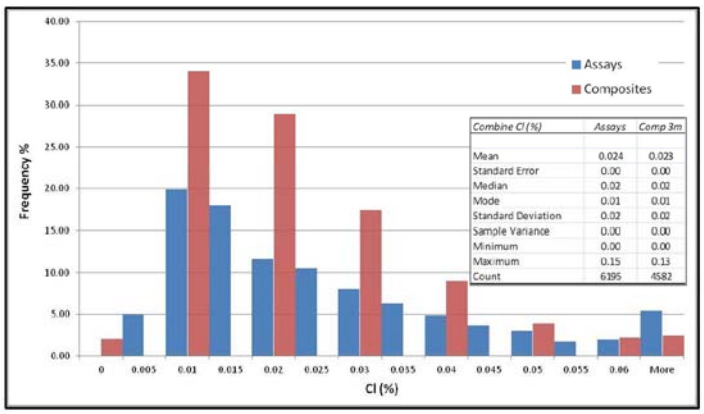
Figure 14-12: Geostatistics for P2O5% between different zones

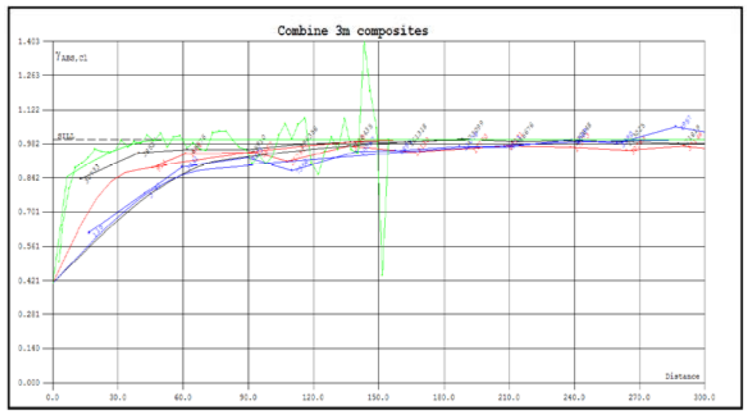
CALIFORNIA ZONE



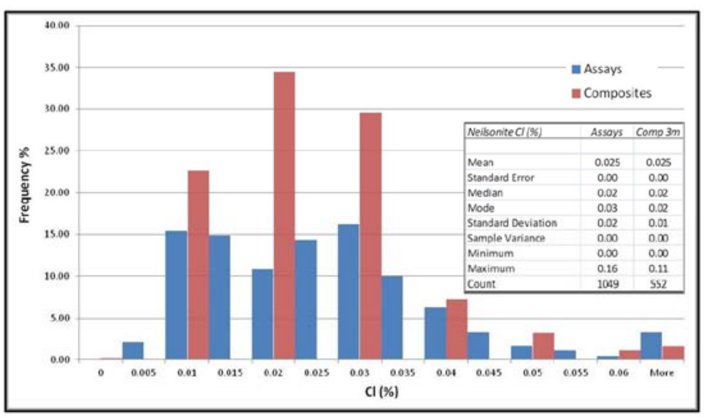


COMBINE ZONE





NELSONITE ZONE



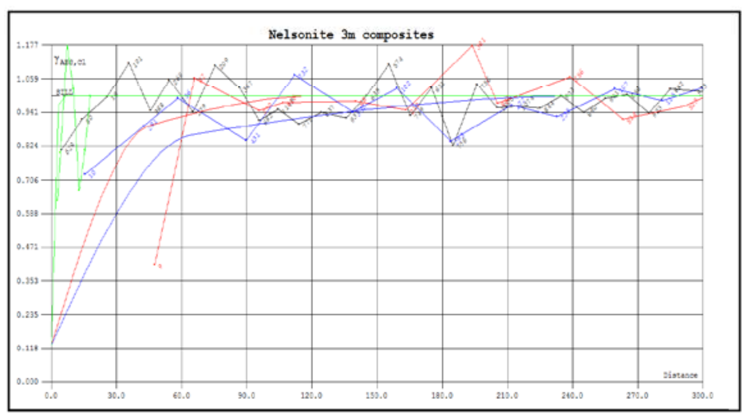
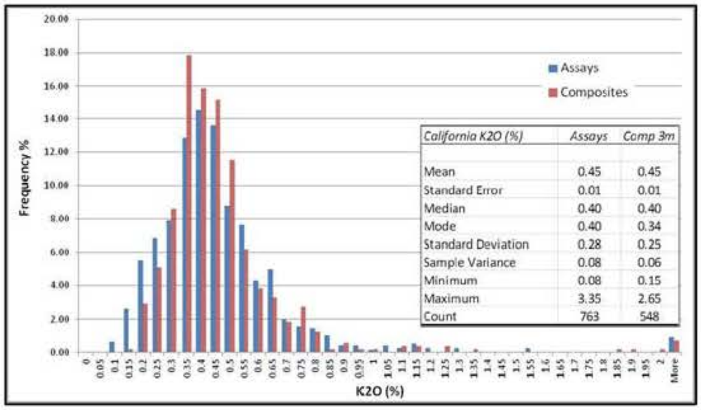
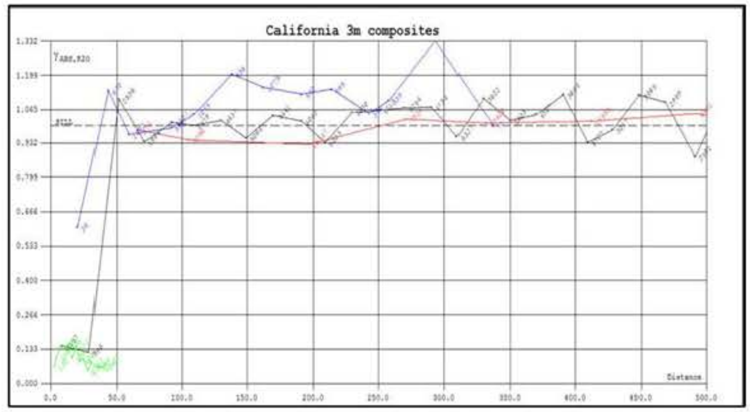


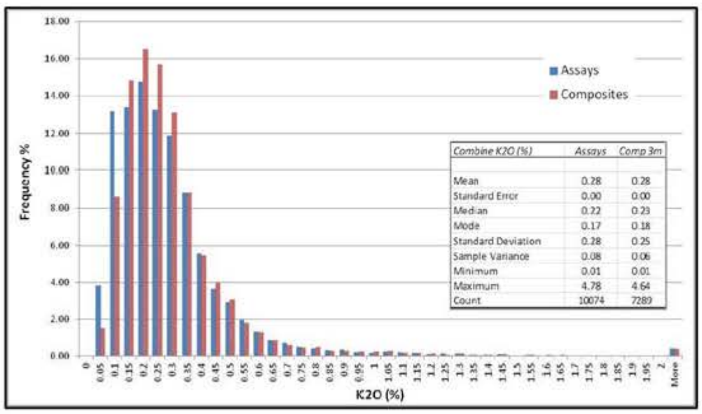
Figure 14-13: Geostatistics for Cl% between different zones

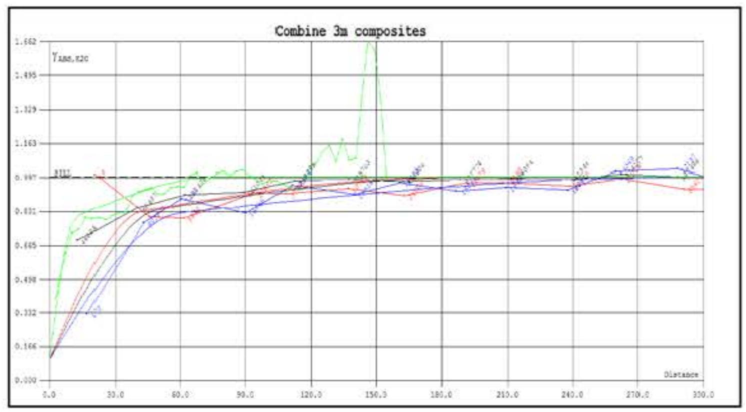
CALIFORNIA ZONE



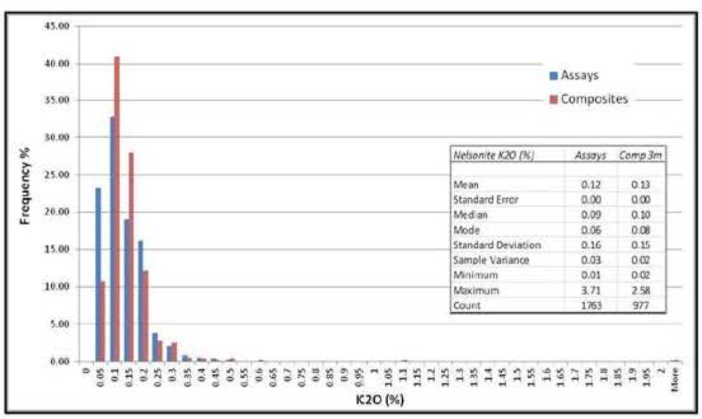


COMBINE ZONE





NELSONITE ZONE



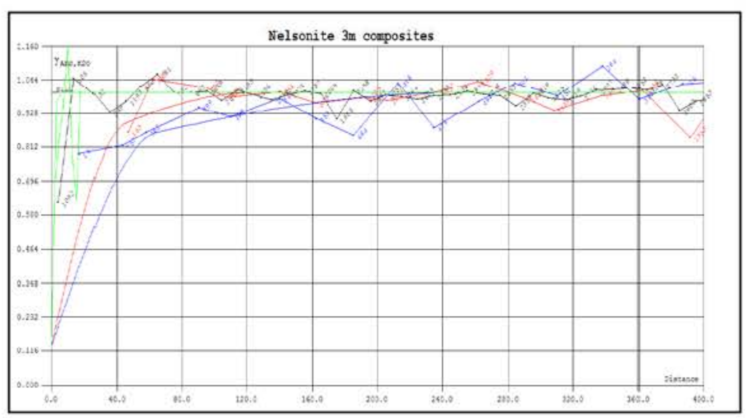
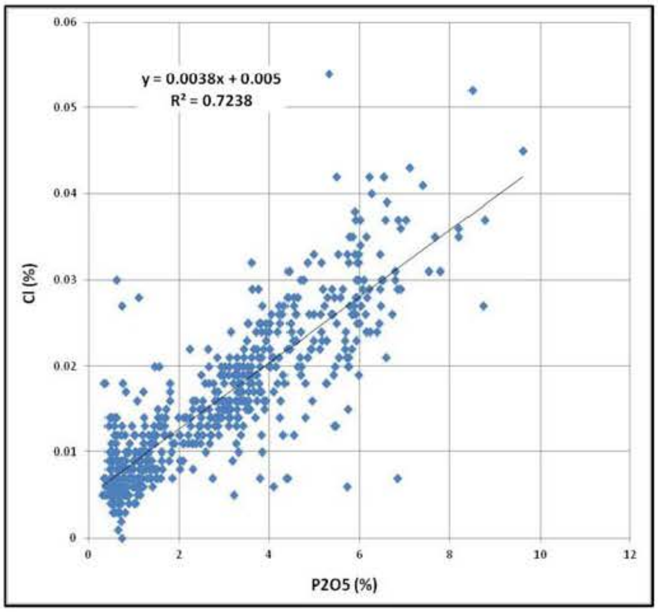
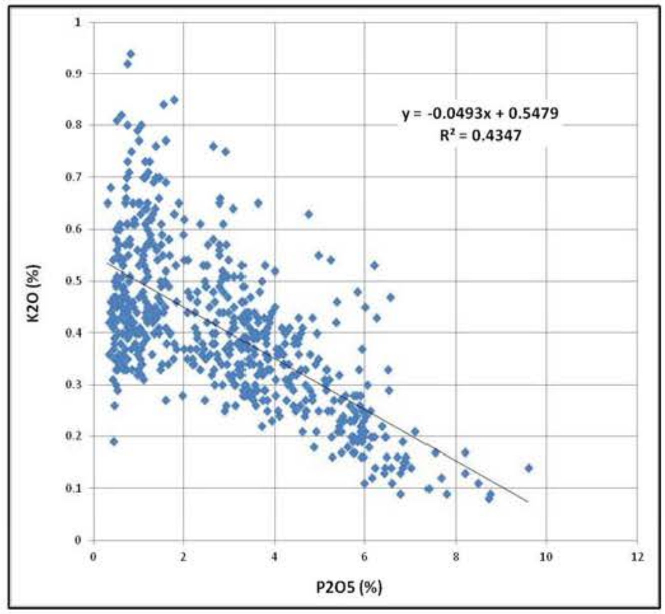


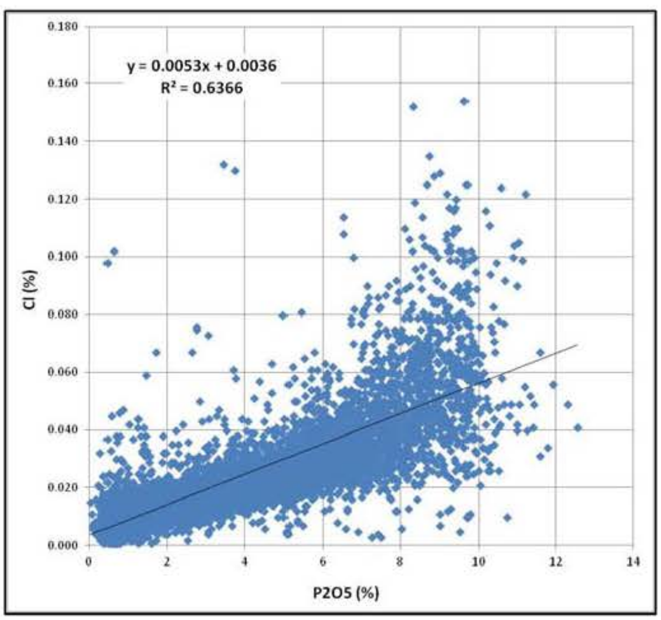
Figure 14-14: Geostatistics for K2O% between different zones

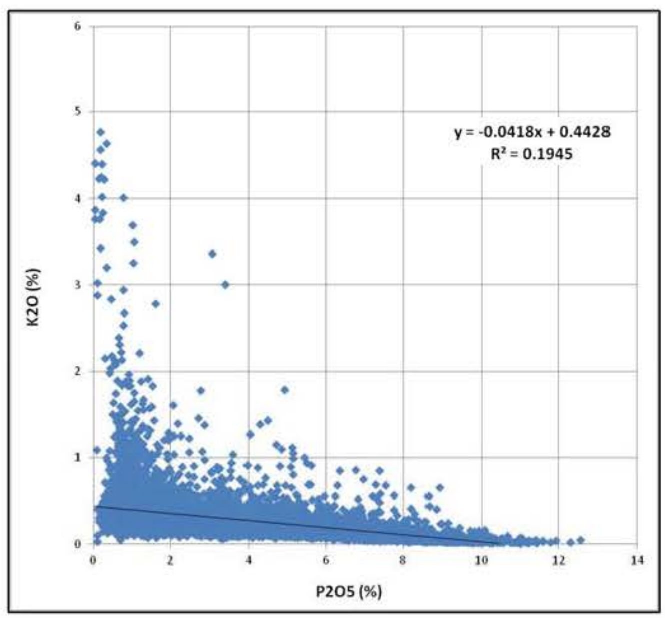
CALIFORNIA ZONE





COMBINE ZONE





NELSONITE ZONE

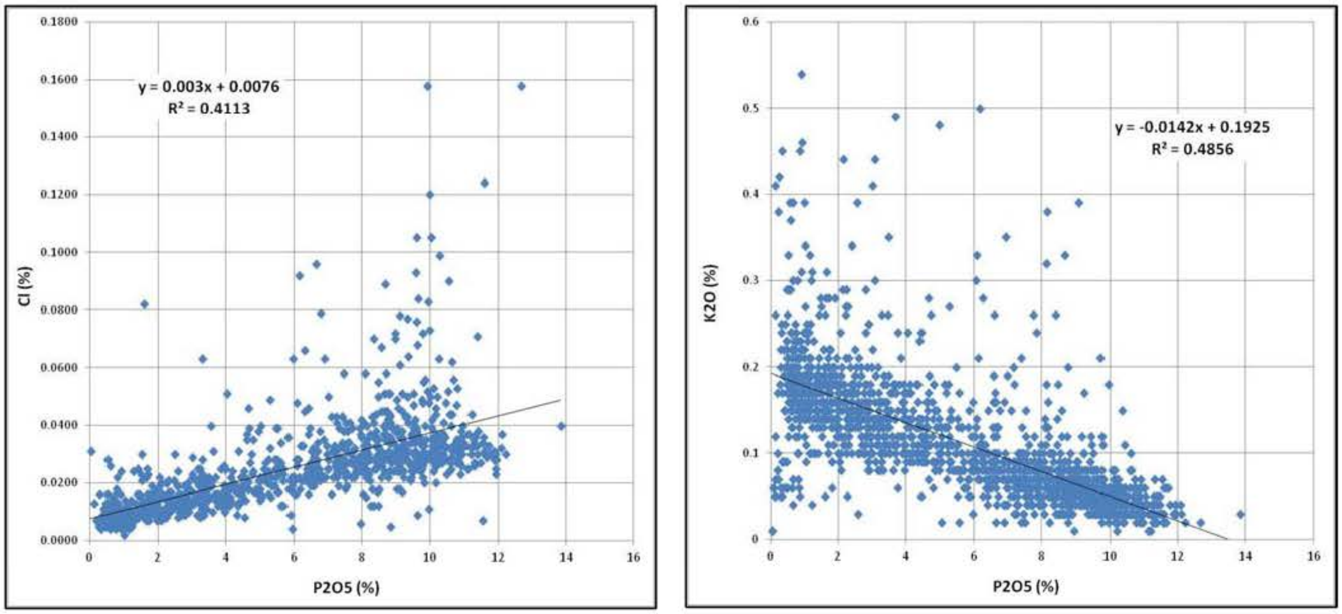


Figure 14-15: Element correlation (P2O5, Cl and K2O) in the different zones

Table 14-1: Variography models

Variographic Model	Nugget	Spherical: 1rst Component				Spherical: 2nd Component				Azimuth	Dip	Spin
		Sill	Max Range	Mid Range	Min Range	Sill	Max Range	Mid Range	Min Range			
California P2O5	0.28	0.55	145	50	10	0.1699	400	100	25	55	0	20
California Cl	0.01	0.50	145	50	10	0.49	400	175	25	55	0	20
California K2O	0.28	0.55	145	50	10	0.1699	400	100	25	55	0	20
Combine P2O5	0.41	0.40	70	60	8	0.1899	500	500	80	145	-25	0
Combine Cl	0.41	0.40	70	80	6	0.1899	300	200	50	145	-25	0
Combine K2O	0.10	0.65	60	50	12	0.25	300	200	80	145	-25	0
Nelsonite P2O5	0.13	0.65	65	45	3	0.2199	250	125	5	145	-20	0
Nelsonite Cl	0.13	0.65	65	45	3	0.2199	250	125	5	145	-20	0
Nelsonite K2O	0.13	0.65	65	45	3	0.2199	250	125	5	145	-20	0

14.6 Specific Gravity

The database transmitted in December 2012 to SGS Geostat contained 348 density measurements made on pulp samples and 583 density measurements made on core samples. The 2 data sets were compared (Figure 14-16) in order to decide if the pulp samples density measurements were going to be used in the interpolation process. The comparison showed that the pulps seemed to be systematically lower than the core samples (pulp mean = 3.25 and core mean = 3.32). This 2.57% difference coupled with the known fact that measurements made on pulps do not account for the in-situ density enable the justification of the rejection of the pulp samples measurement in the interpolation process.

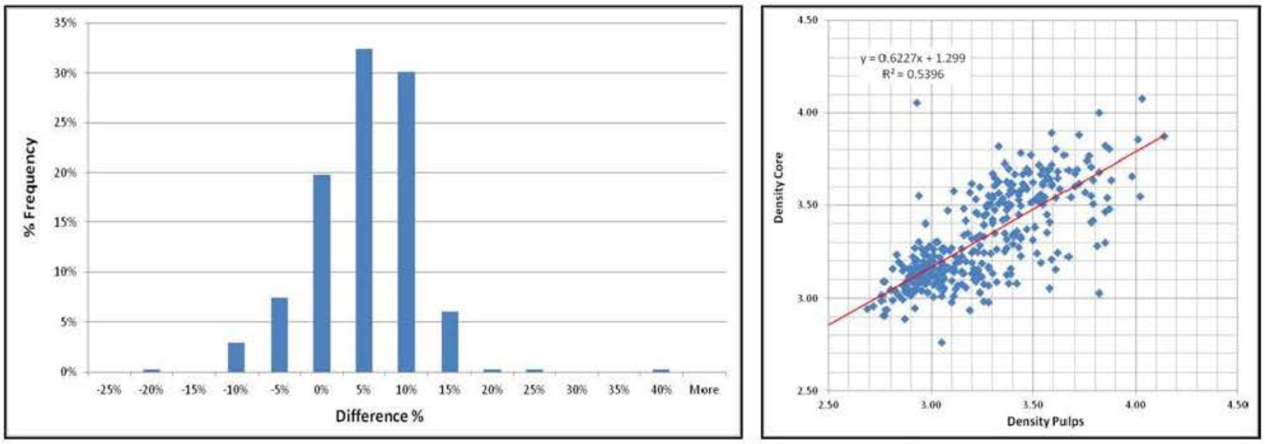


Figure 14-16: Comparison between Density measurements on Core and Pulp samples

The core sample measurements were then used to verify the existence of relationships between whole rock analysis data and density measurements. Since, the main heavy minerals found in the deposit are magnetite and ilmenite (or Fe_2O_3% and TiO_2% (Figure 14-17). The relation between these 2 components and the densities were quite good (R^2>0.90). Hence, a theoretical equation was developed to calculate the density using the assays results for Fe_2O_3% and TiO_2%. This equation takes into account a basic density of 2.6 and the magnetite (density of 5.15) and ilmenite (density of 4.72) content of the sample to estimate a density.

*Density = (((100-(Fe+Ti))*2,6)+((Fe-Ti)*5.15)+(Ti*4.72))/ (100-(Fe+Ti)+(Fe-Ti)+(Ti))*

The densities are then calculated for each of the samples in the database, creating a set of 20,486 calculated density and 583 measured densities. This data set was then transferred in the composites and used to interpolate the densities of each block in the model (Figure 14-18). The interpolated densities were then compared with the calculated densities of each block using their interpolated Fe_2O_3% and TiO_2% values, which showed a good relationship. The interpolated densities of each block are the one used in the model to estimate the tonnage of the resource. The densities vary slightly from the densities used in the past (Table 14-2) where the present average densities are between 3% and 12% lower than the past densities.

Table 14-2: Comparison between past density values and currently used values

Densities	California	Combine	Nelsonite
2012 Mean value	3.24	3.40	3.82
2013 Mean value	**3.15**	**3.18**	**3.40**
% Difference	*3%*	*7%*	*12%*
2013 Standard Error	0.00	0.00	0.00
2013 Median	3.16	3.20	3.43
2013 Mode	3.17	3.18	3.45
2013 Standard Deviation	0.15	0.16	0.21
2013 Sample Variance	0.02	0.03	0.04
2013 Minimum	2.58	2.55	2.54
2013 Maximum	3.57	4.25	4.46
2013 Count	50 171	545 449	61 556

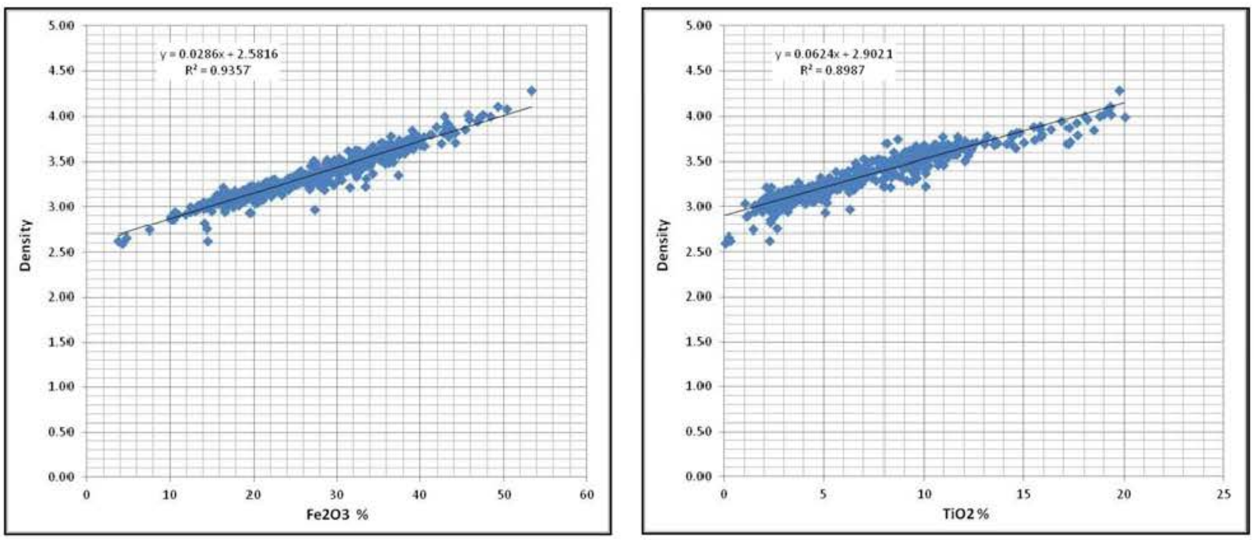


Figure 14-17: Relation between assays and density measurements

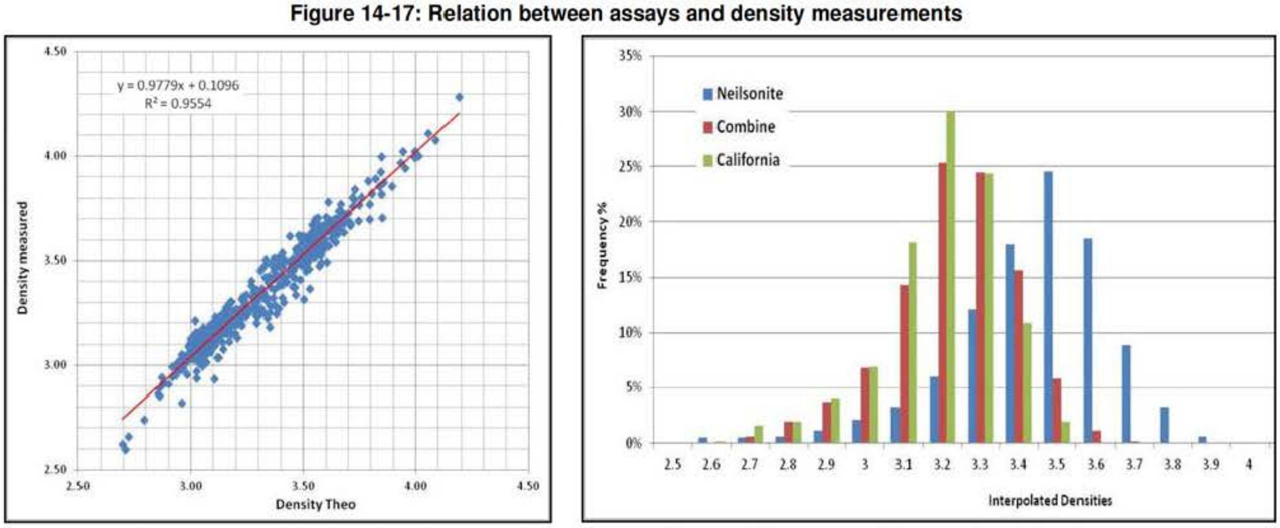


Figure 14-18: Calculated and measured density relations and Histogram of interpolated densities

14.7 Block Model

A block model was generated inside the limits stated in Table 14-3. The block model was rotated in order to follow the general orientation of the deposit. The blocks have a size of 10m along the long axis of the deposit, 5m along the short axis of the deposit and 5m in height. A total of 2,483,911 blocks were generated and then tagged differently depending in which zone they were, creating 4 sub-block models that are to be interpolated separately with different composite sets, ellipses and parameters. The four block models used in this project are:

1) California Zone;
2) Combine Zone;
3) Nelsonite Zone and
4) Waste, corresponding to the inter layers of waste between the zones and outside blocks.

The block model was then extracted using the overburden surface in order to remove any blocks above this surface. No block percentage were used and block tagging was done using the criteria of the block center point being inside or outside of the given envelop.

Table 14-3: Block model grid parameters

GRID	X	Y	Z
Origin	673000	5566000	-300
Size	10	5	5
Discretization	1	1	1
Starting Coordinates	673000	5566000	-300
Starting Indices	1	1	1
Ending Coordinates	686000	5574000	150
Ending Indices	1301	1601	91
***Rotation of 54° clockwise around Z axis**			

14.8 Block Model Interpolation and Classification

In order to interpolate the different block models, different sets of composites, envelops, ellipses and parameters were generated. This process enables to use the specific statistical properties of each zone during the interpolation process. The ellipses sizes were based on the continuities found in the variography study (Table 14-4). The mineralized zones were interpolated using kriging whereas the waste zone is interpolated using Inverse Square Distance methodology (Table 14-5). All of the zones were interpolated using 3 successive passes with more permissive criteria with each passes.

P_2O_5%, Cl% and K_2O% variables were interpolated using kriging whereas the other variables, including density were interpolated using the inverse square distance methodology for the three mineralized block models. All the variables of the waste block model were interpolated using the inverse square distance interpolation method.

On the total 2,859,094 blocks, P_2O_5% and density values were interpolated for 2,537,166 blocks. Cl% values were interpolated for 2,771,847 blocks; most missing values for Cl are found in the north of the "Wedge" area and would need to be re-assayed in order to be interpolated in the next resource estimation (Figure 14-19). K_2O% values were interpolated in 2,807,069 blocks whereas other chemical elements of the assay database were interpolated for 2,391,896 blocks.

During interpolation process, it was noted that a single hole from the database (1166-10-83) shows abnormal chlorine results (average of 0.1069% Cl) over 38.2m. This hole only accounts for an increase of 15% Cl of the California block model when tested by SGS Geostat. Hence, it would benefit Mine Arnaud to re-assay drill hole 1116-10-83 for chlorine values to verify these abnormal results.

Table 14-4: Ellipses parameters

Ellipses	Azimuth		Major Axis	Interm. Axis	Minor Axis
California 1	55°	20° Spin	125m	50m	10m
California 2	55°	20° Spin	200m	80m	15m
California 3	55°	20° Spin	400m	160m	30m
Combine 1	145°	-25° Dip	90m	50m	5m
Combine 2	145°	-25° Dip	150m	90m	10m
Combine 3	145°	-25° Dip	400m	250m	30m
Nelsonite 1	145°	-20° Dip	65m	55m	3m
Nelsonite 2	145°	-20° Dip	250m	125m	5m
Nelsonite 3	145°	-20° Dip	400m	200m	15m
Waste	145°	-20° Dip	300m	300m	50m

Table 14-5: Interpolation parameters

Zones	Passes	Method	Ellipses	Min Comp.	Max. Comp	Max Comp /DDH
California	1	Kriging	California 1	5	10	2
California	2	Kriging	California 2	4	10	3
California	3	Kriging	California 3	2	10	2
Combine	1	Kriging	Combine 1	5	10	2
Combine	2	Kriging	Combine 2	4	10	3
Combine	3	Kriging	Combine 3	2	10	2
Nelsonite	1	Kriging	Nelsonite 1	5	10	2
Nelsonite	2	Kriging	Nelsonite 2	4	10	3
Nelsonite	3	Kriging	Nelsonite 3	2	10	2
Waste	1	ISD	Waste	5	10	2
Waste	2	ISD	Waste	4	10	3
Waste	3	ISD	Waste	2	10	2

Using the continuities found in the variography study, parameters for the classification of the resource were established. The measured category was given for blocks inside envelops were the drilling grid spacing was of 50m and less. The indicated category was given to blocks were the drilling grid spacing was between 50m to 150m. Any blocks outside of these grids were classified in the inferred category. Furthermore, the blocks in the portion of the deposit where no Cl values were analyzed were downgraded to the indicated category even if the drilling grid spacing was sufficient for the measured category. For the total block model, 48% of the blocks are contained in the measured envelop, 35% in the indicated envelop and 17% in the inferred envelop.

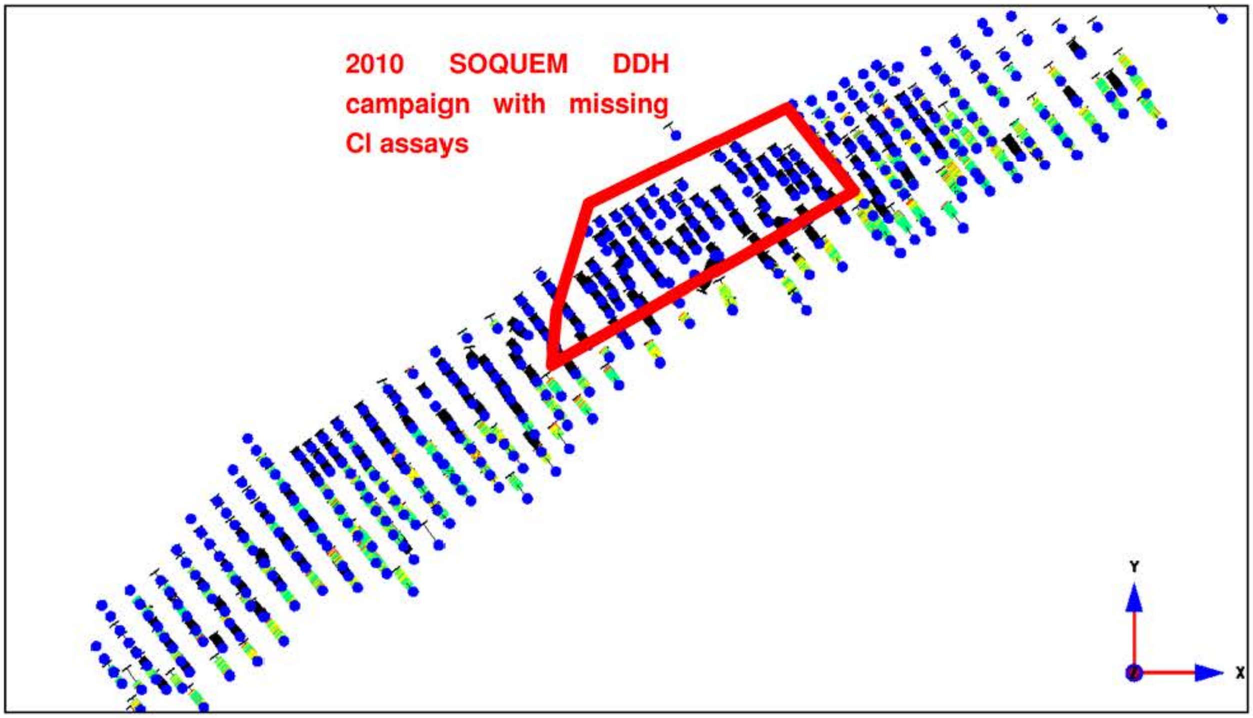


Figure 14-19: Missing Cl data in drill holes

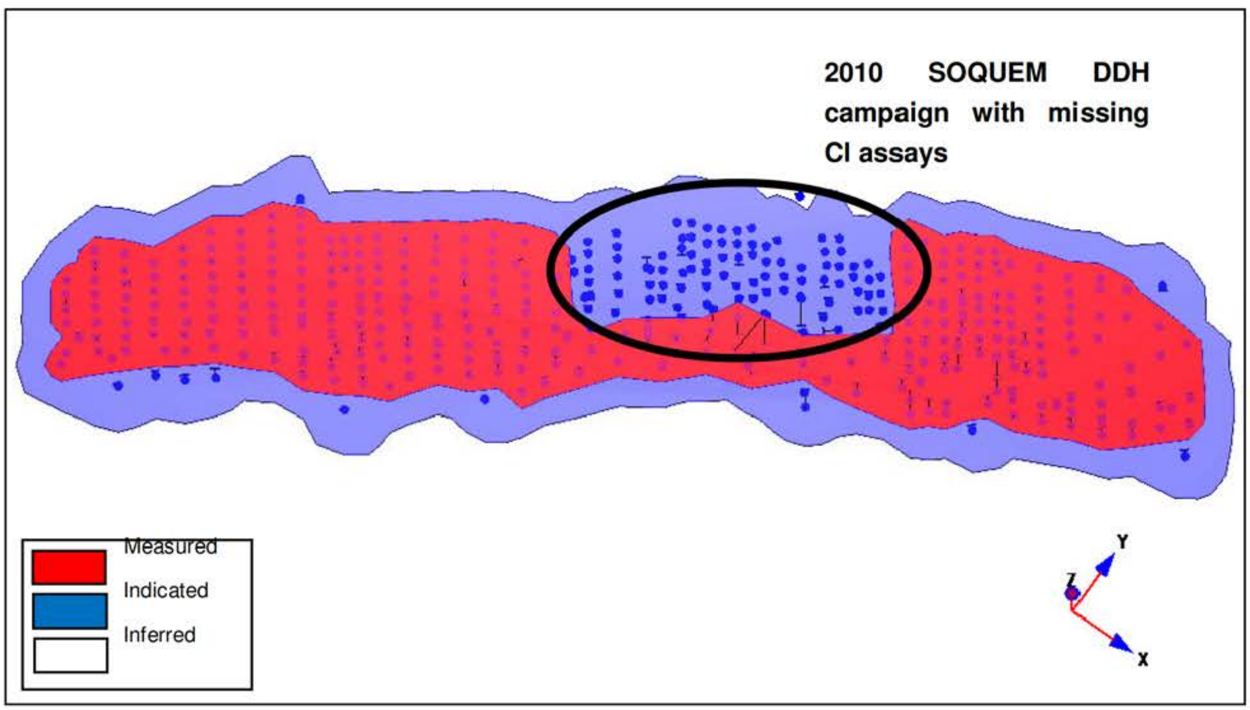


Figure 14-20: Classification envelops for the mineral resource

14.9 Metallurgical Parameters in Block Model

In order to proceed to pit optimization and resources reporting, certain metallurgical parameters had to be imported in the block model and calculated for each blocks from the interpolated data. Industrial minerals have to be evaluated according to the number of tonnes of concentrate produced from a block contained in the resources model. Hence, every block has to be given a Weight Recovery (WRec) and ultimately a monetary value (Bvalue) according to the number of saleable tonnes produced from this particular block. Furthermore, parameters corresponding to the quality of the concentrate produced should be integrated in the model. These values correspond to the P_2O_5 grade of the concentrate (% P_2O_5 Conc) and Cl grade of the concentrate (% Cl Conc).These parameters can then be used in the mine planning in order to determine the optimal mining sequence for producing the best quality concentrate over the life of the mine.

Metallurgical testing done in 2013 (see Section 13) have enabled SGS Geostat to estimate the WRec from the feed P_2O_5 grade. This relationship is based on the relation found in Table 14-6 and can be expressed has following:

$$\text{WRec\%} = 2.233 \times P_2O_5\% \text{ feed} + 0.426$$

The values for the quality of the concentrate are included in the block model but not used for pit optimization processes. The concentrate quality parameters for the moment comprise P_2O_5% Conc and Cl% Conc. These 2 values can also be calculated using modal and statistical tools leading to the final following equations (see section 12.4 for Cl prediction models):

$$P_2O_5\% \text{ Conc} = (P_2O_5\% \text{ feed} \times 90\% \text{ Rec}) / \text{WRec\%}$$

$$\text{Cl\% Conc} = 0.2608 \times K_2O\% \text{ feed} + 0.0503$$

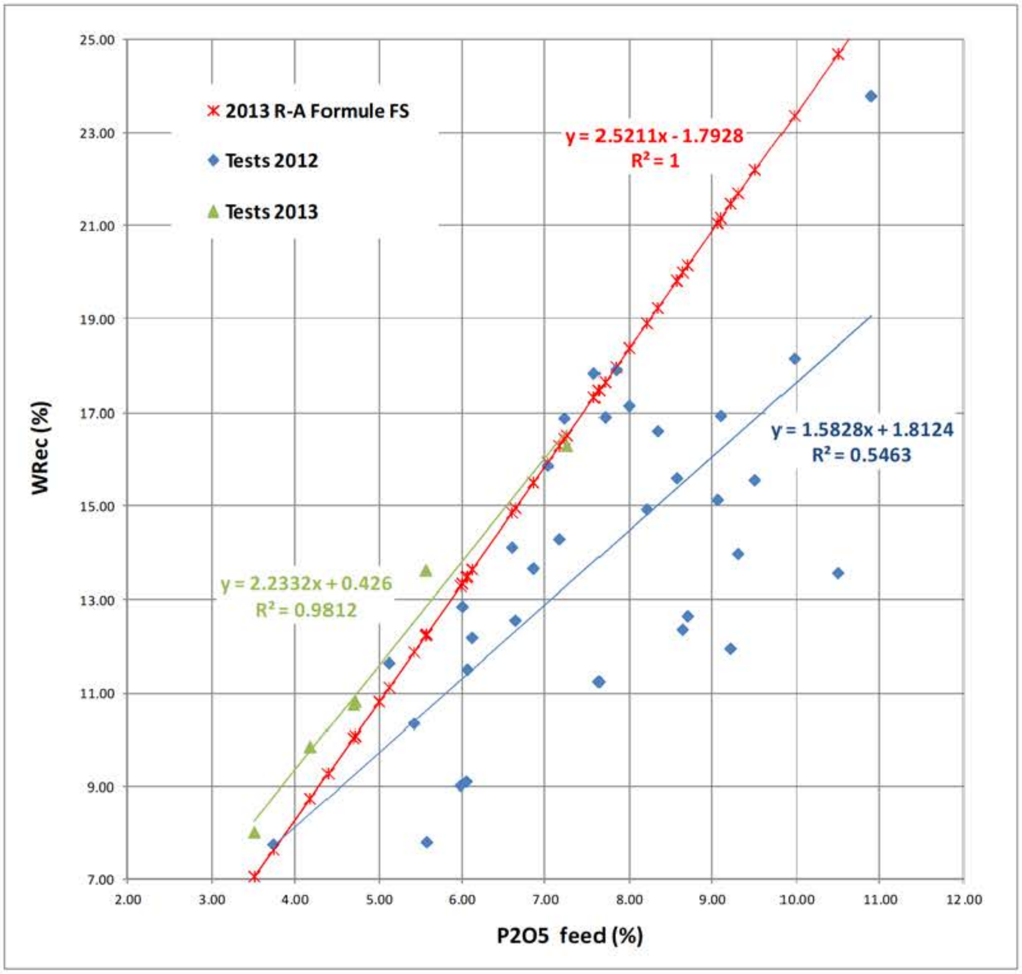


Figure 14-21: P2O5% feed and WRec% from different Metallurgical testing

14.10 Optimization Procedures and Parameters

An open pit optimization was done on the Sept-Îles deposit to test for the NI 43-101 requirements of *"reasonable prospect of economic extraction"* (CIM standards 2012) for resources reporting purposes. The parameters used for this pit optimization are summarized in the following Table 14-6.

This optimization outlines an open-pit shell that generates the maximum economic value. However, this value does not take into account mine planning and time value of money (discounting rate). It is for this reason that there are no guaranties that this shell shall be selected as the base case scenario to develop the mining scenario; thus, to calculate the in-pit reserves.

Table 14-6: Open-pit optimization parameters

Parameters	Units	Combine	Nelsonite
OVB Mining Cost	*Cdn$/tonne*		
Waste Mining Cost	*Cdn$/tonne*		
Mining Recovery	%	98.00	90.00
Mining Dilution	%	2.00	10.00
Vertical Mining Cost Increment	*$/10m*		
Concentrator	*Cdn$/treated*		
G&A Cost	*Cdn$/treated*		
Transport Cost	*Cdn$/treated*		
Ore Mining Increment	*Cdn$/treated*	-	-
Rehab Cost	*Cdn$/treated*		
Total Ore Based Cost	***Cdn$/treated***		
Processing Recovery	%	Variable	Variable
Moisture	%	1.00	1.00
Apatite Concentrate Selling Price	*$/wet tonne*		
Apatite Concentrate Selling Price	*$/dry tonne*		
Payable	%	99.00	99.00
Resulting Apatite Concentrate Selling Price	***$/dry tonne***		
Exchange Rate	*Cdn$: US$*	1:1	1:1

*It was discussed that Ore Mining Increment will be set to 0.00$/t instead of 0.35$/t as it was in the previous optimization

14.11 Mineral Resources

Following the request from Mine Arnaud, the block model was adjusted for the mineral resources report. Due to the low grade nature and high Cl content, the blocks belonging to the California zone and the blocks above the upper limit of the California zone were subtracted from the model and considered as waste material. A total of 149,045 blocks were removed from the mineral resources. **These blocks account for 116 Mt of material with an average grade of 1.74% P_2O_5 and cannot be added to the mineral resources, since they were not taken in account during the pit optimization process.**

The pit shell from the optimization was used to limit the extent of the mineral resources at depth (Figure 14-22). The mineral resources are stated at two different cut off grades, depending on the zones they are part of. The general cut off grade is of 1.76% P_2O_5 except for the blocks inside the Nelsonite envelop where a cut off grade of 1.91% P_2O_5 is used due to the higher dilution factor caused by the thinner layer of Nelsonite. The mineral resources are then reported following their classifications (Table 14-7) which are exclusive categories that cannot be added to one another.

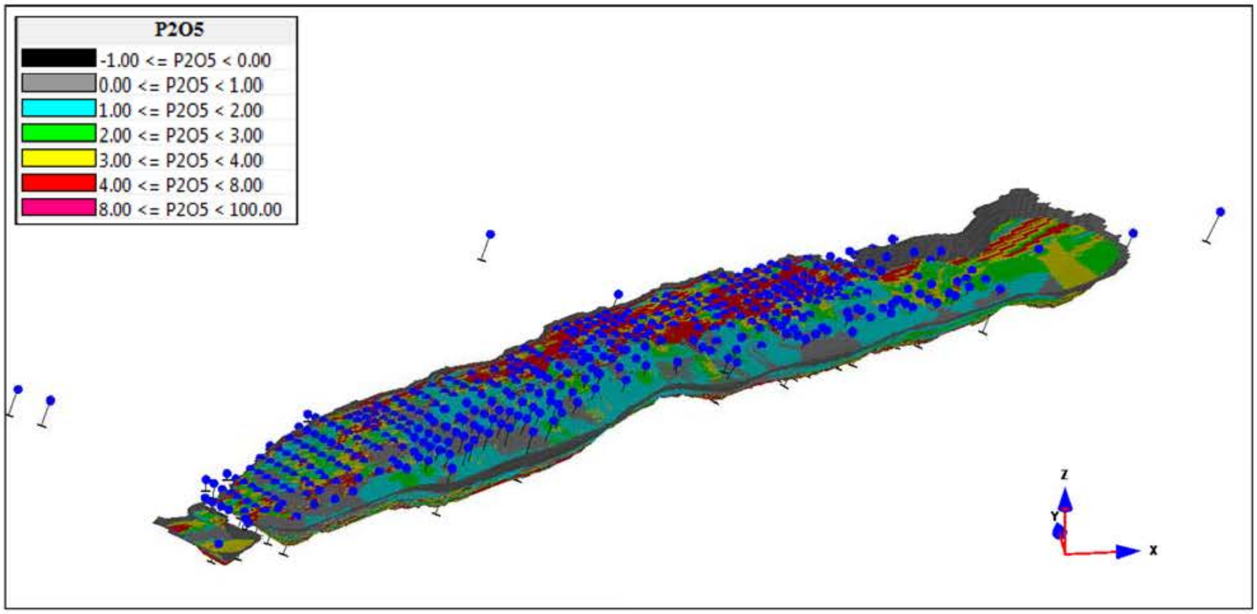


Figure 14-22: Mineral resources block model inside pit shell

The resources stated in the present report have to be treated carefully when reporting and using for further economical analysis. Assumptions regarding the WRec and Rec were made in order to proceed with pit optimization. The statistical relations used in order to determine block WRec and Rec still need to be verified by metallurgical testing. The following Mineral Resources do not represent Mineral Reserves because they have not yet demonstrated economic viability.

Table 14-7: Mineral Resources

Category	Cut Off (P2O5%)	Zones	Tonnage (Mt)	Average P2O5 (%)	Average WRec (%)	Average P2O5 Conc (%)	Average Cl Conc (%)
Measured	1.91	Nelsonite	38.48	5.91	13.62	38.98	0.082
Measured	1.76	Others	332.39	3.95	9.25	38.25	0.124
Measured		**TOTAL**	**370.87**	**4.16**	**9.70**	**38.69**	**0.106**
Indicated	1.91	Nelsonite	9.39	6.22	14.31	39.06	0.087
Indicated	1.76	Others	101.48	4.06	9.48	38.27	0.127
Indicated		**TOTAL**	**110.87**	**4.24**	**9.89**	**38.75**	**0.113**
Inferred	1.91	Nelsonite	-	-	-	-	-
Inferred	1.76	Others	42.76	3.52	8.29	38.11	0.158
Inferred		**TOTAL**	**42.76**	**3.52**	**8.29**	**38.11**	**0.158**

Notes: -The mineral resource estimate has been calculated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definitions Standards for mineral resources in concordance with National Instrument 43-101, Standards of Disclosure for Mineral Projects.

-Mineral resources which are not mineral reserves do not have demonstrated economic viability.

-Inferred mineral resources are exclusive of the Measured and Indicated resources.

-Resources are constrained by Pit Shell and under the bottom surface of the California zone.

-Effective date 10-07-13.

-Others are referring to the material categorize as Combine and Surrounding mineralization.

15. Mineral Reserve Estimate

The reserves derived from the detailed pit design have been estimated in accordance with the definitions and guidelines adopted by the Canadian Institute of Mining, Metallurgy, and Petroleum (CIM Standards on Mineral Resources and Reserves). The reserves are based entirely on measured and indicated resources and were converted as probable and proven reserves respectively. They are also limited to the final pit limit and above the marginal cut-off grade. The marginal cut-off grade (or milling or economical cut-off grade) is used to classify the material inside the pit limits as in-pit reserve or waste. Since the material is located inside the pit, the marginal cut-off grade excludes the mining cost and corresponds to the grade required to cover the costs of processing, G&A, and other costs related to transport. It should be noted that a small portion of the blocks that meet these criteria are nevertheless excluded. This exclusion (22.50 Mt) comes from blocks destined for the low grade stockpile, that when taken on their own (not diluted with the entire low grade stockpile), do not respect the chlorine limit of 0.14 %Cl. The low grade stockpile is an ore reserve having a grade above the marginal cut-off grade but below a fixed grade of 2.94 % P_2O_5. The use of this stockpile is to reserve a tonnage of material considered low grade that will be processed at the end of the open-pit operation.

The mineral reserves (with dilution and ore loss) are therefore equal to 324.4 Mt of ore at an average grade of 4.42 % P_2O_5 using cut-off grades related to the rocktype (1.76 %P_2O_5 and 1.91 %P_2O_5) and represents an operation of 29.4 years. The entire reserve comprises 33.42 million tonnes of apatite concentrate grading 39 %P_2O_5 and with a chlorine content of 0.106 %. Total waste, including rock, inferred resources and overburden, is 261.7 Mt; resulting in a waste to ore ratio of 0.81. The detailed mineral reserve estimate is shown in following Table 15-1.

Table 15-1: Mine Arnaud Project Reserves (presented as mill feed)

Material Type		Cut-off (%P2O5)	Tonnes*	Grade* (%P2O5)	Grade (%Wrec)	Concentrate tonnes	Chlorine** (%Cl)
Ore (Probable Reserves)	Combine	1.76%	54 400 000	4.67%	10.85%	5 900 000	0.112%
	Surrounding material	1.76%	1 900 000	2.39%	5.75%	110 000	0.136%
	Nelsonite	1.91%	8 100 000	5.79%	13.34%	1 080 000	0.081%
	Total		**64 400 000**	**4.74%**	**11.01%**	**7 090 000**	**0.108%**
Ore (Proven Reserves)	Combine	1.76%	227 000 000	4.26%	9.93%	22 540 000	0.110%
	Surrounding material	1.76%	7 000 000	2.50%	6.00%	420 000	0.109%
	Nelsonite	1.91%	26 000 000	5.62%	12.97%	3 370 000	0.073%
	Total		**260 000 000**	**4.34%**	**10.13%**	**26 330 000**	**0.105%**
Ore (Total Reserves)	Combine	1.76%	281 400 000	4.34%	10.11%	28 440 000	0.111%
	Surrounding material	1.76%	8 900 000	2.47%	5.95%	530 000	0.114%
	Nelsonite	1.91%	34 100 000	5.66%	13.06%	4 450 000	0.075%
	Total		**324 400 000**	**4.42%**	**10.30%**	**33 420 000**	**0.106%**

Note: **This reserve includes 2% dilution and 98% mining recovery for Railroad, Upper and Surrounding material ore types and a 10% dilution and 90% mining recovery for the Nelsonite ore type. Surrounding material is referring to mineralized material between zones but excluding the sterilized material as defined in Figure 16-2.**Chlorine grade estimation in concentrate, see section 12.4.*

16. Mining Method

16.1 Introduction

Taking into account the geometry and the depth of the mineralized zone, only open-pit mining has been considered in this study.

16.2 Open-Pit Mining

The near surface resources will be mined by a single large open pit, which will have 28 years of production following a three year construction and pre-production period. The mine plan is based on probable and proven reserves contained in the pit design, which was based on a Lerchs-Grossmann optimized pit shell. Open-pit mining will be conducted by the Project operator from the beginning to the end of the operation. Surface mining will follow the standard practices of an open-pit operation; with conventional drill and blast, load and haul cycle using a drill/truck/shovel mining fleet. The overburden and waste rock material will be hauled to the overburden and waste disposal areas near the pit. The run-of-mine mineralization will be delivered by large mining trucks to the gyratory crusher or stockpiled near the crushing plant.

16.3 Overall Pit-Slope Angle

The selected slope angles were based on a pit slope stability study completed by Ausenco-Vector[4]. The following Table summarizes the selected slope angles. The pit was divided into 6 geotechnical domains and slope attributes were provided for each of these sectors.

[4] Ausenco-Vector, Pit Slope Stability Arnaud Mine Feasibility Study, July 2011

Table 16-1: Recommended Bench Face and Inter-Ramp Angles of Rock Slope

Pit Sector	Average Dip Direction	Critical Failure Mode at 80% Reliability	Maximum Bench Face Angle (deg)	Maximum Inter-Ramp Angle for 10-m Benching[1] (deg)
1	145	Wedge or Planar	70	45
2	160	Wedge	70	45
3	260	Wedge	65	42
4	345	Wedge	60 With overburden sloped back at 21.8 (2.5H:1V)	39
5	330	Wedge	75	47
6	45	Wedge	75	47

1. Bench Widths = 6.5 m

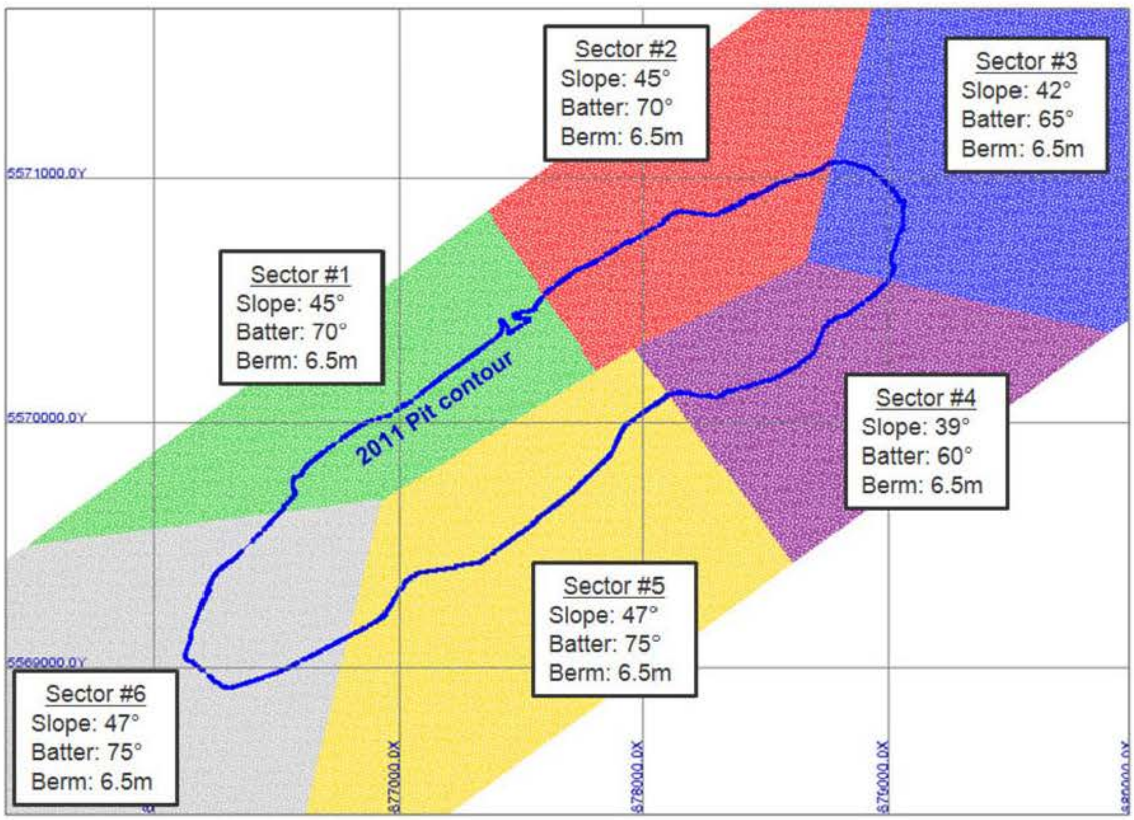


Figure 16-1: Geotechnical Domains

16.4 Pit Optimization Procedure and Parameters

In order to develop an optimal engineered pit design for the Arnaud deposit, an optimized pit shell was first prepared using the Lerchs-Grossman 3D routine in Gems Whittle ("LG 3D"). The basic optimization principle of the algorithm operates on a net value calculation for each block in the model; in other words revenue from sales minus total operating cost mining, processing plus general and administration costs.

In accordance with the guidelines of the NI 43-101 and the Canadian Institute of Mine Metallurgy and Petroleum Definition Standards for Mineral Resources and Mineral Reserves, only blocks classified in the measured and indicated categories are allowed to drive the pit optimizer for this level of study. Inferred resources were not taken into account during the optimization. Additionally, Mine Arnaud indicated that all material categorized in the California mineralization zone and above California upper limit is to be considered as waste material, even if the grade of P_2O_5 was above the cut-off grade. This directive was applied to limit the quantity of chlorine that might be sent to the processing facilities, considering that the California mineralization zone has an important percentage of this deleterious element. Refer to the next Figure for visualization of this constraint.

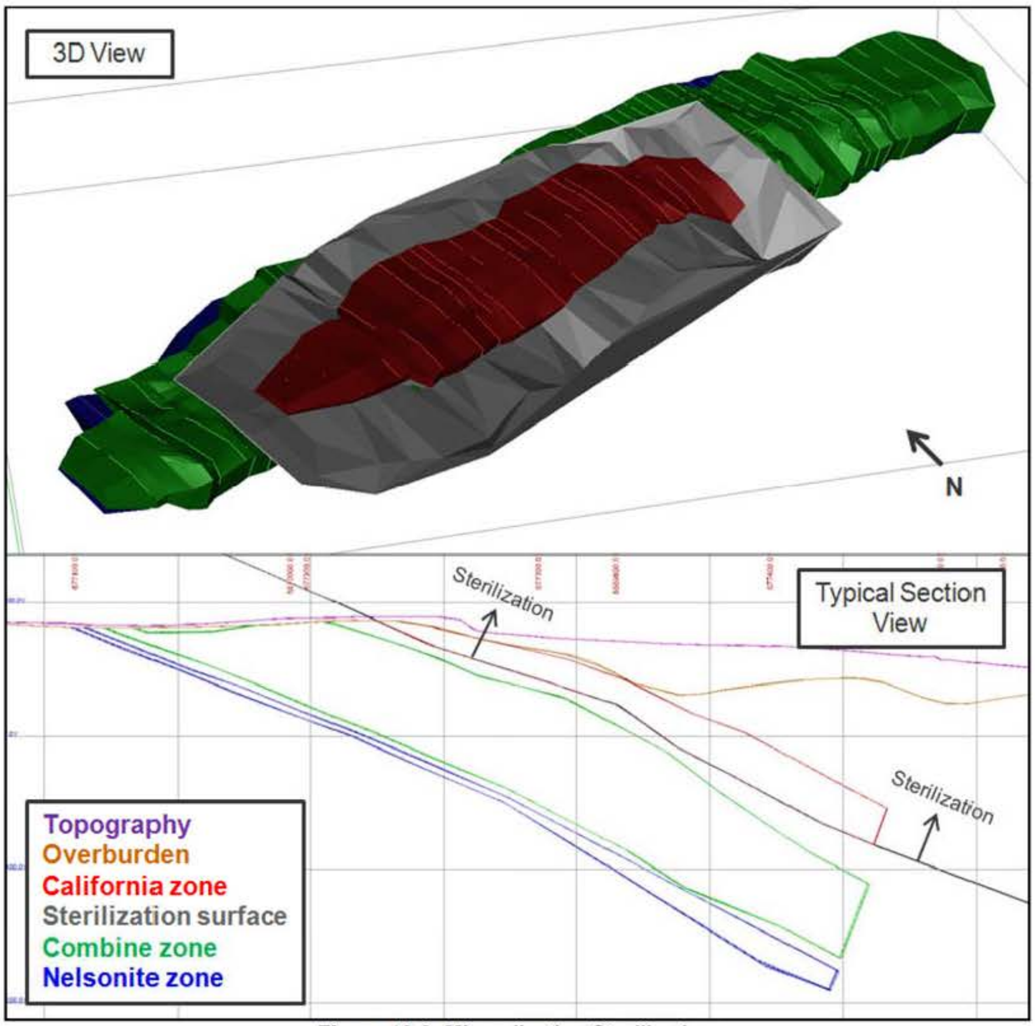


Figure 16-2: Mineralization Sterilization

For the initial optimization, the required parameters were selected by SGS and Mine Arnaud to evaluate the most economic open-pit profile. Although these parameters are not necessarily final, a reasonable degree of accuracy is required, since the analysis is an iterative process. The economic and operating parameters used in the initial optimization are given in the following Table 16-2.

Table 16-2: Open-pit Optimization Parameters

As per directive of Mine Arnaud, SGS limited the lateral extent of the pit optimization as constrained by a Hydro-Quebec power line and a stream.

Note: The economic parameters used at the time of the pit optimization are assumptions and do not corroborate the results stated in the economic model.

In order to simulate a realistic mine plan, SGS and Mine Arnaud used mining dilution and ore loss factors in the Project estimate. Dilution can be viewed as waste that is not segregated from the ore during mining, thus decreasing the grade of the ore and increasing the tonnage of mill feed. Ore losses are due to the inability of the mining method to accurately follow and segregate edges and small irregular offshoots from the main ore body. The selected mining dilution and ore losses are defined as:

Table 16-3: Mining dilution and ore losses

Ore type	Mining dilution (%)	Ore loss (%)	Mining recovery (%)
Combine	2.00	2.00	98.00
Surrounding	2.00	2.00	98.00
Nelsonite	10.00	10.00	90.00

16.5 Cut-Off Grades

The marginal cut-off grade or milling cut-off grade (CoG) is used to classify the material inside the pit limits as in-pit reserve or waste. Since the material is located inside the pit, the marginal cut-off grade excludes the mining cost and corresponds to the grade required to cover the costs of processing, G&A, and other costs related to transport. The marginal cut-off (based on weight recovery) is the grade where:

Total Ore Based Cost ($/t) x (1 + %Mining Dilution) = Concentrate value ($/t) x Weight Recovery (%)

The resulting CoG for all types of mineralized material is therefore equal to:

Table 16-4: Calculated CoG

*

16.6 Pit Optimization Results

Table 16-5: Optimization Results

Shell #	Revenue factor	Ore tonnes	Waste tonnes	Total tonnes	SR t:t	Grade %Wrec	Grade %P2O5	Concentrate tonnes
1	0.32	11 000	4 000	15 000	0.37	17.84	7.80	2 000
2	0.34	250 000	61 000	311 000	0.24	16.29	7.10	41 000
3	0.36	1 200 000	350 000	1 550 000	0.29	15.70	6.84	189 000
4	0.38	2 780 000	1 130 000	3 900 000	0.41	15.39	6.70	427 000
5	0.40	5 650 000	2 450 000	8 100 000	0.43	14.84	6.45	839 000
6	0.42	9 610 000	4 190 000	13 790 000	0.44	14.36	6.24	1 380 000
7	0.44	14 509 000	6 552 000	21 061 000	0.45	13.95	6.06	2 024 000
8	0.46	21 540 000	10 190 000	31 730 000	0.47	13.54	5.87	2 916 000
9	0.48	32 550 000	17 230 000	49 780 000	0.53	13.14	5.69	4 276 000
10	0.50	47 010 000	25 780 000	72 790 000	0.55	12.75	5.52	5 992 000
11	0.52	63 640 000	35 340 000	98 970 000	0.56	12.41	5.37	7 898 000
12	0.54	78 170 000	43 080 000	121 250 000	0.55	12.16	5.25	9 503 000
13	0.56	98 670 000	56 100 000	154 770 000	0.57	11.88	5.13	11 724 000
14	0.58	124 900 000	75 830 000	200 730 000	0.61	11.63	5.02	14 526 000
15	0.60	148 170 000	92 280 000	240 450 000	0.62	11.45	4.94	16 963 000
16	0.62	189 940 000	134 200 000	324 140 000	0.71	11.27	4.86	21 414 000
17	0.64	213 110 000	151 740 000	364 850 000	0.71	11.12	4.79	23 701 000
18	0.66	241 370 000	175 850 000	417 230 000	0.73	10.96	4.72	26 462 000
19	0.68	260 150 000	188 350 000	448 500 000	0.72	10.83	4.66	28 171 000
20	0.70	282 630 000	208 880 000	491 500 000	0.74	10.71	4.61	30 270 000
21	0.72	305 780 000	225 070 000	530 850 000	0.74	10.57	4.54	32 312 000
22	0.74	327 140 000	242 570 000	569 700 000	0.74	10.45	4.49	34 173 000
23	0.76	349 130 000	261 620 000	610 750 000	0.75	10.33	4.44	36 054 000
24	0.78	378 720 000	286 620 000	665 340 000	0.76	10.17	4.36	38 499 000
25	0.80	389 740 000	291 610 000	681 350 000	0.75	10.09	4.33	39 328 000
26	0.82	405 650 000	303 750 000	709 400 000	0.75	10.00	4.29	40 568 000
27	0.84	416 940 000	311 440 000	728 380 000	0.75	9.93	4.26	41 419 000
28	0.86	428 640 000	318 150 000	746 780 000	0.74	9.86	4.22	42 265 000
29	0.88	437 350 000	320 560 000	757 910 000	0.73	9.80	4.20	42 845 000
30	0.90	446 810 000	324 920 000	771 740 000	0.73	9.73	4.17	43 487 000
31	0.92	454 170 000	326 170 000	780 340 000	0.72	9.68	4.14	43 945 000
32	0.94	461 750 000	326 800 000	788 550 000	0.71	9.62	4.12	44 399 000
33	0.96	467 850 000	327 900 000	795 750 000	0.70	9.57	4.09	44 764 000
34	0.98	472 120 000	326 020 000	798 140 000	0.69	9.53	4.08	44 982 000
35	1.00	476 580 000	324 180 000	800 760 000	0.68	9.49	4.06	45 206 000

Note: The difference in the tonnage of shell #35 and the one presented as resource in Section 14 comes from the exclusion of inferred resource in the present exercise.

In order to select an optimal pit shell for developing a life-of-mine scenario, discounted cash flow analysis were performed taking into account the sequence of mining for all the nested pit shells (35 shells created previously) using a fixed concentrate value of Preliminary assumptions made in order to perform those analyses are:

Capital expenditure:

Mill throughput: Year 1: 8,400,000 tonnes

Year 2: 9,250,000 tonnes

Year 3: 10,640,000 tonnes

Year 4+: 11,200,000 tonnes

Discount rate: 5.00 %

Note: The economic parameters used at the time of the pit optimization do not necessarily confirm those stated in the economic model.

Three different mining scenarios were used for the analyses: best case, specified case, and worst case. The best case (mining by layers) and worst case (mining bench by bench) were combined to produce a realistic mining scenario (specified case). This realistic case assumes four mining phases (using shells 7, 10, 13 and 17 for pushbacks). The results of the optimization are summarized below in the next Figure and in the next Table.

Table 16-6: Optimization results (preliminary LOM)

Shell #	NPV - Best M$ disc.	NPV - Spec. M$ disc.	NPV - Worst M$ disc.	Ore Mtonnes	Waste Mtonnes	Total Mtonnes	SR t:t	Grade %Wrec	Grade %P2O5	Concentrate Mtonnes	Mine Life years	IRR - Best %	IRR - Spec. %	IRR - Worst %
	-													

Note: These economic results do not necessarily corroborate those stated in the final economic model because the optimization stage is iterative and used solely to select a scenario.

Shell #35 is the optimal shell as defined previously. All other shells (1 to 34) are smaller in size and are used to evaluate the net present value of the project if mining would stop at this specific shell rather than at the optimal one.

Important note:

This pit optimization was performed based on updated parameters developed by SGS and Mine Arnaud. A previous pit optimization exercise (May 2013) was performed using the same parameters, but with a more conservative weight recovery formula (in others words, less concentrate produced for the same amount of mill feed). This older pit optimization was followed by a base case shell selection and a detailed pit design.

SGS and Mine Arnaud compared the results of this new optimization (July 2013) with the older one (May 2013), and judged that the results are sufficiently similar to continue using the previous optimal pit shell and therefore, the existing detailed pit design (May 2013).

The older pit optimization (May 2013) was developed using the same approach than the one presented above. Based on the older results, SGS and Mine Arnaud selected a specified shell as reference one to develop the pit design. This choice was made in order to limit the size of the

Project, and to preserve a 25+ years mine life; keeping the possibility of a future mine expansion through pushback. Also, as is it the case in the updated optimization, it was notable that the Project NPV, based on specified mining scenario (green curve), was stabilizing between shells #20 to #30. The previous (older) selected shell is shown in the next Figure.

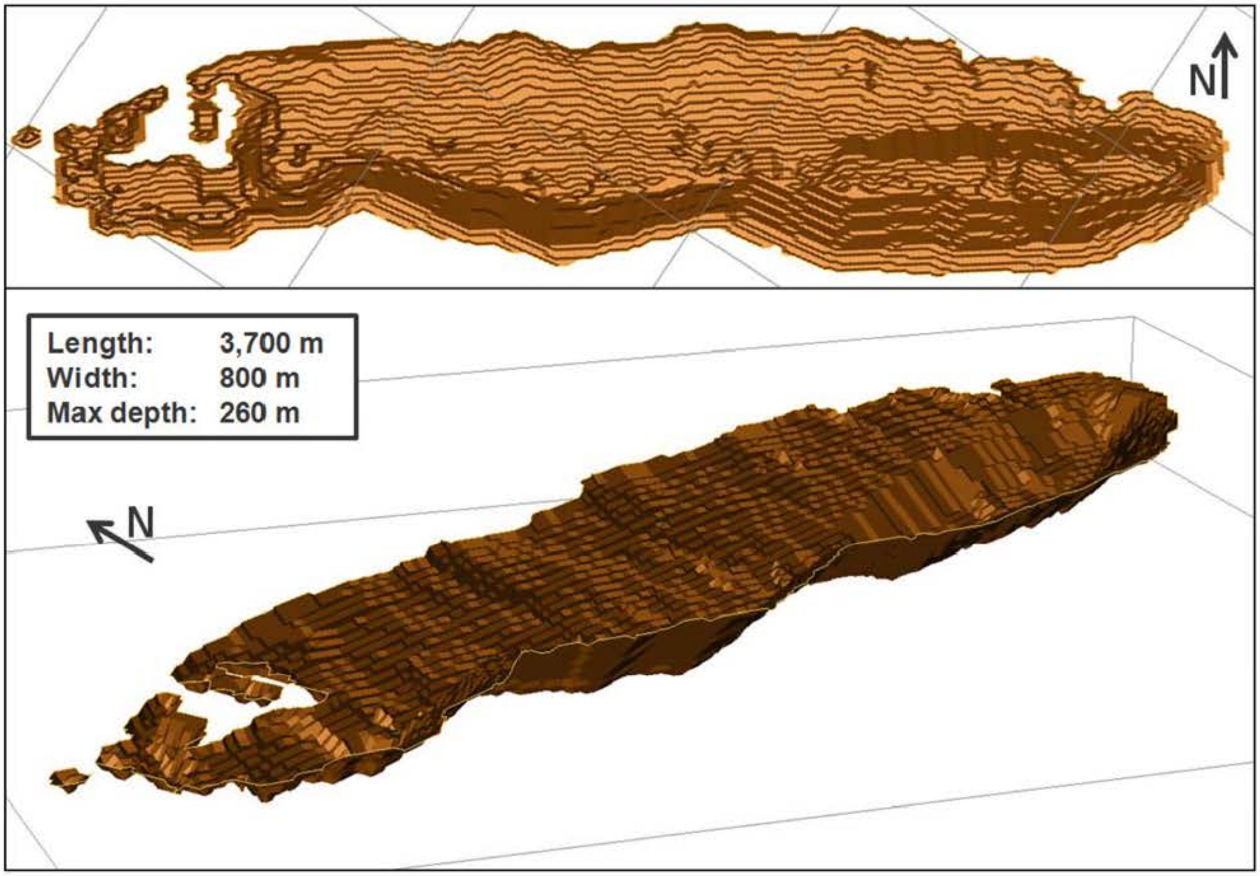


Figure 16-4: Selected Optimized Shell

16.7 Ultimate Pit

16.7.1 Pit Design Parameters

Using the base case optimized pit shell as reference, an open-pit including a ramp and safety berms was designed to develop a realistic mining scenario. The new designed pit will account for the additional waste material coming from the addition of a ramp to the base case shell. The design parameters used are defined as:

Overall slope angle:	Variable (refer to Section 16.3)
Face angle:	Variable (refer to Section 16.3)
Bench height:	5.0 m (footwall) and 10.0 m (center of the deposit)
Safety berm:	6.5 m width (1 safety berm for each 10 m vertical interval)
Ramp grade:	12.0 % (single lane) and 10.0 % (double lane)
Ramp width:	18.7 m (single lane) and 25.4 m (double lane), see following figure

The ramp width supposes that the Project operator will be using CAT 785 hauling truck. The ramp dimensions are also based on Quebec regulation[5] such as:

"*45.1. In addition to the standards prescribed in section 45, haulage roads:*

(2) constructed in an open-pit mine at which operations begins on or after 1 April 1993 and used by motorized vehicles shall have a width at least:

(a) 2 times the width of the widest vehicles if they are single-track roads;

(b) 3 times the width of the vehicles if they are 2-way roads."

"*45.2. Service roads used by motorized vehicles in an open-pit mine shall:*

(1) Be edged by a pile of fill or a ridge where vehicles could fall more than 3 m (9.8 ft). The pile of fill or the ridge shall have a height equal to at least the radius of the largest wheel of any vehicle travelling on the road;

(2) Be off-limits to any vehicle whose width exceeds that of the driving surface;

(3) Be maintained by clearing or scarifying or by spreading an abrasive substance, so as to keep a non-skid surface."

[5] Quebec, Regulation respecting occupational health and safety in mines, June 1st 2013

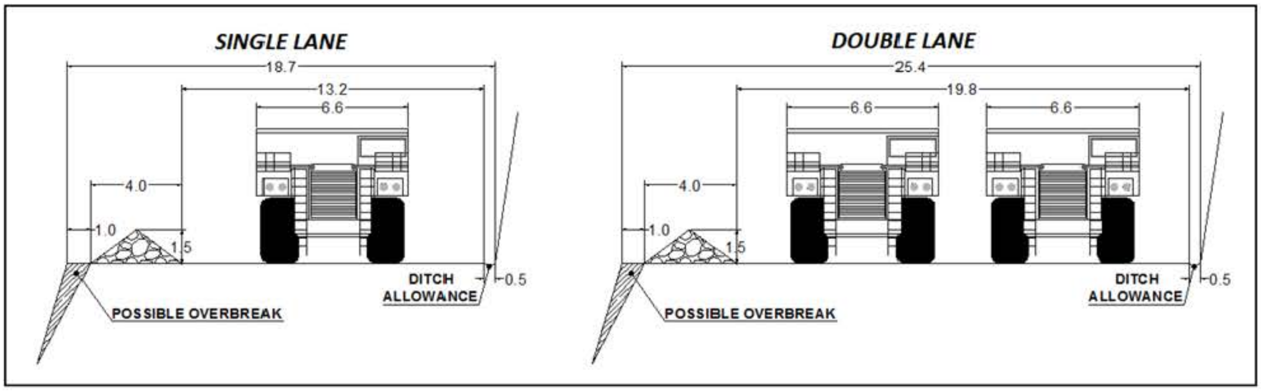


Figure 16-5: Hauling Ramps Specifications

16.7.2 Ultimate Pit Design

The next Figure shows views of the designed open-pit with his dimensions.

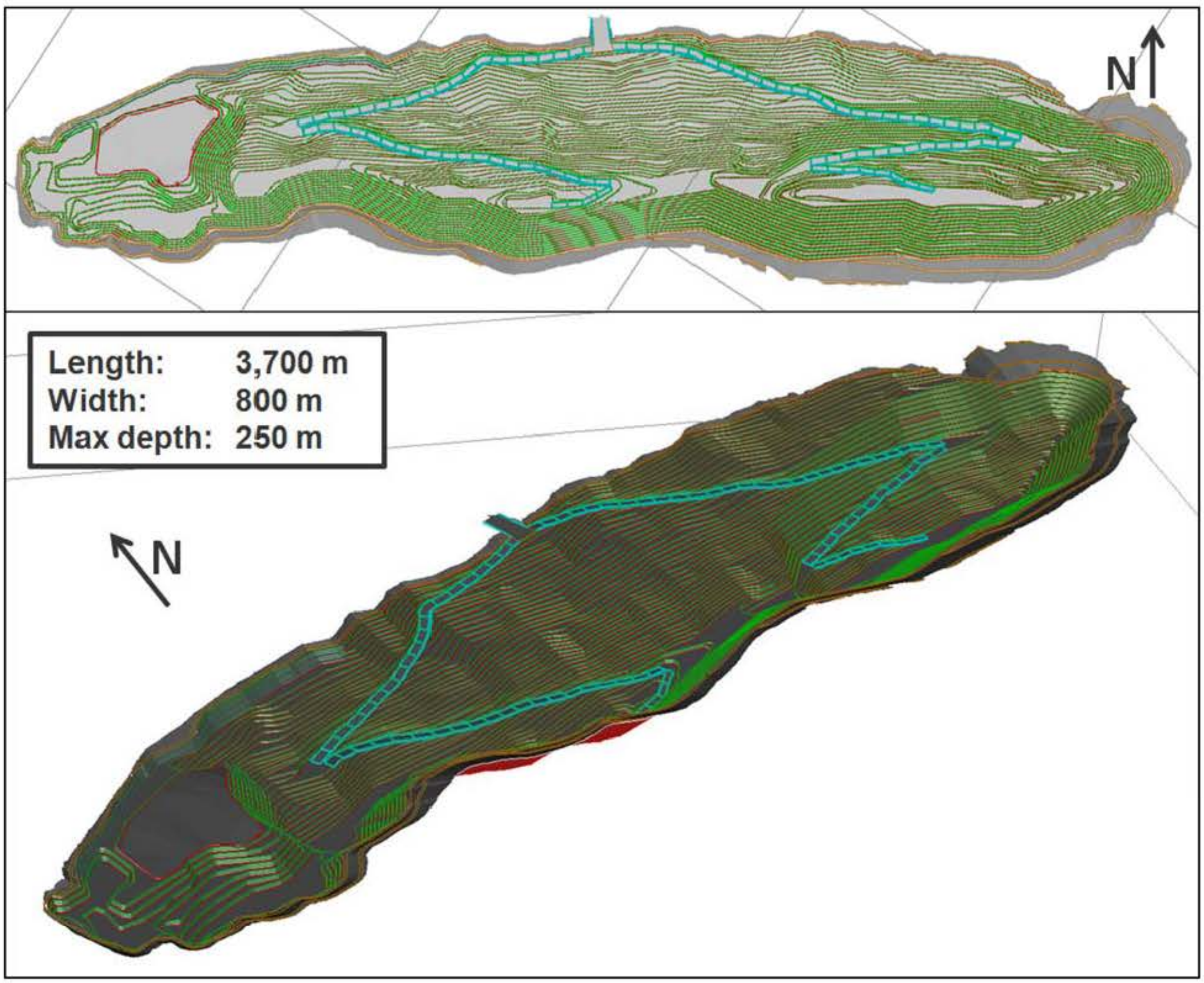


Figure 16-6: Final Pit Design

16.7.3 Mineralization Contained within Pit Design

The tonnage contained inside the pit design is presented in the next Table.

Table 16-7: In-Pit Material

Material Type		Cut-off (%P2O5)	Tonnes	Grade (%P2O5)	Grade (%Wrec)	Concentrate tonnes	Chlorine (%Cl)
Ore (Probable Reserves)	Combine	1.76%	54 400 000	4.67%	10.85%	5 900 000	0.112%
	Surrounding material	1.76%	1 900 000	2.39%	5.75%	110 000	0.136%
	Nelsonite	1.91%	8 100 000	5.79%	13.34%	1 080 000	0.081%
	Total		**64 400 000**	**4.74%**	**11.01%**	**7 090 000**	**0.108%**
Ore (Proven Reserves)	Combine	1.76%	227 000 000	4.26%	9.93%	22 540 000	0.110%
	Surrounding material	1.76%	7 000 000	2.50%	6.00%	420 000	0.109%
	Nelsonite	1.91%	26 000 000	5.62%	12.97%	3 370 000	0.073%
	Total		**260 000 000**	**4.34%**	**10.13%**	**26 330 000**	**0.105%**
Ore (Total Reserves)	Combine	1.76%	281 400 000	4.34%	10.11%	28 440 000	0.111%
	Surrounding material	1.76%	8 900 000	2.47%	5.95%	530 000	0.114%
	Nelsonite	1.91%	34 100 000	5.66%	13.06%	4 450 000	0.075%
	Total		**324 400 000**	**4.42%**	**10.30%**	**33 420 000**	**0.106%**
Waste	Waste rock (*)		**22 500 000**	2.59%	6.22%		0.176%
	Waste rock		**174 700 000**				
	Overburden		64 500 000				
In-pit Total	**All**		**586 100 000**				

**Material is above the P_2O_5 cut-off grade, but does not respect the Chlorine (Cl) upper limit of 0.14%.*

16.8 Mine Development and Production Schedule

The mine development plan includes a number of push-backs, or phases, designed to meet the following objectives:

- Enable the mining of high grade mineralization as early as possible;
- Effectively reduce stripping ratio in the initial mining stage;
- Balance the stripping ratio over the period of the mine life;
- Maintain a minimum mining width between two working phases.

16.8.1 Pushback Width

In order to have a safe operation, a minimum mining width has to be respected when introducing a pushback into an operating pit. An appropriate mining width was determined based on:

- Komatsu PC3000-6 excavator and CAT 785 mining truck;
- 20 m allowance for loader movement;
- 20 m haul road width.

The next Figure illustrates the proposed pushback width to be used in the design of the phase development:

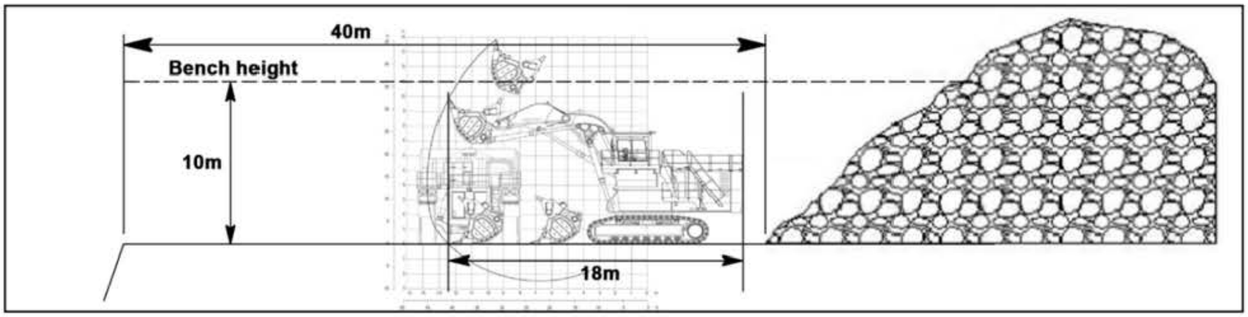


Figure 16-7: Minimum Pushback Width

16.8.2 Mine Development

Four minable phases are proposed to develop the ultimate pit. Each phase, or pushback, is designed with a minimum mining width of 40 m to accommodate the mining equipment that will operate on each working bench.

Phase/Pushback #1

At the beginning of the Project, the mining activities will be concentrated around phase #1 since the shell defined by this phase gives a high achievable grade near surface and at a low waste-to-ore stripping ratio. Prioritizing the mining in this section of the deposit will maximize revenue at the beginning of the Project, thus maximizing the net present value (NPV).

Phase/Pushback #2

Phase #2 is essentially an expansion of Phase #1. A constant difference of 40 meters has been kept during development of the mining plan to limit the number of benches mined simultaneously. This constraint also has the effect of minimizing the variation of stripping ratios from year to year.

Phase/Pushback #3

Phase #3 is essentially an expansion of Phase #2. A constant difference of 40 meters has been kept during development of the mining plan to limit the number of benches mined simultaneously. This constraint also has the effect of minimizing the variation of stripping ratios from year to year.

Phase/Pushback #4 (Optimal Pit Design)

Phase #4 includes the mining of the rest of the mineralized material. The next Figure schematizes these mining phases.

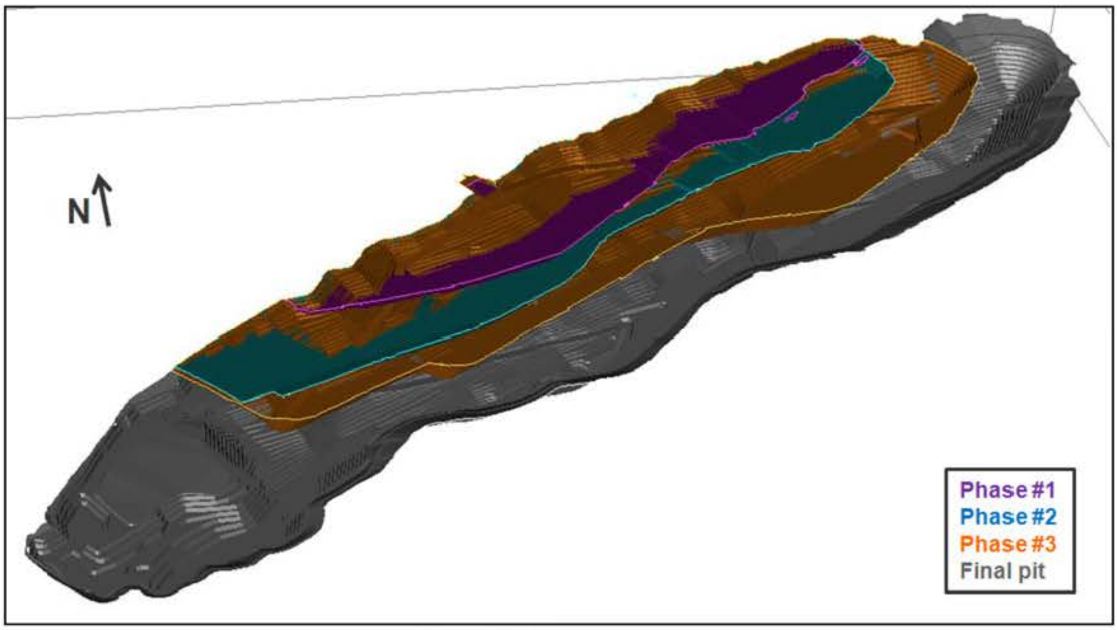


Figure 16-8: Mining Phases

16.8.3 Stockpiles

Two low grade ore stockpiles will be located near the crushing plant. The main use of these stockpiles is to reserve a tonnage of material considered low grade that will be processed at the end of the open-pit operation. This exercise is common in mining operations when the Operator desires to maximize the Project economics and thus minimize the payback period. The use of two stockpiles is planned due to the fact that the overall grade of the low grade ore that will be sent to the piles will have an overall estimated concentrate chlorine content of 0.1473 %, which is just above the limit 0.1400 %. The separation criterion between each stockpile is therefore the estimated concentrate chlorine limit of 0.1400 %. In other words, all the material going to the stockpiles with an estimated concentrate chlorine grade below 0.1400 % will go on a given stockpile, while the remainder will report the other one. At this stage, and in this PFS, only material with estimated concentrate chlorine content below of 0.1400 % is intented to be processed. The high chlorine stockpile was kept in the plan because the current method used for the estimation of chlorine concentrations is considered rather conservative with respect to the actual chlorine concentration. It is possible that this stockpile will eventually be amenable to processing.

16.8.4 Production Schedule

A mine production schedule was prepared for the development and the operation of the Project. In order to develop this mine plan, the following assumptions were made:

- The mine plan was optimized only based on %P2O5 and %Cl (all others elements, such as Fe, Al, etc., were not considered at this stage);
- The maximum chlorine content of the yearly concentrate production should be lower that 0.1400 %;
- Mill throughput: Year 1: 8,400,000 tonnes (75 % of full capacity)
  Year 2: 9,250,000 tonnes (85 % of full capacity)
  Year 3: 10,640,000 tonnes (95 % of full capacity
  Year 4+: 11,200,000 tonnes (100 % of full capacity)
- Highgrading at 2.94 %P_2O_5 was done during years 5 to 20 in order to maximize the Project NPV and IRR;
- Consequently, the use of stockpiles was allowed (without any size limitation);
- The three interim mining phases were used as pushbacks;
- A maximum fixed lead of 20 vertically meters was allowed between mining phases.

The results of the developed schedule are summarized in the next Table and presented in the following Figures. The key findings of the proposed mine plan include:

- Potential Project life of approximately 30 years;
- Potential mill feed over the Project life of 324.4 Mt at 4.42 %P2O5;
- An overall Project stripping ratio of 0.74;
- Potential concentrate production of 33.4 Mt;
- Peak in mining the production at year 8 to 14;
- A well distributed stripping ratio.

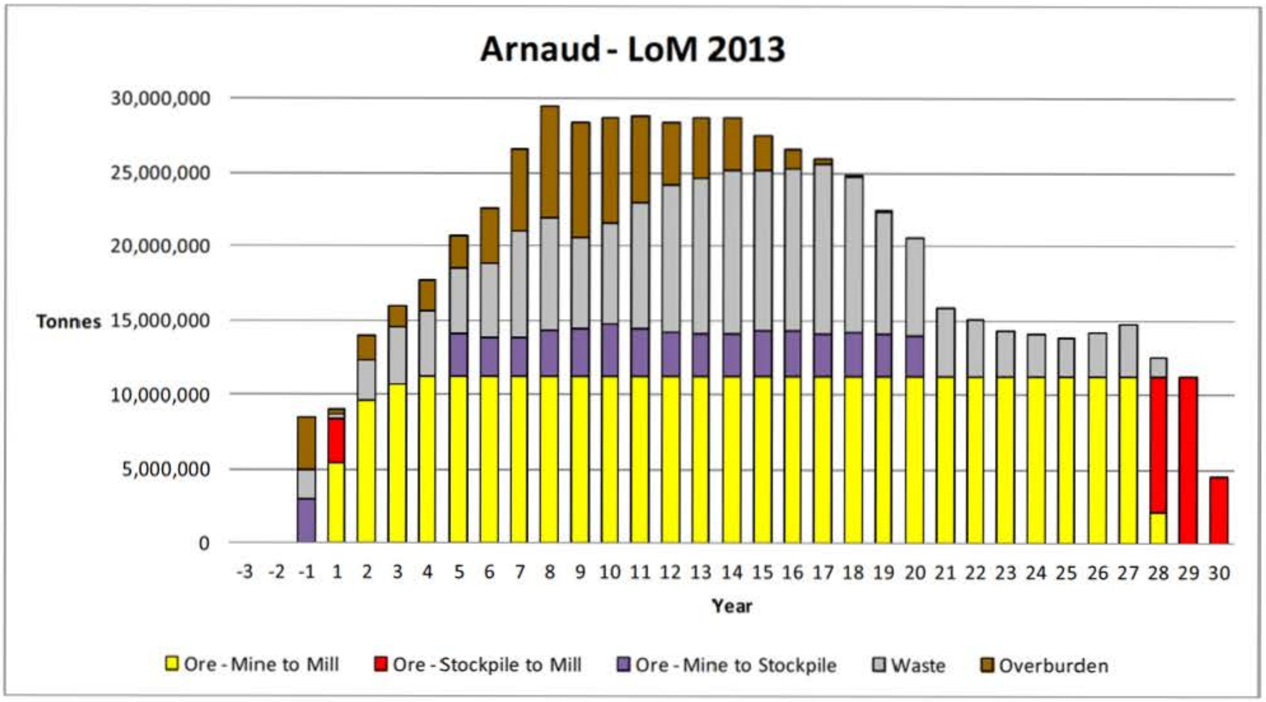


Figure 16-9: Production Schedule

Table 16-8: Production Schedule

MINING

Period (year)	Ore Mine to Mill (tonnes)	Weight Recovery (%Wrec)	Input Grade (%P2O5)	Simulated Output (%Cl)	Ore Mine to Pile (tonnes)	Weight Recovery (%Wrec)	Input Grade (%P2O5)	Simulated Output (%Cl)	Ore Pile to Mill (tonnes)	Weight Recovery (%Wrec)	Input Grade (%P2O5)	Simulated Output (%Cl)
-3												
-2												
-1					3,000,000	11.15%	4.80%	0.0967%				
1	5,399,165	11.15%	4.80%	0.0967%					3,000,000	11.15%	4.80%	0.0967%
2	9,521,152	10.82%	4.66%	0.1043%								
3	10,639,980	10.85%	4.67%	0.1038%								
4	11,200,327	10.47%	4.50%	0.1052%								
5	11,199,806	10.94%	4.71%	0.1082%	2,863,527	5.73%	2.37%	0.1509%				
6	11,200,599	10.77%	4.63%	0.1074%	2,666,485	5.69%	2.36%	0.1441%				
7	11,202,595	11.13%	4.79%	0.1022%	2,675,773	5.64%	2.33%	0.1314%				
8	11,201,529	10.99%	4.73%	0.1045%	3,059,077	5.65%	2.34%	0.1434%				
9	11,200,073	10.56%	4.54%	0.1105%	3,151,675	5.74%	2.38%	0.1462%				
10	11,199,743	10.28%	4.41%	0.1149%	3,483,554	5.68%	2.35%	0.1485%				
11	11,200,138	10.46%	4.49%	0.1132%	3,181,087	5.63%	2.33%	0.1490%				
12	11,199,114	10.71%	4.61%	0.1108%	2,971,987	5.65%	2.34%	0.1444%				
13	11,200,742	10.80%	4.65%	0.1093%	2,869,878	5.67%	2.35%	0.1422%				
14	11,200,083	10.79%	4.64%	0.1075%	2,858,321	5.72%	2.37%	0.1406%				
15	11,200,281	10.69%	4.59%	0.1079%	3,083,210	5.73%	2.37%	0.1415%				
16	11,199,554	10.67%	4.59%	0.1083%	3,110,269	5.72%	2.37%	0.1460%				
17	11,200,396	10.38%	4.46%	0.1113%	2,886,890	5.73%	2.38%	0.1502%				
18	11,199,798	10.30%	4.42%	0.1120%	2,961,672	5.66%	2.35%	0.1561%				
19	11,199,962	10.62%	4.56%	0.1079%	2,893,010	5.55%	2.29%	0.1603%				
20	11,199,659	10.91%	4.70%	0.1036%	2,724,741	5.51%	2.28%	0.1624%				
21	11,200,397	9.89%	4.24%	0.1088%								
22	11,199,478	9.82%	4.21%	0.1109%								
23	11,200,734	10.04%	4.30%	0.1087%								
24	11,199,926	10.61%	4.56%	0.1014%								
25	11,199,879	10.92%	4.70%	0.0970%								
26	11,199,917	11.17%	4.81%	0.0925%								
27	11,200,001	11.53%	4.97%	0.0840%								
28	2,127,116	11.62%	5.01%	0.0732%					9,072,884	5.77%	2.39%	0.1136%
29									11,200,000	5.77%	2.39%	0.1136%
30									4,670,483	5.77%	2.39%	0.1136%
Total	296,492,145	10.67%	4.59%	0.1054%	50,441,157	6.00%	2.49%	0.1417%	27,943,367	5.77%	2.39%	0.1136%

Period (year)	Waste (tonnes)	Overburden (tonnes)	Total mined from open- (tonnes)	Total moved (tonnes)	Stripping ratio (t:t)
-3					
-2					
-1	2,000,000	3,494,360	8,494,360	8,494,360	2.00
1	322,074	286,804	6,008,044	9,008,044	0.08
2	2,833,605	1,602,375	13,957,131	13,957,131	0.49
3	3,894,733	1,368,836	15,903,549	15,903,549	0.51
4	4,455,792	2,037,303	17,693,421	17,693,421	0.61
5	4,419,930	2,178,167	20,661,430	20,661,430	0.87
6	4,921,940	3,729,711	22,518,736	22,518,736	1.06
7	7,127,461	5,568,576	26,574,405	26,574,405	1.44
8	7,575,501	7,661,372	29,497,478	29,497,478	1.73
9	6,244,336	7,749,969	28,346,054	28,346,054	1.63
10	6,820,192	7,218,867	28,722,356	28,722,356	1.66
11	8,558,492	5,812,010	28,751,727	28,751,727	1.64
12	10,008,107	4,185,286	28,364,494	28,364,494	1.59
13	10,542,352	4,075,767	28,688,739	28,688,739	1.61
14	11,145,426	3,433,509	28,637,340	28,637,340	1.60
15	10,913,695	2,277,843	27,475,029	27,475,029	1.48
16	10,937,153	1,383,099	26,630,075	26,630,075	1.40
17	11,530,315	368,511	25,986,113	25,986,113	1.32
18	10,598,554	39,122	24,799,145	24,799,145	1.21
19	8,187,813	1,534	22,282,320	22,282,320	0.99
20	6,602,244		20,526,644	20,526,644	0.83
21	4,622,780		15,823,177	15,823,177	0.41
22	3,870,341		15,069,820	15,069,820	0.35
23	3,107,095		14,307,829	14,307,829	0.28
24	2,863,355		14,063,281	14,063,281	0.26
25	2,672,749		13,872,627	13,872,627	0.24
26	2,975,078		14,174,995	14,174,995	0.27
27	3,569,693		14,769,695	14,769,695	0.32
28	1,375,375		3,502,491	12,575,375	0.12
29				11,200,000	
30				4,670,483	
Total	174,696,182	64,473,022	586,102,505	614,045,872	0.74

PROCESSING

Period (year)	Ore treated (tonnes)	Weight Recovery (%Wrec)	Input Grade (%P2O5)	Concentrate production (tonnes)	Output Grade (%Cl)
-3					
-2					
-1					
1	8,399,165	11.15%	4.80%	936,619	0.0967%
2	9,521,152	10.82%	4.66%	1,030,342	0.1043%
3	10,639,980	10.85%	4.67%	1,154,533	0.1038%
4	11,200,327	10.47%	4.50%	1,172,887	0.1052%
5	11,199,806	10.94%	4.71%	1,224,790	0.1082%
6	11,200,599	10.77%	4.63%	1,206,779	0.1074%
7	11,202,595	11.13%	4.79%	1,247,182	0.1022%
8	11,201,529	10.99%	4.73%	1,231,286	0.1045%
9	11,200,073	10.56%	4.54%	1,183,281	0.1105%
10	11,199,743	10.28%	4.41%	1,150,941	0.1149%
11	11,200,138	10.46%	4.49%	1,171,654	0.1132%
12	11,199,114	10.71%	4.61%	1,199,311	0.1108%
13	11,200,742	10.80%	4.65%	1,210,101	0.1093%
14	11,200,083	10.79%	4.64%	1,208,538	0.1075%
15	11,200,281	10.69%	4.59%	1,196,887	0.1079%
16	11,199,554	10.67%	4.59%	1,194,676	0.1083%
17	11,200,396	10.38%	4.46%	1,162,469	0.1113%
18	11,199,798	10.30%	4.42%	1,153,179	0.1120%
19	11,199,962	10.62%	4.56%	1,189,202	0.1079%
20	11,199,659	10.91%	4.70%	1,222,175	0.1036%
21	11,200,397	9.89%	4.24%	1,108,092	0.1088%
22	11,199,478	9.82%	4.21%	1,099,470	0.1109%
23	11,200,734	10.04%	4.30%	1,124,078	0.1087%
24	11,199,926	10.61%	4.56%	1,188,718	0.1014%
25	11,199,879	10.92%	4.70%	1,223,018	0.0970%
26	11,199,917	11.17%	4.81%	1,251,545	0.0925%
27	11,200,001	11.53%	4.97%	1,291,911	0.0840%
28	11,200,000	6.88%	2.89%	770,815	0.1006%
29	11,200,000	5.77%	2.39%	646,332	0.1136%
30	4,670,483	5.77%	2.39%	269,525	0.1136%
Total	324,435,512	10.30%	4.42%	33,420,335	0.1057%

Important note:

The Stripping Ratio (SR) of 0.74 is attributable to the entire Project. The SR related to the production period, ie. Open-pit mining (years 1 to 28) is equal to 1.02. This last value supposes that the low grade material is considered as waste.

SGS assumed that the diluted material will have a grade of 0.00 $\%P_2O_5$.

SGS assumed that the diluted material will have the same %Cl that the ore blocks where the dilution is applied.

Considering that the overall stockpile estimated concentrate chlorine content was just over the limit of 0.1400 %Cl (approximately 0.147 %Cl without considering year -1), SGS assumed that only the material with a %Cl lower than 0.14 % will be processed. It is for this reason that only 54% of the stockpile tonnage of 50.4 Mtonnes is planned to be sent to the processing plant.

The reason why the remaining portion of the stockpile will eventually be sent to the mill depends on the actual chlorine concentrations that meet Yara's criteria.

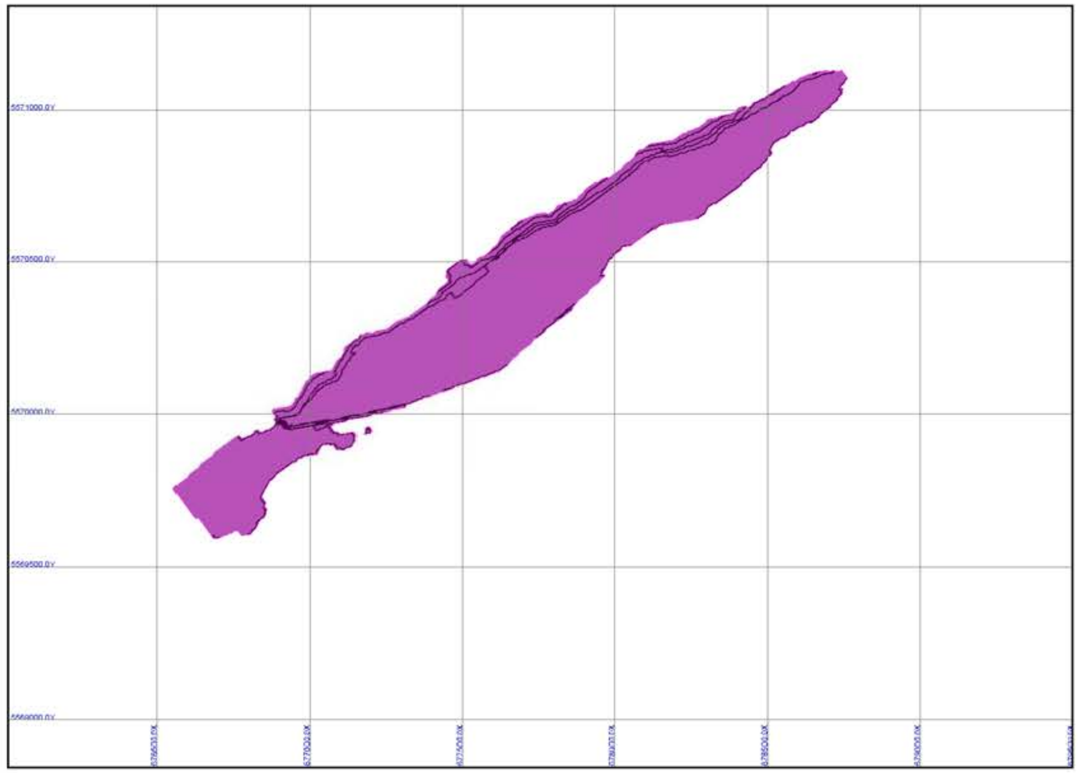


Figure 16-10: Mine Arnaud Open-pit - End of Year 1 Production

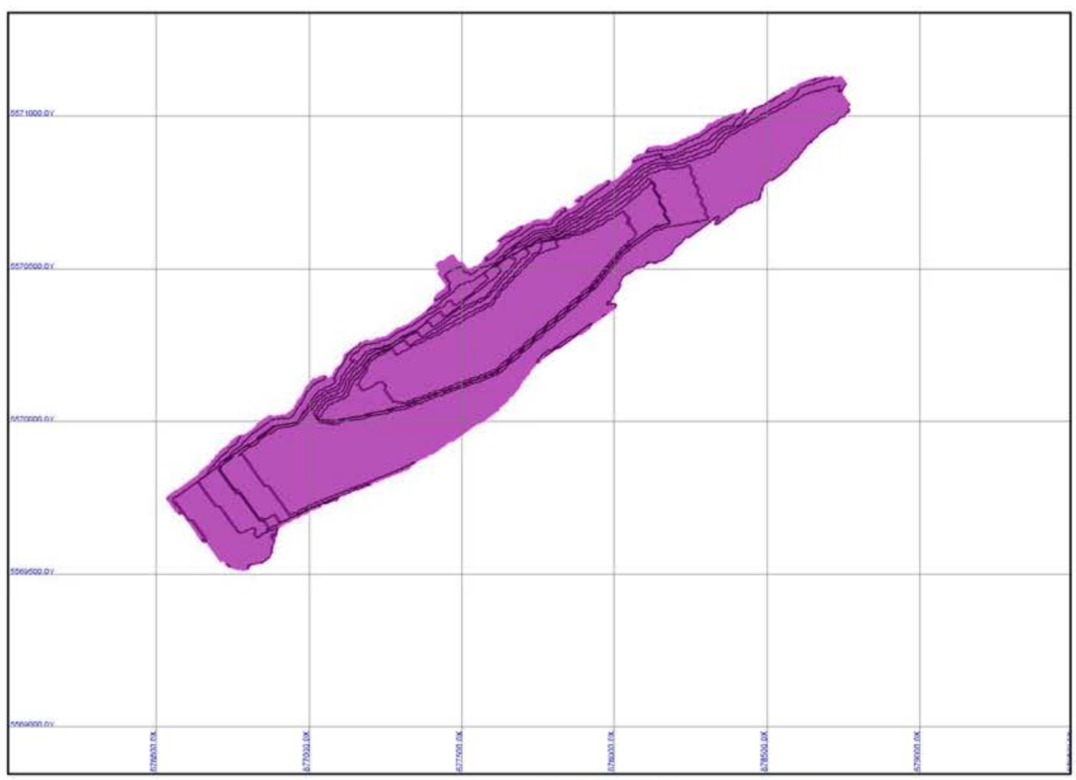


Figure 16-11: Mine Arnaud Open-pit - End of Year 3 Production

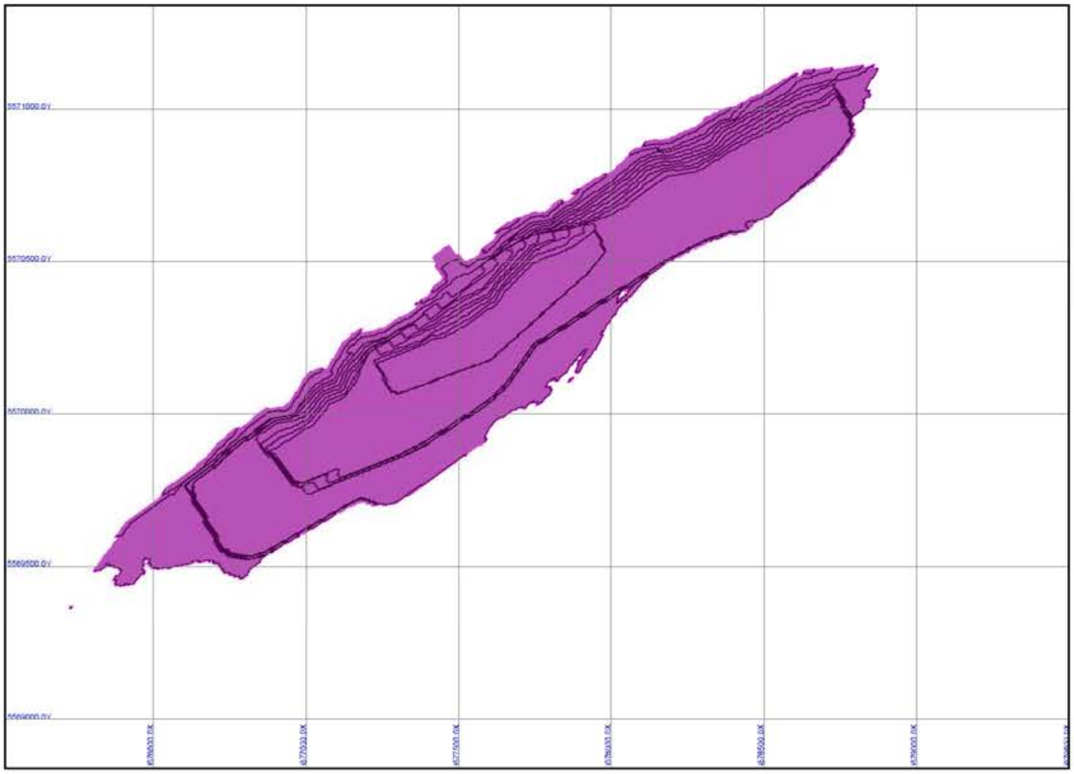


Figure 16-12: Mine Arnaud Open-pit - End of Year 5 Production

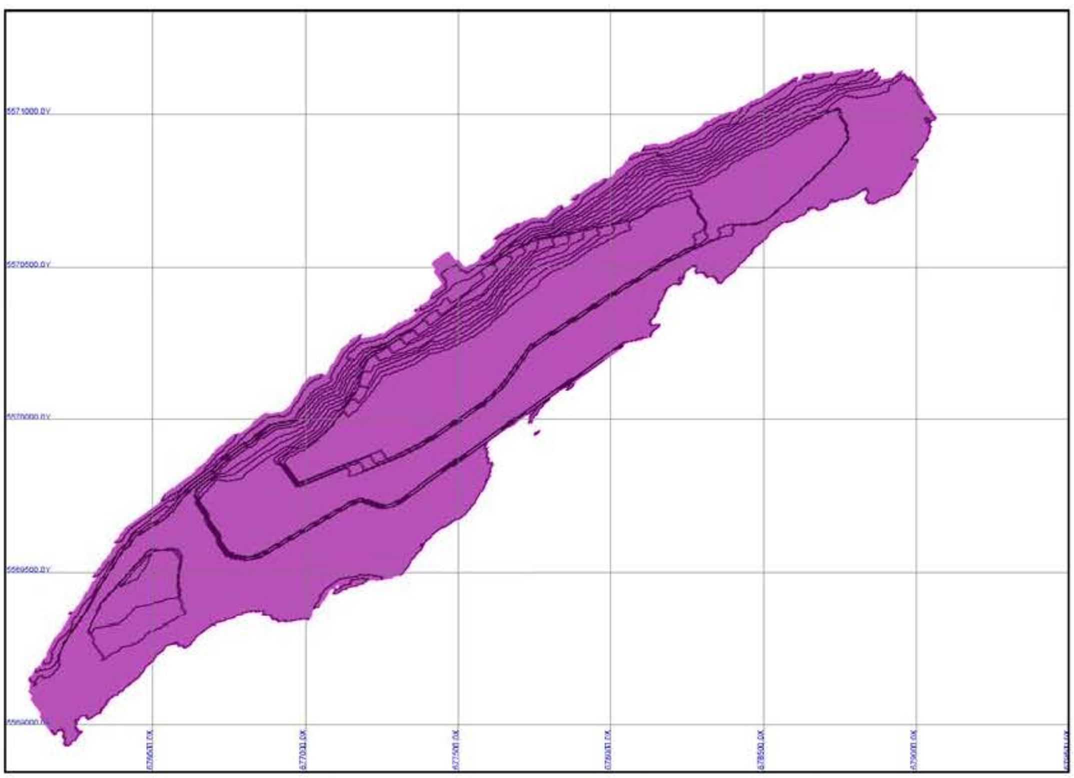


Figure 16-13: Mine Arnaud Open-pit - End of Year 7 Production

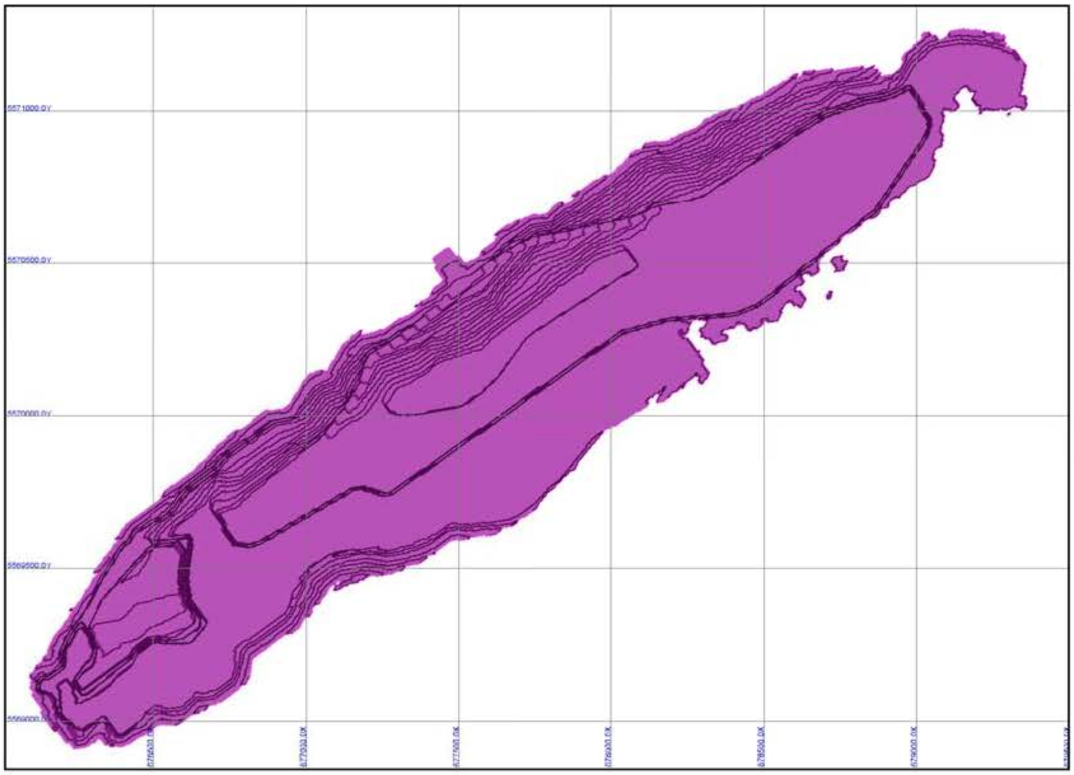


Figure 16-14: Mine Arnaud Open-pit - End of Year 10 Production

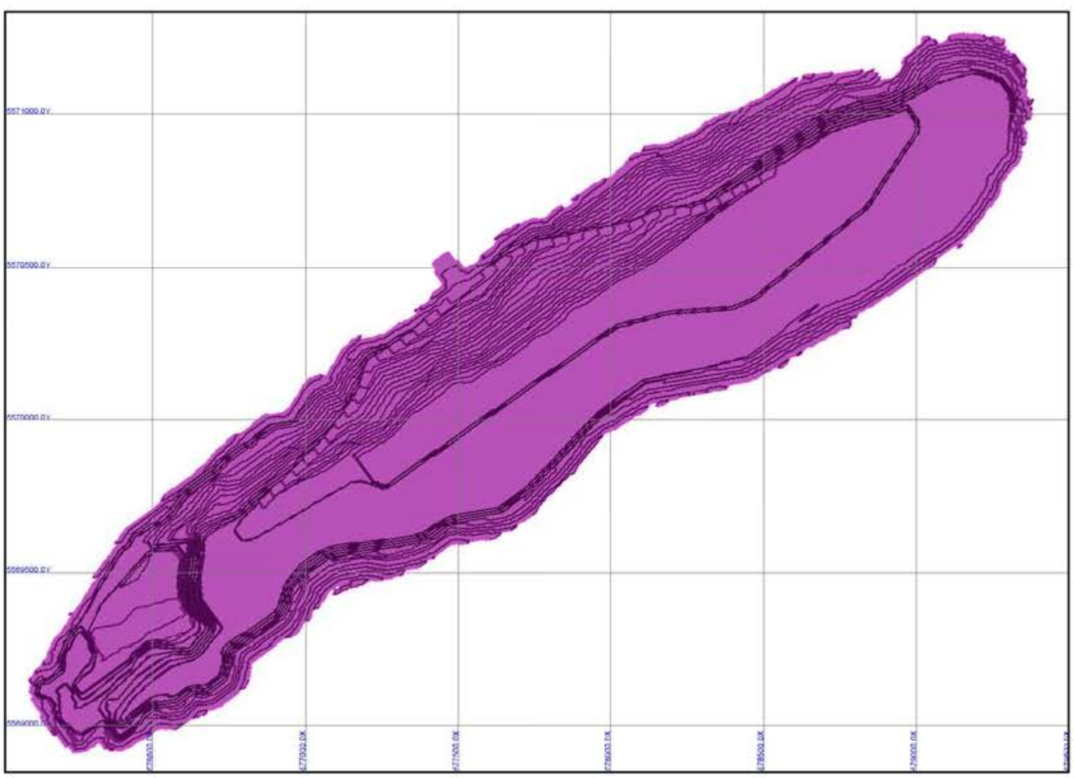


Figure 16-15: Mine Arnaud Open-pit - End of Year 15 Production

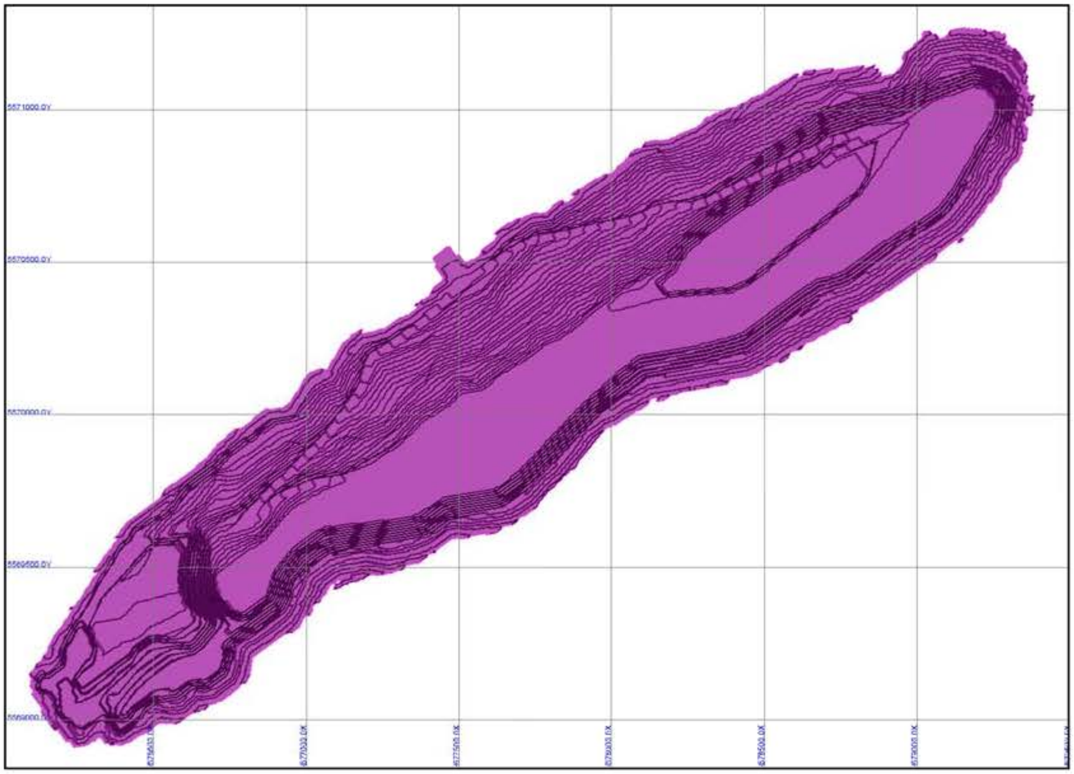


Figure 16-16: Mine Arnaud Open-pit - End of Year 20 Production

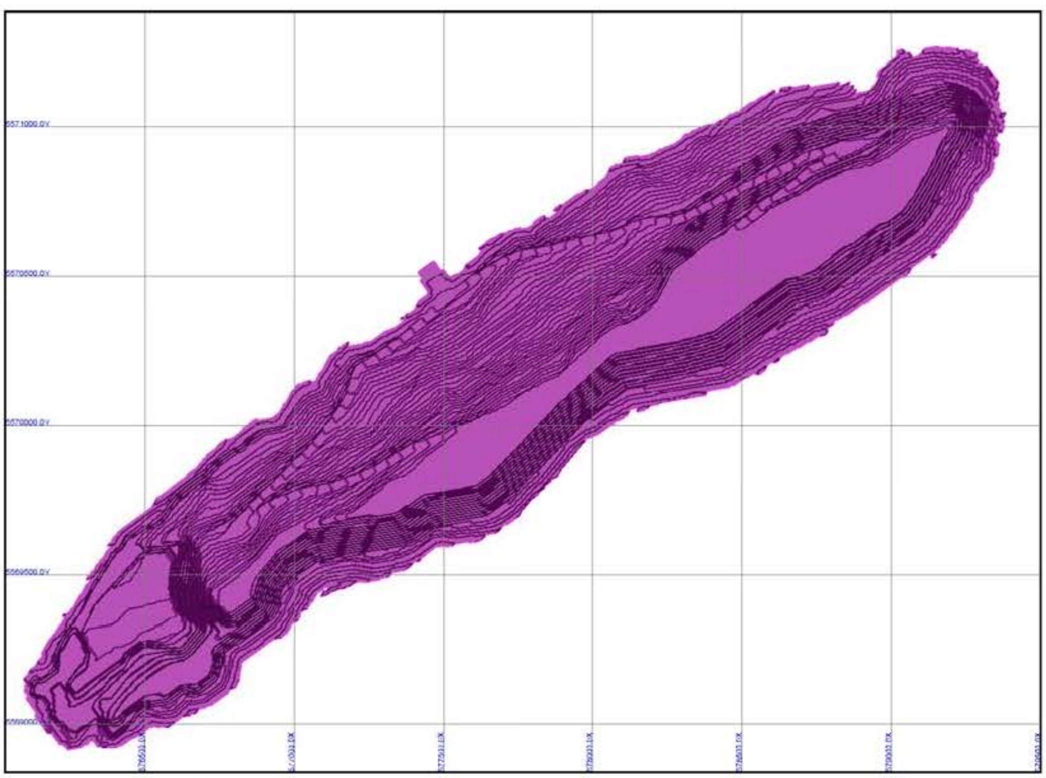


Figure 16-17: Mine Arnaud Open-pit - End of Year 25 Production

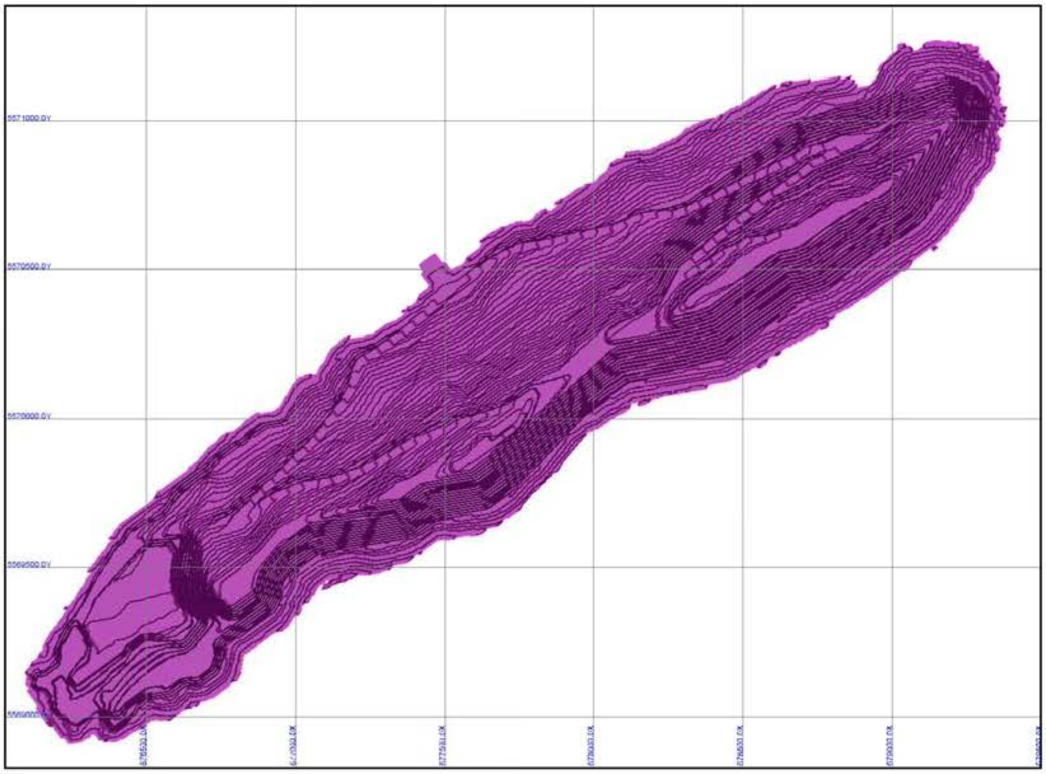


Figure 16-18: Mine Arnaud Open-pit - End of Year 28 Production

16.9 Mining Equipment Requirement

The productivity of equipment has been estimated using industry standard simulation and information supplied by equipment manufacturers. Cycle times were calculated based on average haulage distances to the respective destination for each mining phase. A moisture content of 5.0% has been assumed to convert dry tonnes to wet tonnes for the production estimate on the assumption that resources are reported as dry tonnes.

Even if a small portion of the deposit could potentially be mined using 5 meter benches, the operations are based on a standard production bench height of 10 m. According to the previous FS, RPA recommends further optimization of the bench height relative to the production requirements and equipment selection.

Open pit mining operations are scheduled to operate 24 hours per day with 2 shifts of 12 hours; each requiring four crews. It is estimated that 15 days per year will be lost due to adverse weather conditions and other non-scheduled events. An estimate of calendar time hours is shown in next Table.

Table 16-9: Mining Operating Calendar Time Estimate

Description	Units	Value
Shifts per day	Shifts	2
Hours per shift	Hrs	12
Hours per day	Hrs	24
Operating days per year	Days	365
Holidays	Days	0
Inclement Weather	Days	15
Total	Days	350
Total (for 350 days)	Hrs	8,400

Operating delays that have been used in estimating equipment productivity and requirements are shown next Table.

Table 16-10: Estimated Operating Delays

Description	Units	Drilling	Loading	Hauling
Shifts change	min/shift	30	30	30
Lunch and Other Breaks	Hrs	60	60	60
Re-drill Time	%	5	-	-
Inspection/Refueling	min/shift	20	5	20
Pit Displacement	min/shift	15	15	15
Total delays	min/shift	130	110	125

Additional considerations for estimating the size and number of equipment units required to meet the production schedule are mechanical availability, utilization of available time, and operator efficiency. Next Table lists the mechanical availability by machine hours for the major mining equipment, at the exception of mining trucks. Utilization of available time is based on a 50-minute hour. Operator efficiency starts at 95%, with drills being factored higher to compensate for fewer weather related delays.

Table 16-11: Mechanical Availability of Major Equipment

Operating Hours	Mechanical Availability %		
	Drills	Excavators	Track Dozers
0 - 5000	92	92	90
5000 - 10000	90	90	88
10000 - 15000	88	88	86
15000 - 20000	86	86	84
20000 - 25000	84	84	82
25000 - 30000	82	82	80
30000 - 35000	80	80	78
35000 - 40000	78	78	76
40000 - 45000	76	76	74
45000 - 50000	74	74	72

The equipment purchase schedule was prepared from suppliers' data of speed and payloads for trucks, buckets capacity for excavators and loaders, and drilling rate varying from 0.6 to 0.9 m per minute for blast holes production drilling. SGS estimated the truck fleet based on mechanical availabilities of 97%, 95%, 93% and 91% for Years -1, 1, 2 and 3 respectively, followed by a constant value of 89% for the remaining production years. The significant size of the truck fleet and the renewal of the equipments were the justification for this assumption. The working cycles and productivity were prepared for every working year for ore, waste and overburden. The movements of stockpiles were also taken in consideration for the total estimation of the equipment requirements of the life-of-mine that are presented in following Table. As shown in this table, there is no new equipment purchase after Year 25.

16.9.1 Equipment Purchase Schedule

Table 16-12: Major Open Pit Equipment Purchase Schedule

Equipments	Total	Y(-1)	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19	Y20	Y21	Y22	Y23	Y24	Y25
Drilling																											
Atlas Copco - Pit Viper 235	5	1			1					1						1		1									
Sandvik DR560	3	1									1									1							
Loading																											
Excavator Komatsu PC3000-6	5	1		1								1	1				1										
Wheel Loader - CAT 993K	4	1							1			1								1							
Excavator CAT 374D L (Overburden)	3	3																									
Hauling																											
Mining Truck CAT 785D	22	2	1	1		1	1	1	1			2	2	1	1	1		2	2					3			
Mining Truck CAT 740	8	3							3	2																	
Main Services																											
Motor Grader Cat 16M	6	1				1						1					1					1			1		
Track Dozer CAT D9T	12	2	1						2	1					2	1					2						1
Water Truck CAT 777	2	1													1												
Water Truck Kenworth T800	3	1											1											1			
Shovel CAT 374D L (General services)	2	1												1													

One of additional consideration in the fleet selection is the favorable location of the mine to Sept-Îles, Québec. In the immediate area there are a multitude of mining equipment distributors, service providers, and contractor's setup primarily to service multiple large open pit iron ore mines located a few hundred kilometers to the North, along with more distant operations. For this reason, purchase of equipment backups and spare units has been kept to a minimum.

16.9.2 Drilling & Blasting

Drill pattern design will be dictated by constraints laid out by Hydro-Québec relative to the high tension power line corridor immediately northwest of the open pit development, and Directive 019 of the Ministry of Sustainable Development, Environment and Parks (MSDEP).

In summary, Hydro-Québec restrictions for blasting within the vicinity of power lines are:

- Blasting should not cause any damage to the power line;
- Blasting should not create undesirable and perceptible soil movements within the right-of-way;
- The charge calculation must take into account a maximum particle velocity of 25 mm/s measured at the nearest line support;

- Appropriate protective measures should be taken such as the use of blasting mats, when required;
- No usage of electrical system for firing (electric detonation) due to electric line induction (assumed within the power line right-of-way only);
- Equipment must be installed to record and determine the intensity of transmitted vibrations to the nearest structure.

In summary, restrictions on blasting relative to Directive 019 of the MSDEP are related with ground vibration limits and air blast overpressure for the surrounding inhabitants.

In both cases, the quantity of explosive detonated per delay interval may require adjustment from the proposed production pattern, with the greatest influence being distance from the constraint (assuming all other factors are similar). From the previous FS, RPA strongly recommends further investigation into the potential impacts of the Hydro-Québec restrictions and Directive 019 relative to drill pattern design and blasting, and the implications on operating costs.

Production drilling and blasting of the ore and waste will be carried out in advance of excavation with a diesel powered Atlas Copco Pit Viper 235, or equivalent, which are capable of single pass drilling the 5 to 10 m bench height. A 203 mm drill diameter is specified in the blast design. Over the life-of-mine, three original purchases and two replacement purchases are estimated, this results in an average of just over 40,000 hrs recorded per machine.

A Sandvik DR 560 is recommended for pioneering work, mid range grade control drilling, final wall pre-split drilling, Nelsonite footwall drilling and horizontal drain well drilling. This machine is rated for up to 216 mm holes, however, for the pre-split operations 133 mm holes or similar are anticipated. This drill could also be used as a back-up production unit in case of a major breakdown of one of the two major production drills. Over the life-of-mine, one original purchase and one replacement purchase is estimated.

Explosives supply to the hole is assumed on a contractor basis. Similar budgetary quotes for this service were received from both Dyno Nobel and Orica Canada Inc. 100% use of water proof emulsion has been specified. The explosive supply contractor is responsible for all equipment, buildings, and permits are required to perform the duties of the contract, with staff expected to find their own room and board.

The site preparation for the explosive's site and magazine site is the responsibility of the owner, along with maintenance of the access roads and supplying electricity and water to the site as required. During regular production, it is assumed there will be two rotating day shift crews of three persons each required for operations. The explosive supply contractor is also responsible for the magazines and maintaining supplies. Electric initiation systems are specified for all blasting outside of the Hydro-Québec right-of-way to improve blast fragmentation, reduce ground vibration, and reduce noise levels. Initiation systems layout and firing are the responsibility of the owner.

A powder factor of 0.25 kg/tonne is targeted for ore and waste for production benches. Table 16-13 summarizes typical blast design parameters for the production of both ore and waste.

Table 16-13: Production Blasting Parameters

Description	Units	Ore	Waste
Material Density	t/m3	3.5	30
Powder Factor	kg/t	0.25	0.25
Bench Height	m	10	10
Stemming	m	4.0	4.0
Sub-Grade	m	1.7	1.7
Fallback	cm	10	10
Hole Depth	m	11.7	11.7
Hole Diameter	mm	203	203
Burden	m	5.6	5.8
Spacing	m	6.4	6.7

Geotechnical recommendations call for pre-split drilling and blasting to achieve the final wall slope recommendations, the proposed drilling for final wall pre-split holes, is of approximately 133 mm diameter and 1.5 m spacing. Pre-split requirements are to be determined upon completion of detailed pit design.

In order to manage mining recovery and dilution of Nelsonite ore, a modified pattern design is required for breaking the footwall rock. Projecting the dip of the exposed benches downward to the next bench floor along with the ore control model, the smaller drill will be used to drill on a tightly spaced pattern targeting the footwall. Field experience will determine the optimum hole inclination (i.e. vertical holes versus angle holes perpendicular to the footwall or in between) and pattern burden and spacing. This design should initially be carried out on 5 m benches, with results compared to what can be achieved going to a full 10 m bench height to improve the mining cycle. Grade control for locating the Nelsonite footwall will be a multi-step process dependent on how much variability is observed in the footwall with experience. An initial row of wide spaced blast holes targeting the intersection of the projected footwall with the toe of the next bench are to be drilled with sampling in one meter intervals around the projected contact. Based on sampling results, the projected footwall is refined and blast holes depth is adjusted to maximize mining recovery minimizing back break and resultant dilution. The current operating cost estimate assumes a regular production blast for the Nelsonite footwall. According to the previous FS, RPA recommends estimating quantities of Nelsonite footwall ore requiring a modified pattern design upon completion of detailed pit design to complete a revised operating cost estimate. All drills shall be equipped with GPS and hardware/software for downloading drill patterns and drilling instructions.

16.9.3 Excavators & Loaders

Overburden will be mined with a dedicated equipment fleet including three Caterpillar 374D backhoes equipped with four cubic meter buckets to four pass load the specified haul truck. Each backhoe averages just less than 50,000 hours service. Two more Caterpillar 374D backhoes are specified under auxiliary equipment as general purpose backhoes for the mine site, there are also available as a backup units if required.

Production mining of ore and waste rock is performed by up to three diesel powered Komatsu PC3000-6 excavators, one setup as a backhoe and two as face shovels. Two equipment replacement purchases are required over the life-of-mine. This results in approximately 50,000 hrs being recorded per machine. A 13 m^3 bucket is specified for four pass loading ore while a bucket size of 15 m^3 is required to effectively fill waste rock at four passes. Two excavators Komatsu PC3000-6 will be operating at the same time in ore and waste to meet the annual production requirements for the plant, the stockpiling and the waste disposal.

Face shovels are more suited for mining large continuous blocks of ore and waste. A large waste block is present above the Upper horizon on the high wall side of the deposit, and is more prominent when there is no California present. Large ore blocks exist in the Upper and Railroad (Combine) horizons. The backhoe is better suited for cleaning the ore to waste contacts and mining the narrow inter-burden horizons where greater selectivity is desired. The face shovel will single pass the 10 m bench, while the backhoe requires at least two passes. The excavators will be equipped with GPS units tracking the bucket which will be used to define dig limits from polygons based on the grade control model. For optimization studies, RPA recommends reviewing the use of front end wheel loaders as the primary

Wheel loaders CAT 993K are specified for stockpile reclaim operations. These machines are capable of side loading the truck fleet and feeding ore to the crusher. Based on the stockpile reclaim schedule, two loaders are recommended with two replacement units. This would also allow the front end loaders to step into open pit operations as required to meet production.

16.9.4 **Haul Trucks**

A dedicated truck fleet is matched to the overburden loading equipment. Eight (8) 36 tonne capacity articulated trucks are specified for continuous operation from the start of operations through Year 18. An average of approximately 50,000 hours per truck is recorded during this time. At least four (4) trucks are paired with each loading unit at a time. Working with larger truck size fleet for a portion of the overburden mining might reduce the costs of the overburden stripping operations. The subcontracting alternative should also be considered in future studies.

A fleet of Caterpillar 785D, 133 tonne capacity haul trucks is specified for ore and waste rock haulage. The truck fleet starts at three (3) in Year-1 and hits its peak in Years-17 and 18 with twelve (12) units in operation. The total quantity of trucks to purchase is twenty-two (22) units. As production declines, the peak truck count is still required for numerous years to offset increasing haulage times and reduced mechanical availability. In Year 28, the active truck count starts to fall as production levels decrease faster than the increase in haulage cycle demand. An average replacement life of ten years, the equivalent of ~70,000 hrs per truck is recorded over the life-of-mine.

Pit haul road width of 25.4 m is specified for Caterpillar 785D two way traffic; this includes a single shoulder berm barrier and additional width for drainage of surface water, as illustrated in the above Section 16.7.1.

The following Figure shown below was prepared to illustrate the movement of overburden and hard rock, including ore and waste, with the required quantity of trucks required on a yearly basis.

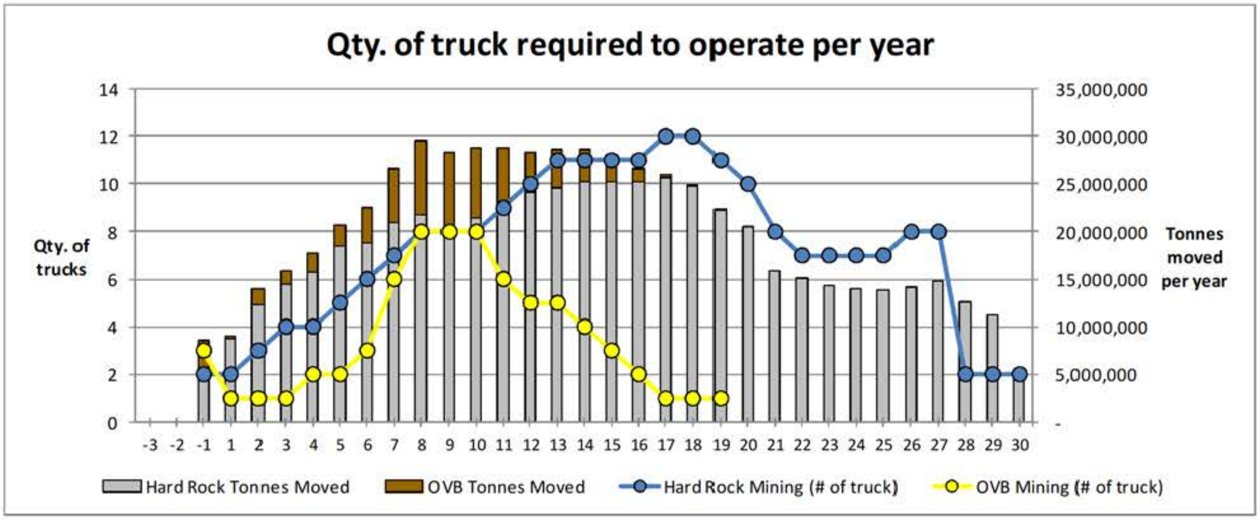


Figure 16-19: Trucks Requirement for OVB and Hard Rock Mining

16.9.5 Service Equipment

A fleet of support and service equipment is provided to assist the mine operation as shown in next Table. This Table also provides the purchase schedule for these equipments consisting of repair service trucks, fuel/lube trucks, auxiliary equipment, buses and light vehicles.

Table 16-14: Mine Arnaud Open Pit Service Equipment Fleet

Description	Total	Y(-1)	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19	Y20	Y21	Y22	Y23	Y24	Y25
Tool Carrier CAT IT62H	10	2					1				1	1				1	1				1	1				1	
Service Truck Kenworth T800	12	2					2					2					2					2					2
Tire Handler Kenworth T800	9	1					1					1	1				1	1				1	1			1	
Fuel/ube Truck Kenworth T800	15	2					2					2		1			2		1			2			1		2
Lowboy 100t	1	1																									
Tower Light	50	6				6				6		2		6		2		6		2		6			2		6
Pickup Trucks	188	20		7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7
Pit Buses	7	1				1				1				1				1				1				1	
Crane 60 tonnes	1	1																									
Welding Machine	7	1				1				1				1				1				1				1	
Compressors	7	1				1				1				1				1				1				1	

16.10 Pit Dewatering

Pit dewatering will be accomplished via a combination of vertical dewatering wells used to lower the water table, horizontal drain holes to depressurize the pit walls, and surface water management.

Thirty (30) dewatering wells will be installed to varying depths with some wells going well below the pit bottom based on the Hydrogeological Investigation Report. Some wells will be lost during development of the open pit with new wells installed as the pit expands. An average discharge rate of 1.5 l/s per well is assumed. Additional shallow overburden dewatering wells and trenches are required, especially around the southeast end and northeast ends of the pit.

Horizontal drain holes in the pit walls are likely to be required as per the Hydrogeological Investigation Report. The smaller drills, Sandvik DR560 is capable of drilling these holes up to approximately 50 m in suitable ground conditions. Based on the final pit depth and rule-of-thumb for horizontal well lengths, the Sandvik Drills should be capable of performing the majority of this operation. However, a specialized rig may be required for longer holes or difficult ground conditions. Approximately 30,000 m of horizontal drain holes drilling has been assumed over the life-of-mine.

The open pit is in an advantageous location for surface water management, as it strikes parallel to a ridge line of topographical highs limiting its surface water catchment area. Water diversion ditches in a few key areas will be used to control surface water running into the pit particularly around the end walls. Surface water will collect in the pit from precipitation, as well as groundwater not removed by the pit dewatering wells. As the footwall of the pit is developed, pumping infrastructure for surface water management is installed early in the mine life, and developed as the pit gets deeper. Other than in the first few years of operations, the pit develops with two bottoms, and opens a considerable amount of strike length. Two main sumps, one for each pit bottom, are estimated. There will also be the ability to move water between sumps if required in the event of an extended shutdown of one sump.

16.11 Hydrological Study

Preliminary Hydrogeological Studies were conducted on the Sept-Îles deposit in conjunction with the pit wall slope stability investigation both performed by Ausenco Vector. Report on the results of the investigation could be found in Appendix 7.2 of the previous Feasibility Study. The hydrogeological field investigation was conducted by AMEC between November 2010 and March 2011 which also included geotechnical borings for overburden characterization and oriented core holes for preliminary pit wall stability design.

The investigation comprised of geologic mapping, packer testing of core holes, test well and observation well/core hole drilling, construction and development, and two aquifer pumping tests. The field investigation data were compiled and analyzed to develop estimates of the potentiometric surface and hydraulic properties in the vicinity of the proposed mine. These data were used to develop a hydrogeological conceptual model (HCM) and a rudimentary groundwater flow model to estimate groundwater inflow to the pit. A preliminary pit dewatering plan was developed based on the results of the field investigation and modeling results.

The hydrogeological data and analysis completed to-date suggests that the overburden and bedrock properties are of moderate permeability. Preliminary hydrogeological modeling based on these parameters suggests that flow rates are anticipated to be on the order of 27 l/s to 55 l/s which will require between 20 and 38 dewatering wells and associated works.

However, additional hydrogeological studies would significantly reduce the uncertainty in the predicted dewatering requirements and hydraulic head distribution during mining activities. This would allow for operational and environmental risks associated with dewatering to be defined and a more efficient and cost-effective dewatering system design with lower capital and operating costs.

17. Recovery Method

The information contained in this section is an up-date of the Roche-Ausenco's Feasibility Study dated February 2012 plus several additional metallurgical tests made later in 2012.

The author of this section is of the opinion that the Mineral Processing part (Chapter 6.4 to 6.8) of Roche-Ausenco Feasibility Study is very commendable and for most of its part could be used as is.

17.1 Introduction

The process plant design criteria are based on Roche-Ausenco's feasibility study and on various sources of information. These sources are:

- Information provided by the client;
- Previous studies;
- Testwork conducted at SGS Lakefield, COREM, or by equipment suppliers before September 2012;
- Testswork conducted at COREM in September and October 2012;
- Testwork conducted at COREM in May and June 2013;
- Roche-Ausenco's calculations, layouts and/or recommendations of phosphate consultants;
- SGS Geostat's calculations
- Industry standard practices or literature.

The plant capacity is established at 11,201,120 tpy ROM, based on an ore processing rate of 1,400 tph, a plant availability of 21.92 hours per day and 365 days of operation per year. This availability has been selected at the request of the client and upon the approval of Roche's process metallurgical engineer. This gives an overall plant availability of 91.34% on a 365 days per year basis. The plant has been sized to meet the criteria and parameters as indicated in Table 17-1. The detailed design criteria are presented in Appendix 6.

Table 17-1: Mine Arnaud General Design Criteria

Parameter		Value	Source
Days per year		365	A – D
Plant Capacity (tph)		1400	C
Process plant availability		91.34%	D
Process plant operation per day (h)		21.92	D
Ore processing per year (tpy)		11,201,120	A - D
Average ore processing per day		30,690	D
Overall concentrate weight recovery		10.67%	D
Concentrate produced total (tpy)		1,195,159	D
Concentrate grade (%P_2O_5)		≥39%	D
%P_2O_5 Recovery		≥90%	D
Tailings produced total		10,005,961	D
Reserves average grade (%P_2O_5)		4.59%	D
Reserves average grade (%Cl)		0.03%	D
Ore type proportion (%)	Nelsonite (S2)	19.4%	A
	Rail Road (SRR)	44.5%	A
	Upper (S3)	36.1%	A
Apatite liberation size (µm)		125	B
Concentrate specifications	P_2O_5 (%)	>32.0%	A
	Fe+Al (%)	<1.0%	A
	Ca/P	<2.2	A
	Cl (%)	<0.1%	A
	Mg (%) – HNO_3	<0.3%	A

Note: *The above criteria are valid for the 28 first years of LOM and do not take into account the low grade (average grade of 2.39% P_2O_5) material from stock piles processed after year 28.*

Source code A: Mine Arnaud

B: Corem

C: Roche

D: SGS

The apatite minerals are, for all practical purposes, liberated at 125 microns and the power consumption for the crushing of the ore is 10.1 kWh/t while the ball mill Work Index is established at 12.5 kWh/t. The concentrate will be dried between 0.5% and 1.5% moisture for ship transportation and subsequent processing by Yara.

17.2 Flowsheets Process Description

The process description was written in association with the schematic drawings listed in Table 17-2. To understand the flowsheets, it would be preferable to read this section using the corresponding simplified process diagram of the different area of the mill after each chapter. Detailed flowsheets supporting this report could be found in Appendix 7 and in the Roche-Ausenco Feasibility Study at Appendix 03.

Table 17-2: Process Flowsheet Drawing List

Drawing No.	Title
2013 – 01 - A	Crushing Schematic Diagram
2013 – 01 - B	Grinding Schematic Diagram
2013 – 01 - C	Low Intensity Mag Separator and Dewatering
2013 – 01 - D	Flotation Schematic Diagram
2013 – 01 - E	Thickening, Filtration, Drying, Ensiling
2013 – 01 - F	Mill Tailings Handling Schematic Diagram
2013 – 01 - G	Tailings Management Schematic Diagram

17.2.1 Crushing and Ore Handling (Drawing 2013 – 01 – A)

The ore will be hauled by 150 tonnes mine trucks (Caterpillar 785D or equivalent). The trucks will discharge on a grizzly set above an out of mine ore hopper. A hydraulic hammer will break down any oversize rocks directly on the grizzly. Ore from the hopper will flow by gravity on a 1,828mm x 16,000mm (6 ft x 52 ft) variable speed apron feeder which in turn will feed a 1,372 mm x 1,905 mm (54" x 75") primary gyratory crusher driven by a 447 kW (600 HP) motor, with an average feed rate of 2,576 tph. The gyratory crusher circuit is overdesigned for the required throughput as per client request, based on his experience for similar operations. The ore will be crushed to a P_{80} of 170 mm and stored in a conical stockpile ahead of the grinding circuit. The crusher will discharge on a belt conveyor, which will transport the ore to the conical stockpile containing approximately 30,000 tonnes of live storage. A storage dome for dust control will cover the stockpile. From the stockpile, the coarse ore is retrieved by five (5) 2,438 mm x 7,924 mm (8 ft x 26 ft) variable speed apron feeders, which will transfer the material onto a 1,067 mm (42") wide belt conveyor carrying the ore to the grinding circuit. A weight scale is used to monitor and control the ore addition into the SAG mill. Each variable speed feeder has a maximum capacity of 1,680 t/h and the mill feed conveyor has a capacity of 1,680 t/h.

The gyratory crusher auxiliaries will include a crusher lubrication unit, an eccentric removal cart, and a hydraulic positioning system. An air compressor, an apron feeder lubrication unit and an apron feeder hydraulic drive unit will also be included. To facilitate the maintenance and operation, the crusher area is serviced by a 60 tonne capacity overhead crane along with a 7.5 tonne capacity monorail hoist.

Two 20,400 m^3/h (12,000 CFM) bag-house dust collector systems will be used for dust control. Four (4) sump pumps for drainage are provided to service the crusher and reclaim tunnel area under the stockpile.

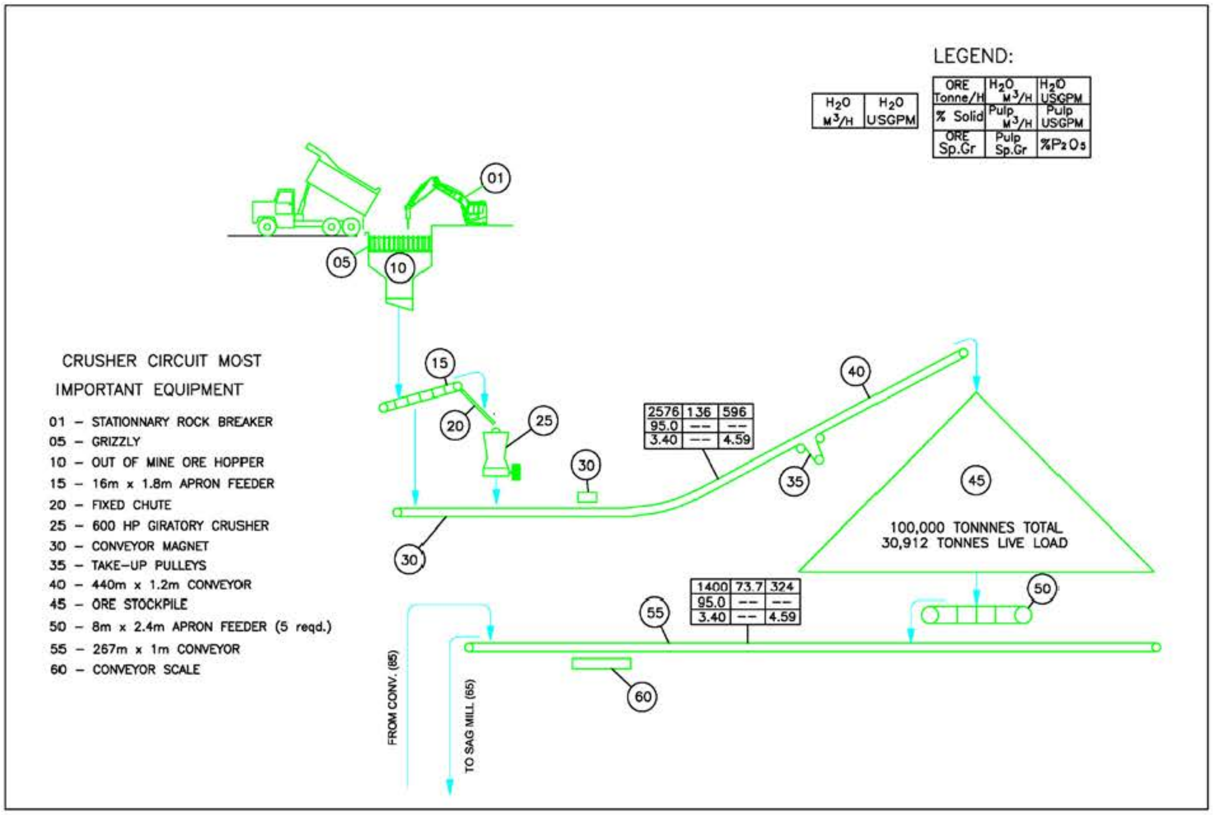


Figure 17-1: Crushing Schematic Diagram

17.2.2 Grinding (Drawing 2013 – 01 - B)

The ore is retrieved from the stockpile at an average nominal rate of 1,400 t/h and fed into a 10,160 mm dia. x 5,330 mm long (34 ft x 17.5 ft) SAG mill driven by dual pinion drives at 6,000 kW (8,000 HP) each. The drive trains are variable speed units. The feed rate to the mill is controlled by the stockpile reclaim conveyor belt scale. Water addition to the mill is proportioned based on the feed rate.

The SAG mill auxiliaries include a ball feed hopper, a stationary feed chute, a retractable feed chute, a jib crane, a mill liner handler, an automatic gear spray system, a portable inching drive, a SAG mill lubrication unit, two (2) jacking cradles, a portable hydraulic jacking unit and a sump pump.

The SAG mill will grind to a P_{80} of 2 mm. The mill discharge will flow onto two (2) double deck 2,440 mm x 7,320 mm (8 ft x 24 ft) vibrating screens with wash water sprays. The screen oversize, or +9.5 mm material is returned to the SAG mill feed. The screen undersize will flow into the SAG mill discharge pump box and pumped to a distributor to feed the two (2) ball mill discharge pump boxes. The two (2) overflow type ball mills, 6,100 mm dia. X 9,140 mm long (F/F) (20.0 ft x 30.0 ft), are each equipped with a 6,000 kW (8,000 HP) fixed speed motors, and are configured in closed circuit with two (2) classifying cyclone clusters. The circulating load is 250%. There are four (4) cyclones (800 mm - 31.5 inch) installed per cluster and three (3) cyclones will be in operation. The cyclone underflow will flow by gravity to their respective mill feed spout. The cyclone overflow will have a fineness of a P_{80} of approximately125 µm.

One liner handler, two hydraulic jacking systems with cradles, two ball storage bins, a ball bucket, an electric magnet, two sump pumps and an automatic gear spray systems ball mill lubrication units for each mill service the grinding area. The grinding area is also serviced by a 40 tonne capacity overhead crane equipped with a 5 tonne capacity auxiliary hoist to facilitate maintenance and operation.

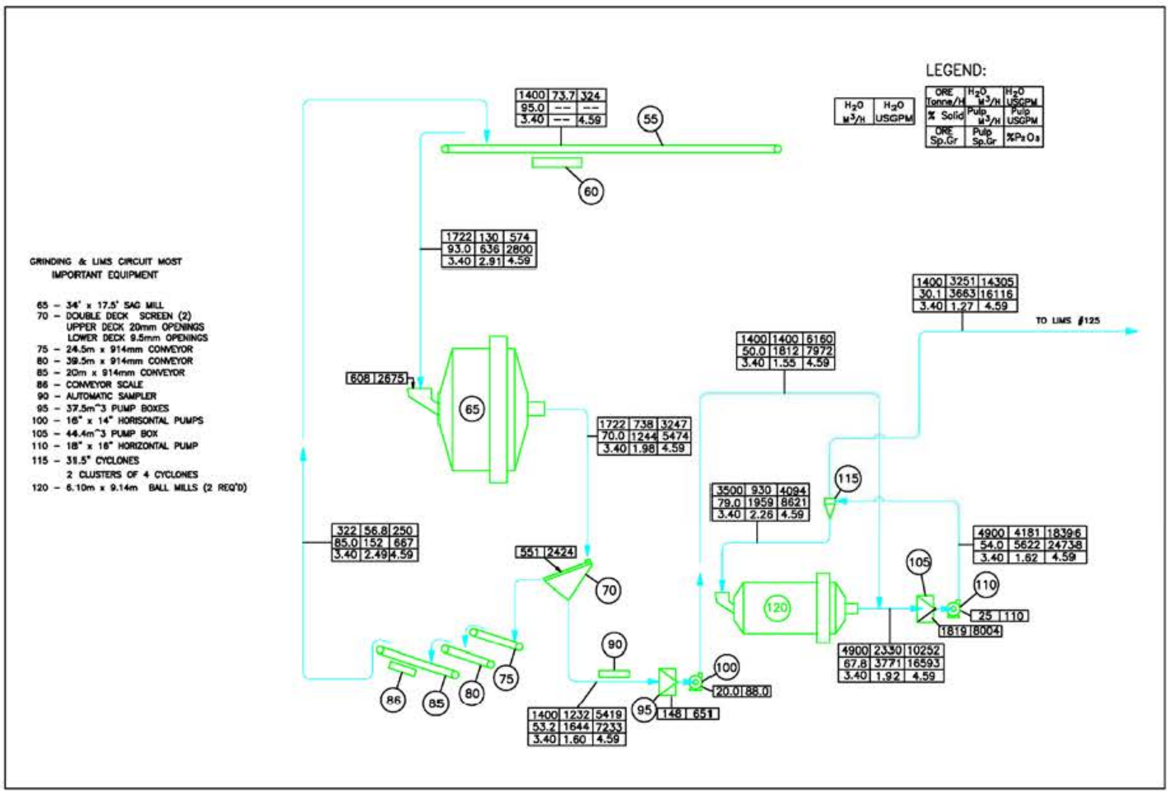


Figure 17-2: Grinding Schematic Diagram

17.2.3 Low Intensity Magnetic Separation and Dewatering (Drawing 2013 - 01 – C)

The overflow from the two classifying cyclone clusters is directed to two distributors and feed (4) double drum Low Intensity Magnetic Separators (LIMS) (1,200 mm diameter x 3,200 mm long) to remove the titano-magnetite. The feed to the drums is diluted to 30% solids. The LIMS is a double drum, counter current tank unit with permanent magnets. The first drum acts as a rougher to capture as much of the titano-magnetite as possible. The first drum is rated at 1,000 gauss. The magnetite material captured in the first drum is reprocessed in the second drum, which has an 800 gauss rating and will operate at a somewhat lower feed rate to minimize the entrapment of apatite. The LIMS concentrate or magnetic product is then thickened in a 23 meters diameter thickener to recover the water, which will be redirected to the process water reservoir. The thickener underflow is pumped to the magnetic tailings ponds.

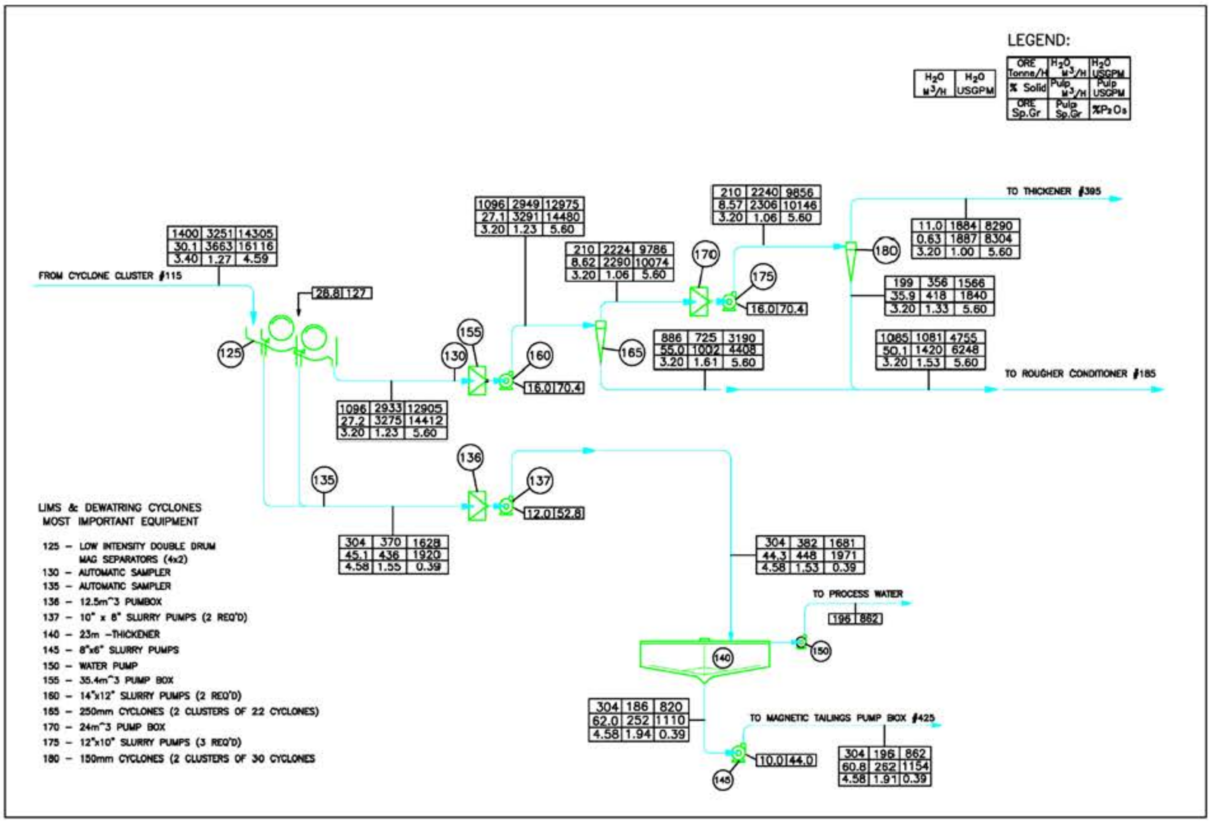


Figure 17-3: Low Intensity Magnetic Separation and Dewatering Schematic Diagram

The non-magnetic material flows by gravity to the dewatering cyclones feed pump box and is pumped to two dewatering cyclone stages. There are two (2) dewatering cyclone clusters per stage; one for each flotation line. There are 22 cyclones (254 mm - 10 inch dia.) installed per cluster and 18 cyclones in operation for the first stage. The second stage has 30 cyclones (152 mm - 6 inch dia.) installed per cluster and 24 cyclones in operation. The overflow of the second stage is directed to the apatite tailings thickener. The dewatering cyclone underflow, from the both stages, is directed to the pH regulation conditioning tank. The particle size distribution for dewatering cyclone feed has a P_{80} of 118 to 125 µm and a P_{50} of 54 to 60 µm.

17.2.4 Flotation (Drawing 2013 – 01 - D)

The underflow from the dewatering cyclone clusters, at 50% solids, flows by gravity to six (6) flotation conditioners. There are two (2) parallel lines of three (3) different conditioning stages in series. Sodium hydroxide (NaOH) is added in the first conditioner tank to reach the desired pH of 10.8. The addition of NaOH is controlled by a pH meter. The pulp exiting the first conditioner will flow to the second conditioner where wheat starch (WW82) will be metered in. Finally, the pulp will flow into a tertiary conditioner where Soybean fatty acid (Liacid 1800) will be metered in. The conditioned pulp will flow to the rougher stage #1 distributor to be diluted to 35% solids using the mill recycle water system and then distributed to rougher stage #1 flotation column cells.

Each line of flotation will have two rougher flotation column cells in parallel followed by a single scavenger column cell. Both lines consist of two (2) 4,880 mm x 14,000 mm rougher flotation column cells in parallel. The underflow from the roughers, or tailings is pumped to a single scavenger flotation column cell (4,880 mm x 14,000 mm). Thus the froth or overflow from the rougher and the scavenger (concentrates) is combined in a pump box and feed two (2) 4,880 mm x 8,000 mm cleaner flotation column cells configured in parallel. Tailings from the cleaner stage are pumped to a conditioner tank where starch (WW82) is metered in. The starch is utilized to depress the remaining iron oxides in the pulp. The conditioned pulp then flows to a 3,670 mm x 14,000 mm cleaner/scavenger flotation column cell for recovery of additional apatite. The cleaner/scavenger is mixed with the rougher – scavenger concentrates to be recycled to the cleaner stage feed. Tailings from the cleaner/scavenger stage are combined with the scavenger tailings and are pumped to the tailings thickener to eventually be discarded as final tailings.

The cleaner concentrate (final concentrate) should have a grade of 39% P_2O_5 containing less than 1% Fe+Al combined with less than 0.3% Mg and a lesser amount than 0.1% Cl. The final apatite concentrate is pumped to the apatite concentrate thickener.

Six (6) low-pressure rotary screw compressors rating 186 kW (250 HP) each will provide the air for the flotation circuit. Four (4) compressors will provide the specified air requirement, a fifth compressor is required to meet the design maximum air requirement and the sixth compressor is a stand-by unit.

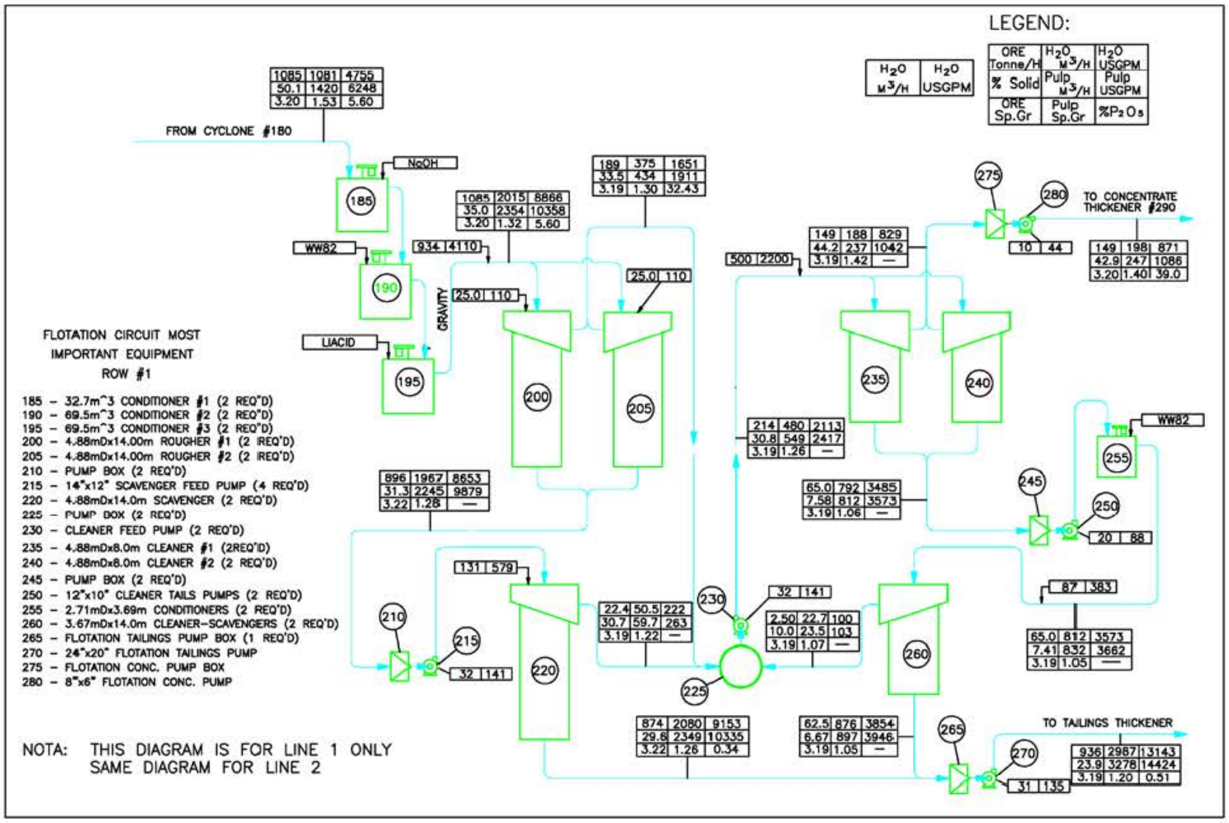


Figure 17-4: Flotation Schematic Diagram

17.2.5 Wet High Intensity Magnetic Separation (WHIMS)

This area is optional for possible future use. Currently, there is no indication that the WHIMS will be required. Nevertheless, depending on the variability of the ore, WHIMS may be required occasionally to meet product grade specifications. The envisioned flowsheet for WHIMS processing of cleaner concentrate consists of pumping the concentrate to the WHIMS feed pump box and then to the WHIMS distributor. The non-magnetic product is then pumped to the final concentrate thickener, and the middling is recycled to the WHIMS feed pump box. The magnetic product reports to the final tailings thickener where it is discarded as waste. Currently, no provision is made other than space in the beneficiation plant for the installation of a WHIMS circuit. Space is planned in the mill to include a WHIMS circuit if ever needed one day, but it is not part of the current process.

17.2.6 Thickening, Filtering, Drying and Ensiling (Drawing 2013 – 01 - E)

The apatite concentrate from the cleaner flotation column cells is pumped to a 20,000 mm diameter apatite concentrate thickener for dewatering. The thickener overflow is directed to the process water reservoir for reuse in the plant. The thickener underflow, at 70% solids, is pumped to two 10,670 mm diameter x 12,800 mm high agitated slurry storage tanks, each providing 8 hours storage capacity.

From the storage tanks, the concentrate slurry is pumped to a horizontal vacuum belt filter. A 186 kW (250 HP) liquid ring vacuum pump services the belt filter. The filtrate from the filter is returned to the apatite thickener. The filter cake, at about 8% moisture, is conveyed to a flash dryer. The belt feeding the dryer surge bin is equipped with a belt scale to monitor the feed rate to the dryer.

From the dryer surge bin, a feed screw conveyor transports the feed material to the venturi, where the feed is evenly distributed into the dryer. The feed encounters the hot process air in the high-speed venturi, where most of the evaporation occurs instantaneously. As the product accelerates into and through the drying column and drying duct, residual moisture will diffuse through the particles and is evaporated from the surface. The dryer will feature a bag house dust collection system to recover dry product from the process air stream prior to discharging to atmosphere. Dried product is discharged using a system of screw conveyors and rotary valves.

A portion of the wet feed from the dryer surge bin is sent to a single-shaft paddle mixer to form a homogenous/uniform product with final moisture of 1% ± 0.5% by mixing with the dried product discharged from the drying system.

The process air is drawn through the entire system by an induced draught fan located downstream of the bag house. Control of the dryer is achieved by maintaining a constant exhaust temperature and modulating the feed rate with respect to the feed screw speed. A 20 MW electric air heater featuring a heater assembly in a "multi stage/multi circuit system" with parallel and series circuit elements which supplies heat to the dryer.

The dryer system discharges are transferred by belt conveyor to two 4,500 tonne capacity storage bins. The transfer conveyor is equipped with a belt scale to monitor plant production. From the storage bins, the apatite is loaded into a freight train containing 45 gondolas (3,281 cubic feet capacity per wagon) for transportation to the port facilities. Each storage bin will be equipped with a discharge air slide system, isolation gates, loading spout and a bin vent dust collector. Detailed description could be found in the Roche-Ausenco feasibility study at section 12.4.1.

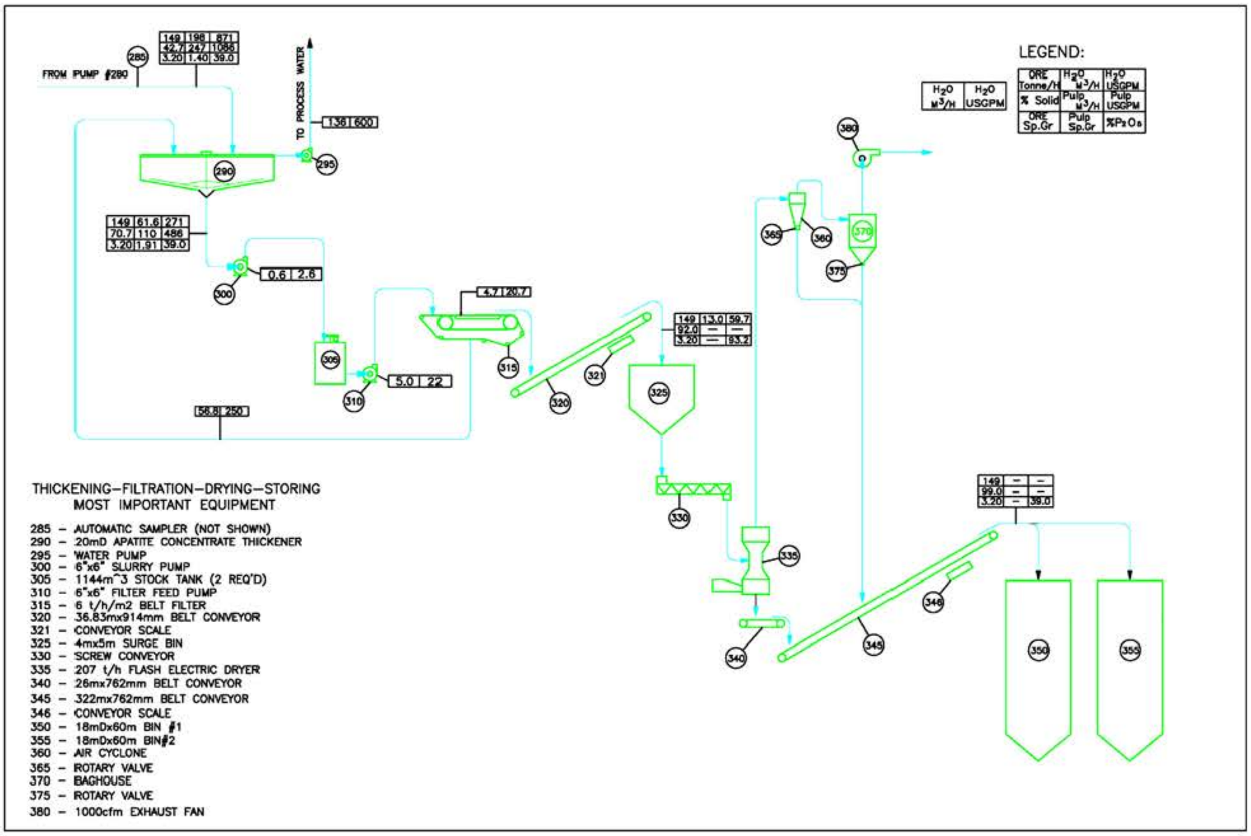


Figure 17-5: Thickening, Filtration, Drying, Ensiling Schematic Diagram

17.2.7 Flotation Tails Thickening and Tailings Handling (Drawing 2013 – 01 - F)

Underflow from the magnetite thickener is pumped to the magnetite tailings pumbox followed by a three stage pumping system carrying the magnetite product to the magnetite tailings pond.

Flotation final tailings along with the second stage dewatering cyclones overflow are pumped to the final apatite 40 meter thickener. Thickener overflow reports to the mill process water tank while thickener underflow is pumped via 3 pumps connected in series to the non magnetite tailings pond.

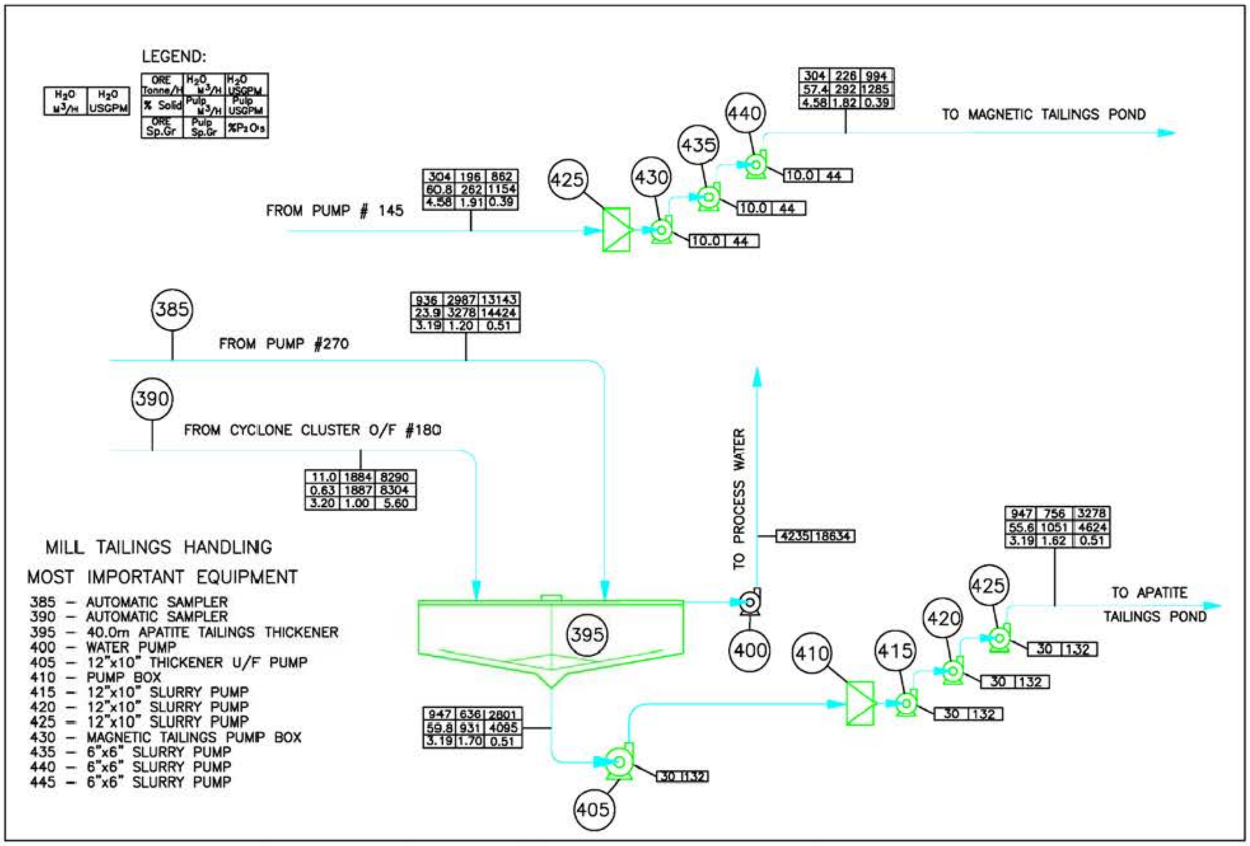


Figure 17-6: Thickening, Filtration, Drying, Ensiling Schematic Diagram

17.2.8 Tailings Management (Drawing 2013 – 01 – G)

Supernatant water from both polishing ponds will be pumped back to the mill process water tank with the thickener overflows. The tailings thickeners are dosed with hydrated lime and flocculent to make sure that all the fines settle and there will be no problem with fines build-up in the plant water system. The tailings thickener underflows, at around 60% solids, are pumped to the tailings disposal pump boxes for pumping to both tailings pond. Whenever tailings pumping is interrupted, the tails line will be drained back to the plant. A floor sump and a vertical sump pump have been provided to pump this material back to the flotation tailings thickener.

The process and domestic water is pumped from barges located on both tailings polishing ponds. The barge houses contain two (2) reclaim water pumps, each with capacities of 700 m^3/h and one (1) de-icing pump. A combined process/fire water reservoir with process water distribution pumps and fire water pumps are also provided. The process water reservoir capacity is 2,560 m^3. Part of the water will be treated prior going back to the concentrator.

The water reservoir will have four vertical turbine pumps to supply process water to meet all plant needs. Four (4) firewater pumps to supply water for firewater protection are also included. One of the pumps from the process water reservoir will feed a water treatment plant and the clean water will be used for reagent preparation, domestic water and gland seals for the slurry pumps. Clean water will be stored in a 5,790 mm diameter x 6,100 mm height tank. The design pressure for the seal water system is 120 psig.

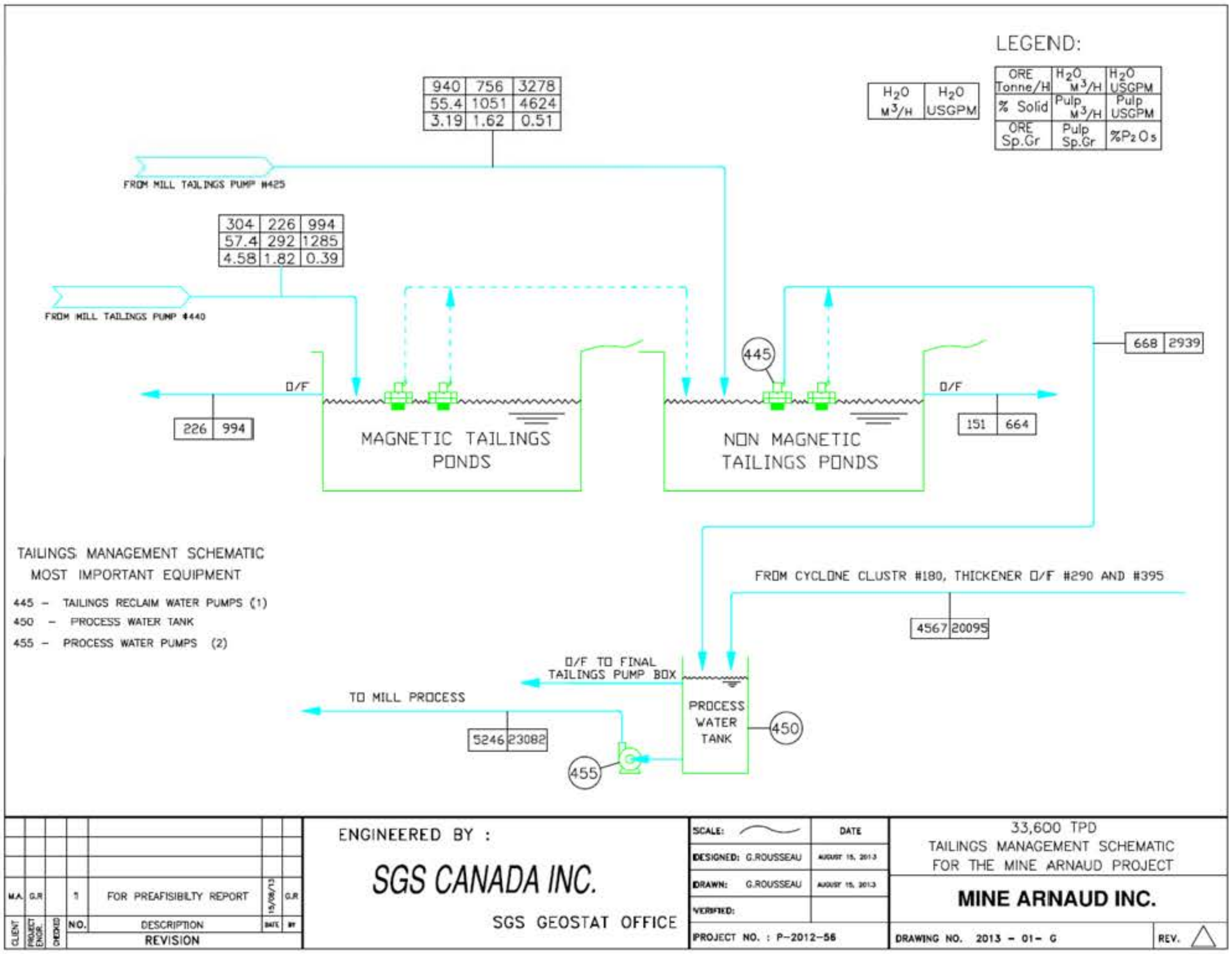


Figure 17-7: Tailings Management (Schematic)

Table 17-3: Mill water balance input parameters

Mean Annual Precipitation (mm)	1,156
Mean Annual Potential Evapotranspiration (mm)	450
Nominal Daily Ore Production (dry metric tonnes)	33,600
Mine Life (years)	28
Apatite concentrate (% by mass)	39%
Apatite Concentrate (m^3/tonne)	0.312 m^3/tonne
% Water in Ore	5%
% Solids in Ore	95%
Non Magnetic Tailings dry density (tonnes/m3)	3.19
Magnetic Tailings dry density (tonnes/m3)	4.58
Water to non magnetic tailings pond (m^3/hour)	745
Water to magnetic tailings pond (m^3/hour)	226
Overall mill process water requirement (m^3/h)	5246
Water recirculate within the mill (m^3/h)	4567
Process mill water from tailings ponds (m3/hr)1	≈≈668 m^3/h (See note 1)
Non magnetic tailings (% by weight of total tailings)	75.3%
Non magnetic tailings production rate	940t/hr
Non magnetic tailings solids content (% by weight)	55.5%
Non magnetic tailings water content	745 m^3/hr
Non magnetic tailings pulp content	1,036 m^3/hr
Non magetic tailings specific gravity (tonne/m^3)	3.19
Magnetic tailings (% by weight of total tailings)	24.70%
Magnetic tailings production rate	304 t/hr
Magnetic tailings solids content (% by weight)	57.4%
Magnetic tailings water content	226 m^3/hr
Magnetic tailings pulp content	292 m^3/hr
Magnetic tailings specific gravity (tonne/m^3)	4.58 t/m^3

NOTES:

1. IT IS ASSUMED THAT ALL OF THE MISSING MILL PROCESS WATER WILL BE SUPPLIED BY BOTH TAILINGS PONDS

2. THE MILL CLEAN WATER WILL ALSO BE SUPPLIED BY THE TAILINGS PONDS AND AFTER TREATMENT WILL SERVICE THE FOLLOWINGS: LUNCH ROOM, TOILETS, SHOWERS, LABORATORIES, REAGENTS PREPARATION, PUMPS GLAND SEALS, ETC.

WHEN USED THE DOMESTIC WATER WILL REPORT TO THE SEPTIC TANK AND LEACHING BED

17.2.9 Reagent Preparation

Wheat Starch (WW82)

This reagent is used for depression of iron oxides during apatite flotation. The starch is received by trucks and unloaded pneumatically into a 60 tonne vertical silo. Wheat starch is reclaimed from the silo by a conveyor into a 5,110 mm dia. X 5,490 mm mixing tank (with agitator) and equipped with a 300 kW electric heater.

Warm fresh water and liquid sodium hydroxide at 10% concentration is added to the mixing tank for dilution and causticization. Strong agitation is required to obtain a homogenous mixture. The 2.5% starch solution is pumped by a 7.5 kW (10 HP) transfer pump to a 5,830 mm dia. X 6,100 mm distribution tank. The solution is distributed through a closed loop pipe system to the flotation circuit addition points by a 7.5 kW (10 HP) pump.

Soybean Fatty Acid (LIACID 1800)

This reagent is an apatite collector for flotation. Soybean Fatty Acid is received as a liquid by trucks. The trucks are unloaded into a 50 tonne storage tank (4,060 mm dia. X 4,860 mm) located indoor. The tank is completed with an outdoor vent to release vapour to the atmosphere and a 200 kW solution heater.

The Soybean Fatty Acid is pumped from storage to a 4,020 mm dia. X 4,480 mm mixing tank equipped with a 3.7 kW (5 HP) transfer pump. The mixing tank will have an agitator and heating coils. Warm fresh water and liquid sodium hydroxide at 10% concentration are added to the mixing tank.

The 2.5% collector solution is transferred into a 4,620 mm dia. X 4,880 mm distribution tank from which the solution is distributed to the flotation circuits addition points by a 7.5 kW (10 HP) distribution pump, through a closed loop pipe system.

Sodium Hydroxide (NaOH)

Sodium hydroxide is used for pH adjustment in the flotation circuit and for the preservation of the starch and the Soybean oil solutions. It is received as a liquid at 50% concentration in 20-tonne tank trucks. The viscosity of the 50% solution rises rapidly below 25°C and the liquid freezes at about 10°C.

The trucks are unloaded into a 4,120 mm dia. x 4,880 mm heated storage tank. From the storage tank the solution is transferred to a 3,410 mm dia. X 4,270 mm distribution tank and diluted to 10% for distribution. At 10% concentration the freezing point is lowered to minus 12°C so that heating of the liquid will not be required. A 7.5 kW (10 HP) transfer pump is used to distribute sodium hydroxide to the starch and fatty acid mixing tanks and to the flotation circuits.

Hydrated Lime

Hydrated lime is used as an aid for flocculation of flotation tailings in the tailings thickener. The lime is received pneumatically by trucks, which discharge into a 60 tonne vertical silo. Lime is reclaimed from the silo by conveyor as required to a 5,110 mm dia. X 5,490 mm mixing tank completed with an agitator.

Warm fresh water is added to the mixing tank for dilution. The lime solution is distributed through a closed loop pipe system to the tailings thickener addition points by a 7.5 kW (10 HP) pump.

Flocculant (Flomin 905 MC)

Flocculant is used to improve the sedimentation and increase the settling velocity of particles. The flocculant will be delivered in 750 kilogram bags. It is emptied in a hopper where the contents are conveyed to the mixing tank through a polymer eductor assembly. The 0.5% w/w flocculant solution is transferred into a distribution tank. The solution is pumped to a secondary dilution system with static mixer that re-dilutes the flocculant at 0.05% w/w prior mixing it to the thickener feed slurries.

17.3 Mass and Water Balances (see above flowsheets)

Based on the design criteria developed for the process plant and on the proposed flowsheet, a mass and water balance for 1,400 tph has been developed for an average P_2O_5 grade of 4.59% and an ore specific gravity of 3.40 t/m^3.

The mass balance of the plant was calculated to provide tonnages and flow rates to different sections and equipment in the plant. However, the throughput, weight recovery and product grade will vary depending on ore characteristics, such as ore hardness, magnetite content and P_2O_5 grade of the feed. Typically, the instantaneous throughput of a SAG-Ball mill grinding circuit varies by ± 15%. Therefore, additional capacity has been included as a requirement when selecting the size of equipment in the downstream process. In some cases, the solids density can be adjusted to a certain limit to compensate for the variation in throughput. The detailed plant mass balance is provided in Appendix. This mass balance from which Tables 17.3 and 17.5 are issued, is based partly on pilot plant results held at COREM and on COREM' reports from projects T1405 and T1518. The results in these tables are deemed representative of the plant first ten years of operation but will vary accordingly to the P_2O_5 grade, ore specific gravity, magnetite content or minerals present in the feed. Material losses to the tailings occur at the LIMS, dewatering cyclones and flotation. The weight recoveries at each step are shown in Table 17-4.

Table 17-4: Weight Recovery by stream

Stream	%Weight from fresh feed
Mill feed	100.00%
LIMS Mag product	21.71%
2nd stage dewatering cyclone	0.79%
Flotation feed (rougher)	77.50%
Rougher concentrate	13.50%
Rougher tails	64.00%
Scavenger tails	1.60%
Cleaner feed	15.29%
Cleaner tails	4.64%
Cleaner/scavenger concentrate	0.18%
Cleaner concentrate (Final concentrate)	10.67%
Scavenger tails	62.43
Cleaner/scavenger tails	4.46%
Final Flotation tailings	66.86%

The 21.7% w/w recovery for the magnetic product at the LIMS circuit is supported by the mass balance computation for a feed grade of 4.59% P_2O_5. This value (probably on the high side) is deemed to be an average for an ore blend consisting of 44.5% Railroad, 19.4% Nelsonite and 36.1% Upper .

A more exact value for the %WRec should be in the 20.0% range since the lock cycle test and the pilot plant run of September-October 2012 reported magnetite concentrate weight recoveries of 20.12 and 20.04% for head grades of 4.69 and 4.62% P_2O_5, thus falling almost perfectly on the regression line of the laboratory bench flotation tests and pilot plant campaigns of 2011 (Figure 17.8).

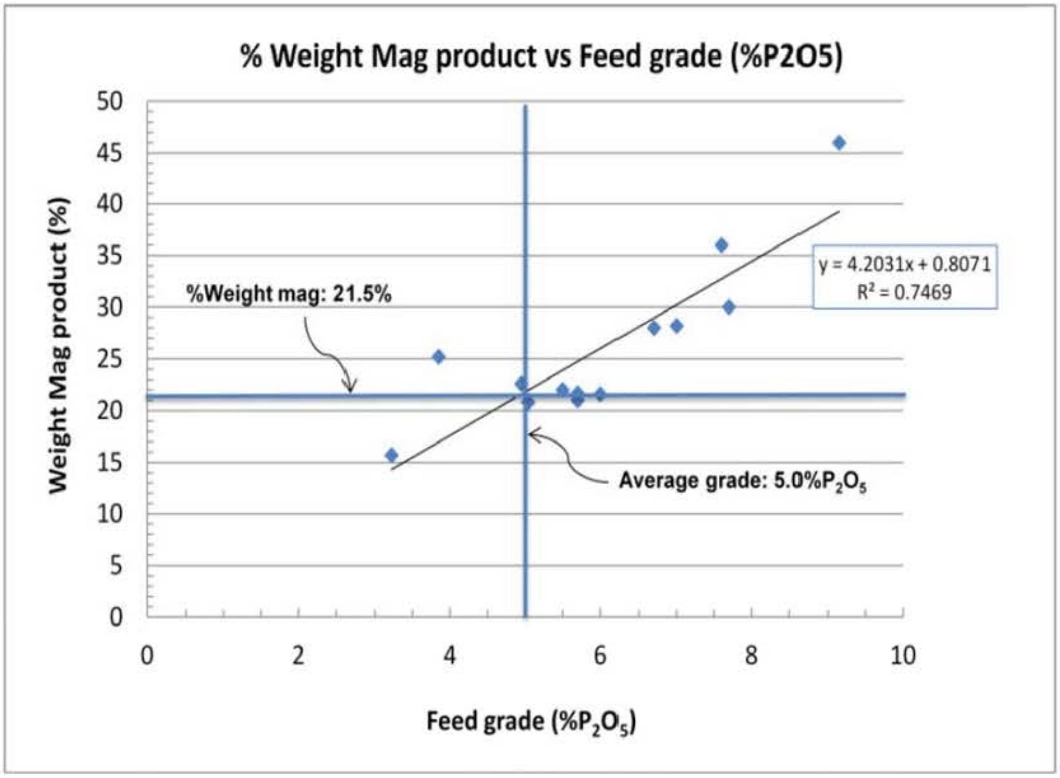


Figure 17-8: Relation of Feed Grade (P2O5) and Magnetic Product Removed at LIMS Circuit

The flotation weight recovery by stream as show in Table 17-5 is based on the results of the COREM's lock cycle test of September 2012 (Corem's Project T1405) plus the five lock cycles tests of June 2013 (Corem's Project T1518). The chemical assays have been balanced and the apatite concentrate weight recovery and P2O5 %recovery for each stream calculated.

Table 17-5: Relation between the % concentrate weight recovery and the head grade

Wrec	P2O5 feed
%	%
8.03	3.51
9.85	4.18
10.75	4.69
10.82	4.71
13.61	5.56
16.29	7.26

Where: %feed P_2O_5 and WRec $R^2 = 0.991$ and %WRec = 2.233 x %Feed P_2O_5 + 0.426

Based on COREM' pilot plant testwork results the P_2O_5 head grade increases by around 1% at the LIMS separation. There is no significant grade change at the dewatering stage. Based on a mill feed of 4.59% and a flotation feed of 5.60% P_2O_5, Table 17-6 tentatively gives an example of the average P_2O_5 grade variations for each flotation stage.

Table 17-6: Average P2O5 grade per process stage

Process stage	%P2O5
Fresh feed	4.59
Flotation feed	5.60
Rougher Concentrate	32.43
Scavenger tails	0.34
Scavenger/Cleaner tails	2.87
Flotation concentrate	39.0
Final tails	0.51

17.4 Water Balance

The process plant water balance is presented in the above flowsheets and is detailed in Appendix 4. This represents an average operating day with 91.34% yearly availability or 21.92 hours per day. This water balance covers only the process. The fire protection system, the global site water balance, which includes the tailings pond precipitation & evaporation, mine dewatering, site drainage and others, is included in the environmental part of the report, Section 20.

17.5 Plant Design Layout

17.5.1 Equipment Sizing and Layout Areas

The equipment list was prepared and the equipment was sized according to the design criteria developed from the flowsheet drawings and the mass balance. Part of the equipment list is provided in Appendix 5 (Process Design Criteria)

However, the throughput will vary depending on ore characteristics. Accordingly, additional capacity has been included as a requirement when selecting the size of the equipment in some cases. The solids density can be adjusted to a certain limit to compensate for this variation. For most of the equipment, a design factor of 10% to 15% was applied to select the equipment size. For the pumps, a 5% factor on the total calculated dynamic head has been applied to select the pump motor horsepower.

17.5.2 Crushing Area

The primary crusher mechanical availability has been estimated at 50% or 12 hour per days. Large crushers require major maintenance and cleanup. To provide the required capacity at the design feed rate and product size, a single 54-75 (1,372 mm x 1,905 mm) gyratory crusher or equivalent is required. Based on a nominal capacity of 2,800 tph, the up-time of the crusher is 92%. The up-time is the percentage of operating time required to crush at nominal capacity the average feed (30,912 tpd) for a time period (12 h/d). Initially Roche-Ausenco sized a smaller crusher, but Mine Arnaud requested a 54-75 crusher, based on its experience with similar operation.

17.5.3 Stockpiling Area

The ore stockpile has a live capacity of approximately one day, or 30,000 tonnes. The ore stockpile reclaim feeders have been selected so that a minimum of one feeder will be in operation at all times to provide feed for the SAG mill. This allows for maintenance of feeders during operation.

17.5.4 Grinding Area

The SAG mill circuit will grind the material from 170 mm to a P_{80} of 2 mm. Based on ore grindability index, simulation work and adding drive loss, the installed power required for grinding 1,400 tph at the SAG mill is estimated at 12 MW (16,000 HP). The SAG mill size is 10,160 mm dia. X 5,330 mm long (34 ft x 17.5 ft) and is driven by dual pinion drives of 6 MW (8,000 HP) each. The SAG Mill will be variable speed for adding flexibility to the grinding circuit. For the secondary grinding mills, the ball mill Work Index, simulation and drive loss have shown that the installed power requirement for grinding the SAG Mill circuit discharge to the required size of 125 microns is 12 MW (16,000 HP). The grinding will be done by two parallel ball mills, with 6 MW (8,000 HP) motors each. Both ball mills size is 6,100 mm dia. X 9,140 mm long (20.0 ft x 30.0 ft). The classification at the SAG Mill will be done by two (2) double deck vibrating screens of 2,440 mm x 7,320 mm (8 ft x 24 ft). The ball mills will be in closed circuit with two (2) classifying cyclone clusters, one per mill, of four (4) 31.5 inch (800 mm) cyclones.

17.5.5 Low Intensity Magnetic Separation Area

From the design criteria, the calculated tonnage going to the low intensity magnetic separation circuits is 1,400 tph at 30% w/w density from which the equipment is sized. Four double drum magnetic separators have been selected (one per flotation line). Each magnetic separator is independent from the others except for the magnetic and non-magnetic product pump box, allowing the bypass of any separator while maintaining magnetic separation in each of the flotation lines.

Two (2) stages of dewatering cyclones are required to dewater and recover more than 96% of the P_2O_5 at the cyclone underflow. The cyclone underflow design percent solid is 50% by weight. For each line a total of 18 units are required at the primary stage and 24 at the second stage. The primary cyclones have a diameter of 250 mm diameter while the cyclones of the secondary stage have a diameter of 150 mm. The cyclone overflows from the primary stage are pumped as feed to the secondary stage. The inlet pressure, for both the primary and secondary stages, is 30 psig.

The thickener sizing for magnetic tailings was done by Roche-Ausenco using the specific design capacity obtained for the combined tailings in the testwork performed by Outotec and Delkor. The thickener for the magnetic product will be operated without flocculant. Incorporating a design factor of 10%, the required thickener diameter was estimated at 23 meters.

17.5.6 Flotation Area

The number of conditioning tanks and sizing is selected in order to have one tank per reagent addition. Roche-Ausenco based the retention time on pilot plant testwork and on the advice of KEMWorks. Table 17-7 gives the effective retention time for each conditioning tank using the calculated flow rate from the mass balance.

Table 17-7: Flotation Conditioning Tanks Retention Time

Tanks	Retention time (minutes)
pH regulation conditioning tanks	2
Starch addition conditioning tanks	5
Collector addition conditioning tanks	5
Cleaner/scavenger conditioning tanks	2

Roche-Ausenco sized the flotation columns according to the following information:

Results of column flotation pilot plant program conducted by COREM personnel;

Discussions between Roche and ERIEZ MINERALS FLOTATION GROUP (EFG);

EFG's past experience with phosphate ores in Brazil and elsewhere.

Due to the relatively coarse particle size distribution, EFG recommends three column cells configured as two (2) Roughers in parallel followed by a single scavenger. This configuration is based on EFG expertise with phosphate ores in order to ensure that the concentrate quality and production rate can be maintained over the entire range of expected operating conditions. The flotation rate (kinetics) for this ore is very fast and therefore residence time requirements will not limit the capacity and performance of the columns. The main design criteria for this application will be the carrying capacity, a measure of the maximum production rate for the column. When designing large diameter columns, it is necessary to de-rate the performance of pilot plant columns to take into account the decreased ratio between the available lip length and column area. The columns have been designed to operate in a range of carrying capacities of 3.0 tph/m2 to a maximum of 4.0 tph/m^2.

Because the final concentrate specifications for Mine Arnaud are very stringent, it is important to maintain a high quality product at all times. This means that there will be variations in the stage recovery, particularly for the coarse size fractions, which could lead to unacceptable low recoveries. By using two columns in series, the first stage (roughers) is operated to produce a very high-grade concentrate and the second stage (scavenger) is operated to regulate the circuit recovery. The effects of variations in the feed grade, feed tonnage and pulp density will be lower than for a single column stage operation.

Based on the pilot plant results, the flotation kinetics for the cleaning stage is very high. Residence time will not be an issue, and limiting factor will be carrying capacity. To ensure a consistent high quality product, Roche chose a slightly lower carrying capacity range and the columns are expected to operate between 2.5 - 3 tph/m^2. EFG helped selecting two columns operating in parallel for the cleaners. Operating at higher carrying capacities could result in a decrease in selectivity due to entrainment of fine impurities. To counter the entrainment, wash water is added more aggressively at the cleaner columns than at the rougher columns.

A cleaner scavenger stage will be utilized to treat the cleaner tailings to recover any free apatite lost in this unit operation. The anticipated mass recovery from the cleaner scavenger is very low and will be recycled directly to the cleaner.

17.5.7 Thickening Area

The concentrate and tailings thickeners were sized by Roche according to the specific design capacity obtained during the testwork performed by Outotec and Delkor. Both thickeners can use the same flocculant (Flomin 905 MC). Incorporating a design factor of 10%, the required concentrate thickener diameter was estimated at 20 meters and 40 meters was required for the tailings thickener.

17.5.8 Filtering Agent

Prior to apatite concentrate filtration, two (2) surge tanks are used, each providing 8 hours of retention time. These tanks will be useful in case of shut down of the filtration or drying circuits due to maintenance.

The apatite concentrate is dewatered to 8% moisture by using a 34.2 m^2 horizontal vacuum belt filter. Roche-Ausenco sized the belt filter according to the testwork done by Delkor with a filtration rate of 6 tph/m^2.

17.5.9 Drying Area

The dryer was sized by FLSmidth, based on 8% moisture from the horizontal belt filter and a final product moisture content of 1% to ± 0.5% by weight. Based on pilot plant testwork, a product conditioning system is required. The system consists of a single-shaft paddle mixer and slip-stream screw conveyor to ensure/control the final product moisture of 1% ± 0.5%. The paddle mixer is designed for extended residence time to ensure a uniform/homogeneous product by providing intimate solids mixing of dryer product at 0.5% moisture and wet feed at 8% moisture.

Electric air heaters of approximately 18.5 MW totals are used as a heat source to heat the ambient air to the required dryer inlet temperature (415°C). The main induced draft fan required to draw the process drying air (175,000 m^3/hr) through the system is designed for heavy-duty application. A single stage bag-house system will collect the product prior to discharging the process gases to the atmosphere.

17.5.10 Reagent Preparation Area

Roche-Ausenco based the sizing of the reagent tanks on the consumptions obtained during the COREM 2011 pilot plant testwork. Table 17-8: Reagents Addition and Consumption gives an indication of the probable reagent consumption but recent LCT's also done at Corem have shown that reagent consumption will vary and will have to be closely monitored to the mineralogy of the ore.

Table 17-8: Reagents Addition and Consumption

Reagent	Addition Point	Consumption (g/t feed)
Depressant – Wheat Starch (WW82)	Starch addition conditioner tank	300 g/t
	Cleaner/Scavenger conditioner tank	25 g/t
Collector – Soy Bean Oil	Collector addition conditioner tank	165 g/t
pH regulator – NaOH	PH regulator conditioner tank	710 g/t
Flocculant – Flomin 905MC	Apatite concentrate thickener	15 g/t
	Tailings thickener	15 g/t
Setting agent – Lime	Tailings thickener	30.8 g/t

Except for flocculant, the others reagents are delivered bulk truck to the mine site.

17.5.11 Silo and Load Out

Silo and load-out facilities at the mine site were sized by Roche-Ausenco based on 2 days of production. There are two (2) silos and each silo has a capacity of 4,500 tonnes. Each silo has a dedusting system, air slide for silo discharge and railcar loading system.

17.5.12 Building Layout

SGS Geostat has reviewed the Mill General Arrangement (Roche-Ausenco' drawings # 1848-03-DR-GE-002, 003 and 004) and is of the opinion that the selected mill machinery and equipment disposition is optimal and if there is no change in the flotation circuit, the mill layout as designed is probably there to stay.

The crusher building will house the gyratory crusher with auxiliary equipment and the tail end of the stockpile feed conveyor. The gyratory crusher building dimensions are 8.4 m x 24 m x 31 m height.

The crushed ore stockpile is contained within a storage dome, 70-meter diameter footprint and a height of 30 meter. Live capacity is approximately 30,000 metric tonnes. Under the stockpile, one concrete reclaim tunnel is installed to recover the stored material. Five (5) apron feeders under the crushed ore stockpile will discharge on the mill feed conveyor.

The main processing building houses the grinding, magnetic separation, flotation circuit, concentrate filtering as well as offices, mechanical and electrical shops for plant maintenance. Two electrical rooms are planned to supply power to the grinding mills, apatite dryer and other various areas. A separate adjacent building is also required for the flash dryer. This building must be separated from the main process building due to the height of the chimney. The dimensions of the building are 63 m x 143 m x 27.8 m high clear under the roof trusses for the grinding section and 22.8 m high for the flotation section. An extension to the building 28 m x 20.4 m x 43 m high clear under the roof trusses will house the dryer on the east side of the main building.

The magnetic tailings thickener, the flotation tailings thickener and apatite thickener are all located outside, on the west and south sides of the building.

17.6 Instrumentation and P&ID

Roche-Ausenco developed the preliminary P&ID's based on the flowsheets and are presented in their Feasibility Study at Appendix 1, Section 03, drawings 1848-03-DR-IC-001 to 007. The P&ID's were necessary to establish the instrument list (Appendix 10.5 of the Feasibility Study), the piping list and the requirement for the number of valves. By using the mass and water balance, it was possible to size the instruments and send RFQs to establish their costs.

All the instruments and control logic will be integrated in PLC/DCS. The instruments will be wired to the PLC/DCS through digital and analogue I/O modules. The process control will be done through operator stations linked to PLC/DCS. A brief description of the principal control loops on each flowsheet is presented in the following sections.

17.6.1 Grinding

The ore feeding the SAG mill will be transported by a conveyor equipped with belt scale. The reject material from the double deck vibrating screens will be also weighed and forwarded to the SAG mill. The two measurements will be used to control the production rate by varying the speed of the feed conveyor. The flow of process water to the SAG mill will be controlled according to the desired percentage solid.

The level of the SAG mill discharge pump box is controlled by modulating the speed of the pump which transfers the slurry to the ball mill discharge pump box. The level of the latter will be maintained with the addition of process water. Variable speed pumps at the ball mill discharge pump box outlet ensure a constant pressure to the classifying cyclone clusters. Flow and density are also measured at the inlet of each cluster. Those measurement points can be used for further control strategies.

17.6.2 Low Intensity Magnetic Separator

The grinding cyclones overflow is pumped through the LIMS double drums. The titano-magnetite material is going to the mag tailings transfer pump box where the level is controlled with the addition of process water. The titano-magnetite is then pumped to the mag tailings thickener. The density at the outlet of the thickener is measured and controlled by modulating the speed of the pump. Two level transmitters are used for slurry level and bed mass load measurement at the bottom of the thickener. Additional instruments are included to measure the torque and the position of the rake mechanism. Those measurements are used for operator information and interlock purposes. The overflow of the thickener is directed to the process water tank.

The apatite slurry from the LIMS flows to the first stage dewatering cyclones feed pump box. The variable speed pump at the outlet is used to control the level of the pump box. The density of the slurry is measured with a coriolis densimeter installed on the line joining the entrance of the first stage dewatering cyclone. The density is controlled by adding process water in first stage dewatering cyclones feed pump box. The overflow of the first stage dewatering cyclones flows to the second stage dewatering cyclones feed pump box. The variable speed pump at the outlet is used to control the level of the pump box. The overflow of the second stage cyclones flows to the dewatering cyclones distributor and from there to the non mag tailings thickener. The underflows of the first and second stage report to the rougher conditioner tank.

17.6.3 Flotation

Chemical treatment must be done on first and second stage dewatering cyclone underflows, where three chemical products are used. Two of them are regulated by flow control loops and their set point will be determined from a production rate ratio. The third product will be added by a control valve and metered by the pH regulation loop.

The treated second stage dewatering cyclone underflow is mixed in the rougher stage #1 distributor. The flotation process uses several flotation column cells but the control strategy is similar from one flotation column cell to the other. The level is measured with an ultrasonic and float assembly and controlled with a control valve at the outlet. The sparging system is created by the combination of the slurry pumped from the bottom of the cell mixed with an air supply. A vortex flow meter and a flow control valve are used to obtain the required air supply. The slurry is pumped with a variable speed pump to ensure constant pressure at the cavitation tube.

17.6.4 Concentrate Thickening, Filtering and Drying

The concentrated slurry coming from the flotation process flows to the apatite concentrate thickener feed tank and is mixed with the flocculent. The thickener used for apatite is similar to the thickener used to treat the titano-magnetite. The same instrumentation is used: density, level, bed mass load, torque and position transmitters and their utilisation are the same.

The concentrate slurry is routed to the horizontal vacuum belt filter. The vacuum under the belt will be controlled with a control valve connected on one side to the vacuum pipe and on the other to atmosphere in order to optimize the operation of the belt filter. The filtered material will be weighed with the belt scale and forwarded to the drying process, allowing to obtain the production rate at this point. The drying process is relatively simple and consists essentially in the handling of the material with the right conveying sequence and interlock. The final belt conveyor is also equipped with the same weighing system for measuring purposes.

17.6.5 Tailings Thickening, Disposal and Water Suppy

The tailings thickener feed tank accepts principally the tailings from the flotation process and the second stage cyclone overflow. The tailings will be thickened the same way as titano-magnetite and apatite slurries. The thickener overflow is recovered in the process water reservoir. Essentially, the instrumentation for this part of the process is composed of level transmitters for the different tanks and tailings ponds. Magnetic flow meters are used to measure the water from the polishing pond discharged to the environment and other tailings that are routed in different ponds.

Process water is composed of the recovered thickener overflow and supernatant waters from the tailings ponds tailings. Water level in the firewater tank is controlled by two control valves with different operating range. A level switch is used to detect a low level of water and open an electro valve to make the filling with fresh water if ever necessary. Process water level is ensured by the overflow of the firewater tank. The pressure will be controlled with control valves for the main process water circuit and the firewater circuit.

17.6.6 Reagents

Each reagent has its own preparation process. Magnetic flow meters, control valves, electro valves and level transmitters will be used to meet the proper recipe for each reagent. In most cases, each reagent will have a controlled pressure after the distributing pump. The pressure will be controlled by a control valve on the recirculation line which returns the product to its own distribution tank.

17.7 Capital and Operating Costs

SGS has reviewed the CAPEX AND OPEX for the constructuction and operation of the mill and found that they are definitely in an acceptable range for the size of the project. Because more tonnes of ore per year will be processed compared to the Roche-Ausenco Feasibility Study, the cost per tonne of feed will be somewhat lower but this cost decrease will be somewhat offset by the increase in the cost of living since the publication of the Roche-Ausenco Report. (See Article 21)

18. Project Infrastructure

Mainly all the information contained in this section was derived from the feasibility study (prepared by Roche-Ausenco) dated of February 2012 and has not been updated. SGS considered that this information is still relevant in the context of this Preliminary Feasibility Study. Only sub-sections 18.1.4 to 18.1.7 were added by SGS.

18.1 Major Site Infrastructures

18.1.1 Bulk Earthworks, Landscaping, Fencing

Following the completion of the geotechnical work and topographic analysis, the site pad has been implemented at a location showing sound properties for the concentrator and administration buildings' foundations. The site pad area is evaluated at 210,000 m^2. It is assumed that 220,000 m^3 of rock will be blasted and 305,000 m^3 of rock will be excavated, of which 90,750 m^3 will be used to fill and build the pad. The top of the pad will be covered with 225 mm thick MG-56 for a total of 113,400 t of material. A standard galvanized steel fence, 1.8 m high with three rows of barbed wire will secure the site pad perimeter. The fence's total length is evaluated at 1,830 m. Access to the site is allowed by two 4-m wide gates and one pedestrian entrance turnstile. A second fence will cover the South, East and West ends of the open pit.

18.1.2 Plant Building

The concentrator plant is divided in two separate buildings, the concentrator and the flash dryer building (Appendix 1, section 03, drawings 1848-03-DR-GE-002 to 004 of the Feasibility Study). The main building covers an area of approximately 9,000 m^2 with a maximum elevation of 30 m, while the flash dryer building covers 570 m^2 and a maximum elevation of 43 m. Due to the significant height of the flash dryer, it is preferable that it has its own structure, isolated from the concentrator building. The flash dryer building's design was based on suppliers' general arrangement drawings and calculations. The concentrator building was based on similar previous projects as well as on current project's equipment layout and calculations. The procedure used to prepare the quantities and costs associated with the civil and structural work is as follows:

- The scope of work for each area was determined from overall general arrangement drawings;
- The geotechnical report from Lab Journeaux Inc. was reviewed and underground conditions were assessed for each area (Appendix 8.1 of the Feasibility Study);
- Preliminary Roche-Ausenco and Vendor drawings for each area were assembled and reviewed;

- Preliminary calculations were performed for structures and foundations from which quantities were determined;
- A vendor quotation was also obtained for the supply of the domed storage structure and cladding;
- Quotations were obtained for the crusher retaining wall, the domed storage structure and cladding, and reviewed with vendors and contractors;
- Quantities arrived at were compared with those culled from Roche-Ausenco records and similar past projects;
- Unit prices were developed by Roche-Ausenco based on similar past projects and reviewed with vendors and contractors. Current unit prices were applied to the estimate.

18.1.3 **Auxiliary Building – Non-Process**

The service and administration building (Appendix 1, Section 04, drawings 1848-04-DR-GA-001 & 002 of the Feasibility Study) will contain offices on the second floor, while laboratory, nursery, storage area, vehicle maintenance area, machine shop, mechanical and electrical rooms, a bay for ambulance and fire truck, a washing bay, as well as employee services (cafeteria, locker rooms, training rooms, etc.) are located on the main floor. Its dimensions and general arrangement are based on approximately 100 workers per shift as well as a maximum of 16 trucks required for the mine operations. The service building has a rectangular shape of 42 m x 96 m. It is assumed that the soil bearing capacity will be sufficient to allow standard foundations (no pilings).

The garage itself will contain the necessary equipment to maintain and repair heavy mining trucks and other mobile equipment (loaders, lift trucks, pickups, etc.). The main equipment being overhead cranes, compressors, machine shop tools, etc. Costs for the equipment were based on a mix of quotations, experience from previous projects and catalogue pricing.

A 20 m x 16 m covered cold storage area and a 20 m x 23 m mechanical inventory pad adjacent to the administration building have also been accounted for in the estimate.

The gatehouse dimensions and general arrangement is based on a recent similar mining project in Sept-Îles area, whose price was updated with inflation. The gatehouse building is rectangular and measures 7.5 m x 24 m. Again, Roche-Ausenco considered that the soil bearing capacity is sufficient to allow for standard foundations (no pilings).

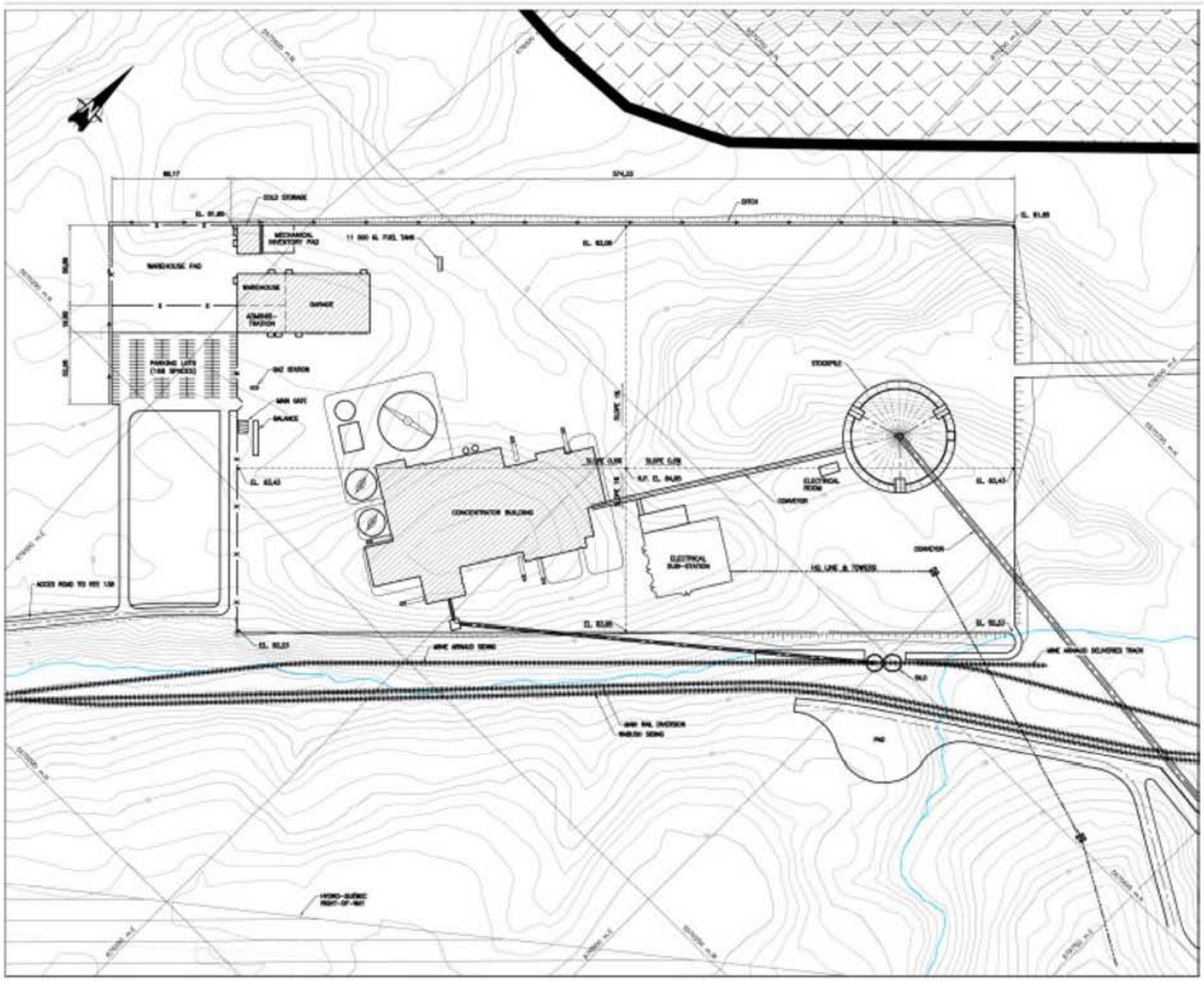


Figure 18-1: Concentrator Area Layout

18.1.4 **Rock Waste Dump**

A waste rock stockpile will be located approximately 1.1 km from the mine entrance. This stockpile will be composed of rock material that does not contain enough mineralized material to be economically processed. It will have a volume of approximately 42,000,000 m^3 and will be strategically located to minimize hauling distances, and thus the size of the mining fleet. The pile will be accessed by a 10 % access ramp and will have an overall slope angle of 32 degrees. In addition of this rock waste dump, it is planned to use the West section of the open-pit to dump waste rock. It will be possible to store approximately 17,000,000 m^3 of material in this area, starting around year 20. Refer to next Figure for a visual sketch of these waste piles.

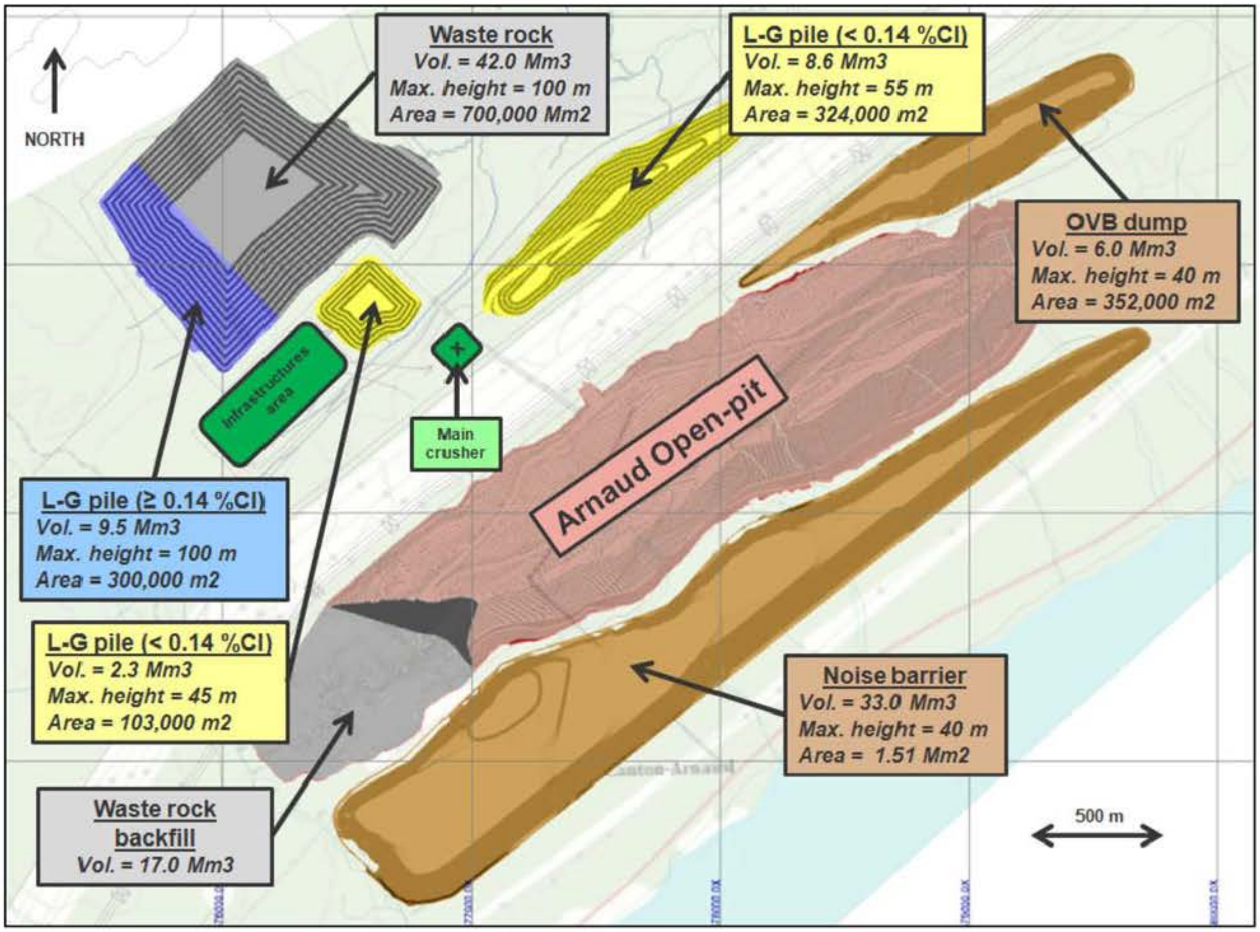


Figure 18-2: Piles locations

18.1.5 Overburden Waste Dump

An overburden material stockpile will be erected at approximately 800 m from the mine entrance/exit.This stockpile will be primarily composed of top soil material that will be remove in order to reach hard rock containing mineralization. It will have a volume of approximately 6,000,000 m^3. The pile will be accessed by a 10 % access ramp and will have an overall slope angle of 20 degrees. Refer to previous Figure for a visual sketch of this waste pile. The stockpile footprint is limited on each side by the final pit crest, the Hydro-Quebec power lines and the railway. The resulting stockpile capacity is therefore not sufficient to dispose all of the overburden material. A second overburden stockpile will be available (refer to next section entitled Noise barrier) and the remaining material is planned to be used to cover a portion of the tailing ponds.

18.1.6 Noise Barrier

A noise barrier will be erected along the South wall of the open pit at approximately 100 m of the open-pit crest. The barrier will be used to shield the project from the local environment (houses, roads, etc.) and to mitigate noise and dust that will be produce by the mining activities. The noise barrier will be composed of a core of waste rock overlain by overburden material. Approximately 33,000,000 m^3 of material will be required for this construction (50 % overburden and 50 % waste rock). The noise barrier will be accesed via 10 % access ramp and will have an overall slope of 20 degrees. Refer to previous Figure for a visual sketch of this infrastructure.

18.1.7 Ore Stockpiles

In order to maximize the profitability of the project, highgrading will be performed between years 5 and 20 of operations. In others terms, the minimum cut-off grade will be raised during those years, allowing to produce more tonnes of concentrate for the same amount of tonnes treated. The material having a grade between the marginal cut-off grade and the raised cut-off grade will therefore be stockpiled near the crushing plant, to be treated at the end of the mining operations (year 28 and subsequent). Through the first 20 years of mining operation, approximately 47.4 Mt of mineralized material will be stockpiled. However, it is important to remember that only 53 % (24.9 Mt grading at 2.39 %P_2O_5 and 0.1136 %Cl) is considered to be treatable in the context of this PFS (refer to Section 16 for more information). Due to physical constraint limitation, two piles of low grade treatable ore will be built in proximity to the main crusher to minimize hauling costs. The piles will be accessed by a 10 % access ramp and will have an overall slope of 32 degrees. The low grade pile that is not considered to be treatable will be in the vicinity of the main waste pile, but disposed of separately. Refer to previous Figure for a visual sketch of these infrastructures.

18.1.8 Roads, Overpasses & Parking

The roads included in this Study are:

- A paved road connecting the Project site to Highway 138, including three culverts;
- A haul road connecting the pit and crusher area to the concentrator and waste rock dump area;
- An overpass which will allow the haul trucks to go over the railway to reach the waste rock dump and concentrator area;
- A road to access the dikes around the tailings basin;
- A road to access the explosives storage area from the concentrator;
- A road to access the crusher from the mine on the eastern side of the site will be added at Year 5.
- A parking lot connected to the main access road is located east of the concentrator and a provision was made for lighting poles.

18.1.9 Fuel Storage

The fuel storage consists of a diesel/gasoline fuel distribution station including:

- Two 40,000 L diesel tanks;
- One high-speed fuel transfer pump for mine trucks;
- One low-speed fuel transfer pump for diesel pick-up trucks, lifts and loaders;
- One 20,000 L gasoline tank;
- One low-speed gasoline transfer pump for gasoline pick-up trucks and small tools running on gasoline;
- One 40,000 L diesel fuel tank for the concentrator generator;
- One fuel management system.

18.1.10 Fresh and Potable Water

In addition to potable water and treated water will be required for process purposes (gland seals and preparation of reagents). However, water recirculation within and to the concentrator will be maximized to reduce water consumption.

For the potable water requirements, a 3 m^3/hr filtration and disinfection unit as well a 12 m^3 potable water tank are planned. A 38-mm HDPE pipe will be installed between the plant and the office building.

18.1.11 Safety and Fire Protection

18.1.11.1 General

The safety equipment includes:

- A 4x4 fire truck, with a mini pump and foam system;
- A 4x4 ambulance;
- A first-aid room as well as provision for safety showers, eye washes and fire extinguishers;
- A 2.45 m (8 ft) high fence to secure the open pit area at the South, East and West ends.
- Fire protection infrastructure and systems will comply with NFPA-13-2007 standards and Québec's National Building Code, 2005.
- Standpipes systems will comply with NFPA-14.
- Water loop system will comply with pertaining AWWA standards and NFPA-24.
- Fire pumps and generator sets will comply with NFPA-20.

18.1.11.2 **Water Supply**

Fire protection water will be supplied by the incoming water main entering the site and will be made up a 10" (254 mm) diameter water loop plus 6" (152.4 mm) and 8" (203.2 mm) diameter branches to different buildings. Water storage capacity for the fire system is build in with the bottom portion fresh water reservoir. This volume could only be accessed through the fire protection system.

This loop will also encompass 14 hydrants strategically located for fire fighters usage.

18.1.11.3 **Sprinklers Systems**

Sprinklers systems will be designed according the following criteria:

- Offices: ordinary risks, 0.10 USGPM density and 1500 sq ft area plus 100 USGPM for standpipes, area per sprinkler head 225 sq ft;
- Concentrator and conveyers: ordinary risks, group 2, 0.20 USGPM density and 1500 sq ft area for wet systems and 1950 sq ft for dry systems plus 250 USGPM for standpipes, area per sprinkler head 130 sq ft;
- Garage: extra hazard risks, 0.25 USGPM density and 2500 sq ft area, plus 500 USGPM for standpipes, area per sprinkler head 100 sq ft.

18.1.11.4 **Standpipes Systems**

Standpipes systems will be provided for tall buildings i.e. more than 46 in height (office building and crusher) and will supply 2 ½ fire hose with 500 USGPM at 100 PSI.

18.1.11.5 **Equipment Included**

- Sprinklers (pendant, recessed or upright depending on use);
- Siamese for fire department connexions;
- Alarm valves;
- Fire extinguishers;
- Siesmic restreints as required;
- Accessories;
- Tests and start-up of all fire protection related equipment and systems;
- Design and drawings as required by NFPA.

18.1.11.6 **Fire Protection Systems Description**

Crusher:

- Dry system with 6" water supply in a heated room;
- Standpipe system;
- One electrical fire pump and one diesel fire pump.
- Crusher conveyer (storage):
- Pre-action system with 6" water supply in a heated room;
- Detection system (protecto-wire).

Garage:

- Wet system with 6" water supply;
- Stand pipe system.

One electrical fire pump and one diesel fire pump Storage:

- Dry system with 6" water supply;
- Standpipe system.

Conveyer (from storage to concentrator):

- Pre-action system and 6" water supply;
- Detection system (protecto-wire).

Electrical room

- Non-automated system.

Concentrator:

- Wet and dry systems depending on area served with 6" water supply;
- Standpipe systems;

One electrical fire pump and one diesel fire pump conveyer (from concentrator to silo):

- Pre-action system with 6" water supply.

Silo - no protection provided:

- Transformers (3 pieces);
- Vortex system.

18.1.12 Domestic Waste Water Treatment

Considering 100 workers per shift, a daily flow rate of 60 m³/day has been calculated from a unit flow rate of 200 L/person/day. In order to treat this flow, a compact mechanized plant is recommended. The equipment will be integrated in a 140-m² building, excluding the underground basins. The system includes:

- One 100 m^3 septic tank with filter;
- One 225 m^3 equalization basin;
- Six Rotofix units;
- One 15 m^3 settling tank;
- One 35 m^3 sludge storage basin;
- One UV rays disinfection unit.
- The effluent of this advanced secondary treatment could be either infiltrated into the ground or released into Ruisseau Clet.

18.1.13 Geotechnical Survey

Geotechnical and hydrological investigation was conducted by NJA & Associates during the spring of 2011. The program was concentrated over a large area north of the Hydro-Québec transmission lines corridor where the crusher, concentrator and other buildings would be located. The program is summarized in Table 18-1. The final report could be found in Appendix 8.1 of the Feasibility Study.

Table 18-1: Geotechnical Tests Location

Location	Nos. of Test Locations
General Plant Site	27
Original Ore Storage Dome	3
Crusher Site	3
Proposed Bridge Site Over the Des Rapides River for West Bound Waste Rock Trucks	2
Railway	23
Load Out Silos	5
Waste Rock Dump Site	5 (4 on north side of creek and 1 on south side of creek)
Tailings Storage Area	9
Original Port Silo Location	2

The program consisted in boreholes and test pits to collect samples for laboratory testing as well as installation of standpipes in some boreholes for water monitoring.

The area is bedrock controlled with a high plateau at about elevation 100 to 110 m. It is dissected by deep, well defined, fault and joint controlled valleys which drain the many small ponds in the area. The major principal joints or fault systems trend northeast-southwest and a secondary family, perpendicular to the major fault lines, trends northwest-southeast. These valleys are at their deepest at about 25 to 30 meters at the northwest-southeast trending Clet Creek and the wider west branch of the Des Rapides River which drains north-eastward.

During the period of glacial retreat, the area was inundated by the sea and saline marine clays were deposited over the area in the inland bays and other depressions on the land. In the uplands, the clay deposits are relatively thin in the depressions, but the clays are much thicker at elevations below about 80 m.

The plant site is located to the south-western end of an elongated bedrock ridge which can be levelled into a platform to provide space for all the structures associated with the concentrator. Large volumes of blasted rock will be available for general backfilling on site for outside storage areas. It can also be used for construction of roads and the railway. For the foundations of the plant, thickener, garages, warehouses and administration building carried on bedrock, an allowable bearing value of 2,000 kPa can be used in the design.

The ore storage shed will be in an area where bedrock is at a shallow depth but drops off steeply at the south-eastern edge of the peat bog at elevation 75 on the west side of the site. As such, it will be important to have the ring foundation for the unheated ore storage shed on bedrock.

For the load out silos located along the spur line as they are about 10 meter in diameter and 45 meter high, they should be founded on bedrock. Their location at a relatively shallow depth to bedrock should allow that a bearing value of 2,000 kPa could be used for the design of footings.

The testing at the location of the crusher indicated bedrock near the surface on the higher ground with deep, soft clays at the south end of the tunnel structure. An additional test hole was done at the base of the rock slope to confirm the depth to rock. This was found to be slightly above the level of the clay filled depression or peat bog further to the south. To ensure that the complete structure would be on bedrock, it was suggested that the crusher be relocated a minimum 20 meter further up the slope to the northeast which is certainly a bedrock ridge. This would place the complete structure on bedrock. At this location, the crusher and accompanying retaining wall for the mine truck approach fills to the crusher should all be on bedrock for which a design bearing value of 2,000 kPa can be used. In order to benefit from the advantages of vertical rock cuts, excavations into bedrock for the crusher building and tunnels should be done using presplitting or preshearing techniques as suggested for the tunnel under the ore storage building.

18.2 Electrical Infrastructures

Details of the proposed electrical equipment are contained in document number 1848-01-DC-EL-001 Electrical Design Criteria (Appendix 10.1 of the Feasibility Study) and in the single line diagrams (Appendix 1, Section 04, drawings 1848-04-DR-EL-001 to 008 and Section 08, drawing 1848-08-DR-EL-006 of the Feasibility Study).

18.2.1 Electrical Distribution at the Mine-Concentrator Site

The power supply for the mine-concentrator site installations will be via one incoming 161 kV overhead transmission line from Hydro-Québec fed from Poste Arnaud 735-161 kV substation. A new 161 kV substation including three 161:13.8 kV 40/53.3 MVA transformers will be provided near the concentrator building. The 161 kV overhead line is entirely under the responsibility of Hydro-Québec.

Substation general arrangements drawings 1848-04-DR-EL-011 – 161kV Substation Plan and 1848-04-DR-EL-012 – 161kV Substation Sections are presented in Appendix 1, section 04 of the Feasibility Study.

The 13.8 kV medium voltage system will be used for the primary distribution voltage and for feeding large loads such as the SAG Mill and Ball Mills.

The 13.8 kV distribution circuits run from the 161kV substation electrical building (located adjacent to 161 kV outdoor substation) to secondary electrical rooms conveniently located to load points close to the areas served, where the 13.8 kV distribution voltage will be converted to 4.16 kV and 600 V, using 13.8-4.16 kV and 13.8-0.6 kV indoor dry-type transformers.

The following secondary electrical rooms are planned:

- Concentrator grinding electrical room;
- Concentrator dryer electrical room;
- Garage, warehouse & administration electrical room;
- Crusher electrical room;
- Stockpile electrical room;
- Rail load-out electrical room;
- Tailings electrical room;
- Mine pit electrical room.

18.2.2 Electrical Distribution at the Port Facilities

The power supply for the port site facilities will be taken from an existing 25 kV line. One 25 kV switchgear will supply the power to the transformers feeding low voltage loads. Two electrical rooms are planned for the port loads:

- Receiving & storage electrical room;
- Shiploader electrical room.

18.2.3 Load Flow and Short-Circuit Study

To establish equipment ratings, preliminary load flow, short-circuit, motor starting and harmonic studies were made for the project. The results of the studies are presented in Appendix 10.4 of the Feasibility Study.

18.2.4 Electrical Load

The electrical installed and running loads for the concentrator-mine site and for the port facilities are indicated in the following tables.

Table 18-2: Concentrator - Mine Site

Area	Electrical Connected kW	Electrical Running kW
Concentrator	74,882	58,333
Rail Load-Out	482	416
Garage, warehouse & administration	2,321	1,893
Crusher	1,848	1,574
Stockpile area	935	792
Tailings	700	205
Mine Pit	237	120
TOTAL	81,405	63,333

Table 18-3: Port Facilities

Area	Electrical Connected kW	Electrical Running kW
Port-Shiploader	2,172	1,844
Port-Receiving	742	634
Port-Storage	944	818
TOTAL	3,858	3,296

The detailed list of loads is shown in the Electrical Load List presented in Appendix 10.3 of the Feasibility Study.

The running load is based on the installed mechanical equipment, efficiencies, load factor and diversity factor, plus an allowance of 10% for contingency. The running loads calculated on the load list are used for transformer and equipment sizing.

The OPEX running load is the same as the electrical running load, but does not include the 10% contingency.

18.2.5 Switchgear and MCC's

The 13.8 kV switchgears at the main substation will supply power to the 13.8 kV switchgears at the concentrator-grinding electrical room and to transformers feeding 4.16 kV and 600 V loads in and around the rest of the plant.

The 13.8 kV switchgears at main substation and at the concentrator-grinding electrical room have been arranged to provide dual sources of supply to each 13.8 kV busbar in the event of loss of one of the incoming breakers.

The 13.8 kV motors for SAG Mill and Ball Mills will be fed from the 13.8 kV switchgear located at the concentrator-grinding electrical room, and the 4.16 kV motors will be fed by 4.16 kV MCCs. Starters for low voltage motors will be grouped in motor control centres ('MCCs'), with incoming breakers located in the electrical rooms and will comprise intelligent combination starters, with circuit breakers for instantaneous protection.

18.2.6 Sag and Ball Mill Motors

The SAG mill motor will be variable speed, wound rotor with a liquid resistance starter (LRS) to reduce starting currents and a slip energy recovery system (SER) to control the speed.

Ball mill motors will be fixed speed, wound rotor with an LRS to reduce starting currents.

All motors are the same size and identical design.

During the detail design phase an alternative option, using squirrel cage induction motors with variable speed drives, for both the SAG and ball mills will be considered to determine if there is a technical or cost advantage to the project.

18.2.7 Flash Dryer

The flash dryer will use electric heating. Multiple 600 V heating elements will be provided with controls and MCC sections as part of the dryer package.

18.2.8 Emergency Power Supply

An uninterruptable power supply (UPS), with sealed batteries, which will supply power to essential loads, will be provided at each substation.

An emergency diesel generator will not be provided at the Port. However, two 13.8 kV emergency diesel generators will be provided at the mine site, in case of power failure, which will automatically start and supply essential loads at the mine site. The generators will be located at the concentrator–grinding electrical room close to the main emergency loads.

Each diesel generator set shall be a fully autonomous system complete with local tank, pumps, piping, fans, control system, disconnect switch, protection and synchronizing panels, etc.

Once normal AC power is restored, the generator set shall be able to synchronize with the power grid.

The generator set shall be able to synchronize with the MV grid for periodic testing purpose.

In the case of loss of normal voltage, all non essential loads will be automatically disconnected before the generators are started and connected to the system.

The circuit breakers and protection will be powered by a direct current ("DC") system comprising sealed battery and charger mounted in the same room as the equipment.

The ventilation system will provide sufficient air changes per hour to allow the UPS batteries and DC batteries to be mounted in the same electrical room as the equipment. During the detail design phase an alternative option will be studied to determine if there is a cost saving in OPEX or CAPEX by installing batteries in a separate room and keeping the main electrical equipment room at a higher temperature in summer and lower temperature in winter.

18.2.9 Lighting Transformers and Distribution

600:600/347 V and 600:208/120 V lighting/utility transformers and distribution panels will supply all lighting and power receptacle loads. Outdoor lighting will be high pressure sodium controlled by a programmable timer, while lighting inside the buildings will be metal halide. Lighting in the electrical rooms will be by fluorescent fixtures. Exit signs and escape route lighting will be provided by individual battery powered halogen lamps with 30 min capacity. LV receptacles of various voltages will be located throughout the plant so that every part of the plant can be reached with an extension cord.

18.2.10 Power Factor Correction

The power factor, as seen by the utility, will be corrected to above 0.95 by the addition of automatic power factor correction capacitors. Detuning reactors will be added in series with each capacitor step to avoid resonance frequencies. The power factor correction capacitors will be sized to provide the desired power factor with the largest capacitor out of service.
Power factor correction equipment will be located at the medium voltage busbars in the Main Substation Electrical Building at the Concentrator-Mine Site, and at the 600V busbars of both Low Voltage Switchgears at the Port electrical buildings.

18.2.11 Grounding

A ground grid will be provided in the 161 kV substation to limit step and touch voltages to acceptable levels.

Interconnected ground loops will be provided around all structures/buildings and the metallic parts of buildings and equipment will be grounded to these loops. A ground loop will be provided inside each electrical room and a ground conductor will be run in the cable trays and connected to this ground loop.

18.2.12 **Cable Reticulation**

In general all cables will be TECK 90 type, with aluminum corrugated armour and will be run in ventilated cable trays. Underground trenches will be used to run the power supply to the tailings area and to the garage, warehouse & administration area.

A single circuit 13.8 kV overhead line with overhead ground wire shall be used for the supply of the mine open pit area. The ground wire shall include a fibre optic cable for control purposes.

18.3 Control and Communication

Details of the proposed control & communication systems and equipments are contained in document number 1848-01-DC-IC-001 Design Criteria Instrumentation and Controls (Appendix 10.2 of the Feasibility Study) and in the system architecture drawings (Appendix 1, Section 04, drawings 1848-04-DR-IC-401 to 403 of the Feasibility Study).
System architecture drawings 1848-04-DR-IC-401 and 1848-04-DR-IC-403 cover the different systems located within the mine area, while drawing 1848-04-DR-IC-402 cover the different systems located within the port area.

18.3.1 **Enterprise Ethernet Networking**

The Enterprise Ethernet Network system will include all the necessary cabling, router, firewall and accessories required for the transmission of data within the plant, as well as providing communication with the external WEB.

IT equipment rooms located in the concentrator building, main office building and port area office will contain the main IT equipments. Some other equipment, such as patch panel and repeater will be located in cabinet located in remote electrical rooms.

Restricted access to IT room will be enforced by means of access control cards and video monitoring.

Firewalls and routers will allow communication within the different system and users within the premises, while preventing intrusion to sensible data from outside. System servers will be used to collect and save data from the different systems.

Administrative network by means of dedicated fibre optic and Cat6 cables will cover all buildings in order to support telephone, intercom, process CCTV, access system, as well as providing link from process network to external WEB.

Process network by means of redundant dedicated fibre optic cables and copper cabling will cover all buildings where process control equipments are located.

18.3.2 Microwave Link

Communication between mine area and port area and regional telephone and internet system will be done by redundant microwave link.

Dedicated communication equipment installed at both site and equipment part of regional infrastructure will be used to achieve the required redundancy and availability.

Main installed equipment will support 100 Mbps, full duplex, while backup equipment will allow 20 Mbps. If required, an upgrade up to 300 Mbps will be possible in the future.

There will be one 100 Mbps microwave link from the mine administrative building to the Sept-Îles network and one 100 Mbps microwave link from the port administrative building to the Sept-Îles network.

There will be one 100 Mbps microwave doubled by a second 20 Mbps microwave links between the mine and port administrative buildings allowing fully redundant communication between areas and regional network.

18.3.3 Telephone and Intercom System

The telephone and intercom system will allow direct communications between the different sectors and rooms within each of the two areas of the project, namely the mine-concentrator area and the port area. Intercom or PA equipment will be installed in noisy area or outside of building, where a telephone set is not practical.

IP telephone equipment, server and handset, and IP intercom equipment will be integrated in order to provide a global communication network. Telephone management system will provide facilities such as directory, forwarding, messaging, usage statistic, transfer call on cell telephone, etc.

18.3.4 Access System

The control access system will permit acknowledgment of entrance and exit of individual and vehicle at the gate entry as well as restricted area such as IT rooms, control room or other sensible areas.

Control of entry and exit at guard house will consist of automatic and manual motorized gates and magnetic cards readers allowing identification of each movement of personal.

Control of entry and exit at control room and IT room will consist of video monitoring and control by means of card reader.

Control access management system will allow production of detailed reports, such as history of personal movement, archiving of video and movement at specific control point, etc.

18.3.5 Radio Communication System

45 Watts mobile radios installed in vehicles and 5 Watts hand held radios will supplement communication between individual and vehicles. Antennas, repeaters and links with internet will provide complete coverage of the mine and port installations. TDMA digital communication provides clearer audio quality, while allowing data communication such as text and GPS based location tracking and telephony integration. Radio system will be compatible with the system used by the Wabush Mine train operators.

18.3.6 CCTV System

Closed circuit television system will be installed to monitor process operation around the crusher, material reclaim, tripper and other equipment as defined during detailed design. All digital cameras will be provided with dust and climate protection as required for proper operation. Cameras will be IP addressable; communication will be by optic fibre, or Cat6 according to distance. Digital video recorder will allow recording and archival of process and restricted area activities when required.

18.3.7 Process Control System

Process control system will consist of redundant operation station located in control rooms, one in the concentrator building, one in the railcar unloading station in the port area. Other non redundant control stations will be located in electrical rooms or by the equipments, such as at the SAG and ball mills, crusher and ship loader.

Process controllers and I/O cabinets will be located in electrical rooms or control cabinets as part of equipment package. Communication between processor and remote I/O cabinet will be redundant, communication with other equipment such as MCC and Switchgear will be non redundant.

18.3.8 Instrumentation

All instruments supplied by package supplier or Owner shall be as possible from the same supplier. Instruments shall be new, selected for industrial usage, optimum accuracy and durability. Instrument material and installation material shall be compatible with both process and environmental conditions. Electronic transmitter shall be preferably of the 2 wire type with 4-20 mA output signals with superimposed HART signal or being directly linkable to the SCADA I/O bus network if applicable. All instruments shall be provided with hazardous area certificates as per applicable standard and SIL certificates when applicable. Whenever applicable, transmitters shall be used in place of switches.

In order to reduce cable entry into electrical and rack room, junction boxes shall be used to group signal from field instruments.

18.3.9 Reference Documents

System architecture drawings 1848-04-DR-IC-401 and 1848-04-DR-IC-403 cover the different system located within the mine area, while drawing 1848-04-DR-IC-402 cover the different system located within the port area.

18.4 Transportation and Relocation Railroad

18.4.1 Background

In the early phase of the Feasibility Study, Mine Arnaud requested Roche-Ausenco to compare four (4) different alternatives for the transportation of apatite concentrate between the concentrator plant and the port facilities:

- Transportation by trucks;
- Transportation by rail;
- Transportation by conveyors;
- Transportation by slurry pipeline.

The comparison had to include CAPEX, OPEX and a financial analysis.

A Preliminary Alternative Transportation Study Report was issued to Mine Arnaud on October 26, 2010 (Appendix 12.1 of the Feasibility Study). The report provided a description of each of the alternatives followed by a direct comparison of the transportation alternatives from the financial, environmental and risk aspects. The report was based on a production and transport of 1 million tonnes of apatite concentrate per year.

Taking advantage of existing rail infrastructure nearby, the rail alternative appeared to be the lowest cost alternative, pending feedback from Wabush Mines. Although its capital cost is much higher that the road alternative, it requires less labour and can benefit from experienced Wabush personnel and existing locomotives and maintenance facilities. It is not the best alternative from an environmental standpoint, but can be considered much better than trucks for the safety and greenhouse gases aspects. From a risk standpoint, it was concluded that the loading and unloading stations needed to be accessible by truck for truck transport in case of problems with rail transport.

Based on the information available at that point and pending feedbacks from Wabush Mines and the Port of Sept-Îles, the rail alternative appeared to be the most interesting option and the one to pursue further for Feasibility Study of the Mine Arnaud Project. The rail alternative is therefore the mode of transportation retained for the Mine Arnaud Project.

Existing Rail Infrastructure and Railroad Relocation

The Mine Arnaud mine site is crossed by the Chemin de Fer Arnaud, a heavy haul type railway owned by Wabush Mines (a subsidiary of Cliffs Natural Resources). The railway is used to transport iron ore to the Wabush Mines and CLM installations in the Pointe Noire area of Sept-Îles. It is also used for access to a ferry-rail dock in the Terminal La Relance of the Port of Sept-Îles. The ferry-rail allows transfer of railcars between the Québec North Shore and the southern shore of the St-Lawrence and is operated by Canadian National (CN). It is used mainly for inbound materials.

At the mine site, the existing railway needs to be relocated, as it is directly over the apatite deposit. The relocation of the railway makes even more possible the use of that railway for transport of apatite concentrate to the Port of Sept-Îles. Indeed, the relocated railroad (or diversion track) can be located close to the Mine Arnaud concentrator plant and designed to accommodate a Mine Arnaud siding. Although there are many different property owners near the mine site to the south, only Wabush Mines has right-of-way for the railway corridor.

18.4.2 Rail Transportation

This section describes the installations required for transporting apatite concentrate by rail between the process plant product storage facilities and the port facilities.

Rail haulage must be contracted out to Wabush Mines as they control the rail corridor and the crews that work on it. The option presented for controlling some costs is to purchase the railcars directly and contract their movement and maintenance.

18.4.2.1 Material – Apatite Concentrate

Apatite concentrate (apatite) is to be transported and is described in the Concentrate Design Criteria (Appendix 12.2 of the Feasibility Study). For handling purposes, apatite is assumed to be a dry easily-flowing powder that should not be leaked, released into the air, moistened, nor contaminated. Roche-Ausenco has therefore assumed apatite will be transported in weather protected conveyors and covered hoppers and handled by gravity loading and unloading, similar to bulk cement.

18.4.2.2 Silos and Rail Load-Out

The silos and railcar load-out area consists mainly of two (2) concrete storage silos of 4,500 t / 2,571 m^3 net capacity each, providing the equivalent of about one (1) day of production each.

The silo floors are elevated to provide room and clearance for the railcars underneath. They are located over the Mine Arnaud siding, located such that the entire train can be loaded without being split up and without affecting the traffic on the main railway line. The silos have also been located taking into account the geotechnical surveys of the area. They are positioned where the geotechnical surveys show rock near the surface.

The silos will be fed from the drying area of the process plant via a transfer tower feeding a 30 inch wide by 320 meter long aerial enclosed belt conveyor with a capacity of 250 tonnes per hour. A reversible 30 inch wide by 12 meter long belt conveyor installed on top of the silos will distribute the material to the appropriate silo.

The silos will be of the controlled flow inverted cone type and will be adequately vented. The technology considered for this Study is the CFI Silo System from FLSmidth (Appendix 12.3 of the Feasibility Study). The silo bottom ring will be fluidized and equipped with multiple discharge spouts, where the apatite will be extracted and conveyed to a central tank via air slides. The central tank, equipped with a de-dusting filter, will feed the downstream airslides which, in turns, will feed a FLSmidth Fluxo Filling Device capable of loading the three (3) railcar openings one after the other (Appendix 12.4 of the Feasibility Study). The spacing between the silos will allow feeding two (2) adjacent railcars at a time.

Both the transfer tower and the silos will house dust collector systems, even if transfer chutes will be designed to minimize dust. The silos will be equipped with bin vent and fan systems.

The area under the silos will be designed will embedded rails to accommodate truck loading in case rail transportation cannot be used.

18.4.2.3 **Haulage Route**

The haulage itinerary will typically comprise of:

- Overhead loading bays at Silos and Railcar Load-out near the process plant;
- Railcars to be removed from the Mine Arnaud mine site loading siding by contracted rail crews for transport and delivery to the port, approximately 17 kilometres away;
- Railcars delivered to unloading facility by contracted rail crews;
- Staff at the unloading facility to unload railcars and prepare them for return to the mine site.
- The route to be used is not congested with traffic, and the rail operator has indicated there is sufficient capacity to accommodate the required service.

18.4.2.4 **Equipment**

<u>Locomotives</u>

The locomotives to perform this movement will be supplied under contract by the rail operator. These locomotives should be capable of handling the 45 loaded cars on the grade present on the Wabush subdivision, each having a capacity to carry 105 tons of apatite concentrate. The two locomotives should be oriented back to back to provide a safe operating position while traveling in either direction.

Railcars

The railcar selected for this operation is manufactured by Trinity Rail and the manufacturer's specification sheet is presented in Appendix 12.5 of the Feasibility Study. This operation requires 45 rail cars in service. With a bad order factor of 5%, the total number of railcars to be purchased is 48.

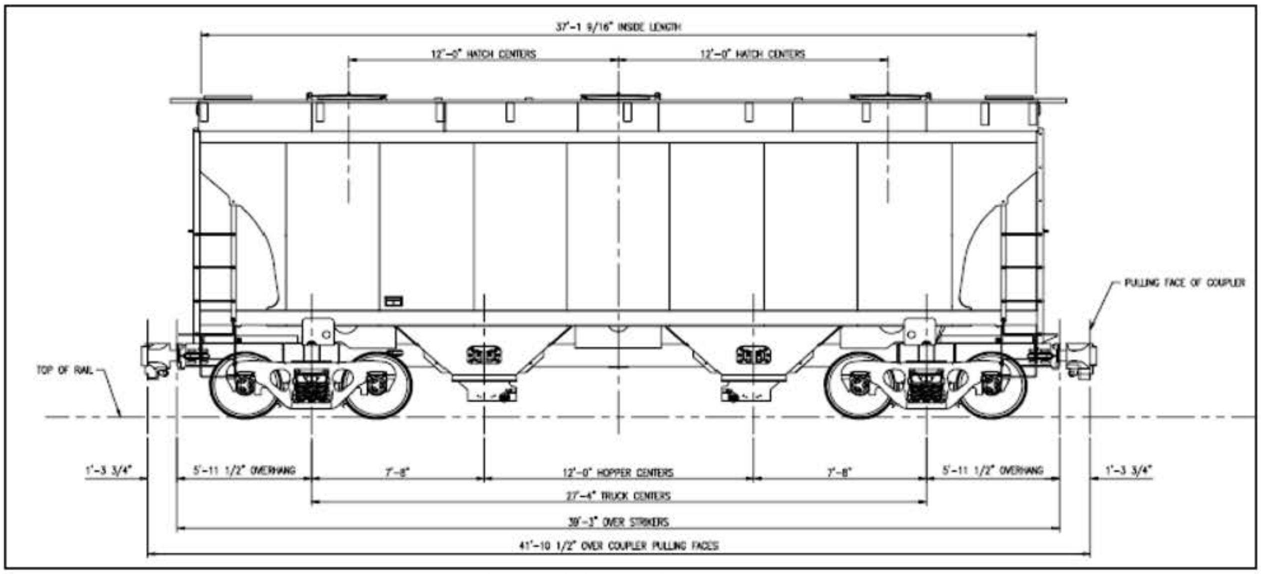


Figure 18-3: Railcar Drawing

18.4.2.5 **Operating Plan**

It was estimated that that there would be 330 working days per year. The additional 35 days of the year account for the lost time of planned and unplanned operational shutdowns. This translates into a 90% operational efficiency.

Table 18-4: Rail service Data outlines the service requirements and assumptions.

Parameter	Unit	Value
Operating days	days/y	330
Tonnes per Year	t/y	1,400,000
Tonnes per Operating Day	t/day	4,242
Tonnes per Rail Car	t/car	105.6
Rail Cars per Day	cars/day	41

Switching service would be provided by Wabush Mines under contract. Wabush Mines would provide the crews and the locomotives and be in charge of the dispatch. Wabush Mines would also control switching priorities.

The 45-car train is expected to be able to handle 1.40 Mt/y of planned annual throughput. The train will spend its idle time at the mine, without locomotives, waiting for product.

A 45-car train has been selected because it makes decent use of the locomotives' power and promotes a schedule of one train per day, which is straightforward to manage. Making decent use of the locomotives' power capacity reduces the number of trips and distance-associated maintenance required. Also, a longer train would require longer sidings at both the mine and port. It might be possible to use as little as 41-car trains, but a 45-car train has been selected for this Feasibility Study to be on the conservative side.

More precise concentrate density measurements would possibly allow to slightly reduce the number of railcars per train.

The unloading process is being designed to unload all 45 rail cars within one 8-hour shift. The rail car unloading station at the port has been designed to handle the annual throughput proposed. Mine staff would be expected to load the train within 12 hours, at the mine. Most of that time, however, will be idle time spent waiting for product. Next Table presents the proposed train cycle time.

Table 18-5: Proposed Cycle

Event	Duration (h/cycle)
Loading Window	12.0
Pick-up from Loader	0.5
Loaded Transit	1.0
Spotting at Unloader	0.5
Unloading Window	8.0
Pick-up from Unloader	0.5
Empty Transit	1.0
Spotting at Loader	0.5
Gross Cycle Time	24.0

18.4.3 Track Standards

Track standards are generally per:

- American Railway Engineering and Maintenance-of-Way Association (AREMA)
- C.N. Engineering Specifications for Industrial Track in U.S.; January 29, 2007.

18.4.4 Diversion Track

The diversion track consists of the portion of the new track that is constructed as to facilitate the re-alignment of existing train traffic, for the purpose of accommodating the proposed mine site.

18.4.4.1 Earthworks

The cut and fill quantities were calculated per the assumed cross-section shown in drawing 1848-04-DR-RA-007 (Appendix 1, Section 04 of the Feasibility Study). This includes ditches for both cut sections and fill sections, with fill section ditches separated from the toe of fill slope by 1.8m. The cut and fill slopes are set at 2:1 with the depth of stripping assumed throughout the entire diversion and siding area to be 0.30m.

18.4.4.2 Profile & Alignment

The alignment departs from the existing track near Mile 6, climbing the adjacent western slope at 1%, and turns to the northwest to follow the Ruisseau Clet valley and crosses below the Hydro-Québec power lines. The diversion then turns back to the northeast and plateaus at elevation 79.10 meters. The load out siding track and the future Wabush passing track branch off at this point. The diversion track alignment then runs roughly northeasterly, with some curves to allow the track to avoid an area of soft ground (as per borehole information) lying north-east of the load out area. The track profile starts a decline of 0.40%. The alignment then turns to the southeast to cross the Hydro-Québec right-of-way again and follows the valley, to tie back in to the existing track. The alignment curves and grades were set to match or improve upon the existing track geometry and conform to the specifications for track standards (see section 12.9). Curvature along the diversion track was limited to 4 degrees or lower. The maximum grades were set at 1.00% (empty) and 0.40% (loaded).

18.4.4.3 Drainage

The diversion track crosses at least two existing waterways and the majority of the track is designed to be constructed on a cross-slope. As mentioned above, the standard cross-section calls for drainage ditches along the entire length. The standard culvert size used for this Study was 900mm; culverts were located in localized low spots to allow the high-side ditch to flow to the low side as well as to provide flow for existing creeks.

18.4.4.4 Overhead Powerline

Hydro-Québec has three 735 kV powerlines within a 370m right-of-way that runs southwest to northeast. This Hydro-Québec powerline right-of-way is crossed twice by the diversion track. In both cases the track crosses the right-of-way within a natural valley to maximize the overhead clearance.

18.4.5 Track Construction Methodology

The existing track is to remain useable during the construction of the diversion track, and the interruption of the track operation (before or after the diversion construction) is to be minimized.

The construction could start with the installation of a temporary turnout at one end of the proposed diversion. The temporary turnout will allow the existing track to remain in use while the construction proceeds towards the other end of the diversion. Completed sections of diversion track can be utilized to import construction material. This may be more accommodating to the contractor, then working from the centre out and having to simultaneously connect to the existing track, at both ends.

As the existing track will be abandoned afterwards, a portion of it can be re-aligned to make up part of the future Wabush siding. It is assumed that generally the rail, fasteners, and ties can be re-used at the tie-in locations. Of course, the existing tack can only be re-aligned once the diversion track is ready to go in service.

The diversion track is to be constructed to final top of rail elevation prior to tying in to the existing track.

18.4.6 Mine Arnaud Siding

The Mine Arnaud Siding starts at diversion track station 3+381 and ends at diversion track station 4+791. It is roughly parallel to the diversion track alignment. The load out siding for Mine Arnaud is sized to accommodate 45 cars plus 2 locomotives both before and after the load out silos. The siding is intended to operate with a loading direction from north-east to south-west. This is to ensure that the south-west portion of the siding bears the heaviest load.

It is preferable that the Mine Arnaud Siding is completed before the diversion track is opened for traffic as to avoid increased safety risks and construction delays.

The dual overhead load out silos will have an enlarged sub-grade pad to allow for an indexer as well as potential for truck access (in case of track service interruption).

18.4.7 General Delivery Track

The General Delivery track is a short stub intended to be a storage track for up to six (6) wagons in total carrying any goods intended for delivery to the east side of the Mine Site, including fuel as well as storage for (3) three of the extra railcars. As per the request of the client, the end of the track is designed to be as close as possible to the fuel storage facility.

18.4.8 Future Wabush Passing Track

The future Wabush passing track is not included in this report, as it is to be completed by others. A portion of the existing track that will be abandoned after the diversion track is in service can be re-aligned to make up part of the future Wabush siding.

18.5 Port Facilities

Apatite concentrate is transported by train up to the Mine Arnaud Port Facilities (the Facilities) located at Terminal La Relance in the Pointe-Noire area of the Port of Sept-Îles, about 17 km away from the mine site. The following (Figure 18-4) shows the location of the Port Facilities vis-à-vis the mine site.

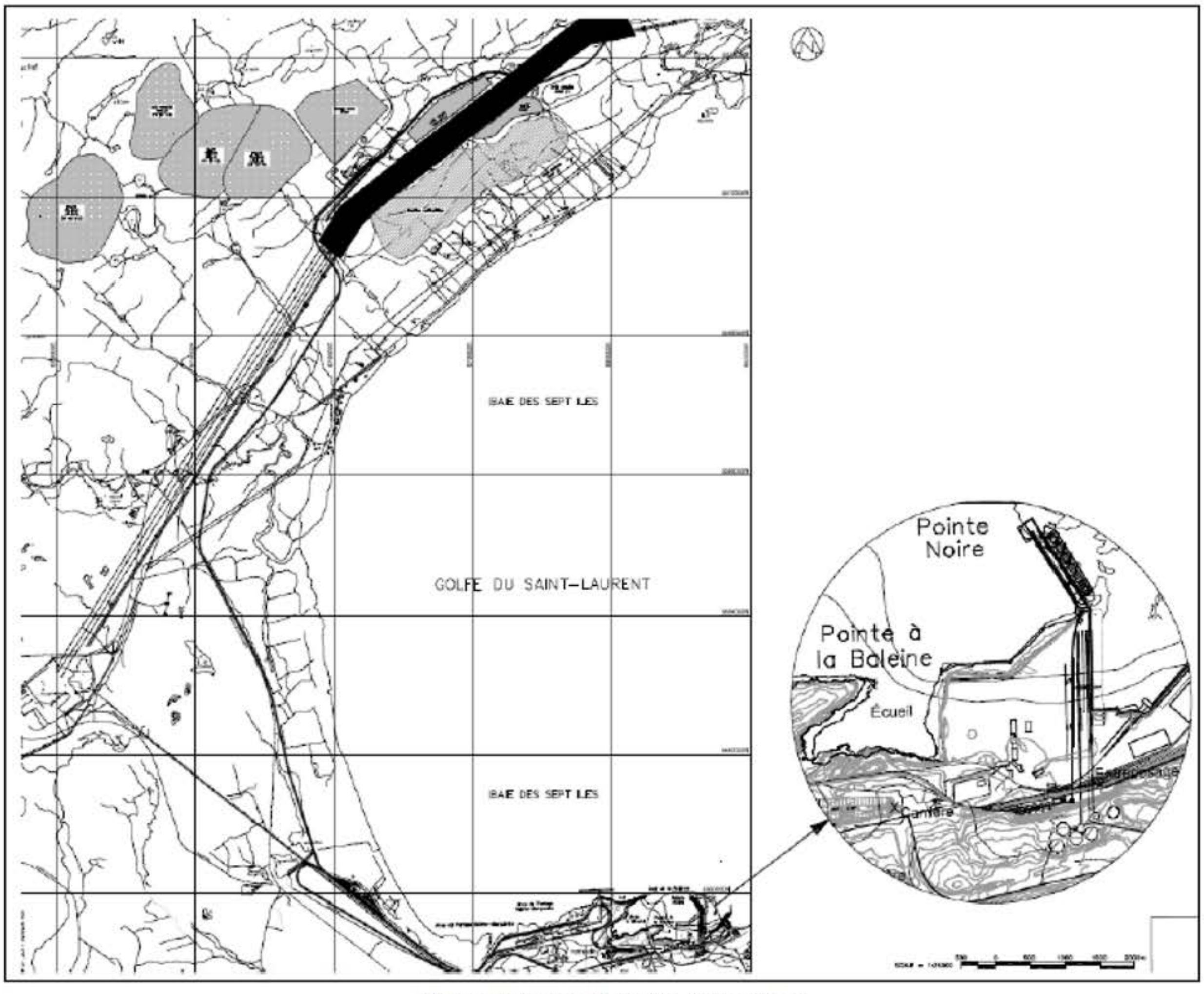


Figure 18-4: Location of Port Facilities

During the course of the Feasibility Study, different concepts for the Port Facilities were considered in cooperation with the Port of Sept-Îles. The following are key design criteria considered:

- The existing berths no. 41 and 42 at the Port cannot be used by Mine Arnaud. Therefore, a new dock located at the end of the existing dock no. 42 is required. The Port of Sept-Îles needs to build the required marine structures that will also eventually be used by other users. The shiploader and tripper conveyor on the new dock need to be designed for a multi-user dock;
- To maximize apatite concentrate handling efficiency at the Port and minimize manpower, a key requirement was not to split up trains and automate the train unloading process as much as possible. Not splitting up trains means that the Facilities require more land at the Port Terminal, especially if a rail loop is considered;
- The facilities include some high vertical structures, such as the silos, that require rock not to be too deep in the ground. The Terminal La Relance is mainly reclaim land with tick filling material except near the entrance of the Terminal on the south side that is not reclaim land.
- In June 2011, the Port of Sept-Îles developed a concept acceptable to them and Mine Arnaud that allows for a new multi-user dock, with Mine Arnaud not having to split up trains. The concept allows high vertical structures to be built directly on the rock.

18.5.1 Marine Structures for the New Dock

The marine structures for the new dock will be built by the Port of Sept-Îles. That part of the scope was only developed up to pre-feasibility level.

Roche-Ausenco designed the marine structures according to the Port Authorities requirements for a multi-user type dock that will allow loading and unloading of different types of cargo and be capable of receiving Panamax type vessels in the future.

A marine geotechnical investigation was performed in the early part of the Feasibility Study to assist in locating and designing the marine structures. The results were published during the winter 2010-2011 (Appendix 8.1 of the Feasibility Study).

A Marine Design Criteria was also developed to collect in one document the design requirements of the site (Appendix 13.1 of the Feasibility Study).

The marine geotechnical investigation together with past bathymetry information and input from the Port of Sept-Îles allowed developing the most likely scenario for dock orientation, location and construction. Roche-Ausenco prepared drawings of the marine structures required for the new dock.

Finally, a CAPEX estimate of the marine structures with an accuracy of +/- 25% was prepared (Appendix 13.2 of the Feasibility Study). The CAPEX for the marine structures is at PFS level and intended for use by the Port of Sept-Îles. It is therefore not included within the Mine Arnaud Project CAPEX estimate. However, it is indirectly included as a portion of the fee from the Port of Sept-Îles in the OPEX estimate.

The marine structures are the responsibility of the Port of Sept-Îles, which is expected to work on the development of the new dock independently, but in parallel with the Mine Arnaud Project.

18.5.2 Concepts Considered for the Port Facilities

While it has been relatively easy to locate the new dock, many different concepts had to be developed to locate the Mine Arnaud Port Facilities on land at the Terminal La Relance.

The first concepts developed based on the requirement of not splitting up trains required a train loop design that would have required either some additional filling of the Bay of Sept-Îles or an expensive railway bridge. The train loop would also have used up most of the land available at the Terminal, including an area currently used by Wabush Mines for inbound bulk materials. Another issue was related to the high cost of constructing silos in filled material.

In June 2011, the Port of Sept-Îles developed a concept acceptable to them and Mine Arnaud that allows Mine Arnaud not to split up trains and build silos directly on rock. The drawings of the Facilities for this Feasibility Study are largely based on the arrangement proposed by the Port of Sept-Îles as shown in the following Figure.

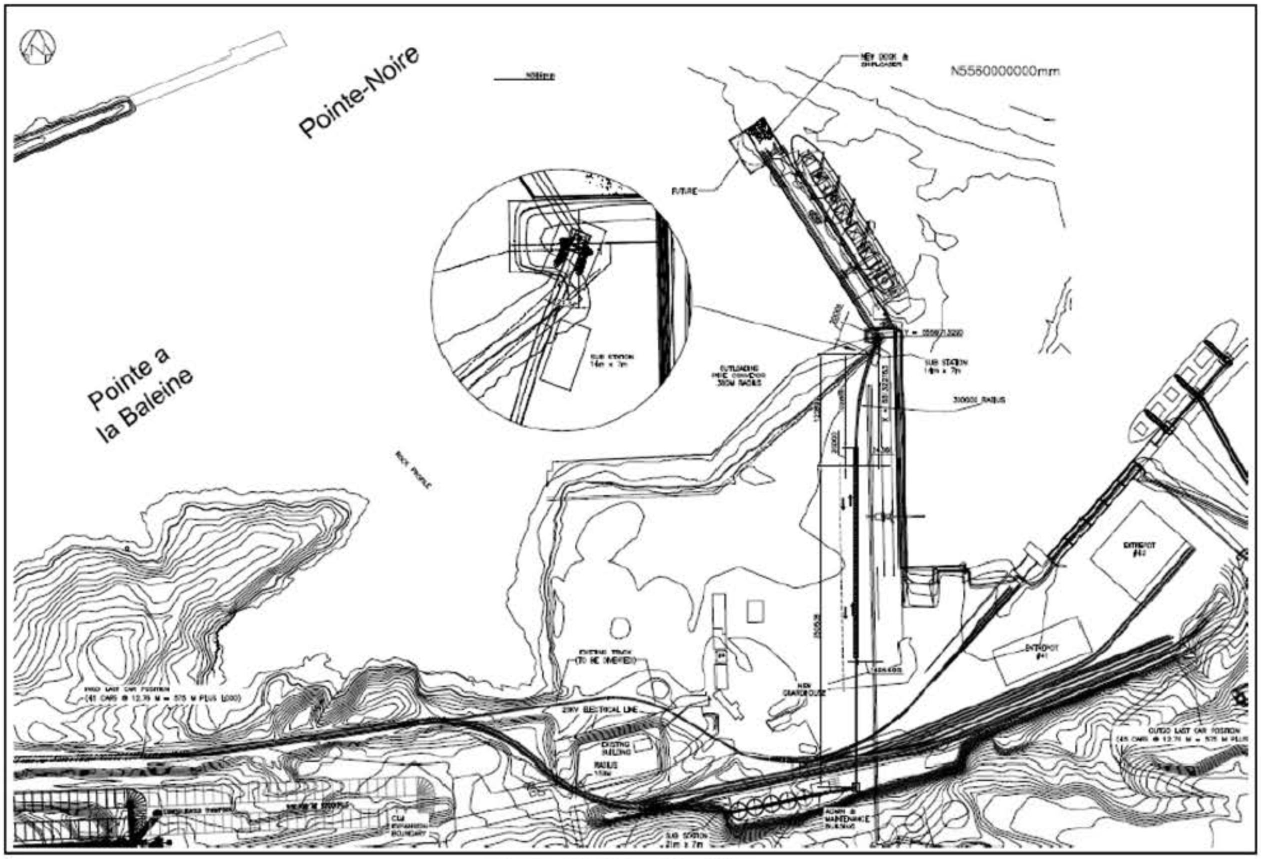


Figure 18-5: Port Facilities

Mine Arnaud rail unloading and storage areas are located in the southern part of the Terminal near the existing rail tracks and where rock is near the surface. An elevated conveyor crosses the Terminal from south to north to tie the unloading and storage area to the shiploader at the new dock.

18.5.3 **Port Track Infrastructure**

The proposed track design underwent a number of revisions prior to the final design. Roche-Ausenco had developed a loop design. Eventually this was discarded in favour of a linear-style yard.

Port track infrastructure consists of a linear unloading track and storage tracks set around a dumping pit and indexer. The design of these tracks has been taken over by the Port of Sept-Îles, whom completed the track alignment layout. Therefore, port tracks have not been defined in details and are not included within the Mine Arnaud Project CAPEX estimate. However, it is indirectly included as a portion of the fee from the Port of Sept-Îles in the OPEX estimate.

One of the requirements of this Project was to automate the train unloading process as much as possible. An indexer is therefore included to control the train displacement as it goes through the unloading station.

The design and cost of the indexer was supplied by Metso based upon the Port of Sept-Îles proposed track layout and resulting profile grade (Metso drawing in Appendix 13.3 of the Feasibility Study). The profile grade suggested that about half of the train will be in a slope going into the unloading station. The indexer will have to move and hold the train on a significant slope and has therefore been sized accordingly.

18.5.4 Material

Apatite concentrate (apatite) is described in the Concentrate Design Criteria (Appendix 12.2 of the Feasibility Study). For handling purposes, apatite is assumed to be a dry easily-flowing powder that should not be leaked, released into the air, moistened, nor contaminated. Roche-Ausenco has therefore assumed apatite will be transported in weather protected conveyors and covered hoppers and handled by gravity loading and unloading, similar to bulk cement.

18.5.5 Concentrate Unloading

Roche-Ausenco analysed the most suitable unloading methods for the railcars. Non-automated tools were assumed to facilitate opening and closing the railcar discharge gates. The combination of an operator-assisted railcar bottom gate opener (Appendix 13.4 of the Feasibility Study) and Boot-Lift type connector (Appendix 13.5 of the Feasibility Study) is believed to insure a clean transfer of apatite concentrate, while minimizing dust and moisture pickup.

The railcar unloading station will consist of a shed housing the indexer, railcar unloading equipment, unloading pit and its ancillary systems. The station will be located as far as possible to the west at the entrance of the Terminal to allow for a complete 45-car train to be unloaded without splitting it up. The station will therefore be located close to the access road of the Terminal.

The railcar unloading station will have a capacity of 750 tonnes per hour. It will include a receiving bin with sufficient volume for two (2) railcars and an extracting screw conveyor. A Beltwall type silo feed conveyor in a gallery will collect concentrate for transport to the silos. The unloading station with have ancillary systems such as a dust collector and sump pump.

18.5.6 Concentrate Storage

The 1200 mm wide by 120 meter long Beltwall type silo feed conveyor with a capacity of 875 tonnes per hour will transfer and lift concentrate at an angle of about 45 degrees to the top of the silos located about eighty (80) meters away for the unloading station.

The apatite concentrate storage at the port will consist of four (4) silos, 18 meter in diameter by 60 meter nominal height for a total of 60,000 tonne of concentrate storage capacity. This is equivalent to one and half the capacity of the largest vessels expected to handle Mine Arnaud apatite for Yara (40,000 tonne vessels). Space will be left for a future silo that would bring total storage capacity to 75,000 tonnes.

At the top of the silos, an enclosed gallery will house a 42 inch wide by 86 meter long tripper belt conveyor with a capacity of 750 tonnes per hour for the transport of apatite concentrate to one silo at a time. The gallery will be equipped with a dust collector. A personnel/freight elevator will be installed along the side of the silos to provide access to the silo roof and to the gallery.

The silos at the port will be similar in design as the load-out silos at the concentrator, although larger in size. The silos will be of the controlled flow inverted cone type and will be adequately vented. The technology considered for this Study is the CFI Silo System from FLSmidth (Appendix 12.3 of the Feasibility Study). The silo bottom ring will be fluidized and equipped with multiple discharge spouts, where the apatite will be extracted and conveyed to a central tank via air slides. The central tank, equipped with a de-dusting filter, will feed the downstream airslides. The silos will be equipped with bin vent and fan systems.

The silo floors will be elevated to provide room and clearance for the silo discharge conveyor underneath.

18.5.7 Concentrate Ship loading

When a vessel will be ready for loading, a 60 inch wide by 137 meter long silo discharge conveyor located underneath the silos will transfer at controlled rate the apatite concentrate from the bottom of one of the silos to a transfer tower located near the silos on the southern side of the Terminal. The shiploading portion of the Facilities will have a capacity of 2,500 tonnes per hour, capable of loading a 40,000 tonne ship in less than 24 hours. The silo discharge conveyor is a conventional enclosed belt conveyor equipped with a belt scale at its head end. The transfer tower will allow feeding the main transfer conveyor of the port facilities. The transfer tower will be equipped with a dust collector.

The main transfer conveyor, which will cross the Terminal La Relance from its southern end to the new dock on the north side, will be an elevated 500 mm diameter pipe conveyor. The 493 meter long conveyor will transfer apatite concentrate from the transfer tower near the silos to a transfer tower located on the new dock. The pipe conveyor will be installed in an unheated enclosed gallery. Near the dock, the pipe conveyor will have a large 300 meter radius to get around the existing Aluminerie Alouette conveyor. Without that radius, an additional transfer tower would need to be built. The elevated pipe conveyor will allow rail and truck traffic underneath just like the existing Aluminerie Alouette conveyor. The transfer tower at the new dock will be equipped with a dust collector, surge bin and reclaim screw conveyor.

In the transfer tower at the dock, the apatite concentrate will be transferred to a 60 inch wide by 248 meter long shiploader tripper conveyor, a conventional enclosed belt conveyor. The shiploader tripper conveyor, installed in an elevated gallery and equipped with a travelling tripper, will feed a travelling shiploader along the new dock.

As the new dock needs to be designed for multiple purpose usage, the shiploader will be elevated and travel on rails flush with the dock floor with the possibility to park it away from the berth in order not to limit access to the berth when not in use. The shiploader boom will have luffing capability. The 55 inch wide by 40 meter long shiploader shuttle conveyor will have a 15 meter long telescoping portion allowing the loading of vessels up to 40,000 tonne in capacity. Apatite concentrate will be loaded via a Cleveland Cascade type telescopic chute to minimize dust emissions. The shiploader boom will also allow handling of mobile equipment for use in the vessels to trim the apatite concentrate.

18.5.8 Port Facilities Services

The Mine Arnaud Facilities will be located mainly at the southern end of the Terminal La Relance near its entrance. A small administration and maintenance building will be built near the silos. A road will allow access to the building even during when unloading of a train.

This Feasibility Study is based on Mine Arnaud having access to the existing port services such as potable water, fire protection water, waste water treatment, etc. However, power will come directly from the Hydro-Québec 25 kV transmission line nearby via a sub-station located near the silos. Please refer to Section 10 of the Feasibility Study for the electrical-automation portion of the Port Facilities.

19. Market Studies and Contracts

In the context of this PFS, SGS did not have a market study in order to predict the apatite concentrate value and to assess the future demand/forecast of this commodity. However, SGS received a document from Yara, which is considered as a partner, an investor and a potential buyer, stipulating:

"Yara has done extensive market price evaluations and confirms that the sales price for Sept-Iles concentrate which is used in the 2012 feasibility study, is in line with our expected price prognosis (no adjustment for reduces P-content is needed as the price is regarded as conservative)."[6]

Given this document, SGS is in the opinion that the assumed value of represents a reasonable base case parameter for this study.

[6] Appendix-2: Ole Bjorn Jenssen, Memo – Yara Position on limits and Process for Mine Arnaud deposit, June 6th 2013

20. Environmental Studies, Permitting and Social or Community Impacts

Mainly all the information contained in this section was derived from the Feasibility Study (prepared by Roche-Ausenco) dated of February 2012 and has not been updated. Original high resolution figure and tables can be found in the feasibility study report from Roche-Ausenco. SGS considered that this information is still relevant in the context of this Preliminary Feasibility Study.

Additionally, the environmental impact study and other complements were performed since the previous Feasibility Study and can be found on Mine Arnaud website (http://www.minearnaud.com/en/impact-study).

20.1 Environmental Baseline Information

Information in this section is a summary of sections of the baseline studies (Roche, 2011a, 2011b, 2011c and 2011d), in which a more detailed description of the environmental setting is provided.

20.1.1 **Quality Air**

The air quality in Sept-Îles is a source of concern for some residents of the municipality. Sept-Îles is characterised by an industrial activity, linked to the presence of metallurgical plants (pelletizing and aluminum plants) and a deep water port for transhipment of large quantities of bulk, which could affect the air quality in the region. In addition to the industrial activities, other sources of atmospheric emissions that are usually found in urban areas, such as traffic and wood firing in residences[7], are also participating to the degradation of air quality.

The MDDEP has implemented an air quality monitoring network in Québec. There is however no measuring station on the north shore, east of the Saguenay River. At the federal level, a similar network is managed by Environment Canada. Sampling stations of the National Air Pollution Surveillance Program (NAPS)[8] on the north shore and the closest of the project site (the Forestville[9] and Mingan[10] stations) are located in rural areas at about 250 km and 165 km respectively from the study area. These stations are therefore not representative of conditions characterising the study area. A global analysis of the air quality in Sept-Îles was however conducted from historical data and an air quality characterisation campaign conducted between June 16 and September 3, 2009 (Couture, 2010). The main findings of the study of Couture (2010) (performed following Thibault (2010)) are:

[7] Wood heating would be responsible for 46% of particulate emissions in Québec and 25% of emissions of volatile organic compounds (VOCs).
[8] The objective of the National air pollution monitoring network is to provide accurate long term data on air quality obtained from standard Canadian methods. The network was established in 1969 to monitor and assess the quality of ambient air in populated areas of the Canada.
[9] Station 51901 is located in a rural forest area, at 15 km from road 385.
[10] Station 55601 is located in a rural undeveloped area.

The results of the characterisation campaign show that air quality criteria and standards are generally met in the city of Sept-Îles;

Over a period of one year, air quality in Sept-Îles is not a problem in regard to particulate matter (total and fine particulate matter) and is comparable to the situation that exists in an urban or peri- urban environments.

On shorter periods of time (hourly or daily) and intermittently, total suspended particulates are high in the southern sector of the municipality and sometimes in the east sector. Dust events occur mainly in dry weather and when wind speed is high (> 15 km/h);

The results of the study on the repercussions of wood heating indicate that particulate matter (mostly the fine fraction) and PAH concentrations in Ferland Park can be quite high, at times during the winter, sufficient to justify preventive actions. Concentrations observed on an annual basis however are not problematic.

Preliminary results simulating the ambient air quality at the limit of the mine property while taking into consideration the Project activities show that total suspended particulates and PM2.5 would be above the ambient air quality standards on very rare occasions (Génivar, 2011a).

Noise

Sound records of ambient noise levels were conducted on a continuous basis from 6 pm on July 19 to 6 pm on July 20, 2011 (Génivar, 2011b). The measures were conducted at three sites. Sites P1 (in the vicinity of the 3685, Highway 138) and P2 (Highway 138, at the entrance to the mine site) are located along the Laure boulevard, while the site P3 is located on Arnaud Street. The results are presented in Table 20-1.

Table 20-1: Ambient Noise Levels Measured Over 24 h

Measuring Site	Ambient noise levels Leq 1h maximum (dBA)1	
	Day (from 7 h to 19 h)	Night (from 19 h to 7 h)
Near 3685, Highway 138 (P1)	65	54
Entrance to the mine site, on Highway 138 (P2)	63	52
In front of the rotary intersection, Arnaud Street (P3)	56	45

[1] *Ambient noise levels rounded to the unit dBA and ref.: $20x10^{-6}$Pa.*

The municipality of Sept-Îles has no regulations limiting quantitatively noise. It has however a regulations for nuisances entitled Règlement concernant la paix, le bon ordre et la sécurité publique (No 2005-63), which present some qualitative clauses relating to ambient noise levels. Criteria used below come from the Directive 019 on the mining industry (Directive 019 sur l'industrie minière (MDDEP, April 2005).

To assess the impacts of the Project on ambient noise levels, the acoustic powers of the main sources of noise relating to the mining activities have been evaluated. Drills, hydraulic mining excavators (RH), off-road trucks and the crusher are the noisiest equipment. These acoustic powers were used to estimate, by simulation of sound propagation, resulting noise levels in the agricultural and forest areas near the mine site. From these simulations, noise levels generated in agricultural and forest areas along route 138 will meet the maximum ambient noise level criteria prescribed by the MDDEP's Directive 019 as well as those of the revised version (June 6, 2006) of the Instruction Note 98-01.

The simulation of sound propagation, taking in consideration mitigation measures (including state of the art mufflers on trucks), shows that noise from the activities at the mine site should meet the maximum levels for residual noise (Génivar, 2011b).

20.1.2 Vibrations

According to the SNC-Lavalin report (2011; Appendix 7.3 of the Feasibility Study), drilling and blasting operations should not cause inconvenience to the nearest residences. Blasting shall however be planned carefully to make sure vibrations and rock projections do not affect Hydro-Québec power lines located near the North sector of the pit. In this sector, the exploitation of 5-m benches is preferred to avoid stone projections from reaching Hydro-Québec structures and equipment, and a further recovery of blasting with flak mattresses and/or a geotextile membrane surface is recommended. From simulation calculations, blasting in the South sector of the pit should not be subject to constraints with regard to the control of vibrations and stones projections because the nearest residences are located at a distance of more than 800 meters from the limit of the pit.

However, airblast overpressure at the first residences might exceed the limit value of 128 dB if blasting clearing axis is oriented towards the residences along Highway 138. Blasting will therefore have to be planned so that clearance axis is perpendicular to Highway 138 (i.e., the free face is oriented towards the West, or oriented North to minimize the air overpressure to the first residences). It would be relevant to mine the deposit progressing along the dominant ore lens, in East-West or North-South direction by favouring shots directed towards the North, except in the sector of the Hydro-Québec power lines, where the height of the banks will have to be limited to 5 meters and the axis of clearance of blasting shall be oriented towards the South to avoid any damage to the Hydro-Québec infrastructure. Weather conditions will require a particular attention to minimize the overpressure at the nearest residences, taking care, to the extent possible, that the blast should be carried out when:

- The sky is clear or clouds are scattered,
- The height of the cloud ceiling is more than 300 meters, and
- Winds speed blowing towards the residences does not exceed 25 km/h.

20.1.3 Hydrogeology and Ground Water Quality

- As part of the Project, detailed geotechnical and hydrogeological studies were conducted in the pit sector and at the sites where the development of associated mining infrastructure is foreseen. In summary, the work involved:
- The drilling and installation of wells in surface deposits and bedrock;
- Pumping tests;
- Permeability tests (packer tests) in existing exploration drill holes.
- The information available on the Project area shows two main hydrogeological systems. The first system, of a confined to semi-confined type, is found deep in the rock formations, while the second system is encountered near the surface and would be semi-confined to unconfined water. It is located in the more permeable horizons of surface geological deposits.

20.1.3.1 Hydraulic Properties

The results of the pumping tests were used to estimate the main hydraulic properties of surface deposits and rock, i.e., the hydraulic conductivity (permeability), transmissivity and storage coefficient. Permeability tests (Packer tests) allowed specifying the spatial variation of hydraulic conductivity in the rock. In general, the results confirm the low hydraulic potential of the till and the rock formations, and values are comparable to those generally reported in the literature for such units (Freeze and Cherry, 1979).

Results indicate that the deposits show hydraulic conductivities ranging between 8.2 x 10-9 m/s and 3.7 x 10-7 m/s, with a geometric mean of 3.5 x 10-8 m/s. Transmissivity values were estimated between 0.02 m^2/d and 1.09 m^2/d. Storage coefficients are estimated to be between 2.5 x 10-4 and 1.1 x 10-3, which suggests semi-captive conditions for these deposits.

The presence of washed deposits in places could explain the higher hydraulic conductivity values measured in the deposits. This would be consistent with the results of the Cogemat study in 1997 (between 10-7 m/s and 10-6 m/s).

The hydrogeological system in the rock has more random hydraulic characteristics. Results from pump tests indicate hydraulic conductivity values ranging from 1.4 x 10-7 m/s to 1.6 x 10-7 m/s. However, the hydraulic conductivity of rock, as measured with the packer tests, varies from 1.0 x 10-10 m/s to 8.9 x 10-6 m/s, with a geometric mean of 2.9 x 10-8 m/s. The hydraulic conductivity would decrease with depth, as indicated by the permeability trials in the drillings (Ausenco Vector, 2011). Transmissivity values were estimated between 0.62 m^2/d and 0.69 m^2/d for the rock. Storage coefficient values lie between 5.0 x 10-5 and 6.8 x 10-5 for the rock, which suggests captive conditions

.

Measures associated with faults and fracture zones are very local. The presence of important fault systems at the site could represent important flow corridors. Future studies are scheduled in the following stages of the Project to examine these structures.

20.1.3.2 **Flow Regime of Groundwater**

Hydrogeological characterisation works at the Project site were used to define the components of the groundwater flow. Knowledge of the different groundwater flow components in the Project sector should allow evaluation of the migration of potential contaminants to the neighbouring environments.

Drilling activities done in 1997 in the pit area showed that the depth of the water table was between 2.4 m and 7.9 m (Cogemat Inc., 1997, in Roche (1997)). Data collected in the new wells (work done as part of this Feasibility Study) also show groundwater depth levels ranging between 2.4 m and 7.7 m. This would correspond to a piezometric level at elevations ranging between 30 m and 80 m (above mean sea level (amsl)). These results seem to suggest that the piezometric surface follows relatively well the topography of the site, with respect to the shallow hydrogeological systems (deposits and rock). Thus, groundwater flow would be in the direction of the Bay of Sept-Îles.

Data collected during the water level survey conducted on February 28, 2011 were used to produce a preliminary groundwater piezometric map (Ausenco/Vector, 2011). The piezometric map confirms that groundwater flow is mainly in the direction of the Bay of Sept-Îles. The flow lines drawn perpendicular to the equipotential lines show aflow direction to the Southeast with a relatively high average horizontal hydraulic gradient of 0.125 m/m. These observations are consistent with the groundwater flow directions modeled by Ausenco Vector (2011).

With such a pattern, the flow is controlled by the groundwater level, where the recharge is done directly by precipitation in sectors showing rock outcrops and/or permeable surface deposits setting topographic highs and hills, while the Bay of Sept-Îles will probably act as a regional discharge zone. Upward hydraulic gradient in some wells on the site suggests a regional discharge of groundwater in the bay.

20.1.3.3 **Classification of Hydrogeological Formations**

The status of a groundwater resource and its relative value varies depending on a number of criteria related to its hydrogeological properties and potential uses. The MDDEP groundwater classification system is a tool to ensure the reconciliation of the uses of the territory with those of groundwater resource according to the following classes (MENV, 1999). The main classes defined within this system are:

- Class I: Aquifer that is an irreplaceable source of water (single source of water supply or supplying a community);
- Class II: Aquifer that is a common source (II a) or a potential source (II b) of water because of its acceptable quality (drinking water with usual treatment) and sufficient quantities (transmissivity greater than 1 m²/d);

- Class III: Aquifer that, although saturated with water, cannot constitute a water supply (III a and III b: poor quality, insufficient quantity or non-economic extraction).

From these definitions, the rock aquifers at the mine site constitute a Class II aquifer that can be used to supply small wells. The source of water for the residents of the municipality of Sept-Îles is however provided by a surface water intake that also serves private properties located south of the mine site, along Highway 138.

Unconsolidated deposits (shallow marine facies) could represent aquifers of Class I or II insofar as they are of sufficient thickness and extent to serve as a source of supply of interest. Till deposits do not form aquifers unless they are sufficiently washed out.

20.1.3.4 **Geochemical Characteristics of Ground Waters**

A preliminary picture of the geochemical characteristics of ground water was established in 1997 using limited available data. Additional sampling is planned (the draft report is still awaited) and should be used to supplement the existing information.

Two boreholes (P-6 and P-7) done on behalf of SOQUEM (Cogemat, 1997) have been sampled in July 1997 to evaluate the quality of groundwater. These two boreholes are located at the southern edge of the pit and intercept groundwater of the bedrock hydrogeological systems. A third sample was taken in November 2011 in an observation well installed at borehole R-11 located along the future segment of the railroad (5 569 897 mN and 675 949 mE; elevation 70.67 m).

The results of this characterisation are presented in Table 20-1. Drinking water quality criteria and groundwater quality criteria were used for comparison.

Groundwater geochemical characteristics in the P-6 and R-11 wells meet the groundwater quality criteria while water in the P-7 well exceeded the criteria used with respect to nickel and, to a lesser extent, copper. It is noted that concentrations of total phosphorus and phosphates are low, which indicates the low solubility of the apatite contained in the bedrock. However, a relatively high concentration (1.5 mg/l) of fluorides is noted in P-7.

Table 20-2: Groundwater Characteristics

Parameter	Unit	Groundwater criteria (µg/l)		Stations (1997)		Station (2011)
		For drinking water purposes	Resurgence in surface waters	P-6	P-7	R-11
Dissolved metals and metalloids						
Arsenic	µg/l	25	340	<1	<1	2,4
Cadmium	µg/l	5	2,1	<0,5	<0,5	< 0,8
Chromium (total)	µg/l	50	-	<1	<1	< 10
Copper	µg/l	1000	7,3	<10	30	< 3,0
Iron	µg/l	-	-	120	30	< 300
Nickel	µg/l	20	260	<5	81	3,8
Lead	µg/l	10	34	<5	<5	< 1,0
Sodium	µg/l	200 000	-	-	-	31 200
Zinc	µg/l	5000	67	<10	<10	< 3,0
Other						
Conductivity	µS/cm	-	-	-	-	597
pH	-	-	-	9,3	7,7	6,94
Chlorides (Cl^-)	µg/l	250 000	860 000	-	-	19 000
Fluorides (total)	µg/l	1500	4000	340	1500	-
Phosphates	µg/l	-	-	20	30	-
Total phosphorus ($P\text{-}PO_4^{-3}$)	µg/l	-	3000	50	150	-
Petroleum hydrocarbons C_{10} - C_{50}	µg/l	-	3500	-	-	1670

Groundwater in surface deposits was not sampled in 1997. In addition to this characterisation effort, samples should be collected from a few wells, some being located downstream from the site, to take into account the potential presence of salt waters near the site. Indeed, the draining of the pit could perhaps affect on the long term groundwater quality by promoting the intrusion of saline waters to the site through surface deposits or rock fractures. An hydrogeological model (Ausenco-Vector, 2011) was partly used to verify the range of the groundwater drawdown surrounding the pit, and its potential impact. Considering the hydraulic properties of the hydrogeological formations, the results of the simulations suggest that the influence of keeping the excavation dry would not reach the coastline (i.e., inland salt-water encroachment is not expected). Moreover, the pit would be sufficiently far from the coastline to theoretically avoid uprising of deep saline groundwater through cracks. However, the hydrogeological characteristics downstream from the pit would benefit from a better definition, namely to validate the results of the hydrogeological simulations.

20.1.3.5 **Groundwater Vulnerability**

The assessment of groundwater vulnerability to a potential contamination from human activities is an approach to implement, if necessary, measures for the protection of the resource. One of the approaches suggested by the MDDEP to assess groundwater vulnerability is the DRASTIC method (Aller et al., 1989).

There is currently little information on the spatial distribution and the value of many of the parameters required to estimate the DRASTIC index. Indeed, accurate data on hydraulic characteristics and thickness of the deposits are required to calculate the DRASTIC index. These are available for the pit sector, but absent for other areas of the mine site. However, it is possible to obtain a first qualitative estimate of the vulnerability, namely when using map showing the distribution of surface deposits.

Sectors characterised by the presence of marine deposits of shallow facies (granular material) will typically present a higher vulnerability where the water level is closer to the surface. A higher vulnerability will also characterise sectors of rocky outcrops and those where the thickness of the deposits above the rock is low. Bedrock aquifer sectors characterised by low permeable till deposits are typically less vulnerable. A better knowledge of the spatial distribution of groundwater vulnerability will be obtained as a result of the complementary hydrogeological characterisation work to be carried out (possibly in 2012) to address the other issues concerning groundwater.

20.1.3.6 **Users of the resource**

Sept-Îles population is fed by a surface water intake. It is located in Lake des Rapides, about 4 km North-East of the future mine area. Various sectors further away from the city are supplied by individual wells, but residences located downstream from the mining site are all supplied from the municipal aqueduct.

Only 25 wells or drillings are reported and recorded in the hydrogeologic information system (S.I.H.) of the Ministère du Développement durable, de l'Environnement et des Parcs (MDDEP) for the Sept-Îles area. Their use is not known (drinking water, irrigation). However, all these wells are located at more than one kilometre of the limits of the mine site and none is located between the mine site and the bay.

For the needs of the Project, the drinking and process water supply will be ensured by a water intake located in the Wabush reservoir erected on the Hall River. However, another option is currently considered to reduce the need of freshwater, avoid the construction of a pumping station at the Wabush reservoir, and greatly reduce the flow of the wastewater effluent to be discharged into Clet Creek. This option considers the possibility of reusing wastewater from the polishing pond (or the water basin into the tailings area) for reagent preparation and to compensate the losses of water from pump gland seals. In this option, a small well at the mine site would be required to provide drinking water to the camp.

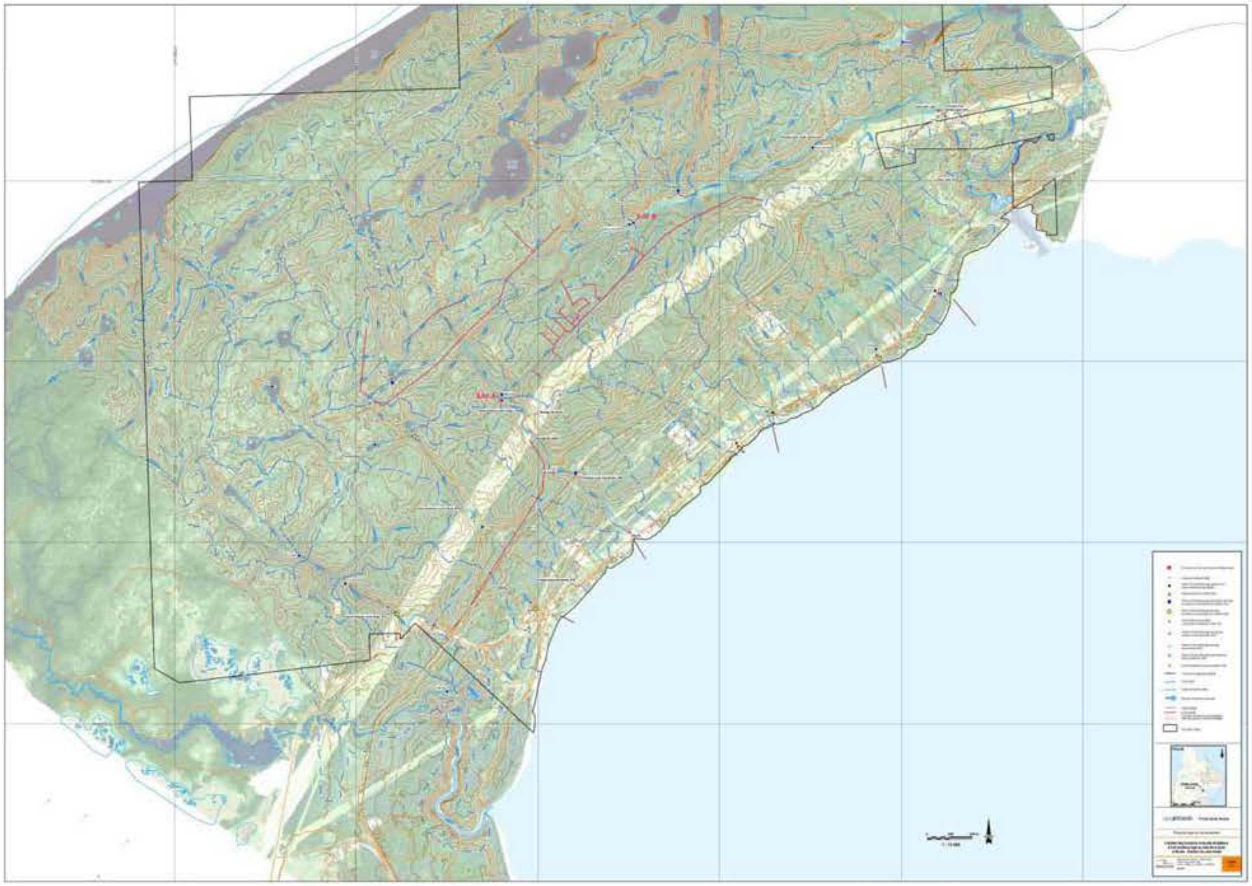


Figure 20-1: Limit of Watersheds and Sampling Stations within the Study Zone – Mine Site Sector

20.1.4 Hydrography and Hydrology

20.1.4.1 Hydrographic Network

The southern part of the future mine site (namely at the pit) drains towards the bay of Sept-Îles via a series of brooks more or less parallel the ones compared to the others. The limits of the watershed of these creeks are indicated on Figure 20-1. The western portion of the mine site drains into the Clet Creek, which flows in the bay. The northern part of the site (possibly including the waste rock dump[11] and a portion of the industrial area) is in the catchment area of an unnamed brook which flows in River des Rapides, approximately 1 km upstream from the Bay of Sept-Îles and about 3 km (as the crow flies) downstream from the municipal water intake.

[11] Surface drainage from the waste rock dump would naturally flow in the unnamed creek (at a point located slightly upstream from the junction with the outlet of Lake Gamache) but it will be accumulated at the foot of a small dam forming a small sedimentation pond and pumped back into the tailings accumulation area.

20.1.4.2 **Hydrological Regime of Watercourses**

According to the section 3.3.2.2 of Directive 019 on the mining industry, the environmental study of a mining project requires the determination of the annual and summer minimum flows (Q2-7, Q5-30 and Q10-7)12 of the streams at the possible discharge sites of mining effluent in the receiving environment. For this analysis, two points were retained (Figure 20-1):

Site A: Possible effluent discharge point of the polishing pond: Watershed: Clet Creek, Catchment area: 373 ha;

Site B13: Located downstream from the waste rock dump: Watershed: Unnamed brook flowing into the Des Rapides River. Catchment area: 289 ha.

20.1.4.3 **Flood Flow**

Considering the small size of the watersheds, the rational method was used in order to determine the flood flows of watercourses draining the mine area.

The calculation of the flood flows, based on these various parameters and assumptions, gives the following results for the different studied periods of recurrence (Taqble 20-3).

[12] Q_{10-7} corresponds to the summer dry-weather flow over seven consecutive days with a decennial recurrence.
[13] Site B is no longer considered as a site for an effluent discharge.

Table 20-3: Rain Intensity and Flood Flow Calculated at Two Sites

	Site A Potential tailings effluent discharge point in Clet Creek (watershed: 3.7 km²)			Site B Site downstream from the waste rock dump, in the unnamed brook (watershed: 2.9 km²)		
Recurrence (years)	Rain intensity (mm/h)	Flood flow (m^3/s)	Flood specific discharge (l/s/ha)	Rain intensity (mm/h)	Flood flow (m^3/s)	Flood specific discharge (l/s/ha)
2	9.1	2.2	5.9	10.9	2.0	6.9
5	11.9	2.8	7.5	14.2	2.6	9.0
10	13.7	3.3	8.8	16.4	3.1	10.7
25	16.0	3.8	10.2	19.2	3.6	12.4
50	17.8	4.2	11.2	21.4	4.0	13.8
100	19.4	4.6	12.3	23.4	4.4	15.2

20.1.4.4 Low Flow

Three methods, each one presenting their forces and gaps, were in turn used to determine the low flows. The method of determination of the low flows based on the frequency analysis of the flows of station 072201 seems the best. The results of this analysis are presented in next Table.

Table 20-4: Minimum Flows at the Limnimetric Station 072201

Flow recurrence (a) – duration (d)	Flow (m^3/s)
Q2-7	1.368
Q5-30	1.120
Q10-7	0.778

20.1.4.5 Downstream Uses

Because the residences located along Highway 138 are fed out of water by the municipal aqueduct, the surface and subsoil waters downstream of the mine site are not the subject to any specific use, except for a small trade (Le Végétarien) which uses surface water to sprinkle its cultures (flowers, fruit and vegetables).

However, the Lake des Rapides, located upstream, is the Sept-Îles water intake for the production of drinking water[14]. In addition, industrial waters from the Wabush Mine pellet plant are drawn from a reservoir located on the Hall River upstream from Highway 138.

20.1.5 Surface Water and Sediment Quality

A surface water and sediment quality characterisation was carried out at the mining site in October 2010 and July 2011. About 20 sampling stations were defined in a way to include the majority of watercourses and waterbodies present in the study area, including reference stations located upstream of the proposed infrastructure sites. Location of the sampling stations is shown on Figure 20-1.

20.1.5.1 Surface Waters

The parameters used for the water and sediment quality monitoring were selected based on recommendation given in the Directive 019 governing mining industry.

Surface waters at the mining site are generally acidic with pH values varying from 4.55 to 6.34. These waters virtually do not contain any suspended solids and are well oxygenated. In addition, the waters present low values of alkalinity (6 mg/l or less) and hardness (14 mg/l or less) and are also low in nutrients, such as nitrogen (Kjeldahl, nitrites/nitrates and ammonia) and phosphorus.

In general, these waters do not contain any metals or metalloids, given that concentrations generally remain below the analytical method detection limits. This is the case for the following elements: Sb, Ag, As, Ba, Cd, Cr, Co, Cu, Hg, Mo, Ni, Pb, Se and Zn. However, waters are characterised by high aluminum contents exceeding, in all samples, the chronic toxicity criteria for the protection of aquatic life, and for some samples, the acute toxicity criteria for aquatic life.

Sediments

Overall, sediments in streams and water bodies are acidic with pH values lower than 5.9. The metal content is low given that the measured concentrations are generally lower than the analytical method detection limits. Only sediments collected at sites PE-1 and PE-3 show copper concentrations slightly above the concentration that may cause occasional effects on organisms. Chromium concentration in sediments in streams R6 and R3 exceeded the threshold concentration that causes an effect on organisms, while sediments at stations R3, R6 and PE-3 have zinc concentrations above the one that may causes rare effects on organisms. In addition, high calcium content was noted in sediments collected at stream R8.

[14] River des Rapides is the outlet of Lake des Rapides. The municipal water intake is located in Lake des Rapides (in the bay *des Crans*, in the south-western part of the lake). The water intake is thus located some 3 km (as the crow flies) upstream from the junction of the unnamed creek and River des Rapides.

Sediments are generally characterised by a high proportion of sand, excepted in some rare instances. Streams R8 and R10 are characterised by a high percentage of gravel (57% and 49%), while streams R6 and R3 and the Lac à l'Anguille show the highest fine sediment contents (49%, 70% and 61% of silt and clay). The sediments at Clet Creek are characterised by a very low content of fine particles.

Table 20-5: Parameters and Measurements as part of the Water and Sediment Quality

Matrix	Water	Sediments
Basic physicochemical parameters	Total alkalinity / Conductivity / BOD5 / COD / Total hardness / Suspended solids / Dissolved oxygen / Redox potential / pH / Total dissolved solids / Temperature / Turbidity	Grain size distribution Organic matter (loss on ignition at 550°C) pH
Nutrients and anions	Total Kjeldahl nitrogen / Ammonia / Nitrates / Nitrites / Total phosphorus / Chlorides / Fluorides / Sulphates	Total phosphorus Total sulphur
Metals and metalloids (extractible totals)	Al, Ag, As, Ba, Ca, Cd, Co, Cr, Cu, Fe, Hg, K, Mg, Mn, Mo, Na, Ni, Pb, Sb, Se and Zn	Al, Ag, As, Ba, Ca, Cd, Co, Cr, Cu, Fe, Hg, K, Mg, Mn, Mo, Na, Ni, Pb, Sb, Se and Zn
Organic substances	Hydrocarbons (C10-C50)	Hydrocarbons (C10-C50)

20.1.6 Biological Environment

Information in this section is a summary of a section of the Baseline Study (Roche, in prep.) in which a more detail description of the environmental setting is provided. This section provides a brief description of the vegetation, benthos, fish, amphibians, reptiles, birds and mammals characterising the study area.

Flora

Section 4 briefly presented the vegetation characterising the mining property and a map showing the distribution of plant species in the study area; the following is a complement to the information presented in this section.

The bay of the Sept-Îles is recognized on a provincial scale for the presence of wetlands of ecological importance. The IBA (Important Bird Area - a science-based initiative to identify, preserve, and monitor a network of sites that provide essential habitats for bird populations) of Sept-Îles covers a surface area of approximately 242 km^2. This IBA, with the marsh, the salted meadows of its bay and its islands, as well as the Checkley plain, constitutes a site of great interest for the water birds and the conservation of the wetlands.

The presence of a wetland of great ecological importance within the study area (the Checkley plain, a flat peat bog officially recognized as being a natural reserve) and of small peat bogs and riparian marshes near the mining site justified the realization of an in situ characterisation and a specific inventory in October 2010 in order to evaluate the ecological value of the wetlands present where mining infrastructure is planned.

The riparian marshes located within the study area are dominated by the speckled alder and are found primarily on the banks of the main brooks, such as Clet Creek. Because of their low specific richness and the presence of several paths, these riparian marshes are of low ecological value at the local and regional scale.

Wooded and open bogs dominated by sphagnum were also observed within the study zone. Less than 20 vascular species were recorded in the wooded bogs of the study area, which corresponds to an ecosystem of very low specific richness. Open bogs offer a greater specific richness, particularly when ponds are present within the bog. No species with particular status were observed in the study area.

In the bay of Sept-Îles, the structure of the littoral plant community corresponds to a succession of salted meadows, salted marshes, mud flats without vegetation, eelgrass beds and, finally, algae beds (Natural Québec/UQCN, 2007).

Fauna

Fauna inventories carried out before 2010 within the study zone are rare. Existing data come mostly from Hydro-Québec within the framework of the Sainte-Marguerite project and from information requests addressed to local, provincial and federal authorities, as the Ministère des Ressources naturelles et de la Faune (MRNF), the Department of Fisheries and Oceans (DFO) and the Association de chasse et pêche Sept-Ilienne Inc. (Zec Matimek). As part of the Arnaud project, fish and bird inventories were thus carried out, as well as opportunistic observations of mammals.

Fish

The study zone comprises some rivers or brooks that flow in the bay of Sept-Îles. In general, the majority of the rivers and watercourses at the mine site and in the region shelter populations of brook trout, a species exploited by sporting fishing, particularly inside the Zec Matimek. Ninespine stickleback is also occasionally captured in the area.

In addition to brook trout, the River des Rapides, the most important tributary of the bay, shelters American eel, Atlantic tomcod as well as rainbow smelt at certain periods of the year. The mouth of the River des Rapides is a winter concentration area for the rainbow smelt, which attracts amateurs of ice fishing.

Birds

Grass beds of the bay of Sept-Îles are recognized as reproduction and resting areas for the waterfowl. These grass beds, composed of Spartina alterniflora, are localized mainly on both sides of the River des Rapides, i.e., between the Clet Creek and the au Foin River. According to seasons, sea ducks (eider, long-tailed duck), dabbling ducks (black duck, pintail and green-winged teal), diving ducks (mergansers, goldeneyes, scaups) as well as various species of water birds could be seen.

With regard to the continental part of the study zone, the potential for waterfowl is considered weak. In spite of this low potential, the peat bogs and ponds with well developed riparian areas (like the shores of the unnamed brook and the peat bog upstream from à l'Anguille Lake) represent habitats favourable for the reproduction of the waterfowl. These habitats are however less important than those of the bay.

Among the avifauna of interest which attends the continental part of the study zone, there is the ruffed grouse, the spruce grouse and the osprey.

Mammals

Terrestrial fauna using the future mine area, or in its vicinity, includes: moose, black bear, weasel, marten, squirrel, otter, hare, red fox, muskrat, beaver and mink. The majority of these species have been hunted or trapped during the last five years, the weasel and the marten being the most captured.

Because of the presence of human activities (Highway 138, power lines, railroad, ATV/snowmobiles), the mine area is not a good habitat for moose. In addition, the map produced by Hydro-Québec about the environmental sensitive elements to the establishment of electric infrastructure (Hydro-Québec, 1987) does not report any wildlife habitat of particular interest for moose in the study area. However, according to the representatives of Zec Matimek, the sector of the Hall Lake, located to the North-West of the study area, would offer some potential for moose hunting.

20.1.7 Human Environment

The Mine Arnaud Project is located within the North Shore administrative region of the Province of Québec. This region is divided into six regional municipalities (MRC), including the MRC of Sept-Rivières whose main city is Sept-Îles[15].

Covering an area of more than 32,000 km^2, the MRC of Sept-Rivières is characterised by a sparsely urbanized coastal strip and a vast hinterland mainly devoted to the exploitation of natural resources (forest, hydropower, wildlife, etc.). In addition to Sept-Îles, the MRC territory also includes the town of Port-Cartier, to the West, and the unorganized territory of Lac-Walker and Rivière-Nipissis, to the North.

[15] Following the municipal mergers that occurred in 2003, the town of Sept-îles includes the former municipalities of Gallix and Moisie.

As well, the study area overlaps with the territory of two Innu[16] communities, known as Uashat and Mani-Utenam. Although separated by some 15 km, the two communities are grouped under one Council forming a single band.

Demographics

Sept-Îles

At the 2006 census, the population of Sept-Îles stood at 25,725 persons, slightly lower than that recorded in the 2001 census done by Statistics Canada. Since then, according to estimates published by the Statistical Institute of Québec, its population has increased steadily year by year to reach 26,196 persons in 2010. This contrasts with the situation on the rest of the North Shore where the regional population declined over the same period.

Its population has a profile somewhat similar to the Province of Québec and Canada in terms of its breakdown by age groups. Individuals under the age of 25 make up for slightly less than one-third of the total population. As for the rest of Québec, the population is aging, however, and this could eventually be reflected in the number of available workers.

If the level of education of the population of Sept-Îles is slightly higher than the rest of the Province in terms of graduates at the vocational and college level, the number of university graduates, however, is much lower (9.6% against 16.5% in Québec). On the other hand, one third of the population aged 15 years and over had no diploma.

The economic turmoil prevailing in Sept-Îles and the surrounding area has a direct effect on the main economic indicators of the population. In 2009, the local labour force was in the range of about 17,000 individuals and nearly 75% of them were employed. The unemployment rate for the combined areas of the North Shore and Northern Québec was 8.8% in 2009. It is generally agreed that unemployment in Sept-Îles is significantly lower than that of the region and some observers even speak of a situation which tends more and more towards full employment. Such a situation is not without its challenges to local businesses and enterprises, regardless of their size.

Uashat mak Mani-Utenam

Indian Register data compiled for the Department of Indian Affairs indicates that the Uashat mak Mani-Utenam community had 3,805 members in 2009 (3,114 on the reserve and 691 off the reserve). Its population continues to increase but at a slower pace than that recorded between 1980 and 2006, when it more than doubled.

[16] Previously known as the Montagnais, the Innu (which means "human being" in Innu-Aimun, their language) are the aboriginal inhabitants of an area they refer to as Nitassinan, which comprises most of the north-eastern portions of the provinces of Québec and some western portions of Labrador.

Like most indigenous communities, the population of Uashat mak Mani-Utenam is rather young, especially when compared to the population of the province of Québec. There is a large youth population, and individuals under the age of 25 make up almost half of all members.

The population of Uashat mak Mani-Utenam is generally less educated than the population of Québec. In fact, while 70% of the Innu 15 years and older have completed elementary school; only 30% have finished high school. Recently, the drop-out and failure rates have raised at all academic levels.

Data from the 2006 census point to a relatively high level of economic activity, a low employment rate and strong unemployment. Median incomes are similar from one reserve to the other and are significantly lower than the median income for the Province of Québec.

As in most Aboriginal communities, the economy in Uashat mak Mani-Utenam chiefly relies on the public sector. The band council, which oversees many economic activities, is the largest employer on the reserve, providing some 400 positions. Commercial fishing is very important to the community, creating mainly seasonal jobs, and Uashat mak Mani-Utenam has a fleet for crab, lobster, shrimp and demersal fish fishing. The community also operates a seafood product plant and plans to develop forestry activities, which are currently limited. Private sector economic activities stem from some thirty private businesses, mainly in construction and services.

Economic Conditions

Unlike some other cities of the North Shore and other regions of Québec, Sept-Îles is currently experiencing a real economic boom. In fact, it seems that for the first time since the early 1980s (date of closure of the IOC pellet plant), the local and regional economy is growing.

The economic structure of the Sept-Rivières regional municipality is mainly based on exploitation and, to a lesser extent, the transformation of its natural resources. Mining, forestry, hydroelectric power, fishing and hunting as well as the processing of iron and aluminum are the main economic activities in the region.

The city of Sept-Îles constitutes a major regional hub. The city's economy is based mainly on aluminum processing, the transformation and shipment of iron mineral resources through its deep water port and the delivery of various services (municipal, governmental, health, education, etc.).

The Alouette aluminum smelter, the largest in the Americas, is the most important employer in Sept-Îles, with nearly 1,000 direct employees. In production since 1992, the company is considered to have allowed the city to come out of economic gloom in the early 1990s. Its importance is such that some believe that almost 20% of local workers depend on it (including the estimated 2,000 indirect jobs it generates).

Although there is no active mine within the territory of Sept-Îles, the city's economy is closely linked to mining activities taking place on the iron deposits of Fermont and Schefferville, in Northern Québec, as well as Labrador City and Wabush, in the province of Newfoundland and Labrador. Iron Ore Company of Canada (IOC), Cliffs Natural Resources (Wabush Mine and Bloom Lake Mine – recently bought from

Consolidated Thompson) carry their respective production to the port terminals located on both sides of the bay of Sept-Îles, for shipment to the world steel industry.

The business relationships developed with mining companies operating in the hinterland and Northern Québec have resulted over the years by the development of several specialized small and medium enterprises (SMEs) offering goods and services to meet the needs of heavy industry (e.g. manufacturing, maintenance, installation of metallic structures, supply of equipment or chemicals, etc.). Sept-Îles thus provides a pool of well-established manufacturing companies serving not only local and regional customers but which are now exporting their expertise in Mexico, the United States, Brazil, Scandinavia, Russia, India and elsewhere through the world in the areas of technologies, products and services especially for companies that mine iron ore.

Finally, the tertiary sector of the Sept-Îles economy is also well on track with the Center for Health and Social Services Sept-Îles, various educational institutions, government offices (federal, provincial, regional and local) which are major employers in the services domain. Added to this are several shopping centers, retail shops, professional offices and financial institutions. Overall, the trade and services industry employs about 10,000 people locally.

Land Use and Occupancy

The site of the Project being developed by Mine Arnaud is located mostly within the boundaries of the town of Sept-Îles (it also partially overlaps the Lac Walker unorganized territory), but well away from the urban core that is located about ten kilometres to the east.

A sparsely populated residential area (about fifty residences) stretches along Highway 138 between the rivers Hall, to the west, and des Rapides, to the east. Within this zone, some agricultural activities have developed over the years, as is the case for a vegetable and horticultural producer (operating under the name Le Végétarien) whose activities take place just east of the Clet Creek.

The northern part of the future mine site intersects the Zone d'exploitation contrôlée (ZEC) Matimek, an area dedicated to the practice of hunting, fishing and recreation activities. Access to the southern part of the ZEC is from Highway 138, via a forest road leading to Lake Hall, or by using a gravel road (known as Chemin Allard), which crosses the Arnaud property to reach the lakes of Sept-Îles hinterland. Some of the lakes and rivers within the limits of the study area - including Gamache sector - are also used to practice sport fishing. Territories adjacent to them are popular for hunting big game and small game.

The Project site also overlaps with three trap lines (No. 09-11-320, 09-11-321 and 09-11-0326) on which fur-bearing animals are trapped by non-Aboriginal residents of the Sept-Îles area. Inventories have also located some hunting and fishing camps and lodges within the study area.

The snowmobile trail Trans-Québec no. 3 follows an axis roughly parallel to Highway 138 within the right-of-way of a 161 kV power line; from there, its path leads into the back country to reach the Lac des Rapides area. Outside the winter season, the same path is also used by all-terrain vehicles (ATV) enthusiasts.

Finally, the study area is crossed by six transmission lines and distribution grouped within two different right-of-way.

Transportation

The North Shore road network is essentially made up of Highway 138, a road of national significance which stretches just over 800 km between Tadoussac to Natashquan. This road, which is the only major highway along the coast, crosses the southern part of the study area. The annual average daily traffic (AADT) is estimated at a little below 5,000 vehicles per day.

The secondary transport network is deployed from Highway 138. One of these access infrastructures, known as the Pointe-Noire road, provides access to the Cliffs Natural Resources pellet plant, the "Pointe-Noire" area of the port of Sept-Îles and the Alouette aluminum smelter.

A private railroad operated by the company Arnaud Railway (property of Cliffs Natural Resources) crosses the study area. This railway, which has an approximate length of 38 km, connects the industrial and port facilities of Pointe-Noire to the regional rail network (QNS & L). Every day, the train makes four trips back and forth between Pointe-Noire and Sept-Îles Junction (QNS & L terminal). For 2006-2010, information obtained from the company indicates that about 6 million tons of concentrate are transported each year. Because of the many projects under development north of Sept-Îles, Arnaud Railway expects the number of daily round trips will double or even triple over the 2011-2015 period.

Sept-Îles enjoys a strategic position at the entrance of the Gulf and the Saint Lawrence Seaway, which gives it a privileged status and makes it a crossroads of international trade for goods in bulk. The site is characterised by a semi-circular bay about 10 km in diameter, deep water, and accessibility throughout the year. The Port of Sept-Îles currently ships about 24 million tonnes per year. Due to the start of production of new deposits, the port could receive within three years an additional 30 million tons coming from the Fermont-Labrador City and Schefferville areas. Also, if the various projects currently under development in the Schefferville area (Taconite, KéMag and LabMag) were to materialize, an additional 22 million tonnes could be added by 2016, bringing total shipments to almost 80 million tonnes per year. At this rate, the port of Sept-Îles will soon be the largest mining port in North America.

Finally, the Sept-Îles airport is the hub for air service throughout the North Shore. Connected daily to Québec and Montréal, the airport also serves Northern Québec and Labrador. Sept-Îles has one of the busiest air traffic in eastern Canada and it is still growing.

20.2 Environmental Characterization Program (Ore, Overburden, Waste Rock and Tailings)

20.2.1 General

This section presents the results of the environmental characterisation program conducted on the solids to be excavated (overburden, waste rock and ore) and on tailings[17] (solid and liquid phases) to be produced. These analyses have been conducted to determine whether or not the material can present environmental risks because of its metal content (the environmentally available fraction) or relatively high concentrations of some other substances (fluorides, selenium and petroleum hydrocarbons), its leaching properties, and/or its potential to generate acid mine drainage. The specific objectives of this geochemical characterisation program are to:

Classify mine waste according to the Québec Directive 019 sur l'industrie minière and the Guide de valorisation des matières résiduelles inorganiques non dangereuses de source industrielle comme matériau de construction for waste management planning (including possible valorization avenues), and

Determine probable mining effluent quality and identify chemicals of environmental interests related to water and mine waste management.

In addition to helping design appropriate accumulation areas that take into account the properties of the material to be accumulated, and assessment of the project impacts, the characterisation of the ore and the inorganic waste materials of industrial source (overburden, waste rock and tailings) will help determine whether these materials can be valorized.

The methods used to characterise materials are consistent with the Guide de valorisation des matières résiduelles inorganiques non dangereuses de source industrielle comme matériau de construction (Guide for the valorisation of residual inorganic non-hazardous materials of industrial source as construction material) (Ministère de l'Environnement du Québec, 2002) and with the most recent methods advocated by the Centre d'expertise en analyse environnementale du Québec (CEAEQ).

Process Generating Residual Materials

The description of the way residual materials to be managed are produced allows preliminary assessment of the various contaminants likely to be found.

Overburden will be removed from the pit with mechanical shovels, while extraction of waste rock and ore will require the use of explosives based on ammonium nitrate and diesel (ANFO). This material will be

[17] Ore samples were treated in pilot plants to determine the best ore processing method. During the piloting phase, tailings pulp samples were collected.

sent to the various dedicated accumulation areas; the ore to the accumulation area located near the crusher, while waste rock and overburden are sent to accumulation areas around the pit.

The ore will be treated at the concentrator where it will be crushed, ground, and subject to a magnetic separation followed by a flotation in basic conditions. Ore grinding will bring the material to a size equivalent to fine sand (about 170 µm). Magnetic separation will allow the removal of the magnetic portion of the ore (magnetite and titanomagnetite). Following flotation, the apatite concentrate will be thickened, belt filtered and dried in an electric furnace. It will then be stored in silos near the loading area to be shipped in closed railcars to silos located at the port of Sept-Îles. The two rejects produced during the apatite concentration process (the mag tails and the float tails) will be sent through pipes as pulp to separated tailings accumulation areas. The ore processing circuit allows the recovery and/or neutralization of reagents prior to the discharge of the tailings into the accumulation areas (through water recirculation).

Reagents

The reagents used in flotation are: wheat starch18, soybean oil (a natural vegetable oil), sodium hydroxide19 and a flocculent (possibly Flomin 905, an anionic polymer soluble in water and non-toxic). MSDS (material safety data sheets) and the coding used by the SIMDUT (Système d'information des matières dangereuses utilisées au travail; Information system on hazardous materials used in workplaces) are a source of information facilitating classification and reducing the need for chemical analyses of hazardous substances. Sodium hydroxide is the only reagent that is assimilated to a hazardous substance.

Hazardous Substances

When considering the properties of a hazardous material, as defined in section 4 of the Regulation respecting hazardous materials, one concludes that overburden, waste rock, ore and tailings are not hazardous materials.

20.2.2 Environmental Characteristics of Ore

Elemental Composition

Table 20-6 shows the elemental composition of the ore. Most of the parameters analysed on the ore samples show results below the A quality criteria for soil presented in the Soil protection and rehabilitation of contaminated sites policy (MDDEP, 2001). When considering mean concentrations, the elements

[18] Wheat starch is a natural product often used in food.

[19] Sodium hydroxide or caustic soda (NaOH) is used in large quantities by several industries, primarily as a basis for the manufacture of pulp and paper, chemical products and plastics, soaps and detergents, some artificial textiles, and aluminum (bauxite processing). It is also used to regulate the pH and regenerate ion exchange resins in water treatment stations. This product is corrosive.

analysed all meet their respective A-criterion for soil protection, except for manganese, cobalt and copper.

Leaching Tests

Results from the TCLP, SPLP and CTEU-9 leaching tests conducted on ore samples are shown in Table 20-7. All of the parameters tested on the leachates meet the environmental standards and criteria used except for copper and cobalt, which may sometimes be found in concentrations above the criteria for the protection of ground waters, namely for CTEU-9 tests. It should be noted that cobalt does not leach in such a manner as to exceed the criterion for the protection of ground waters when using the TCLP and the SPLP tests, and that the mean value for the CTEU-9 test remains below the criterion. In fact, the only mean value exceeding a criterion relates to the copper for the TCLP (EPA 1311) test.

As for mercury, the only two results exceeding the criterion are not representative of the majority of the results and too close to the analytical detection limit to be meaningful.

Table 20-6: Elemental Composition of Ore Samples

Element *	Unit	Detection limit	Québec - Soil protection criterion [1]			Results				Results							Statistics (n=11)		
						Ore samples from COREM analysed in June 2011				Ore samples from COREM analysed in November 2011									
			A-Criterion	B-Criterion	C-Criterion	S2 Drill Core	S3 Drill Core	RR Drill Core	RR/S2/S3 (Composite) Drill Core	Bulk S2	Bulk S3	Bulk S4	Drill core S2	Drill core S3	Drill core S4	Drill core railroad	Min	Max	Mean
Metals and metalloids																			
Aluminum	mg/kg	20	-	-	-	3 600	2 900	3 000	3 100	3 700	3 000	2 800	5 400	3 600	2 300	3 900	2 300	5 400	3 391
Silver	mg/kg	0,8	2	20	40	< 0,5	< 0,5	< 0,5	< 0,5	< 0,8	< 0,8	< 0,8	< 0,8	< 0,8	< 0,8	< 0,8	< 0,5	< 0,8	< 0,8
Arsenic	mg/kg	5	10	30	50	< 5	< 5	< 5	< 5	< 5	< 5	< 5	< 5	< 5	< 5	< 5	< 5	< 5	< 5
Barium	mg/kg	5	200	500	2 000	13	46	23	34	65	50	50	19	56	29	28	13	65	38
Cadmium	mg/kg	0,5	0,9	5	20	< 0,1	< 0,1	< 0,1	< 0,1	< 0,5	< 0,5	< 0,5	< 0,5	< 0,5	< 0,5	< 0,5	< 0,1	< 0,5	< 0,5
Calcium	mg/kg	30	-	-	-	77 000	65 000	76 000	73 000	56 000	48 000	37 000	55 000	58 000	53 000	66 000	37 000	77 000	60 364
Chromium	mg/kg	2	45	250	800	4	3	4	4	12	9	< 2	8	5	2	7	<2	12	5
Cobalt	mg/kg	2	15	50	300	***43***	***45***	***54***	***51***	***34***	***37***	***24***	***39***	***44***	***37***	***50***	***24***	***54***	***42***
Copper	mg/kg	2	50	100	500	32	***70***	43	48	***240***	***200***	***130***	26	***57***	***78***	37	26	***240***	***87***
Tin	mg/kg	4	5	50	300	1	2	1	2	< 4	< 4	< 4	< 4	< 4	< 4	< 4	1	< 4	< 4
Iron	mg/kg	10	-	-	-	41 000	76 000	72 000	75 000	70 000	80 000	69 000	59 000	95 000	75 000	86 000	41 000	95 000	72 545
Manganese	mg/kg	2	1 000	1 000	2 200	460	***1 300***	***1 100***	1 000	1 000	***1 300***	***1 100***	530	***1 500***	1 000	***1 200***	460	***1 500***	***1 045***
Mercury	mg/kg	0,02	0,4	2	10	< 0,01	< 0,01	< 0,01	< 0,01	< 0,02	< 0,02	< 0,02	< 0,02	< 0,02	< 0,02	< 0,02	< 0,01	< 0,02	< 0,02
Molybdenum	mg/kg	1	6	10	40	< 0,5	< 0,5	< 0,5	< 0,5	< 1	< 1	< 1	< 1	< 1	< 1	< 1	< 0,5	< 1	< 1
Nickel	mg/kg	1	30	100	500	19	10	20	16	***47***	***32***	4	21	13	8	23	4	***47***	19
Lead	mg/kg	5	50	500	1 000	< 1	< 1	< 1	< 1	< 5	< 5	< 5	< 5	< 5	< 5	< 5	< 1	< 5	< 5
Selenium	mg/kg	1	3	3	10	1,1	1,3	1	1,1	1	< 1	< 1	< 1	< 1	< 1	< 1	< 1	1,3	< 1
Strontium	mg/kg	-	-	-	-	180	140	190	170	-	-	-	-	-	-	-	140	190	170
Titanium	mg/kg	5	-	-	-	830	760	1 100	1 500	1 500	1 300	1 400	1 600	1 000	1 100	850	760	1 600	1 176
Uranium	mg/kg	5	-	-	-	< 2	< 2	< 2	< 2	< 5	< 5	< 5	< 5	< 5	< 5	< 5	< 2	< 5	< 5
Zinc	mg/kg	10	100	500	1 500	49	63	65	61	70	77	76	73	90	87	88	49	90	73
Others																			
Chlorides	mg/kg	1	-	-	-	-	-	-	-	3	4	1	13	8	5	63	1	63	14
Sulfides	mg/kg	-	-	-	-	-	-	-	-	0,45	0,74	1,89	572	107	33,6	168	0,45	572	126
Fluor	mg/kg	-	-	-	-	-	-	-	-	340	3 040	2 750	2 980	3 820	3 720	3 400	340	3 820	2 864
Fluorides	mg/kg	1	200	400	2000	-	-	-	-	< 1	1	2	< 1	< 1	< 1	< 1	< 1	2	<1
pH	pH	-	-	-	-	-	-	-	-	5,76	5,32	5,20	9,03	9,11	7,31	8,59	5,20	9,11	-
Total phosphorous	mg/kg	20	-	-	-	-	-	-	-	24 000	20 000	15 000	23 000	24 000	23 000	27 000	15 000	27 000	22 286

[1] Criteria are from the *Politique de protection des sols et de réhabilitation des sites contaminés* (MDDEP, 2001). The A-criteria represent natural background concenterations within the Grenville geological province.

[2] CCME; Tableau sommaire des recommandations canadiennes pour la qualité de l'environnment; http://st-ts.ccme.ca/ (Site visited on May 3, 2011)

* Digestion according to the M.A. 200 method

Values exceeding the A criterion are in bold and italics

Values exceeding the B criterion are shaded, in bold and italics

**See Roche-Ausenco feasibility study for high resolution image*

Table 20-7: Results of the Leaching Tests (EPA 1311, 1312 and CTEU-9) on Ore Samples

Parameter	Unit	Critère eaux souterraines [1]	Standards canadien [2] Concentration moyenne mensuelle maximale permise	Normes québécoises d'eau potable	Critère pour les résidus miniers à risques élevés [3]	S2 EPA 1311	S2 EPA 1312	S2 CTEU-9	S3 EPA 1311	S3 EPA 1312	S3 CTEU-9	RR EPA 1311	RR EPA 1312	RR CTEU-9	RRS2/S3 (Composite) EPA 1311	RRS2/S3 (Composite) EPA 1312	RRS2/S3 (Composite) CTEU-9
						Ore samples from COREM analysed in June 2011											
Metals and metalloids																	
Aluminium	mg/l	0.75	-	-	-	0.57	0.52	0.8	0.44	0.19	0.15	0.42	0.13	< 0.08	0.44	0.20	0.09
Argent	mg/l	0.00002***	-	-	-	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01
Arsenic	mg/l	0.34	0.50	0.005	5	< 0.004	< 0.004	< 0.004	< 0.004	< 0.004	< 0.004	< 0.004	< 0.004	< 0.004	< 0.004	< 0.004	< 0.004
Baryum	mg/l	5.3***	-	1	100	0.15	< 0.005	0.031	0.21	< 0.005	0.006	0.16	< 0.005	< 0.005	0.17	0.011	0.006
Cadmium	mg/l	0.0021***	-	0.005	0.5	< 0.002	< 0.002	< 0.002	< 0.002	< 0.002	< 0.002	< 0.002	< 0.002	< 0.002	< 0.002	< 0.002	< 0.002
Calcium	mg/l	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Chrome	mg/l	-	-	0.05	5	< 0.007	< 0.007	< 0.007	< 0.007	< 0.007	< 0.007	0.007	0.007	< 0.007	< 0.007	< 0.007	< 0.007
Cobalt	mg/l	0.5	-	-	-	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01
Cuivre	mg/l	0.0073***	0.30	1	-	***0.04***	< 0.02	< 0.02	< 0.02	< 0.02	< 0.02	***0.02***	< 0.02	< 0.02	< 0.02	< 0.02	< 0.02
Étain	mg/l	-	-	-	-	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05
Fer	mg/l	-	-	-	-	5	< 2	< 2	7	< 2	< 2	6	< 2	< 2	6	< 2	< 2
Manganèse	mg/l	-	-	-	-	0.72	< 0.01	< 0.01	0.97	< 0.01	< 0.01	0.94	< 0.01	< 0.01	0.91	< 0.01	< 0.01
Mercure	mg/l	0.00013	-	0.001	0.1	< 0.0001	***0.0002***	***0.0002***	< 0.0001	< 0.0001	< 0.0001	< 0.0001	< 0.0001	< 0.0001	< 0.0001	0.0001	< 0.0001
Molybdène	mg/l	2	-	-	-	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01
Nickel	mg/l	0.260***	0.50	-	-	0.03	< 0.006	< 0.006	0.03	< 0.006	< 0.006	0.03	< 0.006	< 0.006	0.03	< 0.006	< 0.006
Plomb	mg/l	0.034***	0.20	0.01	5	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01
Sélénium	mg/l	0.020	-	0.01	1	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005
Titane	mg/l	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Uranium	mg/l	-	-	0.2	2	-	-	-	-	-	-	-	-	-	-	-	-
Zinc	mg/l	0.067***	0.5	-	-	< 0.2	< 0.2	< 0.2	< 0.2	< 0.2	< 0.2	< 0.2	< 0.2	< 0.2	< 0.2	< 0.2	< 0.2
Others																	
Fluorures	mg/l	4	-	1.5	150	0.2	0.1	1.4	0.2	0.1	1.2	0.2	0.1	1.1	0.2	0.1	1.1
Phosphore total (P)	mg/l	3**	-	-	-	0.22	0.10	-	0.27	0.12	-	0.18	0.11	-	0.19	0.42	-

Parameter	Bulk S2 EPA 1311	Bulk S2 EPA 1312	Bulk S2 CTEU-9	Bulk S3 EPA 1311	Bulk S3 EPA 1312	Bulk S3 CTEU-9	Bulk S4 EPA 1311	Bulk S4 EPA 1312	Bulk S4 CTEU-9	Drill core S2 EPA 1311	Drill core S2 EPA 1312	Drill core S2 CTEU-9	Drill core S3 EPA 1311	Drill core S3 EPA 1312	Drill core S3 CTEU-9	Drill core S4 EPA 1311	Drill core S4 EPA 1312	Drill core S4 CTEU-9	Drill core railroad EPA 1311	Drill core railroad EPA 1312	Drill core railroad CTEU-9	Average (n = 11) EPA 1311	Average (n = 11) EPA 1312	Average (n = 11) CTEU-9
	Ore samples from COREM analysed in November 2011																							
Metals and metalloids																								
Aluminium	0.51	0.19	< 0.8	0.40	< 0.08	< 0.8	0.70	< 0.08	2.1	0.78	0.09	< 0.8	0.57	0.37	< 0.8	0.43	0.19	< 0.8	0.47	0.28	< 0.8	0.53	0.28	0.52
Argent	< 0.01	< 0.01	< 0.1	< 0.01	< 0.01	< 0.1	< 0.01	< 0.01	< 0.1	< 0.01	< 0.01	< 0.1	< 0.01	< 0.01	< 0.1	< 0.01	< 0.01	< 0.1	< 0.01	< 0.01	< 0.1	< 0.01	< 0.01	< 0.1
Arsenic	< 0.004	< 0.004	< 0.04	< 0.004	< 0.004	< 0.04	< 0.004	< 0.004	< 0.04	< 0.004	< 0.004	< 0.04	< 0.004	< 0.004	< 0.04	< 0.004	< 0.004	< 0.04	< 0.004	< 0.004	< 0.04	< 0.004	< 0.004	< 0.04
Baryum	0.20	0.011	0.06	0.19	0.017	0.05	0.37	0.017	0.06	0.12	< 0.005	< 0.05	0.22	< 0.005	< 0.05	0.19	< 0.005	< 0.05	0.15	< 0.005	< 0.05	0.19	0.01	< 0.05
Cadmium	< 0.002	< 0.002	< 0.02	< 0.002	< 0.002	< 0.02	< 0.002	< 0.002	< 0.02	< 0.002	< 0.002	< 0.02	< 0.002	< 0.002	< 0.02	< 0.002	< 0.002	< 0.02	< 0.002	< 0.002	< 0.02	< 0.002	< 0.002	< 0.02
Calcium	17	8	110	17	9	140	8	4	64	26	6	< 20	26	3	< 20	11	3	< 20	24	4	< 20	18	5	51
Chrome	< 0.007	< 0.007	< 0.07	< 0.007	< 0.007	< 0.07	< 0.007	< 0.007	< 0.07	< 0.007	< 0.007	< 0.07	< 0.007	< 0.007	< 0.07	< 0.007	< 0.007	< 0.07	< 0.007	< 0.007	< 0.07	< 0.007	< 0.007	< 0.07
Cobalt	0.09	0.02	0.3	0.12	0.05	***0.9***	0.09	0.04	***0.9***	0.01	< 0.01	< 0.1	< 0.01	< 0.01	< 0.1	0.03	< 0.01	< 0.1	0.01	< 0.01	< 0.1	0.03	0.01	0.2
Cuivre	***0.80***	***0.03***	< 0.2	***0.46***	< 0.02	< 0.2	***0.17***	< 0.02	***0.3***	***0.04***	< 0.02	< 0.2	< 0.02	< 0.02	< 0.2	< 0.02	< 0.02	< 0.2	< 0.02	< 0.02	< 0.2	***0.12***	< 0.02	< 0.2
Étain	< 0.05	< 0.05	< 0.5	< 0.05	< 0.05	< 0.5	< 0.05	< 0.05	< 0.5	< 0.05	< 0.05	< 0.5	< 0.05	< 0.05	< 0.5	< 0.05	< 0.05	< 0.5	< 0.05	< 0.05	< 0.5	< 0.05	< 0.05	< 0.5
Fer	< 2	< 2	< 20	3	< 2	< 20	3	< 2	< 20	5	< 2	< 20	8	< 2	< 20	5	< 2	< 20	7	< 2	< 20	5	< 2	< 20
Manganèse	0.80	0.27	3.9	0.87	0.47	8.5	1.1	0.48	11	0.85	< 0.01	< 0.1	1.0	< 0.01	< 0.1	0.83	0.02	< 0.1	0.96	< 0.01	< 0.1	0.87	0.13	2.0
Mercure	< 0.01	< 0.01	< 0.1	< 0.01	< 0.01	< 0.1	< 0.01	< 0.01	< 0.1	< 0.01	< 0.01	< 0.1	< 0.01	< 0.01	< 0.1	< 0.01	< 0.01	< 0.1	< 0.01	< 0.01	< 0.1	< 0.01	< 0.01	< 0.1
Molybdène	< 0.01	< 0.01	< 0.1	< 0.01	< 0.01	< 0.1	< 0.01	< 0.01	< 0.1	< 0.01	< 0.01	< 0.1	< 0.01	< 0.01	< 0.1	< 0.01	< 0.01	< 0.1	< 0.01	< 0.01	< 0.1	< 0.01	< 0.01	< 0.1
Nickel	0.059	0.013	0.20	0.050	0.024	0.41	0.020	0.007	0.16	0.016	< 0.006	< 0.06	0.022	< 0.006	< 0.06	0.029	< 0.006	< 0.06	0.026	< 0.006	< 0.06	0.031	0.007	0.10
Plomb	< 0.01	< 0.01	< 0.1	< 0.01	< 0.01	< 0.1	< 0.01	< 0.01	< 0.1	< 0.01	< 0.01	< 0.1	< 0.01	< 0.01	< 0.1	< 0.01	< 0.01	< 0.1	< 0.01	< 0.01	< 0.1	< 0.01	< 0.01	< 0.1
Sélénium	< 0.005	< 0.005	< 0.05	< 0.005	< 0.005	< 0.05	< 0.005	< 0.005	< 0.05	< 0.005	< 0.005	< 0.05	< 0.005	< 0.005	< 0.05	< 0.005	< 0.005	< 0.05	< 0.005	< 0.005	< 0.05	< 0.005	< 0.005	< 0.05
Titane	< 0.05	< 0.05	< 0.5	< 0.05	< 0.05	< 0.5	< 0.05	< 0.05	< 0.5	< 0.05	< 0.05	< 0.5	< 0.05	< 0.05	< 0.5	< 0.05	< 0.05	< 0.5	< 0.05	< 0.05	< 0.5	< 0.05	< 0.05	< 0.5
Uranium	< 0.02	< 0.02	< 0.2	< 0.02	< 0.02	< 0.2	< 0.02	< 0.02	< 0.2	< 0.02	< 0.02	< 0.2	< 0.02	< 0.02	< 0.2	< 0.02	< 0.02	< 0.2	< 0.02	< 0.02	< 0.2	< 0.02	< 0.02	< 0.2
Zinc	< 0.2	< 0.2	< 2	< 0.2	< 0.2	< 2	< 0.2	< 0.2	< 2	< 0.2	< 0.2	< 2	< 0.2	< 0.2	< 2	< 0.2	< 0.2	< 2	< 0.2	< 0.2	< 2	< 0.2	< 0.2	< 2
Others																								
Fluorures	0.3	0.2	0.2	0.3	0.1	0.2	0.8	0.3	3	0.3	0.1	2.5	0.3	< 0.1	3.0	0.3	0.1	3.7	0.2	0.1	2.4	0.3	0.1	1.7
Phosphore total (P)	< 0.02	0.09	0.03	< 0.02	0.03	0.03	< 0.02	< 0.02	< 0.02	0.19	0.13	0.47	0.25	0.15	0.96	0.23	0.13	0.92	0.18	0.14	0.82	0.18	0.13	0.44

Les valeurs excédant un des critères sont ombragées et en caractères gras et italiques.

[1] Politique de protection des sols et de la réhabilitation des sites contaminés (MDDEP, 2001). Pour la résurgence des eaux de surface

[2] Directive 019 sur l'industrie minière (MDDEP, 2005).

[3] Règlement sur les effluents des mines de métaux

* Ce critère augmente avec la dureté. La valeur indiquée correspond à une dureté de 50 mg $CaCO_3$/l. Voir « Critères de qualité de l'eau de surface au Québec » (MDDEP, 2009).

** Le critère de phosphore total vise à la base à limiter la croissance excessive d'algues et de plantes aquatiques dans les cours d'eau. Un critère plus sévère s'appliquerait à l'occasion de la résurgence de l'eau souterraine dans un cours d'eau s'écoulant vers un lac ou à l'occasion de la résurgence de l'eau souterraine dans un lac. Ces situations sont traitées sur une base de cas par cas.

*** Ce critère varie selon les teneurs en chlorures, voir « Critères de qualité de l'eau de surface au Québec » (MENV 2001). La valeur citée dans le tableau correspond à une concentration en chlorures de 2000 µg/L.

See Roche-Ausenco feasibility study for high resolution image

Acid Generation Potential

Results of the acid generation potential test on ore samples are presented in Table 20-8. A material (such as ore, waste and tailings) can be defined as potentially acid generating when:

- It contains sulfur in concentrations above 0.3%, and
- It shows a net acid neutralization potential20 below 20 kg CaCO3/t or an ANP:AGP ratio below 3.

From these results, we see that ore has no acid generation potential, and the mean total sulphur content (0.19%) remains below 0.2%.

Table 20-8: Acid Generation Potential of Ore Samples

Parameter	Unit	Criterion [1]	Results							Statistics
			SGS - Lakefield			COREM				
			PP-16 S2/3 Comp Feed	PP-21 S2 Feed	PP-23 S4 Feed	S2 - Drill Core	S3 - Drill Core	RR - Drill Core	Blend RR/S2/S3 Drill	Average (n = 7)
Acid generating potential										
pH	pH units	-	7,85	8,65	8,31	8,76	8,66	8,82	8,69	-
Acid neutralization potential (ANP)	kg $CaCO_3$/t	-	32,9	33,0	27,4	37,00	37,06	40,44	40,07	35,4
Acid generating potential (AGP)	kg $CaCO_3$/t	-	1,75	0,40	1,91	8,44	7,50	7,50	8,44	5,13
Net acid neutralization potential (ANP-AGP	kg $CaCO_3$/t	> 20	31,2	32,6	25,5	28,56	29,56	32,94	31,63	30,3
Ratio (ANP/AGP)	unitless	> 3	18,8	83,1	14,4	4,38	4,94	5,39	4,75	19,4
Total sulphur	%	< 0,3	0,132	0,036	0,136	0,27	0,24	0,24	0,27	0,19

[1] From Directive 019 on the Mining Industry (MDDEP, 2005)

Values exceeding a criterion (anyone of the criterion) are in bold and italics.

A material (such as ore, waste and tailings) can be defined as potentially acid generating when:

- it presents a sulphur concentration above 0,3%;
- it shows a net acid neutralization potential below 20 kg $CaCO_3$/t or an ANP:AGP ratio below 3.

**See Roche-Ausenco feasibility study for high resolution image*

20.2.3 Environmental Characteristics of Overburden

Elemental Composition

Table 20-9 shows overburden's elemental composition. The results show that all of the parameters analysed on grab samples are below the quality criteria used, except for barium, which sometimes exceeds the provincial soil protection A-criterion, and boron, which sometimes exceeds the federal soil quality guidelines for the protection of environmental and human health (for agricultural use). It should however be noted that:

[20] The net acid neutralization potential is defined as ANP-AGP, where ANP stands for « Acid neutralization potential » and AGP stands for « Acid generating potential », both expressed as kg CaCO3/t. A static acid generation potential test, such as the modified Sobek test used here, allows determining both the ANP and AGP of a rock sample.

The mean barium concentration remains below the A-criterion;

- There are no soil quality guidelines for the protection of environmental and human health (for industrial and/or commercial uses);
- Overburden is not intended to be used for agricultural purposes.

As a conclusion, results show there is no environmental concern with this material, so that it could be used for civil constructions with no restriction on the mine property[21] (see Section 20.2.7for more details).

Leaching Tests

Results from the 1311, 1312 and CTEU-9 leaching tests conducted on overburden samples are shown in Table 20-10.

All leachates produced from TCLP (as well as SPLP and CTEU-9) tests report constituent concentrations below the limits in Directive 019 defining high-risk residues. Hence, overburden is not a high-risk waste.

All of the parameters tested on the leachates meet all the environmental standards and criteria used, except for copper and aluminum, which may sometimes be found in concentrations above the provincial criterion for the protection of ground waters. It should be noted however that the results for copper and mercury are too close[22] to the analytical detection limit to be totally meaningful. Moreover, as indicated above, the copper (and mercury) content in the solid (in mg/kg) remains below the provincial A-criterion for soil protection.

As for aluminum, one should note that concentrations of this element in surface waters near the mine site are naturally quite high[23]. In fact, aluminum concentrations are always above the federal and provincial criteria for the protection of aquatic life (chronic toxicity) and occasionally exceed the provincial criterion for the protection of aquatic life (acute toxicity), which is the same as the one used for the protection of ground waters (0.75 mg/l). It is thus not surprising to observe aluminum concentrations in the leachates above the criterion for the protection of ground waters.

[21] When the chemical composition of a sample is below the A-criteria, the material is considered a low-risk waste, and no aquifer protection measures are required. On the contrary, when the concentration of some elements exceeds the respective A-criterion, TCLP leaching test results need to be considered to determine whether the material should be assimilated to a high-risk waste, a leachable waste or a low-risk waste.

[22] Except possibly for one result (with the SPLP) out of 27, where a copper concentration of 0.10 mg/l is noted.

[23] Indeed, aluminum concentrations measured in October 2010 in streams and water bodies surrounding the mine site varied from 0.32 mg/l to 0.86 mg/l (mean = 0.57 mg/l; n = 14) (Roche, January 2011). In July 2011, it varied from 0.41 mg/l to 0.91 mg/l (mean = 0.62 mg/l; n = 22) (Roche, in prep.).

Table 20-9: Elemental Composition of Overburden Samples

Element *	Unit	Québec soil protection criteria [1]			CCME - Soil Quality Guidelines for the Protection of Environmental and Human Health [2]				Results									Statistics		
					Zoning				Overburden (n=9)									Overburden (n=9)		
		A-Criterion	B-Criterion	C-Criterion	Agricultural	Residential/ parkland	Commercial	Industrial	BH-4, SS-7, 23'6"-25'-6"	BH-5, SS-5, 15'-17'	BH-6, SS-25, 68'-70'	BH-6, SS-5, 13'-15'	BH-10, SS-15, 68'3"-70'3"	BH-10, ST-8, 33'3"-35'3"	BH-10, ST-4, 13'3"-15'3"	BH-9, ST-4, 13'6"-13'8"	BH-9, ST-9, 38'6"-40'6"	Min	Max	Mean
Aluminum	mg/kg	-	-	-	-	-	-	-	4 300	5 500	3 800	14 000	19 000	21 000	20 000	21 000	23 000	3 800	23 000	14 622
Antimony	mg/kg	-	-	-	20	20	40	40	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1
Arsenic	mg/kg	15	30	50	12	12	12	12	0,5	<0,5	<0,5	<0,5	3,3	<0,5	4,0	<0,5	<0,5	<0,5	4,0	<1,2
Baryum	mg/kg	**265**	500	2 000	750	500	2 000	2 000	29	38	35	240	***310***	***340***	***320***	***330***	***390***	29	***390***	226
Beryllium	mg/kg	-	-	-	4	4	8	8	0,2	0,2	0,1	0,5	0,6	0,6	0,6	0,6	0,7	0,1	0,7	0,5
Boron	mg/kg	-	-	-	**2**	-	-	-	<2	<2	<2	<2	***5***	***7***	***3***	***3***	***6***	<2	***7***	***<4***
Cadmium	mg/kg	1,5	5	20	1,4	10	22	22	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1
Calcium	mg/kg	-	-	-	-	-	-	-	2 100	2 400	2 700	6 200	7 100	8 100	7 500	7 400	7 400	2 100	8 100	5 656
Chromium	mg/kg	85	250	800	64	64	87	87	7	8	8	29	37	43	42	42	46	7	46	29
Cobalt	mg/kg	25	50	300	40	50	300	300	5	4	6	12	15	16	15	15	17	4	17	12
Copper	mg/kg	100	100	500	63	63	91	91	16	16	8	39	25	29	25	30	31	8	39	24
Iron	mg/kg	-	-	-	-	-	-	-	8 200	9 600	9 200	29 000	35 000	43 000	38 000	41 000	52 000	8 200	52 000	29 444
Lead	mg/kg	50	500	1 000	70	140	260	600	2	3	2	8	8	7	7	7	8	2	8	6
Manganese	mg/kg	1 000	1 000	2 200	-	-	-	-	92	110	98	480	620	690	630	660	770	92	770	461
Mercury	mg/kg	0,4	2	10	-	-	-	-	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01
Molybdenum	mg/kg	6	10	40	5	10	40	40	<0,5	<0,5	<0,5	0,9	0,9	1,1	0,8	0,8	0,8	<0,5	1,1	<0,8
Nickel	mg/kg	100	100	500	50	50	50	50	5,8	5,1	5,4	11	13	14	14	22	14	5	22	12
Selenium	mg/kg	3	3	10	1	1	2,9	2,9	<0,5	<0,5	<0,5	<0,5	<0,5	<0,5	<0,5	<0,5	<0,5	<0,5	<0,5	<0,5
Silver	mg/kg	2	20	40	20	20	40	40	0,5	<0,5	<0,5	<0,5	<0,5	<0,5	<0,5	<0,5	<0,5	<0,5	0,5	<0,5
Strontium	mg/kg	-	-	-	-	-	-	-	9	12	10	44	47	56	49	49	51	9	56	36
Thallium	mg/kg	-	-	-	1	1	1	1	<0,1	<0,1	<0,1	0,2	0,3	0,3	0,3	0,3	0,3	<0,1	0,3	<0,3
Tin	mg/kg	5	50	300	5	50	300	300	1	2	1	2	2	2	1	1	1	1	2	1
Titanium	mg/kg	-	-	-	-	-	-	-	410	450	490	2 200	2 800	3 100	2 800	3 000	3 400	410	3 400	2 072
Uranium	mg/kg	-	-	-	23	23	33	300	<2	<2	<2	<2	<2	<2	<2	<2	<2	<2	<2	<2
Vanadium	mg/kg	-	-	-	130	130	130	130	22	19	26	52	62	81	76	79	85	19	85	56
Zinc	mg/kg	230	500	1 500	200	200	360	360	16	18	16	63	76	76	76	77	85	16	85	56

[1] The criteria used come from the *Soil protection and rehabilitation of contaminated sites policy* (MDDEP, 2001). These quite stringent criteria are used to guide the management of contaminated sites and mine waste. The A criteria represents maximum natural background concentrations encountered among the five Quebec's geological provinces. Within the A-B range, the material (soil) can be used with no restriction on commercial/industrial lots, except if the activity results in an increase in soil contamination. Within the B-C range, the material can be used as backfill (only) on commercial/industrial lots, except if the activity results in an increase in soil contamination. Above the C criteria, the material should be decontaminated or disposed of in a landfill authorized to receive contaminated soils.

[2] CCME; Summary Table for the Canadian Environmental Quality Guidelines; http://st-ts.ccme.ca/ (Website consulted on May 3, 2011)

* MA. 200 Digestion method

Values exceeding a criterion (anyone of the criteria) are in bold, italics and highlighted.

**See Roche-Ausenco feasibility study for high resolution image*

Table 20-10: Results of the Leaching Tests (EPA 1311, 1312 and CTEU-9) on Overburden Samples

Paramètre	Unité	Critères: Critère eaux souterraines [1]	Critères: Normes canadienne Concentration moyenne mensuelle maximale permise	Critères: Critère pour les résidus miniers à risques élevés [2]	Critères: Normes québécoises d'eau potable	Résultats: BH-4, SS-7, 23'8"-25'8" EPA 1311	EPA 1312	CTEU-9	BH-6, SS-6, 16'-17' EPA 1311	EPA 1312	CTEU-9	BH-6, SS-26, 68'-70' EPA 1311	EPA 1312	CTEU-9	BH-8, SS-6, 13'-16' EPA 1311	EPA 1312	CTEU-9	BH-10, SS-16, 68'0"-70'0" EPA 1311	EPA 1312	CTEU-9	BH-10, ST-8, 30'0"-36'0" EPA 1311	EPA 1312	CTEU-9	BH-10, ST-4, 13'3"-16'3" EPA 1311	EPA 1312	CTEU-9	BH-8, ST-4, 13'6"-15'6" EPA 1311	EPA 1312	CTEU-9	BH-8, ST-8, 38'6"-40'6" EPA 1311	EPA 1312	CTEU-9	Statistiques: Moyennes (n = 9) EPA 1311	EPA 1312	CTEU-9
Métaux et métalloïdes																																			
Aluminium	mg/l	0,75	-	-	-	0,33	<0,08	***3,7***	0,57	<0,08	0,73	0,37	<0,08	<0,08	0,41	***20***	***1,2***	***0,82***	***21***	0,57	***0,89***	***25***	***7,8***	***0,87***	0,11	***1,1***	0,72	0,20	***0,81***	***1,1***	***15***	0,70	0,60	***9***	***1,8***
Argent	mg/l	0,00062***	-	-	-	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01
Arsenic	mg/l	0,34	0,50	5,0	0,025	<0,004	<0,004	<0,004	<0,004	<0,004	<0,004	<0,004	<0,004	<0,004	<0,004	<0,004	<0,004	<0,004	0,005	0,017	<0,004	<0,004	<0,004	<0,004	<0,004	<0,004	<0,004	<0,004	<0,004	<0,004	<0,004	<0,004	<0,004	0,002	0,004
Baryum	mg/l	6,7***	-	100	1	0,23	<0,005	0,045	0,25	<0,005	0,02	0,22	0,006	0,065	0,3	0,31	0,032	0,14	0,29	0,024	0,14	0,34	0,11	0,17	0,013	0,038	0,13	0,012	0,061	0,24	0,24	0,020	0,20	0,13	0,06
Cadmium	mg/l	0,0021***	-	0,5	0,005	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002
Chrome	mg/l	-	-	5,0	0,05	<0,007	<0,007	<0,007	<0,007	<0,007	<0,007	<0,007	<0,007	<0,007	<0,007	0,029	<0,007	<0,007	0,019	<0,007	0,015	0,026	0,011	<0,007	<0,007	<0,007	<0,007	<0,007	<0,007	<0,007	0,013	<0,007	0,005	0,012	0,004
Cobalt	mg/l	0,5	-	-	-	<0,01	<0,01	<0,01	0,03	<0,01	<0,01	0,03	0,01	0,12	<0,01	0,01	<0,01	0,02	0,02	<0,01	<0,01	0,02	<0,01	0,02	<0,01	0,08	0,01	<0,01	0,05	0,02	0,01	<0,01	0,02	0,01	0,03
Cuivre	mg/l	0,0073***	0,30	-	1	***0,04***	<0,02	<0,02	<0,02	<0,02	<0,02	<0,02	<0,02	<0,02	<0,02	***0,03***	***0,05***	<0,02	***0,10***	***0,02***	<0,02	***0,10***	***0,03***	***0,05***	<0,02	***0,02***	***0,06***	<0,02	***0,03***	<0,02	***0,06***	<0,02	***0,02***	***0,04***	***0,02***
Étain	mg/l	-	-	-	-	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05
Fer	mg/l	-	-	-	-	<2	<2	3	<2	<2	<2	<2	<2	<2	10	20	<2	12	28	<2	21	39	13	<2	<2	<2	<2	<2	<2	74	26	<2	14	13	3
Manganèse	mg/l	-	-	-	-	0,12	<0,01	0,04	0,38	<0,01	0,02	0,14	0,10	0,77	0,68	0,38	0,11	0,82	0,61	0,12	0,96	0,80	0,24	1,1	0,50	3,9	0,68	0,26	3,3	2,7	0,54	0,33	0,84	0,36	0,98
Mercure	mg/l	0,00013	-	0,1	0,001	<0,0001	<0,0001	***0,0002***	<0,0001	<0,0001	<0,0001	0,0001	<0,0001	<0,0001	<0,0001	<0,0001	<0,0001	<0,0001	<0,0001	<0,0001	<0,0001	<0,0001	<0,0001	<0,0001	<0,0001	<0,0001	<0,0001	<0,0001	***0,0002***	<0,0001	<0,0001	<0,0001	0,0001	<0,0001	0,0001
Molybdène	mg/l	2	-	-	-	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	0,04	<0,01	<0,01	0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	0,01
Nickel	mg/l	0,260***	0,50	-	-	0,012	<0,006	<0,006	0,009	<0,006	<0,006	0,051	0,02	0,2	0,025	0,019	<0,006	0,046	0,028	<0,006	0,033	0,036	0,013	0,076	0,033	0,26	0,032	0,012	0,15	0,046	0,021	0,007	0,04	0,02	0,07
Plomb	mg/l	0,034***	0,20	5,0	0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	0,02	<0,01	<0,01	0,02	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	0,01	<0,01	<0,01	0,01	<0,01
Sélénium	mg/l	0,020	-	1,0	0,01	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005
Zinc	mg/l	0,067***	0,5	-	-	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	0,3	<0,2	<0,2	<0,2	<0,2	<0,2	> 0,04 < 0,21
Autres																																			
Fluorures	mg/l	4	-	150	1,5	0,1	<0,1	0,2	0,1	<0,1	0,1	0,1	<0,1	0,1	0,2	0,1	0,2	0,7	0,5	0,8	0,6	0,8	2,5	0,5	0,2	1	0,5	0,2	0,6	0,6	1,2	0,7	0,4	0,4	0,7
Phosphore total (P)	mg/l	2****	-	-	-	0,03	0,06	1,2	<0,02	0,03	1,2	<0,02	0,03	0,07	0,07	2,1	0,91	0,19	1,9	0,51	0,08	1,6	1,4	0,06	0,07	0,05	0,06	0,05	0,06	0,09	1,3	0,31	0,08	0,79	0,63

Les valeurs excédant un des critères sont ombragées et en caractères gras et italiques.

[1] Politique de protection des sols et de la réhabilitation des sites contaminés (MDDEP, 2001). Pour la résurgence des eaux de surface

[2] Directive 019 sur l'industrie minière (MDDEP, 2005).

[3] Règlement sur les effluents des mines de métaux

*** Ce critère augmente avec la dureté. La valeur indiquée correspond à une dureté de 50 mg $CaCO_3$/l. Voir « Critères de qualité de l'eau de surface au Québec » (MDDEP, 2009). Notez que les conclusions demeurent inchangées en utilisant une dureté de 10 mg/l.

**** Le critère de phosphore total vise à la base à limiter la croissance excessive d'algues et de plantes aquatiques dans les cours d'eau. Un critère plus sévère s'appliquerait à l'occasion de la résurgence de l'eau souterraine dans un cours d'eau s'écoulant vers un lac ou à l'occasion de la résurgence de l'eau souterraine dans un lac. Ces situations sont traitées sur une base de cas par cas.

See Roche-Ausenco feasibility study for high resolution image

Table 20-11: Elemental composition of waste rock samples

Element [a]	Unit	Québec soil protection criteria [1]			CCME - Soil Quality Guidelines for the Protection of Environmental and Human Health [2]				Results												Statistics		
					Zoning				Waste rock (n=12)												Waste rock (n=12)		
		A-Criterion	B-Criterion	C-Criterion	Agricultural	Residential/ parkland	Commercial	Industrial	26350	27294	27266	26334	24561	24574	21205	24541	21293	21282	24519	21261	Min	Max	Mean
Aluminum	mg/kg	-	-	-	-	-	-	-	3 800	4 300	2 300	3 400	8 850	1 500	2 500	3 400	5 700	5 500	4 600	3 100	1 500	8 850	4 079
Antimony	mg/kg	-	-	-	20	20	40	40	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1
Arsenic	mg/kg	15	30	50	12	12	12	12	<0,5	<0,5	<0,5	<0,5	<0,5	<0,5	<0,5	1,0	<0,5	<0,5	<0,5	<0,5	<0,5	1,0	<0,5
Baryum	mg/kg	265	500	2 000	750	500	2 000	2 000	7	110	57	15	17	6	34	42	11	35	46	57	6	110	36
Beryllium	mg/kg	-	-	-	4	4	8	8	<0,1	<0,1	<0,1	<0,1	0,2	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1	0,2	<0,1
Boron	mg/kg	-	-	-	*2*	-	-	-	< 2	< 2	***19***	2	***13***	<2	<2	<2	***11***	***4***	<2	<2	< 2	***19***	***5***
Cadmium	mg/kg	1,5	5	20	1,4	10	22	22	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1
Calcium	mg/kg	-	-	-	-	-	-	-	2 700	8 100	85 000	12 000	16 000	8 400	33 000	21 000	13 000	11 000	6 500	13 000	2 700	85 000	19 142
Chromium	mg/kg	85	250	800	64	64	87	87	21	41	4	12	11	1	<1	11	4	29	58	25	1	58	20
Cobalt	mg/kg	*25*	*50*	300	*40*	*50*	300	300	***44***	21	***32***	24	***29***	***78***	20	15	***32***	***43***	14	21	14	***78***	***31***
Copper	mg/kg	*100*	*100*	500	*63*	*63*	*91*	*91*	13	41	***100***	52	12	55	4	***380***	11	34	11	36	4	***380***	62
Iron	mg/kg	-	-	-	-	-	-	-	84 000	41 000	53 000	39 000	56 000	51 000	100 000	30 000	57 000	75 000	23 000	36 000	23 000	100 000	53 750
Lead	mg/kg	50	500	1 000	70	140	260	600	<1	<1	<1	<1	<1	<1	<1	<1	<1	<1	<1	<1	<1	<1	<1
Manganese	mg/kg	*1 000*	*1 000*	2 200	-	-	-	-	820	500	740	610	490	270	***2 200***	200	430	***1 100***	180	420	180	***2 200***	663
Mercury	mg/kg	0,4	2	10	-	-	-	-	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01
Molybdenum	mg/kg	6	10	40	5	10	40	40	<0,5	<0,5	<0,5	<0,5	<0,5	<0,5	<0,5	<0,5	<0,5	<0,5	<0,5	<0,5	<0,5	<0,5	<0,5
Nickel	mg/kg	100	100	500	*50*	*50*	*50*	*50*	24	34	7,3	46	7,8	<0,5	<0,5	16	<0,5	***75***	***51***	23	7	***75***	24
Selenium	mg/kg	*3*	*3*	10	*1*	*1*	*2,9*	*2,9*	<0,5	<0,5	***3,9***	<0,5	<0,5	<0,5	1	0,8	<0,5	<0,5	<0,5	<0,5	<0,5	***3,9***	0,7
Silver	mg/kg	2	20	40	20	20	40	40	<0,5	<0,5	<0,5	<0,5	<0,5	<0,5	<0,5	<0,5	<0,5	<0,5	<0,5	<0,5	<0,5	<0,5	<0,5
Strontium	mg/kg	-	-	-	-	-	-	-	23	32	180	47	72	24	81	57	38	57	31	32	23	180	56
Thallium	mg/kg	-	-	-	1	1	1	1	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1	<0,1
Tin	mg/kg	5	50	300	5	50	300	300	<1	<1	<1	<1	<1	<1	<1	1	<1	1	1	<1	<1	1	<1
Titanium	mg/kg	-	-	-	-	-	-	-	5 100	980	1 000	680	550	2 200	830	810	2 700	490	950	1 100	490	5 100	1 449
Uranium	mg/kg	-	-	-	23	23	33	300	<2	<2	<2	<2	<2	<2	<2	<2	<2	<2	<2	<2	<2	<2	<2
Vanadium	mg/kg	-	-	-	*130*	*130*	*130*	*130*	***280***	110	76	64	***155***	***160***	7	***150***	***170***	120	110	100	7	***280***	125
Zinc	mg/kg	230	500	1 500	200	200	360	360	42	53	62	32	51	48	68	43	44	52	32	30	30	68	46

[1] The criteria used come from the *Soil protection and rehabilitation of contaminated sites policy* (MDDEP, 2001). These quite stringent criteria are used to guide the management of contaminated sites and mine waste. The A criteria represents maximum natural background concentrations encountered among the five Quebec's geological provinces. Within the A-B range, the material (soil) can be used with no restriction on commercial/industrial lots, except if the activity results in an increase in soil contamination. Within the B-C range, the material can be used as backfill (only) on commercial/industrial lots, except if the activity results in an increase in soil contamination. Above the C criteria, the material should be decontaminated or disposed of in a landfill authorized to receive contaminated soils.

[2] CCME; Summary Table for the Canadian Environmental Quality Guidelines; http://st-ts.ccme.ca/ (Website consulted on May 3, 2011)

[a] MA. 200 Digestion method

Values exceeding a criterion (anyone of the criteria) are in bold, italics and highlighted.

**See Roche-Ausenco feasibility study for high resolution image*

Acid Generation Potential

Analyses to determine the acid generation potential of overburden were not conducted as this fine and oxidized material is not prone to acid generation, namely considering the inert properties of the rocky substratum.

20.2.4 Environmental Characteristics of Waste Rock

Elemental Composition

Table 20-6 shows waste rock elemental composition. Some of the metals analysed (manganese, cobalt and copper) rarely exceed the provincial B-criterion for soil protection, whereas some other elements (cobalt, vanadium, boron, copper, nickel and selenium) may exceed at least one of the federal soil quality guidelines for the protection of environmental and human health.

On average, however, cobalt is the only element that exceeds the A-criterion for soil protection (but it remains below the B-criterion), and none of the elements exceed the federal guidelines for a commercial or an industrial use of the land. Indeed, boron is the only element exceeding the Canadian soil quality guideline for the protection of environmental and human health (for an agricultural usage of the land).

Leaching Tests

Results from the 1311, 1312 and CTEU-9 leaching tests conducted on waste rock samples are shown in

Table 20-12. None of the leachates from the waste rock samples show concentrations exceeding the criteria used to define a high-risk waste, and hence, the material is not a high-risk waste. As well, none of the results shows concentrations exceeding the Canadian or provincial standards for a mining effluent.

In fact, all of the parameters tested on the leachates meet the environmental standards and criteria used24, except for copper and aluminum, which are occasionally found in concentrations above the environmental criteria for the protection of groundwater.

It should be noted that the mean copper concentration in the leachate from the TCLP test is however too close25 to the analytical detection limit to be meaningful (and that the mean copper concentration in the leachate from the two other leaching tests is below the analytical detection limit).

As for aluminum, mean concentrations in the leachate is above the criterion for the protection of soil for the TCLP test, but not for the SPLP and the CTEU-9 tests.

[24] Including the Québec standards for drinking water.
[25] Except for one result out of 36, where a copper concentration of 0.15 mg/l is noted, but this is clearly the case for the mean concentration of 0.03 mg/l, when compared to a method detection limit of 0.02 mg/l.

Also note that cobalt does not leach, although it is present in the solid at concentrations exceeding the A-criterion for soil quality.

Table 20-12: Results of the Leaching Tests (EPA 1311, 1312 and CTEU-9) on Waste Rock Samples

Paramètre	Unité	Critères				Résultats											
		Critère eaux souterraines [1]	Standards canadien [3] Concentration moyenne mensuelle maximale permise	Critère pour les résidus miniers à risques élevés [2]	Normes québécoises d'eau potable	26350			27294			27266			26334		
						EPA 1311	EPA 1312	CTEU-9	EPA 1311	EPA 1312	CTEU-9	EPA 1311	EPA 1312	CTEU-9	EPA 1311	EPA 1312	CTEU-9
Métaux et métalloïdes																	
Aluminium	mg/l	***0,75***	-	-	-	***2,7***	***0,94***	0,71	***1,8***	***0,81***	***1,4***	0,49	0,15	<0,08	***1,4***	***0,78***	***0,78***
Argent	mg/l	0,00062 [aaa]	-	-	-	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01
Arsenic	mg/l	0,34	0,50	5,0	0,025	<0,004	<0,004	<0,004	<0,004	<0,004	<0,004	<0,004	<0,004	<0,004	<0,004	<0,004	<0,004
Baryum	mg/l	5,3 [aaa]	-	100	1	0,053	<0,005	<0,005	0,14	<0,005	0,012	0,23	<0,005	<0,005	0,14	<0,005	<0,005
Cadmium	mg/l	0,0021 [aaa]	-	0,5	0,005	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002
Chrome	mg/l	-	-	5,0	0,05	<0,007	<0,007	<0,007	<0,007	<0,007	<0,007	<0,007	<0,007	<0,007	<0,007	<0,007	<0,007
Cobalt	mg/l	0,5	-	-	-	0,01	<0,01	<0,01	<0,01	<0,01	<0,01	0,01	<0,01	<0,01	<0,01	<0,01	<0,01
Cuivre	mg/l	***0,0073*** [aaa]	0,30	-	1	***0,04***	<0,02	<0,02	<0,02	<0,02	<0,02	<0,02	<0,02	<0,02	<0,02	<0,02	<0,02
Etain	mg/l	-	-	-	-	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05
Fer	mg/l	-	-	-	-	11	<2	<2	12	<2	3	10	<2	<2	9	<2	<2
Manganèse	mg/l	-	-	-	-	0,98	<0,01	<0,01	1,6	<0,01	0,05	3,1	<0,01	<0,01	0,80	<0,01	0,01
Mercure	mg/l	***0,00013***	-	0,1	0,001	<0,0001	<0,0001	<0,0001	<0,0001	***0,0002***	<0,0001	<0,0001	0,0001	<0,0001	0,0001	<0,0001	<0,0001
Molybdène	mg/l	2	-	-	-	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01
Nickel	mg/l	0,260 [aaa]	0,50	-	-	0,018	<0,006	<0,006	0,020	<0,006	<0,006	0,013	<0,006	<0,006	0,018	<0,006	<0,006
Plomb	mg/l	0,034 [aaa]	0,20	5,0	0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01
Sélénium	mg/l	0,020	-	1,0	0,01	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005
Zinc	mg/l	***0,067*** [aaa]	0,5	-	-	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2
Autres																	
Fluorures	mg/l	4	-	150	1,5	<0,1	<0,1	<0,1	0,2	<0,1	0,5	0,4	0,2	0,8	0,2	<0,1	0,3
Phosphore total (P)	mg/l	3 [bbb]	-	-	-	<0,02	0,03	0,33	<0,02	0,14	***4,7***	0,55	0,18	0,74	0,05	0,1	***4,8***

Paramètre	Résultats											
	24561			24574			21205			24541		
	EPA 1311	EPA 1312	CTEU-9	EPA 1311	EPA 1312	CTEU-9	EPA 1311	EPA 1312	CTEU-9	EPA 1311	EPA 1312	CTEU-9
Métaux et métalloïdes												
Aluminium	***1,1***	***1,2***	0,61	***3,4***	***0,79***	0,20	***1,1***	***0,79***	0,45	***1,3***	***0,79***	0,67
Argent	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01
Arsenic	<0,004	<0,004	<0,004	<0,004	<0,004	<0,004	<0,004	<0,004	<0,004	<0,004	<0,004	<0,004
Baryum	0,18	0,18	<0,005	0,076	<0,005	<0,005	0,42	<0,005	<0,005	0,20	<0,005	<0,005
Cadmium	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002
Chrome	<0,007	<0,007	<0,007	<0,007	<0,007	<0,007	<0,007	<0,007	<0,007	<0,007	<0,007	<0,007
Cobalt	0,02	0,02	<0,01	0,09	<0,01	<0,01	0,01	<0,01	<0,01	<0,01	<0,01	<0,01
Cuivre	<0,02	<0,02	<0,02	***0,15***	<0,02	<0,02	<0,02	<0,02	<0,02	<0,02	<0,02	<0,02
Etain	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05
Fer	7,5	8	<2	10	<2	<2	8	<2	<2	8	<2	<2
Manganèse	1,1	1,0	0,01	0,62	<0,01	<0,01	0,86	<0,01	<0,01	0,93	<0,01	0,01
Mercure	0,0001	<0,0001	<0,0001	<0,0001	<0,0001	<0,0001	<0,0001	<0,0001	<0,0001	0,0001	<0,0001	<0,0001
Molybdène	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01
Nickel	0,01	0,01	<0,006	0,011	<0,006	<0,006	0,009	<0,006	<0,006	0,016	<0,006	<0,006
Plomb	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01
Sélénium	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005
Zinc	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2
Autres												
Fluorures	0,1	<0,1	0,2	0,4	<0,1	0,2	0,5	0,1	0,5	0,2	<0,1	0,3
Phosphore total (P)	0,03	0,085	***4,05***	0,1	<0,02	0,07	0,3	0,15	0,36	0,1	0,13	***11***

Paramètre	Résultats												Statistics		
	21293			21282			24519			21261			Average (n = 12)		
	EPA 1311	EPA 1312	CTEU-9	EPA 1311	EPA 1312	CTEU-9	EPA 1311	EPA 1312	CTEU-9	EPA 1311	EPA 1312	CTEU-9	EPA 1311	EPA 1312	CTEU-9
Métaux et métalloïdes															
Aluminium	***0,98***	0,28	<0,08	***1,3***	0,30	0,38	***1,7***	***1,2***	***2,4***	***1,7***	0,53	***0,90***	***1,6***	0,72	0,72
Argent	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01
Arsenic	<0,004	<0,004	<0,004	<0,004	<0,004	<0,004	<0,004	<0,004	<0,004	<0,004	<0,004	<0,004	<0,004	<0,004	<0,004
Baryum	0,070	<0,005	<0,005	0,068	<0,005	<0,005	0,13	<0,005	0,013	0,13	<0,005	0,005	0,153	0,017	<0,005
Cadmium	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002
Chrome	<0,007	<0,007	<0,007	<0,007	<0,007	<0,007	0,011	<0,007	<0,007	<0,007	<0,007	<0,007	<0,007	<0,007	<0,007
Cobalt	0,10	<0,01	<0,01	0,04	<0,01	<0,01	<0,01	<0,01	<0,01	0,02	<0,01	<0,01	0,03	<0,01	<0,01
Cuivre	<0,02	<0,02	<0,02	<0,02	<0,02	<0,02	<0,02	<0,02	<0,02	***0,07***	<0,02	***0,02***	***0,03***	<0,02	<0,02
Etain	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05
Fer	17	<2	<2	18	<2	<2	8	<2	3	17	<2	<2	11	<2	<2
Manganèse	2,0	<0,01	<0,01	1,5	<0,01	<0,01	0,72	0,01	0,03	0,98	<0,01	0,02	1,3	0,09	0,01
Mercure	<0,0001	<0,0001	<0,0001	<0,0001	<0,0001	<0,0001	<0,0001	<0,0001	<0,0001	<0,0001	<0,0001	<0,0001	<0,0001	<0,0001	<0,0001
Molybdène	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01
Nickel	0,020	<0,006	<0,006	0,015	<0,006	<0,006	0,019	<0,006	<0,006	0,014	<0,006	<0,006	0,015	<0,006	<0,006
Plomb	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01
Sélénium	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005
Zinc	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2
Autres															
Fluorures	0,1	<0,1	0,1	0,1	<0,1	0,2	0,1	<0,1	0,1	0,3	0,1	0,6	0,2	<0,1	0,3
Phosphore total (P)	<0,02	<0,02	0,17	0,04	0,1	1,2	0,04	0,18	2,1	0,14	0,29	***12***	0,12	0,12	***3,5***

Les valeurs excédant un des critères sont ombragées et en caractères gras et italiques.

[1] Politique de protection des sols et de la réhabilitation des sites contaminés (MDDEP, 2001). Pour la résurgence des eaux de surface

[2] Directive 019 sur l'industrie minière (MDDEP, 2005). TCLP signifie "Toxicity Characteristic Leaching Procedure", une procédure utilisée pour déterminer la mobilité des constituants présents dans la roche.

[3] Règlement sur les effluents des mines de métaux

[aaa] Ce critère augmente avec la dureté. La valeur indiquée correspond à une dureté de 50 mg $CaCO_3$ / l. Voir « Critères de qualité de l'eau de surface au Québec » (MDDEP, 2009).

[bbb] Le critère de phosphore total vise à la base à limiter la croissance excessive d'algues et de plantes aquatiques dans les cours d'eau. Un critère plus sévère s'appliquerait à l'occasion de la résurgence de l'eau souterraine dans un cours d'eau s'écoulant vers un lac ou à l'occasion de la résurgence de l'eau souterraine dans un lac. Ces situations sont traitées sur une base de cas par cas.

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Table 20-13: Results of the Acid Generation Potential Test on Waste Rock Samples

Parameter	Unit	Criterion [1]	Results												Statistics
			Waste rock samples												Average (n = 12)
			26350	27294	27266	26334	24561	24574	21205	24541	21293	21282	24519	21261	
Acid generating potential															
Paste pH	pH units	-	9,08	9,79	9,05	9,65	9,66	8,95	9,26	9,69	8,68	8,97	9,69	9,53	9,33
Acid neutralization potential (ANP)	kg $CaCO_3$/t	-	24,1	19,2	32,2	22,7	23,3	18,4	27,8	19,5	30,0	47,7	12,2	21,5	24,9
Acid generating potential (AGP)	kg $CaCO_3$/t	-	1,56	<0.6	7,50	0,31	2,81	11,25	1,88	2,50	3,75	5,94	<0.6	1,56	3,91
Net acid neutralization potential (ANP-AGP)	kg $CaCO_3$/t	> 20	22,5	***19,2***	24,7	22,4	20,5	***7,2***	25,9	***17,0***	26,3	41,7	***12,2***	***19,9***	21,6
Ratio (ANP/AGP)	unitless	> 3	15,4	> 31,9	4,3	72,6	8,3	***1,6***	14,8	7,8	8,0	8,0	> 20,3	13,7	15,5
Sulphate sulphur	%	-	0,02	0,02	0,01	0,02	0,01	0,18	0,10	<0.01	<0.01	<0.01	<0.01	<0.01	0,05
Sulphide sulphur	%	-	0,05	<0.02	0,24	0,01	0,09	0,36	0,06	0,08	0,12	0,19	<0.02	0,05	0,13
Total sulphur	%	< 0,3	0,07	<0.02	0,25	0,03	0,10	***0,54***	0,16	0,08	0,12	0,19	<0.02	0,05	0,16

[1] From Directive 019 on the Mining Industry (MDDEP, 2005)

A material (such as ore, waste and tailings) can be defined as potentially acid generating when:

- it presents a sulphur concentration above 0,3%, and
- it shows a net acid neutralization potential below 20 kg $CaCO_3$/t or an ANP:AGP ratio below 3.

Notes:

Total sulphur, done by Leco at Acme Labs.

Calculations:

*Sulphide sulphur is based on difference between total sulphur and sulphate sulphur.

**AGP is based on sulphide sulphur.

***NNP (Net Neutralization Potential) is based on difference between ANP and AGP.

References:

Reference for Modified ABA NP Method (Maxxam SOP No. 7150): MEND Acid Rock Drainage Prediction Manual, MEND Project 1.16.1b (pages 6.2-11 to 17), March 1991.

**See Roche-Ausenco feasibility study for high resolution image*

Acid Generation Potential

Results of the acid generation potential test conducted on waste rock samples, which are shown in Table 20-13, indicate that waste rock present no potential for acid generation, except for one (sample 24574) out of the twelve samples analysed. When looking at the mean results, waste rock shows no acid generation potential, with a net acid neutralization potential of 21.6 kg CaCO3/t and a mean total sulphur content that remains below 0.2%.

As a conclusion, waste rock could be used for civil constructions with no restriction on the mine property (see Section 20.2.7for more details).

20.2.5 Environmental Characteristics Of Tailings

Elemental Composition

Table 20-14 shows the elemental composition of flotation tailings. Some of the elements analysed (manganese, cobalt and copper) frequently exceed their respective provincial B-criterion for soil protection, whereas nickel, chromium and zinc occasionally exceed their respective A-criterion. On average though, only manganese and copper in the float tails exceed the provincial B-criteria for soil protection, whereas cobalt exceeds the A-criterion. Mean sulphur concentration (460 mg/kg, or 0.46%) exceeds the A-criterion, but it remains well below the 0.3% criterion used to consider a material has the potential to leach acid[26] (see section 0).

As for the magnetic tailings (Table 20-15), three samples have been analysed. The results show that cobalt exceeds the B-criterion for soil protection, while chromium, copper and zinc exceed the A-criterion.

Leaching Tests

Results from the 1311, 1312 and CTEU-9 leaching tests conducted on flotation tailings samples are shown in Table 20-16. All of the parameters tested on the leachates meet the environmental standards and criteria used[27], except for copper (EPA 1311 and EPA 1312), aluminum (CTEU-9) and phosphorous (CTEU-9), which are found in concentrations above the provincial groundwater quality criterion. Concentration of total phosphorous in the EPA 1311 and EPA 1312 leachates remains below the groundwater quality criterion; for the CTEU-9 leachate however, mean concentration of total phosphorous exceeds (by a factor of four) this criterion.

[26] It should also be noted that sulphides concentrations remain very low.
[27] Including drinking water standards.

As for the magnetic tailings (Table 20-17), copper concentrations in the leachate from the EPA 1311 test exceed the groundwater quality criterion, but concentrations remain below the detection limit when performing the two other tests (EPA 1312 and CTEU-9). Mean zinc concentration in the EPA 1311 leachate is too close to the detection limit to be truly meaningful, and additional samples should be analysed to confirm this tendency. Concentration of total phosphorous (and fluorides) in the three types of leachates always remains below the groundwater quality criteria.

Table 20-14: Elemental Composition of Flotation Tailings Samples

Élément [a]	Unité	Limite de détection	Critères			Résultats									Statistics		
			Québec - Critères de protection des sols [1]			Ore samples from COREM analysed in June 2011				Ore samples from COREM analysed in November 2011					Floatation tailings (n=9)		
			Critère A	Critère B	Critère C	40276-4 (Scav. cleaner tails)	40276-5 (Blend final tail scav)	40276-6 (Prod. conc. mag Lims)	40276-7 (Prod. Final tails)	Final tails (pail1/3)	Final tails (pail 2/3)	Final tails (A)	Final tails (B)	Scavenger tails	Min	Max	Mean
Métaux et métalloides																	
Aluminium	mg/kg	20	-	-	-	4 200	1 800	2 400	1 600	2 900	3 100	3 400	3 700	7 700	1 600	7 700	3 422
Argent	mg/kg	0,8	2	20	40	<0,5	<0,5	<0,5	<0,5	< 0,8	< 0,8	< 0,8	< 0,8	< 0,8	<0,5	< 0,8	< 0,8
Arsenic	mg/kg	5	10	30	50	7	6	<5	<5	< 5	< 5	< 5	< 5	< 5	< 5	7	< 5
Baryum	mg/kg	5	200	500	2 000	99	40	14	34	51	54	61	70	130	14	130	61
Cadmium	mg/kg	0,5	0,9	5	20	0,1	<0,1	<0,1	<0,1	< 0,5	< 0,5	< 0,5	< 0,5	< 0,5	<0,1	0,1	<0,5
Calcium	mg/kg	30	-	-	-	25 000	7 700	2 400	5 700	5 300	5 900	11 000	13 000	40 000	2 400	40 000	12 889
Chrome	mg/kg	2	45	250	800	***91***	22	***69***	14	19	19	22	29	***120***	14	***120***	45
Cobalt	mg/kg	2	15	50	300	***57***	***42***	***32***	***40***	***49***	***51***	***55***	***52***	***75***	***32***	***75***	***50***
Cuivre	mg/kg	2	50	100	500	***320***	***130***	***55***	***110***	***150***	***150***	***150***	***150***	***440***	***55***	***440***	***184***
Étain	mg/kg	4	5	50	300	2	2	2	1	< 4	< 4	< 4	< 4	< 4	1	2	< 4
Fer	mg/kg	10	-	-	-	100 000	100 000	180 000	100 000	110 000	110 000	120 000	110 000	120 000	100 000	180 000	116 667
Manganèse	mg/kg	2	1 000	1 000	2 200	***1 400***	***1 700***	840	***1 700***	***2 000***	***2 100***	***2 300***	***2 100***	***1 900***	840	***2 300***	***1 782***
Mercure	mg/kg	0,02	0,4	2	10	<0,01	<0,01	<0,01	<0,01	< 0,02	< 0,02	< 0,02	< 0,02	< 0,02	<0,01	< 0,02	< 0,02
Molybdène	mg/kg	1	6	10	40	1	<0,5	2	<0,5	< 1	< 1	< 1	< 1	1	<0,5	2	<1
Nickel	mg/kg	1	30	100	500	29	22	23	23	29	***31***	***34***	***32***	***47***	22	***47***	30
Plomb	mg/kg	5	50	500	1 000	8	2	<1	1	< 5	< 5	< 5	< 5	9	<1	9	<5
Sélénium	mg/kg	1	3	3	10	0,9	<0,5	<0,5	<0,5	< 1	< 1	< 1	< 1	1	<0,5	1	<1
Strontium	mg/kg	-	-	-	-	69	23	8	17	-	-	-	-	-	8	69	29
Thorium	mg/kg	-	-	-	-	1,6	0,8	<0,5	<0,5	-	-	-	-	-	<0,5	1,6	0,7
Titane	mg/kg	5	-	-	-	1 100	660	10 000	590	870	960	980	1 000	1 500	590	10 000	1962
Uranium	mg/kg	5	-	-	-	<2	<2	<2	<2	< 5	< 5	< 5	< 5	< 5	< 5	< 5	< 5
Zinc	mg/kg	10	100	500	1 500	80	67	89	66	98	100	***110***	100	***140***	66	***140***	94
Autres																	
Chlorures (Cl)	mg/kg	1	-	-	-	-	-	-	-	5	7	7	8	10	5	10	7
Fluor	mg/kg	-	-	-	-	-	-	-	-	310	310	420	480	1700	310	1700	644
Fluorure	mg/kg	1	200	400	2000	-	-	-	-	3	2	2	2	2	2	3	2
pH	pH	-	-	-	-	-	-	-	-	7,98	7,61	7,97	8,31	8,38	7,61	8,38	-
Phosphore	mg/kg	20	-	-	-	-	-	-	-	2 100	2 400	4 100	5 100	14 000	2 100	14 000	5 540
Soufre (S)	mg/kg	100	400	1000	2000	-	-	-	-	330	340	330	350	***950***	330	***950***	***460***
Sulphures	mg/kg		-	-	-	-	-	-	-	0,56	0,52	0,77	0,54	0,30	0,30	0,77	0,54

[1] Les critères utilisés proviennent de la politique de protection des sols et de réhabilitation des sites contaminés (MDDEP, 2001). Ces critères sont utilisés pour guider la gestion des sites contaminés et des résidus miniers. Le critère A représente les concentrations maximales naturelles rencontrées entre les cinq provinces géologiques du Québec. Dans la gamme A-B, le matériel (sol) peut être utilisé sans restriction sur les terrains commerciaux et industriels, sauf si l'activité se traduit par une augmentation de la contamination des sols. Dans la gamme B-C, le matériel peut être utilisé comme matériel de remblai (seulement) sur les terrains commerciaux et industriels, sauf si l'activité se traduit par une augmentation de la contamination des sols. Au-dessus du critère C, le matériel doit être décontaminé ou éliminé dans site autorisé à recevoir des sols contaminés.

[2] CCME; Tableau sommaire des recommandations canadiennes pour la qualité de l'environnment; http://st-ts.ccme.ca/ (Site consulté le 3 mai 2011)

[a] Digestion selon la méthode M.A. 200

Les valeurs dépassant le critère A sont en gras et en italiques

Les valeurs dépassant le critère B sont en gras et en italiques

Les valeurs dépassant le critère C sont en gras et en italiques

**See Roche-Ausenco feasibility study for high resolution image*

Table 20-15: Elemental Composition of Magnetic Tailings Samples

Element [a]	Unit	Detection limit	Criteria			Results (COREM samples)		
			Québec - Soil protection criterion [1]			Samples analysed in June 2011	Samples analysed in November 2011	
			A-Criterion	B-Criterion	C-Criterion	40276-6 (Prod. conc. mag Lims)	Mag tails (2011/04/11) (A)	Mag tails (2011/04/11) (B)
Metals and metalloids								
Aluminium	mg/kg	20	-	-	-	2 400	3 200	3 200
Silver	mg/kg	0,8	2	20	40	<0,5	< 0,8	< 0,8
Arsenic	mg/kg	5	10	30	50	<5	< 5	< 5
Barium	mg/kg	5	200	500	2 000	14	18	16
Cadmium	mg/kg	0,5	0,9	5	20	<0,1	< 0,5	< 0,5
Calcium	mg/kg	30	-	-	-	2 400	6 900	5 500
Chromium	mg/kg	2	45	250	800	***69***	***110***	***100***
Cobalt	mg/kg	2	15	50	300	***32***	***72***	***68***
Copper	mg/kg	2	50	100	500	***55***	***93***	***89***
Tin	mg/kg	4	5	50	300	2	< 4	< 4
Iron	mg/kg	10	-	-	-	180 000	170 000	170 000
Manganese	mg/kg	2	1 000	1 000	2 200	840	930	960
Mercury	mg/kg	0,02	0,4	2	10	<0,01	< 0,02	< 0,02
Molybdenum	mg/kg	1	6	10	40	2	1	1
Nickel	mg/kg	1	30	100	500	23	23	23
Lead	mg/kg	5	50	500	1 000	<1	< 5	< 5
Selenium	mg/kg	1	3	3	10	<0,5	< 1	< 1
Strontium	mg/kg	-	-	-	-	8	-	-
Thorium	mg/kg	-	-	-	-	<0,5	-	-
Titanium	mg/kg	5	-	-	-	10 000	9 300	10 000
Uranium	mg/kg	5	-	-	-	<2	< 5	< 5
Zinc	mg/kg	10	100	500	1 500	89	**160**	**160**

[1] Criteria are from the *Politique de protection des sols et de réhabilitation des sites contaminés* (MDDEP, 2001). The A criteria represent natural background concentrations encountered within the Grenville geological province (Sept-Îles is located within the limit of the Grenville geological province). Within the A-B range, the material (soil) can be used with no restriction on commercial/industrial lots, except if the activity results in an increase in soil contamination. Within the B-C range, the material can be used as backfill (only) on commercial/industrial lots, except if the activity results in an increase in soil contamination. Above the C criteria, the material should be decontaminated or disposed of in a landfill authorized to receive
[2] CCME; Tableau sommaire des recommandations canadiennes pour la qualité de l'environnment; http://st-ts.ccme.ca/ (Site visited on May 3, 2011)

[a] Digestion according to the M.A. 200 method

Values exceeding the A criterion are in bold and italics

Values exceeding the B criterion are shaded, in bold and italics

**See Roche-Ausenco feasibility study for high resolution image*

SGS

Table 20-16: Leaching Tests Results (EPA 1311, 1312 and CTEU-9) on Flotation Tailings Samples

Paramètre	Unité	Critères: Critère eaux souterraines [1]	Critères: Norme canadienne [4] Concentration moyenne mensuelle maximale permise	Critères: Critère pour les résidus miniers à risques élevés [2]	Critères: Norme québécoise d'eau potable	Résultats – COREM (Samples analysed in June 2011): 40276-4 (Scav. cleaner tails) EPA 1311	EPA 1312	CTEU-9	40276-5 (Blend final tail scav) EPA 1311	EPA 1312	CTEU-9	40276-7 (Prod. Final tails) EPA 1311	EPA 1312	CTEU-9
Métaux et métalloïdes														
Aluminium	mg/l	0,75	-	-	-	0,49	***0,91***	<0,08	0,55	0,5	***1,3***	0,43	0,45	***1,7***
Argent	mg/l	0,00062[aaa]	-	-	-	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01
Arsenic	mg/l	0,34	0,50	5,0	0,025	<0,004	<0,004	<0,004	<0,004	<0,004	0,009	<0,004	<0,004	0,009
Baryum	mg/l	5,3[aaa]	-	100	1	0,32	0,007	0,008	0,19	<0,005	0,014	0,15	<0,005	0,016
Cadmium	mg/l	0,0021[aaa]	-	0,5	0,005	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002
Calcium	mg/l	-	-	-	-	-	-	-	-	-	-	-	-	-
Chrome	mg/l	*	*	5,0	0,05	<0,007	<0,007	<0,007	<0,007	0,007	<0,007	<0,007	<0,007	0,009
Cobalt	mg/l	0,5	-	-	-	0,25	<0,01	<0,01	0,14	<0,01	0,02	0,08	<0,01	0,03
Cuivre	mg/l	0,0073[aaa]	0,30	-	1	***0,11***	***0,12***	***0,03***	***0,05***	***0,06***	***0,27***	***0,06***	***0,04***	***0,29***
Étain	mg/l	-	-	-	-	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05	<0,05
Fer	mg/l	-	-	-	-	<2	4	<2	3	3	7	2	3	11
Manganèse	mg/l	-	-	-	-	3	0,04	<0,01	1,3	0,04	0,12	0,74	0,03	0,16
Mercure	mg/l	0,00013	-	0,1	0,001	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01
Molybdène	mg/l	2	-	-	-	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01
Nickel	mg/l	0,260[aaa]	0,50	-	-	0,16	<0,006	<0,006	0,077	<0,006	0,01	0,048	<0,006	0,015
Plomb	mg/l	0,034[aaa]	0,20	5,0	0,01	<0,01	<0,01	<0,01	<0,01	<0,01	0,02	<0,01	<0,01	0,01
Sélénium	mg/l	0,020	-	1,0	0,01	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005	<0,005
Titane	mg/l	-	-	-	-	-	-	-	-	-	-	-	-	-
Uranium	mg/l	-	-	2	0,2	-	-	-	-	-	-	-	-	-
Zinc	mg/l	0,067[aaa]	0,5	-	-	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2
Autres														
Fluorures	mg/l	4	-	-	1,5	0,2	0,1	0,2	0,3	0,1	0,5	0,2	0,1	0,7
Phosphore total (P)	mg/l	3[bbb]	-	-	-	0,09	2	<0,02	0,31	1,2	***19***	0,34	0,82	***25***

Paramètre	Unité	COREM Float tails - 2011/05/12 (Samples analysed in November 2011): Final tails (pail1/3) EPA 1311	EPA 1312	CTEU-9	Final tails (pail 2/3) EPA 1311	EPA 1312	CTEU-9	Final tails (A) EPA 1311	EPA 1312	CTEU-9	Final tails (B) EPA 1311	EPA 1312	CTEU-9	Scavenger tails EPA 1311	EPA 1312	CTEU-9	Moyennes (n = 8) EPA 1311	EPA 1312	CTEU-9
Métaux et métalloïdes																			
Aluminium	mg/l	0,39	0,36	***1,8***	0,38	0,37	***1,2***	0,34	0,20	***1,4***	0,37	0,20	***1,1***	0,57	0,09	< 0,8	0,44	0,39	***1,2***
Argent	mg/l	< 0,01	< 0,01	< 0,1	< 0,01	< 0,01	< 0,1	< 0,01	< 0,01	< 0,1	< 0,01	< 0,01	< 0,1	< 0,01	< 0,01	< 0,1	< 0,01	< 0,01	< 0,1
Arsenic	mg/l	< 0,004	< 0,004	< 0,04	< 0,004	< 0,004	< 0,04	< 0,004	< 0,004	< 0,04	< 0,004	< 0,004	< 0,04	< 0,004	< 0,004	< 0,04	< 0,004	< 0,004	< 0,04
Baryum	mg/l	0,13	< 0,005	< 0,05	0,14	< 0,005	< 0,05	0,14	< 0,005	< 0,05	0,13	< 0,005	< 0,05	0,30	< 0,005	< 0,05	0,19	< 0,005	< 0,05
Cadmium	mg/l	< 0,002	< 0,002	< 0,02	< 0,002	< 0,002	< 0,02	< 0,002	< 0,002	< 0,02	< 0,002	< 0,002	< 0,02	< 0,002	< 0,002	< 0,02	< 0,002	< 0,002	< 0,02
Calcium	mg/l	13	< 2	< 20	15	2	< 20	18	< 2	< 20	21	< 2	< 20	150	20	140	43	5	36
Chrome	mg/l	< 0,007	< 0,007	< 0,07	0,020	< 0,007	< 0,07	< 0,007	< 0,007	< 0,07	< 0,007	< 0,007	< 0,07	< 0,007	< 0,007	< 0,07	< 0,007	< 0,007	< 0,07
Cobalt	mg/l	0,07	< 0,01	< 0,1	0,09	< 0,01	< 0,1	0,08	< 0,01	< 0,1	0,09	< 0,01	< 0,1	0,23	< 0,01	< 0,1	0,13	< 0,01	< 0,1
Cuivre	mg/l	***0,07***	***0,02***	< 0,2	***0,11***	***0,02***	< 0,2	***0,20***	< 0,02	< 0,2	***0,21***	< 0,02	< 0,2	***0,36***	< 0,02	***0,2***	***0,15***	***0,04***	< 0,2
Étain	mg/l	< 0,05	< 0,05	< 0,5	< 0,05	< 0,05	< 0,5	< 0,05	< 0,05	< 0,5	< 0,05	< 0,05	< 0,5	< 0,05	< 0,05	< 0,5	< 0,05	< 0,05	< 0,5
Fer	mg/l	< 2	< 2	< 20	2	2	< 20	< 2	< 2	< 20	< 2	< 2	< 20	< 2	< 2	< 20	< 2	2	< 20
Manganèse	mg/l	0,83	0,03	0,2	0,93	0,03	0,1	0,61	0,01	0,1	0,67	0,01	< 0,1	2,6	< 0,01	< 0,1	1,3	0,02	0,1
Mercure	mg/l	< 0,01	< 0,01	< 0,1	< 0,01	< 0,01	< 0,1	< 0,01	< 0,01	< 0,1	< 0,01	< 0,01	< 0,1	< 0,01	< 0,01	< 0,1	< 0,01	< 0,01	< 0,1
Molybdène	mg/l	< 0,01	< 0,01	< 0,1	< 0,01	< 0,01	< 0,1	< 0,01	< 0,01	< 0,1	< 0,01	< 0,01	< 0,1	< 0,01	< 0,01	< 0,1	< 0,01	< 0,01	< 0,1
Nickel	mg/l	0,045	< 0,006	< 0,06	0,050	< 0,006	< 0,06	0,040	< 0,006	< 0,06	0,043	< 0,006	< 0,06	0,14	< 0,006	< 0,06	0,08	< 0,006	< 0,06
Plomb	mg/l	< 0,01	< 0,01	< 0,1	< 0,01	< 0,01	< 0,1	< 0,01	< 0,01	< 0,1	< 0,01	< 0,01	< 0,1	< 0,01	< 0,01	< 0,1	< 0,01	< 0,01	< 0,1
Sélénium	mg/l	< 0,005	< 0,005	< 0,05	< 0,005	< 0,005	< 0,05	< 0,005	< 0,005	< 0,05	< 0,005	< 0,005	< 0,05	< 0,005	< 0,005	< 0,05	< 0,005	< 0,005	< 0,05
Titane	mg/l	< 0,05	0,11	< 0,5	< 0,05	0,10	< 0,5	< 0,05	0,06	< 0,5	< 0,05	0,06	< 0,5	< 0,05	< 0,05	< 0,5	< 0,05	< 0,05	< 0,5
Uranium	mg/l	< 0,02	< 0,02	< 0,2	< 0,02	< 0,02	< 0,2	< 0,02	< 0,02	< 0,2	< 0,02	< 0,02	< 0,2	< 0,02	< 0,02	< 0,2	< 0,02	< 0,02	< 0,2
Zinc	mg/l	< 0,2	< 0,2	< 2	< 0,2	< 0,2	< 2	< 0,2	< 0,2	< 2	< 0,2	< 0,2	< 2	0,2	< 0,2	< 2	< 0,2	< 0,2	< 2
Autres																			
Fluorures	mg/l	0,3	0,2	0,9	0,3	0,2	0,7	0,2	0,1	0,5	0,2	0,2	0,6	0,3	0,2	0,2	0,3	0,2	0,54
Phosphore total (P)	mg/l	0,11	0,74	***9,6***	0,10	0,60	***9,6***	0,20	0,56	***17***	0,17	0,59	***23***	0,06	0,40	0,14	0,17	0,86	***13***

Les valeurs excédant un de ces critères sont en gras, italiques et surlignés.

[1] Politique de protection des sols et de la réhabilitation des sites contaminés (MDDEP, 2001). Pour la résurgence des eaux de surface

[2] Directive 019 sur l'industrie minière (MDDEP, 2005).

[3] Directives environnementales, sanitaires et sécuritaires pour les mines (IFC, December 2007)

[4] Règlement sur les effluents des mines de métaux

[5] Conseil canadien des ministres de l'environnement (CCME), 2007. Recommandations canadiennes pour la qualité des eaux: protection de la vie aquatique (http://www.ec.gc.ca/ceqg-rcqe/Water_f.pdf).

[aaa] Ce critère augmente avec la dureté. La valeur indiquée correspond à une dureté de 50 mg $CaCO_3$ / l. Voir « Critères de qualité de l'eau de surface au Québec » (MDDEP, 2009).

[bbb] Le critère de phosphore total vise à la base à limiter la croissance excessive d'algues et de plantes aquatiques dans les cours d'eau. Un critère plus sévère s'appliquerait à l'occasion de la résurgence de l'eau souterraine dans un cours d'eau s'écoulant vers un lac ou à l'occasion de la résurgence de l'eau souterraine dans un lac. Ces situations sont traitées sur une base de cas par cas.

[ccc] Ce critère varie selon les teneurs en chlorures, voir « Critères de qualité de l'eau de surface au Québec » (MENV 2001). La valeur citée dans le tableau correspond à une concentration en chlorures de 2000 µg/L.

**See Roche-Ausenco feasibility study for high resolution image*

Table 20-17: Leaching Tests Results (EPA 1311, 1312 and CTEU-9) on Magnetic Tailings Samples

Paramètre	Unité	Critères: Critère eaux souterraines [1]	Critères: Norme canadienne [4] Concentration moyenne mensuelle maximale permise	Critères: Critère pour les résidus miniers à risques élevés [2]	Critères: Norme québécoise d'eau potable	Résultats: (Analysed in June 2011) 40276-6 (Prod. conc. mag Lims)			Résultats: Samples analysed in November 2011 – Mag tails (2011/04/11) (A)			Samples analysed in November 2011 – Mag tails (2011/04/11) (B)			Average (n = 3)		
						EPA 1311	EPA 1312	CTEU-9	EPA 1311	EPA 1312	CTEU-9	EPA 1311	EPA 1312	CTEU-9	EPA 1311	EPA 1312	CTEU-9
Métaux et métalloïdes																	
Aluminium	mg/l	0,75	-	-	-	0,3	<0,08	0,12	0,43	<0,08	< 0,8	0,44	0,27	< 0,8	0,39	0,27	< 0,8
Argent	mg/l	0,00062 [aaa]	-	-	-	<0,01	<0,01	<0,01	< 0,01	< 0,01	< 0,1	< 0,01	< 0,01	< 0,1	< 0,01	< 0,01	< 0,1
Arsenic	mg/l	0,34	0,50	5,0	0,025	<0,004	<0,004	<0,004	< 0,004	< 0,004	< 0,04	< 0,004	< 0,004	< 0,04	< 0,004	< 0,004	< 0,04
Baryum	mg/l	5,3 [aaa]	-	100	1	0,11	0,01	0,029	0,14	0,010	0,05	0,14	< 0,005	< 0,05	0,13	0,01	< 0,05
Cadmium	mg/l	0,0021 [aaa]	-	0.5	0,005	<0,002	<0,002	<0,002	< 0,002	< 0,002	< 0,02	< 0,002	< 0,002	< 0,02	< 0,002	< 0,002	< 0,02
Calcium	mg/l	-	-	-	-	-	-	-	9	6	29	7	4	< 20	8	5	< 20
Chrome	mg/l	-	-	5,0	0,05	<0,007	<0,007	<0,007	< 0,007	< 0,007	< 0,07	< 0,007	< 0,007	< 0,07	< 0,007	< 0,007	< 0,07
Cobalt	mg/l	0,5	-	-	-	0,07	0,06	0,31	0,19	0,06	0,3	0,19	0,01	< 0,1	0,15	0,04	0,2
Cuivre	mg/l	0,0073 [aaa]	0,30	-	1	***0,19***	***0,03***	***0,08***	***0,06***	< 0,02	***0,2***	***0,09***	< 0,02	< 0,2	***0,11***	< 0,02	< 0,2
Étain	mg/l	-	-	-	-	<0,05	<0,05	<0,05	< 0,05	< 0,05	< 0,5	< 0,05	< 0,05	< 0,5	< 0,05	< 0,05	< 0,5
Fer	mg/l	-	-	-	-	<2	<2	<2	< 2	< 2	< 20	< 2	2	< 20	< 2	< 2	< 20
Manganèse	mg/l	-	-	-	-	0,98	0,97	4,1	0,97	0,54	3,5	0,82	0,11	0,6	0,92	0,54	2,7
Mercure	mg/l	0,00013	-	0,1	0,001	<0,01	<0,01	<0,01	< 0,01	< 0,01	< 0,1	< 0,01	< 0,01	< 0,1	< 0,01	< 0,01	< 0,1
Molybdène	mg/l	2	-	-	-	<0,01	<0,01	<0,01	< 0,01	< 0,01	< 0,1	< 0,01	< 0,01	< 0,1	< 0,01	< 0,01	< 0,1
Nickel	mg/l	0,260 [aaa]	0,50	-	-	0,044	0,034	0,17	0,044	0,014	0,09	0,046	< 0,006	< 0,06	0,045	< 0,006	< 0,06
Plomb	mg/l	0,034 [aaa]	0,20	5,0	0,01	<0,01	<0,01	<0,01	< 0,01	< 0,01	< 0,1	< 0,01	< 0,01	< 0,1	< 0,01	< 0,01	< 0,1
Sélénium	mg/l	0,020	-	1,0	0,01	<0,005	<0,005	<0,005	< 0,005	< 0,005	< 0,05	< 0,005	< 0,005	< 0,05	< 0,005	< 0,005	< 0,05
Titane	mg/l	-	-	-	-	-	-	-	< 0,05	< 0,05	< 0,5	< 0,05	0,09	< 0,5	< 0,05	0,06	< 0,5
Uranium	mg/l	-	-	2	0,2	-	-	-	< 0,02	< 0,02	< 0,2	< 0,02	< 0,02	< 0,2	< 0,02	< 0,02	< 0,2
Zinc	mg/l	0,067 [aaa]	0,5	-	-	<0,2	<0,2	***0,3***	***0,5***	< 0,2	< 2	***0,2***	< 0,2	< 2	***0,3***	< 0,2	< 2
Others																	
Fluorures	mg/l	4	-	-	1,5	0,2	<0,1	0,2	0,1	< 0,1	0,1	0,2	0,1	0,2	0,2	< 0,1	0,2
Phosphore total (P)	mg/l	3 [bbb]	-	-	-	0,56	0,22	0,17	< 0,02	< 0,02	0,03	0,02	0,26	0,02	0,20	0,16	0,07

Les valeurs excédant un de ces critères sont en gras, italiques et surlignés.

[1] Politique de protection des sols et de la réhabilitation des sites contaminés (MDDEP, 2001). Pour la résurgence des eaux de surface

[2] Directive 019 sur l'industrie minière (MDDEP, 2005).

[3] Directives environnementales, sanitaires et sécuritaires pour les mines (IFC, December 2007)

[4] Règlement sur les effluents des mines de métaux

[5] Conseil canadien des ministres de l'environnement (CCME), 2007. Recommandations canadiennes pour la qualité des eaux: protection de la vie aquatique (http://www.ec.gc.ca/ceqg-rcqe/Water_f.pdf).

[aaa] Ce critère augmente avec la dureté. La valeur indiquée correspond à une dureté de 50 mg $CaCO_3$ / l. Voir « Critères de qualité de l'eau de surface au Québec » (MDDEP, 2009).

[bbb] Le critère de phosphore total vise à la base à limiter la croissance excessive d'algues et de plantes aquatiques dans les cours d'eau. Un critère plus sévère s'appliquerait à l'occasion de la résurgence de l'eau souterraine dans un cours d'eau s'écoulant vers un lac ou à l'occasion de la résurgence de l'eau souterraine dans un lac. Ces situations sont traitées sur une base de cas par cas.

[ccc] Ce critère varie selon les teneurs en chlorures, voir « Critères de qualité de l'eau de surface

**See Roche-Ausenco feasibility study for high resolution image*

Acid Generation Potential

Results of the acid generation potential tests conducted on combined tailings samples, which are shown in Table 20-18, indicate that tailings present no potential for acid generation, and the mean total sulphur content (0.07%) remains below 0.2%. As ore was shown to have no potential to generate acid, these results are not surprising.

Table 20-18: Results of the Acid Generation Potential Test on Tailings Samples

Parameter	Unit	Criterion [1]	Results – SGS - Lakefield: PP-16C Comb Tls	Results – SGS - Lakefield: PP-21 Comb Tls	Results – SGS - Lakefield: PP-23 Comb Tls	Statistics: Average (n = 3)
Acid generating potential						
pH	pH units	-	8,79	9,30	8,43	-
Acid neutralization potential (ANP)	kg $CaCO_3$/t	-	30,5	23,1	19,2	24,3
Acid generating potential (AGP)	kg $CaCO_3$/t	-	1,34	0,31	1,57	1,07
Net acid neutralization potential (ANP-AGP)	kg $CaCO_3$/t	> 20	29,2	22,8	***17,6***	23,2
Ratio (ANP/AGP)	unitless	> 3	22,7	74,5	12,3	36,5
Total sulphur	%	< 0,3	0,094	0,025	0,096	0,072

[1] From Directive 019 on the Mining Industry (MDDEP, 2005)

Values exceeding a criterion are in bold and italics.

A material (such as ore, waste and tailings) can be defined as potentially acid generating when:

- it presents a sulphur concentration above 0,3%; and
- it shows a net acid neutralization potential below 20 kg $CaCO_3$/t or an ANP:AGP ratio below 3.

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20.2.6 Environmental Characteristics of Tailings Water

Water samples28 corresponding to the supernatant of the tailings pulp were analysed and results are presented in Table 20-19.

Tailings waters present a pH of 10.4 to 10.7, corresponding to the pH at which the apatite is floated. During the operations, this pH will naturally drop below 9.5 before the final effluent will be discharged to the environment, namely because of acidic rain falling into the accumulation areas and acidic surface waters29.

Total suspended solids (TSS) in the tailings water remained above 15 mg/l, likely because sedimentation time given during the preparation of supernatant samples was too short for the solids contained in the tailings pulp samples to settle down sufficiently.

Aluminum concentrations remained relatively high in the tailings water, usually above the criterion for the protection of ground waters. Most (about 90%) of this aluminum reports to the dissolved phase, since this metal is quite soluble at such basic pH levels (ca. 10.5). The reduction in pH-levels (down to ca. 9.5) prior to the discharge of the final effluent will help reducing these dissolved aluminum concentrations below the criterion.

All the other parameters, including copper and cobalt which both remained below detection limits despite the presence of suspended solids, met the environmental quality criteria.

[28] These samples were obtained during the pilot tests conducted by COREM to optimize ore processing. Analyses on tailings water collected during the pilot tests conducted at SGS Lakefield were not used as the treatment process was quite different from the one selected for the project.

[29] In the unlikely event that pH at the effluent remains above 9, a treatment (such as CO2 bubbling) will be possible within the polishing pond to reduce pH levels.

Table 20-19: Results of the Environmental Characterisation on Tailings Water

Paramètre	Unité	Critères				Résultats			Statistics
						COREM			
		Critère pour les résidus miniers à risques élevés [2]	Concentration maximale permise dans un échantillon instantané [4]	Protection de la vie aquatique [5]	Dir. 019	40276-1	40276-2	40276-3	Average (n = 3)
Caractéristiques physico-chimiques de base									
DBO_5 (Demande biochimique en oxygène)	mg O_2/l	-	-	-	-	45	61	51	52
pH	pH	-	***6,0 - 9,5***	***6,5 - 9,0***	***6,5 - 9,0***	***10,4***	***10,6***	***10,7***	***10,6***
Alcalinité totale	mg $CaCO_3$/l	-	-	-	-	110	130	120	120
Solides totaux en suspension (TSS)	mg/l	-	30	-	***15***	14	***27***	***21***	***21***
Solides totaux dissous	mg/l	-	-	-	-	430	450	430	437
DOC (Demande chimique en oxygène)	mg/l	-	-	-	-	95	98	86	93
Dureté totale	mg $CaCO_3$/l	-	-	-	-	54	68	74	65
Hydrocarbures pétroliers totaux (C_{10}-C_{50})	µg/l	-	-	-	-	<100	<100	<100	<100
Nutriments et ions									
Azote totale Kjeldahl (TKN)	mg/l	-	-	-	-	2	1	<1	1
Chlorures	mg/l	-	-	-	-	63	61	60	61
Sulfates (SO_4^-)	mg/l	-	-	-	-	89	93	90	91
Fluorures	mg/l	150	-	-	-	1,2	1,1	1,1	1,1
Nitrites (NO_2^-) + Nitrates (NO_3^-)	mg N/l	-	-	-	-	<0,2	0,4	<0,2	0,2
Phosphore total (P)	mg P/l	-	-	-	-	0,9	1,6	2,7	1,7
Métaux et métalloïdes									
Aluminium	mg/l	-	-	***0,005 [S]***	-	***0,21***	***0,31***	***0,36***	***0,29***
Arsenic	mg/l	5,0	1,00	0,005	0,2	<0,001	<0,001	0,0011	0,0004
Cadmium	mg/l	0,5	-	0,000017 [V] [W]	-	<0,0003	<0,0003	<0,0003	<0,0003
Calcium	mg/l	-	-	-	-	21	26	32	26
Chrome	mg/l	5,0	-	-	-	<0,005	<0,005	<0,005	<0,005
Cobalt	mg/l	-	-	-	-	<0,005	<0,005	<0,005	<0,005
Cuivre	mg/l	-	0,60	0,002 [Z]	0,3	<0,005	<0,005	<0,005	<0,005
Fer	mg/l	-	-	-	3	0,65	1,1	1,9	1,2
Magnésium	mg/l	-	-	-	-	1,6	1,1	1,5	1,4
Manganèse	mg/l	-	-	-	-	0,013	0,022	0,035	0,023
Mercure	mg/l	0,1	-	0,0001	-	<0,0001	<0,0001	<0,0001	<0,0001
Molybdène	mg/l	-	-	0,073	-	0,0072	0,0083	0,0077	0,0077
Nickel	mg/l	-	1,00	0,025 [CC]	0,5	<0,002	<0,002	<0,002	<0,002
Plomb	mg/l	5,0	0,40	0,001 [DD]	0,2	<0,001	<0,001	<0,001	<0,001
Potassium	mg/l	-	-	-	-	3,4	3,2	3,2	3,3
Silicium	mg/l	-	-	-	-	6,7	7,5	7,6	7,3
Sodium	mg/l	-	-	-	-	120	130	120	123
Zinc	mg/l	-	1,00	-	-	<0,007	<0,007	0,0094	<0,007

Les valeurs excédant un des critères sont ombragées et en caractères gras et italiques.

[1] Politique de protection des sols et de la réhabilitation des sites contaminés (MDDEP, 2001). Pour la résurgence des eaux de surface

[2] Directive sur l'industrie minière (MDDEP, 2005).

[3] Directives environnementales, sanitaires et sécuritaires pour les mines (IFC, December 2007)

[4] Règlement sur les effluents de mines de métaux

[5] Conseil canadien des ministres de l'environnement (CCME), 2007. Recommandations canadiennes pour la qualité des eaux: protection de la vie aquatique (http://www.ec.gc

[a] Ce critère augmente avec la dureté. La valeur indiquée correspond à une dureté de 50 mg $CaCO_3$ / l. Voir « Critères de qualité de l'eau de surface au Québec » (MDDEP, 2009

[S] 0,005 mg/l, quand le pH est < 6,5 , $[Ca^{2+}]$ < 4 mg/l et le COD < 2 mg/l; 0,100 mg/l quand le pH plus grand ou égal à 6,5, $[Ca^{2+}]$ est plus grand ou égal à 4 mg/l, et le COD est plus grand ou égal à 2 mg/l.

[W] Recommandatrion pour le Cadmium (µg/l) = $10^{(0,86 [\log \text{dureté}]-3,2)}$

[Z] Ce critère varie en fonction de la dureté: 0,002 mg/l pour une dureté entre 0-120 mg/l $CaCO_3$; 0,003 mg/l pour une dureté entre 120-180 mg/l $CaCO_3$; 0,004 mg/l pour une dureté >180 mg/l $CaCO_3$

[CC] Ce critère varie en fonction de la dureté: 0,025 mg/l pour une dureté entre 0-60 mg/l $CaCO_3$; 0,065 mg/l pour une dureté entre 60-120 mg/l $CaCO_3$; 0,110 mg/l pour une dureté entre 120-180 mg/l $CaCO_3$; 0,150 mg/l pour une dureté >180 mg/l $CaCO_3$

[DD] Ce critère varie en fonction de la dureté: 0,001 mg/l pour une dureté entre 0-60 mg/l $CaCO_3$; 0,002 mg/l pour une dureté entre 60-120 mg/l $CaCO_3$; 0,004 mg/l pour une dureté entre 120-180 mg/l $CaCO_3$; 0,007 mg/l pour une dureté > 180 mg/l $CaCO_3$

**See Roche-Ausenco feasibility study for high resolution image*

20.2.7 Conclusions from the Environmental Characterization Program

Ore Characteristics and Management

- Mean ore concentrations of metals and metalloids all meet the A-criteria for the protection of soils, except for manganese, cobalt and copper.
- The leaching of copper from ore is observed with the TCLP test only.
- Because of the possibility of copper leaching from the ore, surface drainage from the ore stockpiles should be collected in order to make sure this water transit to the polishing pond[30] for a treatment if need be. This is already taken into account in the project design.
- Ore has no acid generation potential.

Overburden Characteristics and Management

- Mean concentration of elements in the overburden always remains below the provincial soil protection A-criteria. Hence, there is no environmental concern with overburden, and this waste is considered at low risk according to Directive 019.
- All of the parameters tested on the leachates meet the environmental standards and criteria used, except for copper and aluminum, which may sometimes be found in concentrations above the environmental criteria for the protection of ground waters.
- It should be noted however that the results for copper are too close to the analytical detection limit to be totally meaningful. Moreover, as indicated above, the copper content in the solid (in mg/kg) remains below the provincial soil protection A-criterion.
- As said above, aluminum concentrations in surface waters near the mine site are naturally quite high, and it is thus not surprising to find relatively high concentrations in the leachates. One should note that, as aluminum is a major constituent of the earth crust, there is no provincial soil protection criterion for this element.

Valorization of the Overburden

Since the overburden:

- Is not a hazardous material;
- Is not contaminated by organic compounds;
- Has no potential to generate acid drainage;
- Presents mean concentration of elements in the overburden always remaining below the provincial soil protection a-criteria;

[30] The transit of surface drainage from the ore stockpile to the polishing pond may be possible by letting the drainage flow into the pit (and then pump it with mine waters), or by accumulating it into a small basin and pump it directly into the tailings area.

It is classified as a Category I waste, as per the Guide de valorisation des matières résiduelles inorganiques non dangereuses de source industrielle comme matériaux de construction. As such, this material can be used for civil constructions with no restriction, including:

- Road construction and maintenance;
- Dike construction and maintenance;
- To fabricate concrete;
- As ballast31 for the construction of a railroad;
- Construction on industrial and commercial lots (and residential lots, for material showing a grain size over 5 mm).

Since overburden contains boron concentrations above the CCME soil quality guideline32 for the protection of environmental and human health, it is suggested to avoid its use in zones defined as agricultural lands.

Waste Rock Characteristics and Management

Even though some of the elements (manganese, cobalt and copper) exceed on rare occasions the provincial B-criterion for soil protection, mean concentration of elements in waste rock always remain below the provincial soil protection A-criteria, except for cobalt, and none of the elements exceed the federal guidelines for a commercial or an industrial use of the land.

All of the parameters tested on the leachates meet the environmental standards and criteria used, except for copper, which may sometimes be found in concentrations above the environmental criteria for the protection of groundwaters. It should be noted that these results are however too close to the analytical detection limit to be meaningful. Moreover, as indicated above, the copper content in the solid (in mg/kg) remains below the provincial soil protection A-criterion.

Cobalt does not leach, although it is present in the solid at concentrations exceeding the A-criterion for soil quality.

Waste rock shows no acid generation potential.

Hence, there is no environmental concern to use (with no restriction) waste rock for civil construction on the mine property and elsewhere.

[31] Ballast is a free-draining granular material used as a loadbearing material in railway tracks. It is composed of medium to coarse gravel-sized aggregates (10–60 mm), with a small percentage of cobble-sized particles.
[32] For agricultural use.

Valorization of the Waste Rock

Since waste rock:

- Is not a hazardous material;
- Is not contaminated by organic compounds;
- Presents a mean total sulphur content below 0.2% and has no potential to generate acid drainage;

Presents mean concentration of elements always below the provincial soil protection A-criterion (except for cobalt, which remains below the B-criterion);

Presents leachates for the three tests performed (tclp, splp and cteu-9) with concentrations below the Québec standards for drinking water; and

- Presents a grain size distribution greater than 2.5 mm (for at least 90% of the material, by weight);

It is classified as a Category I waste, as per the Guide de valorisation des matières résiduelles inorganiques non dangereuses de source industrielle comme matériau de construction. As such, this material can be used for civil constructions with no restriction on the mine property, including:

- Road construction and maintenance;
- Dike construction and maintenance;
- To fabricate concrete;
- As ballast for the construction of a railroad;
- As an abrasive aggregate for use on roads during winter;
- For sandblasting concrete and steel works;
- For construction on industrial and commercial lots (and residential lots, for material showing a grain size over 5 mm);
- To cover residual materials in a sanitary landfill or a dry material deposit.

Tailings Characteristics and Management

On average, manganese and copper concentrations in the float tails exceed the provincial B-criterion for soil protection, whereas cobalt exceeds the A-criterion.

As for the magnetic tailings, the results show that cobalt exceeds the B-criterion for soil protection, while chromium, copper and zinc exceed the A-criterion.

For float tails, all of the parameters tested on the leachates meet the environmental standards and criteria used[33], except for copper (EPA 1311 and EPA 1312), aluminum (CTEU-9) and phosphorous (CTEU-9), which are found in concentrations above the provincial groundwater quality criterion. Concentration of total phosphorous in the EPA 1311 and EPA 1312 leachates remains below the groundwater quality criterion; for the CTEU-9 leachate however, mean concentration of total phosphorous exceeds (by a factor of four) this criterion.

Copper concentrations in the leachate from the EPA 1311 test on magnetic tailings exceed the groundwater quality criterion, but concentrations remain below the detection limit when performing the two other tests (EPA 1312 and CTEU-9). Concentration of total phosphorous (and fluorides) in the three types of leachates always remains below the groundwater quality criteria.

According to Directive 019, and based on the existing results, the floatation tailings should be considered as leachable (for copper). Hence, level-A protection measures should be put in place at the accumulation areas to prevent the migration of contaminants (copper) into ground waters. As such, the tailings accumulation area should be characterised by a maximum daily soil percolation rate of 3.3 l/m².

Tailings show no acid generation potential.

Management and Valorization of the Tailings

Protection Measures at the Tailings Accumulation Areas

Based on the existing but scarce results, both tailings types should be considered as leachable (for copper)[34]. Hence, level-A protection measures should be put in place at the tailings accumulation areas to prevent the migration of contaminants (copper) into ground waters. As indicated in Directive 019 (in-house translation of the original text follows):

Management of leachable tailings (and waste rock) must be designed to meet a maximum daily percolation rate of 3.3 l/m² for the bottom of the tailings accumulation area. This percolation flow is established from a tailings management model providing a 3-m thick layer of clay with a hydraulic conductivity of 10-6 cm/s and a mean hydraulic load of 10 m in height.

Using a model, the proponent must demonstrate that sealing measures will help avoiding any significant deterioration in groundwater quality, or a change of site or even a new design of tailings (and waste rock) management is necessary.

The modeling study will highlight the fact that the site-specific hydrogeological conditions, the physicochemical nature of the substrate on which or in which tailings will be disposed of, and the mine

[33] Including the Québec drinking water standards.
[34] Large scale in situ tests with piles of tailings (or crushed and grinded ore if tailings material is not available) exposed to natural weathering agents over a long period of time (e.g. 4-6 months) are recommended to infirm/confirm the results obtained in the laboratory and have a better understanding of the dynamics of the possible leaching phenomena and the factors involved.

waste management design, including water management, allow compliance with the criteria set out for the protection of ground water.

Thus, accumulation areas must be managed and operated so as to avoid any significant deterioration in the quality of ground water during and after his operation[35].

Possible Usage of Tailings

Since both flotation and magnetic tailings:

- Are not a hazardous material;
- Are not contaminated by organic compounds;

Present a mean total sulphur content below 0.2% and have no potential to generate acid drainage;

Present mean concentration of elements always below the provincial soil protection A-criterion (except for copper in the floatation tailings, which remains below the c-criterion, and copper and cobalt in the magnetic tailings, which remains below the B-criterion);

- Present leachates for the three tests performed (TC
- LP, SPLP and CTEU-9) with concentrations below the Québec standards for drinking water; and Present a grain size distribution smaller than 2.5 mm;

They are classified as a Category II waste, as per the Guide de valorisation des matières résiduelles inorganiques non dangereuses de source industrielle comme matériau de construction.

Hence, tailings could be used for dike construction and maintenance on the mine property, as long as geotechnical specifications are met.

[35] It should be noted however that a hydrogeological model based on the available but scarce information (about surface deposits characterising the tailings accumulation area) demonstrated that the development and operation of the tailings area should not result in any ground water deterioration since daily percolation rate of natural surface deposits is below 3.3 l/m².

Tailings Water

The pH of the water associated with the floatation tailings is about 10.4 to 10.8, which corresponds to the pH at which the apatite is recuperated. This high pH often allows keeping dissolved metal concentrations very low, as their solubility is generally minimal at high pH levels. This is particularly the case for cadmium, nickel and zinc. As for copper, which tend to leach during the laboratory tests conducted under acidic (TCLP and SPLP) or neutral (CTEU-9) conditions, maximum precipitation of cupric hydroxide occurs at pH 8.1.

Aluminum concentrations are relatively high in the tailings water samples, above the criterion for the protection of ground waters; this aluminum being mostly dissolved. At the discharge of the final effluent, aluminum concentrations should naturally be reduced to acceptable levels since reduced pH-levels (below 9.5) and TSS levels (below 15 mg/l) will characterise the final effluent.

20.3 Major Environmental Impacts and Issues

The main sensitive elements of the environment characterising the project are:

- Residents located along route 138 in the Arnaud Canton;
- The use of the territory (including sport fishing and hunting within the Matimek ZEC (Zone d'exploitation contrôlée), snowmobile and ATV activities) and road transportation on Highway 138;
- Ambient air quality and noise;
- Fish habitats and wetlands;
- The three 735 kV and the 161 kV electrical transmission lines located in the northern part of the mining property;
- The railway used to transport iron ore from Labrador to the Wabush Mines pelletizing plant in Pointe-Noire.

A review of the major environmental impacts, by environmental component, is provided hereafter.

20.3.1 Aquatic Environments and Fish Habitats

The clearing and preparation of the site, the construction of various project components, including the construction of the new railway segment and the dismantling of the former one, are likely to affect surface water quality. The removal of the vegetation cover increases runoff and suspended solids in water since surfaces are more susceptible to erosion.

The main impacts on aquatic habitats are the loss and alteration of fish habitat. A characterisation of the fish habitat was conducted in October 2010 and July 2011 to assess the impacts of the Project and mitigating its effects on fish habitat. The effects of an increase in turbidity are a reduction in food consumption by fish because of the difficulty to locate its preys, passing on the growth rate of the individual and gonads, and a possible reduction in reproduction success. Moreover, increased particles in the water and sedimentation in spawning areas can have a direct effect on the survival of the eggs.

In operation, surface water quality will be modified by the discharge of the final effluent from the polishing basin built downstream from the tailings accumulation area. Mine water from the pit will be pumped to the tailings accumulation area for recirculation to the concentrator or treatment at the polishing pond. Mine effluents will be controlled prior to discharge into the natural environment (into Clet Creek).

Waste rock and tailings have no acid generation potential. Waste rock, overburden and mag tails do not leach metals. Float tails could leach copper, but additional analyses are required (and underway) to confirm this. The main impact relating to the discharge of the final effluent will be linked to the increase in the flow, in suspended solids and possibly copper into Clet Creek. No significant impact should be noted on the streams located downstream from waste rock and overburden accumulation areas.

No significant effect resulting from the operations is foreseen on aquatic wildlife (benthos and fish) who frequent the Bay of Sept-Îles or streams draining into the Bay.

20.3.2 Ground Water

Maintaining the pit dry will lead to a drawdown of the water table around the pit. In addition to a local change around the pit itself, no effect is expected on the groundwater quality. There is no use of ground water downstream from the pit.

The deposition of float tails that would leach copper in the accumulation area could contaminate ground water. However, the nature of surface deposits in the sector where the tailings management facilities will be developed is such that an eventual contaminant migration into ground water will be impeded. Indeed, based on the few boreholes and tests pits done in this sector, it has been determined that the daily percolation rate through the soils in place, being below 3.3 l/m^2, meet the applicable level A sealing measures established in Directive 019.

20.3.3 Vegetation and Wildlife Habitats

The destruction of riparian vegetation in some areas and the deforestation of other sectors will result in a loss of vegetation (forest communities and wetlands). Several equivalent habitats found around the mine will however continue to be available.

Deforestation will result in habitat loss for birds and terrestrial mammals that use the area, for reproduction or else. However, the environment does not present specific features from surrounding areas. In addition, the woodland is relatively homogeneous and biological diversity is relatively low. Affected wildlife should be able to easily move from these impacted sectors to equivalent neighbouring ones.

The development of the Project infrastructure, namely the tailings management facilities, will result in the destruction of parcels of wetlands. A compensation for these losses will be required by the provincial government as a result of the application of section 22 of the Environmental Quality Act and the development of recent management tools, namely the Guide d'élaboration d'un plan de conservation des milieux humides.

20.3.4 Exploitation of Wildlife Resources

The activities to be carried out during the construction phase, namely deforestation and construction of the facilities and infrastructure, are susceptible to have a direct or indirect impact on the exploitation of wildlife resources. These activities, which will lead to a net loss of wildlife habitat and increased noise levels, will cause the dispersion of the game animals to other parts of the territory and will reduce the extent of exploitable areas for hunting big and small game and the trapping of fur animals.

20.3.5 Recreation, Leisure and Resort

Construction work will affect recreational activities, leisure and resort in the study area. These impacts will be particularly felt by users of the Trans-Québec network and members of Ook-Pik snowmobile club, which will see their trail sectioned by the pit development. The relocation of the trail will be necessary to ensure the continuation of the practice of ATVs and snowmobile in the sector.

20.3.6 Transportation and Traffic

During the construction phase, the impact of the Project on the transportation and regional traffic will be associated with the transportation of wood (recovered when clearing the site) to Port-Cartier and construction materials to the site. To limit the importance of this impact (increased heavy traffic on Highway 138) and, above all, to ensure the safety of road users, a number of measures should be taken in collaboration with the Québec Ministry of Transportation[36].

During the operations, the main impacts of the Project will be associated with the transportation of the apatite concentrate between the mine site and the Port of Sept-Îles (Pointe Noire). However, the effects will be minimal since transportation of the concentrate is by rail.

20.3.7 Archaeological Resources

Since no site with an archaeological potential has been identified in sectors affected by mining activities (with the exception of the overburden accumulation areas (#1 and #2) and the low-grade ore stockpile #2), the clearing of the site and the construction of infrastructure risk little impact on archaeological

[36] For example, an acceleration lane and a siding for trucks could be built at the intersection of the access road and Highway 138 to make a left turn without affecting the flow of traffic on the 138.

resources. At these sites, test pits or trial trenches will be required prior to development of these accumulation areas to statute on the presence or absence of archaeological artefacts.

20.3.8 Landscape

In construction, the impact of the Project on the landscape will be associated with deforestation activities and construction of the various project components (accumulations areas, mine roads and buildings). Measures will be required to protect trees, which will be kept on the edge of the site to preserve the natural character of the landscape.

During the operations, visual impacts will be primarily related to the presence of mining facilities and infrastructure in the landscape. To provide maximum integration of mining facilities in the landscape, the following measures will be adopted:

The establishment of waste rock and overburden dumps which summits do not rise beyond the crest of the dominant hills located in the background;

Progressive revegetation of the tailings accumulation areas and the overburden and waste rock piles to prevent contrasts which can be generated with the woodlands surrounding;

The preservation of existing woodlots located between route 138 and the mine site;

The establishment of visual screens (using vegetation) (between route 138 and the mine site) complementary to the existing woodlots, specifically in areas where the existing forest is sparse or inexistent and where permanent observers (residents) are able to see project components;

Appropriate control of lighting (while ensuring the safety of workers) to limit the effect of contrast generated by a light halo that could be seen at night.

20.3.9 Economy and Workforce

The implementation of the Mine Arnaud Project will generate two types of economic benefits in the Sept-Îles region and the province of Québec. The first type corresponds to the economic benefits related to the construction of infrastructure. These benefits will be felt during the period of construction (about two years). The second type is derived from the operation of the mine, and benefits will be recurrent on an annual basis over the Project life duration.

The construction of mining infrastructure will maintain or create many jobs. In terms of creating wealth in Québec, the value of goods and services produced in Québec for the construction, its suppliers and their employees, as well as mine site workers will be very important. The Sept-Îles region should take advantage of a significant part of these benefits.

Moreover, the construction of mining infrastructure will generate tax revenues for the two levels of Government.

The operation of the mining site will result in the creation or maintenance of many permanent direct jobs. The wealth created in the region of Sept-Îles and in Québec, the value of the goods and services produced in Québec to supply the mine, its suppliers and their employees, will be important and will be generated by mining activities and purchases of goods and services by the mine. In addition, the mine will pay a property tax to the city of Sept-Îles.

20.3.10 Noise

Deforestation and the construction of infrastructure will have an impact on the sound environment. The transportation of wood and materials used for the site construction will represent the main sources of noise. However, transportation activities should be limited to the period from 6 AM to 6 PM.

The level of noise from the mine site could result in increased ambient sound level outside the mine property. The noisiest activities are ore extraction (including blasting) and crushing, as well as trucking at the mine site. Modeling of ambient sound levels taking into account these activities (and assuming a number of mitigation measures) indicates that the impacts of the Project on this environmental component are not significant. Otherwise, the necessary measures will be implemented to ensure compliance with eligible sound noise limits.

20.3.11 Air Quality

During construction, transportation (of wood and construction materials) will represent the main vectors of dust in the atmosphere.

The furnace used to dry the apatite concentrate will be equipped with dust collectors so that emissions of particles in the atmosphere will be near zero. Emissions of dust from the dirt roads will be controlled by addition of water (and possibly calcium chloride) on the road surface. Emissions of dust from the tailings accumulation area should be limited to restricted portions of the residues, namely because a significant portion of the area will remain wet and because of the progressive restoration efforts. It should be noted that in summer, the strong winds come from East. Dust emissions would therefore be in West direction where there is neither infrastructure nor residence.

20.3.12 Quality Of Life

During construction, some of the planned activities will affect the quality of life of residents established along Highway 138. These impacts will be associated with the increase of traffic due to heavy traffic incoming and outgoing of the site, increased ambient sound levels due to mine activities (including blasting (noise and vibration)), dust emissions and changes affecting the visual aspects of the landscape. Measures should be taken to limit the potentially negative effects for the residents of the area.

As the concentrate should be delivered by trains, heavy traffic will be limited to deliveries of goods and services required for operations, such as explosives, reagents for the plant and other essential products.

20.4 Legal Environmental Context and Permitting Procedure

The legal framework for construction and operation of the projected facilities is a combination of provincial, national, and municipal policies, regulations and guidelines. The design and the environmental management of the Project must be done in accordance with this legal framework.

20.4.1 Québec Procedure Relating to the Environmental Assessment of the Project

20.4.1.1 Overview

Section 31.1 of the Environment Quality Act (EQA) states that: "No person may undertake any construction, work, activity or operation, or carry out work according to a plan or program, in the cases provided for by regulation of the Government without following the environmental impact assessment and review procedure and obtaining an authorization certificate from the Government."

Moreover, section 2 of the Regulation Respecting Environmental Impact Assessment and Review provides the list of projects subject to the environmental impact assessment and review procedure, namely:

"(n.8) the construction of an ore processing plant for metalliferous ore or asbestos ore, where the processing capacity of the plant is 7,000 metric tons or more per day, or any other ore[37], where the processing capacity of the plant is 500 metric tons or more per day;

(p) the opening and operation of a metals mine[38] or an asbestos mine that has a production capacity of 7,000 metric tons or more per day, or any other mine[39] that has a production capacity of 500 metric tons or more per day."

Thus, the Project is subject to the provincial environmental impact assessment and review procedure.

Section 31.2 of the EQA states that: "Every person wishing to undertake the realization of any of the projects contemplated in section 31.1 must file a written notice with the Minister describing the general nature of his project; the Minister, in turn, shall indicate to the proponent of the project the nature, the scope and the extent of the environmental impact assessment statement that he must prepare."

A project notice was tabled on December 14, 2010 to the MMDEP. A week later, following the study of the project notice, a Directive defining the required scope and contents of the environmental impact assessment of the project was sent by the MDDEP to Mine Arnaud.

[37] Other than a metalliferous, asbestos or uranium ore.
[38] "Mine" means all the surface and underground infrastructures used for the extraction of ore.
[39] Other than a metal, an asbestos or an uranium mine.

Under the Canada-Québec Agreement on Environmental Assessment Cooperation of May 2004, the MDDEP sent a copy of the project notice to the Canadian Environmental Assessment Agency so that it is determined if the Project is also subject to the Canadian Environmental Assessment Act. As the Project is subject to the federal procedure (see below), the Project will be subject to a cooperative environmental assessment, and project notice will be registered in the public registry under the Canadian Environmental Assessment Act.

The MDDEP may have to consult with Aboriginal groups involved in the environmental assessment of the Project. The project notice could then be transmitted to one or more Aboriginal communities to inform them of a potential project and consult to this effect. Mine Arnaud will be advised if its Project is subject to consultation with Aboriginal.

General Contents of an Environmental Impact Assessment Statement

Section 3 of the *Regulation respecting environmental impact assessment and review* defines the contents of an environmental impact assessment statement:

A description of the project mentioning, in particular, the desired objectives, the site ..., the project timetable, any subsequent operation and maintenance activities, the amounts and characteristics of types of borrowed materials required, power sources, methods of management of waste or residue other than road construction residue, transportation activities inherent in the construction and subsequent operation of the project, any connection with land use planning and development plans, urban zoning plans or agricultural zoning and reserved areas within the meaning of the act to preserve agricultural land ...;

A qualitative and quantitative inventory of the aspects of the environment which could be affected by the project, such as fauna, flora, human communities, the cultural, archaeological and historical heritage of the area, agricultural resources and the use made of resources of the area;

A list and evaluation of positive, negative and residual impacts of the project on the environment, including indirect, cumulative, latent and irreversible effects...;

A description of the different options to the project, in particular regarding its location, the means and methods of carrying out and developing the project, and all other variables in the project as well as reasons justifying the option chosen;

A list and description of measures to be taken to prevent, reduce or attenuate the deterioration of the environment, including ... In particular, any equipment used or installed to reduce the emission, deposit, issuance or discharge of contaminants into the environment, any control of operations and monitoring, emergency measures in case of accident, and reclamation of the area affected.

Summary of the Environmental Impact Assessment Statement

Section 4 of the *Regulation respecting environmental impact assessment and review* indicates that an environmental impact assessment statement prepared pursuant to Section 31.1 of the *Environment*

Quality Act must be accompanied by a non-technical summary of the main elements and conclusions of the studies, documents or research. The summary is published separately.

Contents of the Directive

The Directive specific to the Mine Arnaud Project received in December 2010 is identical to the document entitled Directive *for an environmental impact study of a mining project* (drafted in French only) (MDDEP, August 2005).

Analysis of the Environmental Impact Study

It is very important to have a good project definition to produce an impact study taking into account the project to be carried out and who will be considered acceptable by the authorities early in the procedure. Following receipt of the impact study, the MDDEP will conduct an analysis of its admissibility. This analysis includes the consultation of many departments and agencies. Generally, the promoter can expect to receive questions and comments to be addressed before the impact study can be determined eligible. Following the response to this first set of questions, it is possible that a second series of questions comes up. To avoid delays associated with this procedure, it is particularly important that, at the outset, impact analysis respond more specifically as possible to the directive issued by the MDEEP.

Public Consultation

Section 31.3 of the EQA states that: "After receiving the environmental impact assessment statement, the Minister shall make it public and indicate to the proponent of the project to initiate the stage of public information and consultation provided for by regulation of the Government.

Thus, when the impact study is found to be admissible, the Minister asked the Bureau d'audiences publiques sur l'environnement (BAPE) to prepare the case for public consultation. Consultation of the file lasts 45 days (Section 11 of the Regulation respecting environmental impact assessment and review).

Public Hearings

Section 31.3 of the EQA also states that: "Any person, group or a municipality may, within the time prescribed by regulation of the Government, apply to the Minister for the holding of a public hearing in connection with such a project. Unless he considers such application to be frivolous, the Minister shall direct the Bureau to hold a public hearing and report its findings and its analysis thereof to him."

Public hearings are governed by the Règles de procédure relatives au déroulement des audiences publiques (Q-2, r. 45).

Following public hearings, the BAPE commissions table his report to the Minister of the MDDEP. The time frame given to the commission to realize its mandate and table his report is four (4) months.

The Minister then has sixty (60) days to make public the report of the BAPE.

<u>Governmental Decision</u>

From the BAPE report and the environmental analysis report from the MDDEP, the Minister analyses the case and make a recommendation to the Government. As specified in section 31.15 of the EQA[40], the Government makes its decision by Decree: it authorizes the project, with or without changes and conditions he determines, or refuses it. The maximum period between the publication of the BAPE report and the Government's decision is not specified in the EQA or its regulations.

<u>Application for Certificates of Authorization</u>

After the Government Decree and until the construction of the project begins, the promoter shall submit to the Regional Directorate of the MDDEP all the plans and specifications for construction to obtain the required permits.

20.4.2 Federal Procedure Relating to the Environmental Assessment of the Project

As prescribed by section 5 of the *Canadian Environmental Assessment Act*, a federal environmental assessment will be required for the project since federal authorities will be required to issue permits or licences prescribed by regulation. The federal assessment is also required because part of the Project will be located on federal land administered by the Sept-Îles Port Authority and will require the issuance of a lease agreement.

Since the Project is already subject to the Québec environmental assessment and review procedure under the *Regulation respecting environmental impact assessment and review*, there is only a single impact study produced for the two instances, as per the Canada-Québec agreements in this regard.

The Canadian Environmental Assessment Agency who will be responsible for the coordination of federal environmental assessment indicated that a comprehensive study level assessment would be required for the project given that the mine would require a facility for the extraction of at least 200 000 m^3/a of mine water (which is assumed to be groundwater by the Agency).

On November 25 2011, Environment Canada informed Mine Arnaud that the Project was considered as a "metal mine" and hence subject to the *Metal Mining Effluent Regulations.* As such, it is now possible to accumulate tailings into fish habitats provided that the water bodies are listed in Schedule 2 of the Regulations.

[40] Section 31.5: *Where the environmental impact assessment statement is considered satisfactory by the Minister, it is submitted together with the application for authorization to the Government. The latter may issue or refuse a certificate of authorization for the realization of the project with or without amendments, and on such conditions as it may determine. That decision may be made by any committee of ministers of which the Minister is a member and to which the Government has delegated that power.*

20.5 Environmental Design Criteria and Guidelines

Environmental design criteria and guidelines provided in this section come from various sources and are use to govern the environmental management of the Mine Arnaud Project.

20.5.1 Ambient Air Quality and Atmospheric Emissions

Projects with significant sources of air emissions, and potential for significant impacts to ambient air quality, should prevent or minimize impacts by ensuring that emissions do not result in pollutant concentrations that reach or exceed relevant ambient quality guidelines and standards. The standards and environmental criteria identified hereafter should provide information necessary for the selection of equipment needed to control particulate and gaseous emissions from various sources.

Point sources are characterised by the release of air pollutants typically associated with the combustion of fossil fuels, such as nitrogen oxides (NO_x), sulphur dioxide (SO_2), carbon monoxide (CO), and particulate matter (PM), as well as other air pollutants including metals that may be associated with the industrial process. Emissions from point sources should be avoided and controlled according to good international industry practice (GIIP) applicable to the relevant industry sector.

The atmospheric emission standards are applicable to the point source emissions i.e., discrete, stationary, identifiable sources of emissions that may release pollutants to the atmosphere, such as the stack of the treatment plant and transfer points of conveyors handling bulk material. Standards are applied directly to the source (at the stack), whereas ambient air quality guidelines are applicable at the limits of the property (or at the point of impingement (POI)) and aim at protecting the environment as well as the populations.

Workplace Exposure

The requirements of the provincial regulations regarding air quality in the workplace are specified in the Québec *Regulation respecting occupational health and safety* and by the *Regulation respecting occupational health and safety in mines*. Exposure standards appear in Annex I to the first regulation.

Ambient Air Quality Criteria and Standards

Concentrations of contaminants in ambient air, measured at the project property boundary, should not exceed ambient air quality standards that appear in Schedule K of the *Clean Air Regulation*, which introduced standards for 81 new contaminants.

Atmospheric Emissions

Dust Emissions

Section 12 of the *Clean Air Regulation* indicates that particle emissions from the transfer, fall or handling of materials including aggregates, ashes, fertilizers, mine tailings, ore, ore concentrate, ore slag, or iron concentrate pellets must not be visible more than 2 m from the emission point. As well, dust emission from the various sources at the mine site should not result in particulate concentrations above the above-mentioned ambient air quality standards.

To limit dust emissions of dust from these sources, dirt road, accumulation areas and other surfaces may be sprayed with water and/or dust suppressants, increasing the moisture of the ore and tailings storage areas. As well, dust collectors will be installed at material transfer points.

Industrial facilities, activities and processes must not emit or have the effect of emitting particles into the atmosphere in a concentration greater than 30 mg/m^3 of dry gas for each emission point, namely for the preparation, concentration, agglomeration or drying of ore or ore concentrate, and for drilling other than the drilling of a water supply well.

Greenhouse Gases

Greenhouse gases in the atmosphere trap energy from the sun and thus contribute to climate change. Main greenhouse gases are carbon dioxide, methane, nitrous oxide, sulphur hexafluoride, perfluorocarbons and hydrofluorocarbons. There is no standard or guideline for greenhouse gas emissions applicable to any single project, but the adoption of good environmental practices for the reduction of GHG emissions is strongly recommended.

Environmental Standards pertaining to Atmospheric Emissions from Off-Road Compression-Ignition Engines

Environmental standards pertaining to atmospheric emissions from transportation (off-road engines, locomotives and ships) have been adopted by the Canadian federal authorities. Prior to the *Canadian Environmental Protection* Act 1999 (CEPA 1999), there was no federal authority for regulating emissions from off-road engines such as those typically found in construction, mining, and forestry machines. Under the December 2000 *Ozone Annex to the 1991 Canada-United States Air Quality Agreement*, Canada committed to establishing emission regulations (under CEPA) for new off-road engines that aligned with the US federal EPA requirements.

The *Off-road compression-ignition engine emission Regulations* (SOR/2005-32) introduced emission standards for diesel engines used in off-road applications such as those typically found in construction, mining, and forestry machines. As the title suggests, *the Off-road compression-ignition engine emission Regulations* apply only to what are commonly referred to as compression ignition or diesel engines. Specifically, the regulation applies to "*reciprocating, internal combustion engines, other than those that operate under characteristics significantly similar to the theoretical Otto combustion cycle and that use a spark plug or other sparking device*". The regulations specifically exempt engines:

Regulated by the on-road vehicle and engine emission regulations;

Designed to be used exclusively in underground mines;

Designed to be used in a vessel and for which the fuel, cooling and exhaust systems are integral parts of the vessel.

20.5.2 Environmental Standards Pertaining To Ambient Noise Levels and Vibrations

Noise standards aim at controlling the ambient noise levels and emissions from industrial facilities. This section presents a review of the environmental standards adopted by the provincial authorities pertaining to ambient noise levels. There are no federal criteria for ambient noise since this aspect of the environment is regulated by provincial and municipal authorities. The reader should note that this section does not take into account Hydro-Québec specific requirements regarding noise and vibrations near transmission lines.

The current approach to developing criteria for noise impact assessments of development projects in most jurisdictions around the world, involves using the current background sound levels and the type or intensity of land-use, as the basis of setting a maximum sound level which can be contributed by the project. The intention is, generally, that the new project will not either cause an increase in existing background sound levels by more than some acceptable amount, or sound levels to exceed a sound level considered acceptable for the type of area where residential receivers are located. The facility must be built, equipped and operated so that its operation will not generate noises or vibrations likely to affect the health, the peace, or the persons living in the surrounding areas. Acoustic baffles should be installed if required. These constraints are designed to preserve the existing and future noise amenity of the area.

The provincial requirements regarding noise and vibrations are indicated in Directive 019. Although Directive 019 has no force of law, the Mine Arnaud Project should meet the spirit and the letter of this Directive to obtain governmental approval.

Continuous Noise

From Directive 019, one can deduct that at any time and place along the northern limit of the private lots (located on the southern portion of the mining property, along Road 138), maximum noise level should remain below the following maximum allowable noise levels : 40 dBA at night (from 19 h to 7 h) and 45 dBA during daytime (from 7 h to 19 h).

Annex IV of Directive 019 defines in detail the method to be used for noise level determination.

Vibrations and Noise during Blasting

Directive 019 indicates the rules a mine operator should follow (during the development of the deposit as well as commercial operations) about vibrations and noise during blasting. It indicates that the operator should realize an auto-surveillance monitoring and keep in a register (for at least the last two years of monitoring) all the data relating to this monitoring of the blasting operations (including vibration velocities, ground vibration frequencies, airblast overpressures and blasting pattern).

Assuming there is an impact point[41,42] within 1 km from the mine site43, the operator must install a surveillance network of ground vibrations and airblast overpressures near the housings or artesian wells (generally between one and three stations installed at the closest housings to the mine).

For an open pit:

- The maximum ground vibration velocity (due to blasting operations) at the impact point is 12.7 mm/s;
- The maximum threshold of airblast overpressure at any housing is 128 (linear) decibels;
- Since there are housings within 1 km from the mine, it is forbidden to blast during night time (from 7PM to 7AM).

20.5.3 Drinking Water

Drinking water standards aim at protecting the human health and preserve the aesthetic quality of the water (odour, taste, etc.). In most cases, surface waters are usually treated while groundwater may be used without preliminary treatment.

Drinking water sources, whether public or private, should at all times be protected so that they meet or exceed applicable provincial acceptability standards.

Provincial Regulatory Requirements

Regulatory requirements for drinking water are presented in the Regulation respecting the quality of drinking water (Q-2, r.40). Water quality standards and sampling frequencies are indicated. The proposed amendment of the regulation, published in the Gazette officielle du Québec on November 24, 2010, is also regarded as having the force of regulations.

Federal Guidelines

Federal guidelines are presented in the Guidelines for Canadian Drinking Water Quality of Health Canada and Canadian Environmental Quality Guidelines (organoleptic or aesthetic criteria) of the Canadian Council of Ministers of the Environment (CCME).

[41] An impact point is defined as any housing equipped with a groundwater supply system and a private sewage disposal system, artesian well, campground, or public institution.
[42] With the exception of a housing or an artesian well owned by (or rented to) the mine operator or owner, or dwellings at a mine camp.
[43] A mine site is defined as: a lot used or that have been used for exploration works or the development of a mineral deposit, mining operations or ore treatment. It includes the mines, surface infrastructure, accumulation areas (of ore, enriched ore, concentrate, waste, tailings, and overburden) and treatment basins as well as cleared or disturbed areas, including ditches that are adjacent to the land and works.

Environmental Design Criteria

Environmental design criteria set from the standards and criteria contained in the above-mentioned provincial regulations and federal guideline documents are presented in Table 20-20. Values that correspond to good environmental practices are also proposed for some parameters.

Table 20-20: Environmental Design Criteria Relevant to Drinking Water

Parameter	Maximum Value	Source
Antimony (mg/l)	0.006	RQEP*
Arsenic (mg/l)	0.010	PRQEP*
Barium (mg/l)	1	RQEP*
Boron (mg/l)	5	RQEP*
Bromates (mg/l)	0.010	RQEP*
Cadmium (mg/l)	0.005	RQEP*
Chloramines (mg/l)	3	RQEP*
Chlorides (mg/l)	250	Health Canada
Chromium (total) (mg/l)	0.05	RQEP*
Conductivity (µS/cm)	< 1 500	Good environmental practices**
Copper (mg/l)	1.0	RQEP*
Cyanides (CN-) (mg/l)	0.2	RQEP*
Fluorides (total) (mg/l)	1.5	RQEP*
Mercury (mg/l)	0.001	RQEP*
Lead (mg/l)	0.01	RQEP*
Selenium (mg/l)	0.01	RQEP*
Sodium (mg/l)	200	Health Canada
Sulfides (H2S) (mg/l)	500	Health Canada
Uranium (mg/l)	0.02	RQEP*
Zinc (mg/l)	5.0	Health Canada
Atypical colonies (UFC/100ml)	200	RQEP*
Total hardness (mg CaCO3/l)	< 180-200	Good environmental practices**
Iron (total) (mg/l)	0.3	CCME*
Manganese (total) (mg/l)	0.05	CCME*
Nitrates et nitrites (mg N/l)	10	RQEP*

Parameter	Maximum Value	Source
Nitrites (mg N/l)	1	RQEP*
pH	6.5 - 8.5	RQEP*
Suspended solids (mg/l)	< 500	Good environmental practices**
Temperature (°C)	< 15°C	Good environmental practices**
Sulfates (mg/l)	0.05	Good environmental practices**
Enterococci	0	RQEP*
Coliphage virus	0	RQEP*
Faecal coliforms (UFC/100 ml)	0	RQEP*
E. coli	0	PRQEP*
Total coliforms (UFC/100 ml)	10	RQEP*
Turbidity (UTN)	Between 0.2 and 5.0 UTN (according to the treatment technology used)	PRQEP*
* RQEP: Regulation respecting the quality of drinking water PRQEP: Bill modifying the Regulation respecting the quality of drinking water CCME: Canadian Council of Ministers of the Environment		
** These limit values represent good environmental practices		

20.5.4 Mining Effluents

As per the Metal Mining Effluent Regulations, an effluent is any effluent[44] that contains a deleterious substance. Effluent standards specify the permissible concentrations of substances in effluents discharged to a lake, a river, or the marine environment.

Regulatory Requirements

In Québec, surface drainage from mine site is regulated by the *Metal Mining Effluent Regulations* and the Québec Directive 019. The *Metal Mining Effluent Regulations* will apply to the Mine Arnaud Project should it is considered a metal mine project. The applicable standards included in Directive 019 are the same as the ones included in the federal regulations.

[44] Namely an effluent from a hydrometallurgical facility (and any other treatment facility other than a sewage treatment facility), a milling facility, mine water, tailings impoundment area, treatment ponds, seepage, and surface drainage.

Environmental Design Criteria

The environmental design criteria applicable to mine effluents, presented in

Table 20-21, are extracted from Directive 019.

In addition, it is noted that environmental discharge objectives (OER) applicable to mine effluents will be established by the MDDEP, when the characteristics of effluents (flow and quality) and the receiving environment (bathymetry, hydrodynamic conditions, capacity of dilution, concentration in the environment) will be known. The OER are designed to meet the water quality criteria set by the MDDEP for the protection of aquatic life. The different potential uses of the environment or the resources to protect are fishing, and the protection of aquatic fauna and flora.

Table 20-21: Quality Standards Applicable to Mine Effluents

Parameter	Monthly mean concentration (mg/l)	Maximum concentration – Grab sample (mg/l)
pH	-	Between pH 6.0 and 9.5
Arsenic	0.2	0.4
Copper	0.3	0.6
Iron	3.0	6.0
Nickel	0.5	1.0
Lead	0.2	0.4
Zinc	0.5	1.0
Hydrocarbons (C10-C50)	-	2.0
Suspended solids	15	30
Acute toxicity (rainbow trout and Daphnia)	-	Absence of toxicity

From Directive 019 sur l’industrie minière (MDDEP, avril 2005)

20.5.5 Hazardous Materials

Provincial regulatory requirements from the management of hazardous materials are contained in the *Regulation respecting hazardous materials*. The main provisions of this regulation are summarized as follows:

- Prohibition to dispose of hazardous materials in the environment (Section 8);
- Measures to be taken in the event of accidental spills (Section 9);
- Measures to be taken in the event of termination of activities or dismantling of buildings (Section 13);
- Prohibition to use oil as a dust suppressant (Section 14).

20.5.6 Environmental Quality Criteria

Quality guidelines exist for surface water, sediment, soils and ground water. Results obtained as part of the monitoring of the receiving environment (to be done during the duration of the Project) should be compared to these criteria to assess the quality of the environment and to note any potential degradation. These guidelines are derived from recommendations of the CCME (2011) and the MDDEP (2009).

Surface water quality criteria are intended to protect aquatic life and to prevent any contamination that may affect current and future human consumption of aquatic resources as well as recreational activities such as fishing, water activities and swimming.

Sediment criteria aim at supporting and maintaining aquatic life associated with bottom sediments in both marine and fresh water environments. Other criteria are available for dredged material management.

Soil quality criteria provide protection to human health and ecological receptors. Soil criteria are used for the protection, maintenance, and improvement of specific uses of land and water. They are usually used as benchmarks to evaluate the need for site restoration with respect to specified land uses (e.g. residential, commercial or industrial). These guidelines are used to assess the general degree of contamination at a site, and to determine the need for follow up actions.

The groundwater quality guidelines correspond to the criteria contained in the *Politique de protection des sols et de réhabilitation des terrains contaminés* (Policy for the protection of soils and the rehabilitation of the contaminated lands) of the MDDEP. Groundwater quality guidelines concern water consumption and resurgence in surface water.

20.5.7 Stream Crossings

Regulatory Requirements and Good Environmental Practices

At the Provincial Level

- The Regulation respecting standards of forest management for forests in the domain of the State (R.S.Q., c. F-4.1) has approximately 150 guidelines, for which more than half concern the protection of the aquatic environments. Intervention guidelines focus particularly on:
- The protection of shores, water bodies and water courses;
- The protection of water quality;
- The selection of routes and construction of roads.

The Guide pour l'aménagement des ponts et des ponceaux dans le milieu forestier (Guide for the installation of bridges and culverts in the forest environment) discusses the factors to be taken into account to choose the type of work to build, the steps to follow for a project of this nature, materials to use, and solutions to a series of problems that may occur.

At the Federal Level

DFO recommends the criteria and measures presented in the document entitled *Bonnes pratiques pour la conception et l'installation de ponceaux de moins de 25 mètres* (Good practices for the design and installation of culverts less than 25 meters). DFO considers that meeting the design criteria and measures presented in this document allows, among other things, to ensure the free passage of fish.

Environmental Design Criteria

With respect to stream crossings, the project must comply with the regulatory requirements and best environmental practices indicated in the Section 0.

20.5.8 Deforestation Works

For an amendment to the Migratory Birds Regulations, the Canadian Wildlife Service began a study about terrestrial birds nesting in forest environments and the level of risk to destroy nests within various regions of Québec. Québec has been divided into 4 geographical areas, and in zone 3[45], restriction periods for deforestation are from May 1 to August 15.

[45] Essentially the northern portions (or stands) in Zone 2 areas and portions of fir forests of the Abitibi-Témiscamingue, Saguenay-Lac-Saint-Jean, Côte-Nord, Bas-Saint-Laurent and Gaspésie regions.

20.6 Environmental Principles and Mitigation Measures

Environmental health and safety (EHS) issues associated with mining activities may occur during the various phases of a project, from exploration to post-closure. These are presented below, and recommendations for the management of these issues are also provided. Whenever applicable, these principles should apply to the project.

20.6.1 Mining Operations

The following environmental principles apply to mining operations:

Maximize in-pit waste rock piling to minimize footprint of waste rock piles and reduce OPEX and fuel burning relating to waste rock haulage;

Maintain a non-extractable vegetated buffer zone of not less than 150 m around the mining area and 100 m along the roadway;

Store topsoil and overburden separately for use as backfill material during progressive reclamation and replace in their natural sequence;

Do not operate any mine within 100 m of a major creek (greater than 10 m wide) unless adequate protection of that water source can be demonstrated;

Avoid pushing the topsoil and overburden close to trees to ensure easy retrieval and no unnecessary destruction of flora;

Topsoil will be reused for restoration purposes by spreading it over areas to be re-vegetated.

20.6.2 Waste Management

The following environmental principles apply to waste management:

Store solid waste at a central location and dispose of appropriately;

Leave the mine sites in a state that is compatible with future land use.

20.6.3 Fuel And Hazardous Material Management

The following environmental principles apply to fuel and hazardous material management:

All fuel or lubricant storage facility must have a containment bund or device of adequate capacity. It must have an impervious surface for the handling of all oils, lubricants and fuels. The containment bund surrounding a fuel depot must represent a volume of at least 110% of the volume of the largest tank in the depot in order to contain any eventual spill;

Waste oil must be collected and stored on site in appropriate containers until ready for disposal and/or reuse;

All the necessary precautionary measures to avoid losses to the environment are taken during the transport of fuel to site, transfer, use, and handling onsite;

A program to ensure regular maintenance of machinery and equipment is implemented to prevent and minimize leakages and emissions;

Regular training sessions are given to employees to make sure they familiar with the regulations and safety measures respecting hazardous materials.

20.6.4 Protection of Surface Waters (Streams, Rivers and Lakes)

Accumulation areas for waste rock and tailings should be located at a minimal distance of 60 meters from the high water line of any water body or stream (30 m when it is not possible to meet the 60 m distance and if a sound justification is provided).

For mine roads, a protection of 60 m from a lake or a steady flow stream (30 meters from a stream showing intermittent flow) should be maintained. This distance is measured from the natural high water mark to the road ditch (on the side of the stream). Of course, roads must occasionally cross streams, this is inevitable, but this should be done perpendicular (to the extent possible) to it. As well, roads should avoid running along shores (within a distance of 60 m) over a length of more than 300 m.

20.6.5 Tailings Management

Opportunities to prevent or reduce wastewater pollution through measures such as recycle/reuse within the facility, or process modification (e.g. change of technology or operating conditions/modes) should be identified. Recommended practices for water management include:

Establishing a water balance (including probable climatic events) for the mine and related process plant circuit and use this to inform infrastructure design;

Minimizing the amount of make-up water;

Consider reuse, recycling, and treatment of process water where feasible (e.g. return of supernatant from tailings pond to process plant);

Consider the potential impact to the water balance prior to commencing any dewatering activities;

Consultation with key stakeholders (e.g. government, civil society, and potentially affected communities) to understand any conflicting water use demands and the communities' dependency on water resources and/or conservation requirements that may exist in the area.

20.7 Environmental Management Plan

The environmental management plan (EMP) to be developed should establish a commitment from the developer to undertake certain tasks and bear some specific responsibilities regarding environmental management of the project. Realistic measures that can be incorporated into the environmental permit should be sought.

The environmental monitoring program for the mine will serve as an integral part of the operation and should be implemented to generate information for environmental management and reporting. The monitoring plans encompass the monitoring of the physical, biological and socioeconomic changes in the environment and the surveillance of project activities, including the mitigation measures. The operational monitoring will include those variables important for day-to-day operations such as processing methods and camp facilities.

Monitoring will be conducted during all phases of the project; design, construction, operation and closure. The monitoring program will play a key role in ensuring that the trends for specific parameters are tracked and will provide information on compliance with legislative norms, set guidelines or desirable operational limits and will provide a basis for corrective actions and modification of activities if necessary. Sampling intensity during monitoring will depend on the time and location of the development activities and results derived from monitoring data.

20.8 Mining Site Rehabilitation Plan

The Mining Act (L.R.Q., C. M-13.1) is another important piece of provincial legislation concerning the management of mining activities in the Province of Québec. "The object of this Act is to promote prospecting, mineral exploration, and development and operation of underground reservoirs, taking into account other possible uses of the land in the territory" (s.17).

Section 232.1 of the Act states that:

"Land rehabilitation and restoration work must be carried out, in accordance with the plan approved by the Minister. The obligation shall subsist until the work is completed or until a certificate is issued by the Minister under Section 232.10."

The land rehabilitation and restoration work to be conducted must be planned and approved by the MRNF (Department of Natural Resources and Wildlife). Indeed, according to Section 232.2 of the Act: "Every person to whom Section 232.1 applies must submit a rehabilitation and restoration plan to the Minister for approval before commencing mining activities."

Hence, as part of the project, a rehabilitation plan will have to be prepared (and approved by the MRNF). The rehabilitation and restoration plan should be elaborated in accordance with the provincial Guidelines for Preparing a Mining Site Rehabilitation Plan and General Mining Site Rehabilitation Requirements (MRNF and MDDEP, 1997) which provides the proponents with the rehabilitation requirements. The financial feasibility of the project will have to take into account the costs of all the work needed for the rehabilitation of the mining site.

20.8.1 General Principles

The aim of mining site rehabilitation is to restore the site to a satisfactory condition by:

- Eliminating unacceptable health hazards and ensuring public safety;
- Limiting the production and circulation of substances that could damage the receiving environment and, in the long-term, trying to eliminate maintenance and monitoring;
- Restoring the site to a condition in which it is visually acceptable to the community;
- Reclaiming the areas where infrastructures are located (excluding the accumulation areas) for future use.
- Specific objectives are to:
- Restore degraded environmental resources and uses of the land;
- Protect important ecosystems and habitats of rare and endangered flora and fauna, which favours the reestablishment of the biodiversity;
- Prevent or minimise future environmental damage;
- Enhance the quality of specific environmental resources;
- Improve the capacity of eligible organizations to protect, restore and enhance the environment; and
- Undertake resource recovery and waste avoidance projects and to prevent and/or reduce pollution.
- The general guidelines of a rehabilitation plan include:
- Favour progressive restoration to allow a rapid reestablishment of the biodiversity;
- Monitoring and surveillance program;
- Maximisation of the recovery of previous land uses;
- Research new vocations for land uses;
- Habitat rehabilitation using operational environmental criteria;
- Ensure sustainability of the results of the restoration efforts.

The mining site rehabilitation plan focuses on land reclamation, reclamation of tailings area and water basins, and surface drainage patterns to prevent erosion. At the end of the activities, the mining site rehabilitation makes sure the project result in a minimum of disturbance. Site inspections before relinquishing the property to the Government will be conducted.

20.8.2 Conceptual Mining Site Rehabilitation Plan

Progressive Rehabilitation and Restoration

Progressive restoration is always favoured in order to rapidly reach the objectives of the rehabilitation program and help in an early habitat reestablishment to increase biodiversity.

Progressive restoration will be possible at the tailings management facilities since tailings are accumulated in distinct cells forming the whole tailings accumulation area. When the storage capacity of a cell is attained, a complete restoration of the cell may be achieved.

Progressive restoration is also possible for the waste rock piles (or sectors in it), when at their maximum capacity.

20.8.2.1 Final Rehabilitation and Restoration

The conceptual final rehabilitation and restoration plan can be summarized as follow:

Tailings Accumulation Cells and Waste Rock Piles

Exposed surfaces of the accumulation areas (tailings accumulation cells, waste rock and overburden piles) will be covered with a layer of top soil/overburden and revegetated.

Tailings dams and the accumulation areas will be covered with a layer of top soil/overburden and seeded so that vegetation will cover and stabilise the surfaces to prevent erosion.

The threshold of spillway of these dams will be lowered as much as possible to minimise the possibility of accumulation of water in the various cells of the tailings pond.

Waste rock accumulation areas will be contoured to flatten them out and to further stabilise them against erosion in a way to promote revegetation, that is, with berms and a gentler slope than the natural slope on a rock pile, in compliance with MRNF safety criteria. No rock pile stability work will therefore be required. The restoration will involve covering the waste rock pile with a layer of about 40 cm of overburden as a substratum for revegetation and seeding plant species and self-sustaining nutrients.

Water in the polishing pond will be pumped out and the dam will be breached to allow free-flowing of surface runoff. The area of the polishing pond will be covered with topsoil, and seeded to help vegetation growth which will consolidate the material that has accumulated into the pond.

Industrial wastewater treatment unit will be dismantled.

Access and Haul Roads

- Access road to the site will be left intact and retrocede to a responsible entity (e.g., the municipality of Sept-Îles, or ZEC Matimek) to provide a main access to the ZEC.
- On-site haul roads and other mine roads will be scarified and revegetated.

Industrial Complex and Buildings

No building will be left in place. Whenever possible, buildings will be sold with the equipment they contain, completely or partially. During dismantling works, beneficiation/recycling of construction material will be maximized. Remaining waste will be disposed of in a landfill. This landfill will be covered and revegetated.

- All equipment and machinery will be sent out of the site for sale or recycling.
- Explosives magazine and related facilities will be dismantled.

The facilities for drinking water supply and domestic wastewater treatment may be transferred to a competent administrative authority or will be dismantled.

- Infrastructure relating to electricity supply and distribution will be dismantled if of no use for other parties.

Open Pit

The open pit will no longer be dewatered, and the pit will eventually fills up with groundwater and precipitation (rain and snow falls) to become a water body (a small lake).

- Security bunds will be constructed around the pit to prevent easy access.

Environmental Aspects

- Drainage
  - Whenever possible, surface water drainage pattern will be re-established to conditions similar to the original hydrological system.
- Topsoil Management
  - During site construction and orebody stripping, overburden and topsoil will be salvaged separately and used for revegetation purpose. Slopes of the overburden storage area and flat surfaces will be seeded and revegetated.
- Waste Management
  - Demolition waste will be:
    - Decontaminated when required;
    - Recycled when cost-effective;

        - Disposed of or burned on site;
        - Buried in an appropriate site.
    - All non contaminated waste will be sent to the landfill.
- Hazardous Materials
- Facilities containing petroleum products, chemicals, solid waste, hazardous waste, and/or contaminated soil or materials will be dismantled and managed according to regulatory requirements.
- All hazardous waste will be managed according to existing laws and regulations and will be sent out of the site.

Final restoration of the mine site and port facilities will be completed within three years following the end of commercial production.

20.8.3 Monitoring Program and Post-Closure Monitoring

Physical Stability

- The physical stability of the tailings dams and the waste rock piles will be assessed, and signs of erosion will be noted. This surveillance will be conducted on an annual basis for three years following mine closure.

Environmental Monitoring

- Geotechnical and hydrogeological monitoring of tailings area waste and overburden piles will be carried out for a three year period.
- Monitoring of water quality (surface and groundwater) will continue for three years after the restoration work.

Agronomic Monitoring

The agronomic monitoring program aims to assess the effectiveness of revegetation that is done as part of the mining rehabilitation efforts.

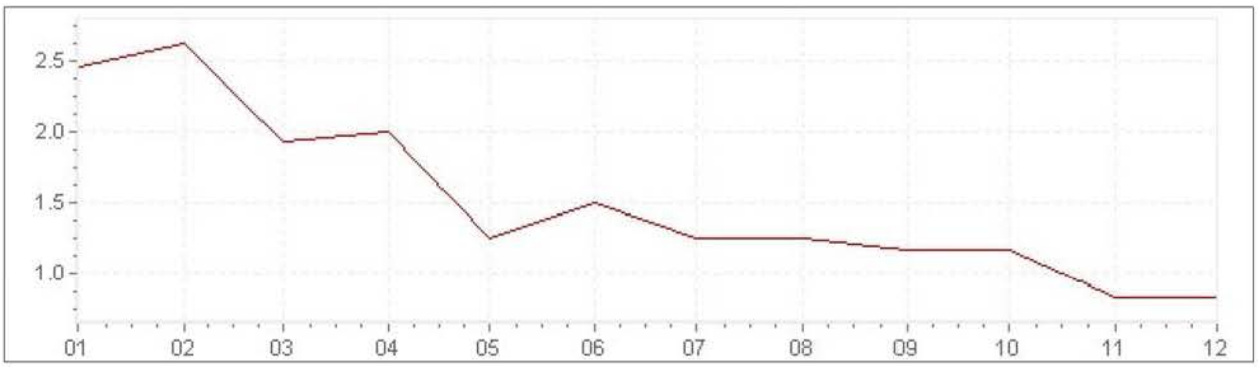
To document the success of the revegetation efforts over the accumulation areas, agricultural monitoring will be undertaken following the establishment of a vegetative cover on the areas subject to the progressive restoration program. Monitoring will be conducted annually for three years following the implementation vegetation.

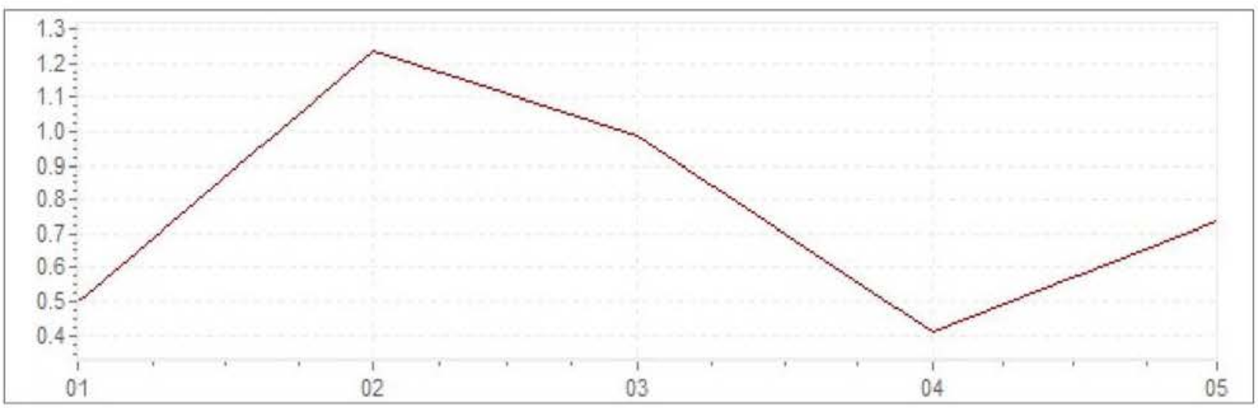
Once the mine site closed, the restoration plan will be implemented and the vast majority of the site will be revegetated. Revegetation success will be monitored for three years. If require, reseeding will be carried out on spots where revegetation is not deemed satisfying.

21. Capital and Operating Costs

21.1 Capital Expenditure (CAPEX)

21.1.1 Introduction





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21.1.2 Summary

	Amount ($)

21.1.3 Accuracy of the Estimate

21.1.4 Direct Costs

21.1.4.1 Mining

Freight

Resale Value

21.1.4.2 Process Facilities

Construction Labour Rates

Table 21-2: All-in labour rates

Note 1: Per Quebec Construction Collective Agreement as of May 1st, 2011

Work Week

In the Province of Québec, a standard normal working schedule for construction is based on five (5) days of one (1) shift of eight (8) hours per day, which represent forty (40) working hours per week at regular time. This work week schedule reflects the basic conditions of the Québec Construction Collective Agreement.

However, Roche-Ausenco determined that most of the major projects in the Sept-Îles area are presently offering fifty-eight (58) working hours per week schedule in order to attract workers. The fifty-eight (58) hour schedule corresponds to five (5) days at ten (10) hrs (Monday to Friday), plus eight (8) hours on Saturday. Sunday is a day off.

For that reason, three different labour rates are provided in the above previous Table.

- Construction base rates as of May 1st, 2011 as per the Québec Construction Collective Agreement;
- All-In Rates as described in Yara's description for All-In Labour Rates, based on forty (40) hours per week;
- As an alternative and in order to secure workers availability for the project, All-In Labour Rates based on fifty-eight (58) hours per week is also presented.

With this breakdown, Mine Arnaud can better understand the potential impacts on CAPEX of workers' availability. After discussions with Mine Arnaud, it was agreed to use in the estimate the All-In Labour Rates based on fifty-eight (58) hours per week.

At the time of the FS preparation, the work demand was probably higher that it is now following a slowdown in the overall mining industry, by selecting to retain the fifty-eight (58) hours per week option for the construction period; Mine Arnaud is on the very safe side of the Cost Estimate. The working schedule will have to be negotiated with the union representing the majority of the workers on site at the time of construction and could result in a different work week, offering reduced labour rates, and cost savings for the project owners.

Man-Hours and Productivity

No productivity factor other than the increased rate between the 40-hour and 58-hour week, has been applied separately to take into account the local conditions for construction work at Sept-Îles. The impact on changing from a 40-hour week to a 58-hour week is an average increase of 26 per cent in the labour rate. There should also be a drop in productivity. The literature on the effect of scheduled overtime on labour productivity refers to a possible drop of 25-30 per cent in productivity for the 58-hour schedule.

No separate productivity factor was used for winter work outside. For this estimate, it is assumed that minimal work will take place in the worst winter months. Winter work usually results in a lower productivity in the range of 75%.

For this Greenfield project, no allocation has been included to take into account the lost of productivity on working 2 shifts of 10 hours at regular rates. This usually corresponds to 16.2 % of the man-hours.

Contractor Expenses at Site

The all-In labour rates are inclusive of a significant portion of the contractor expenses at site, but not all of them. We are identifying hereafter, the inclusions and exclusions of the all-in labour rates of the Contractor Expenses at site.

The following items are included within the All-In Labour Rates:

- Mobilisation and demobilisation of contractor's personnel;
- Room and Board, living allowances and transportation cost for all personnel;
- Transportation to and from project site;
- Safety clothing and safety supplies;
- Contractor's indirect personnel (Foreman, General Foreman & Superintendent);
- Contractor's site supervision personnel;
- Contractor's head office overhead, expenses and insurance;
- Consumables including welding rods, sealant, adhesives and lubricants;
- Contractor's temporary facilities.

The following items are excluded from the All-In Labour Rates and need to be estimated separately.

- Daily pre start safety meetings;
- Weekly tool box meetings;
- Safety induction sessions;
- Special safety trainings;
- Other time consuming activities other than real installation hours.

Unit Prices

The unit prices for concrete and steel are a sensitive issue in the Sept-Îles area and can largely vary from year to year in relation with the overall construction activities. The unit prices used for this Study are presented in the two following Tables. These prices have been validated during the Feasibility Study. Detailed information is included in the FS Appendix 16.4.

Table 21-3: Unit price (steel)

Item	Price
	$5,000/t

Table 21-4: Unit price (civil and site preparation)

Item	Price

Common Construction Equipment

The All-In Labour Rates are inclusive of a significant portion of the construction equipment at site, but not all of them. We are identifying hereafter, the inclusions and exclusions of the All-In Labour Rates with regards to the common construction equipment on site.

The following items are included within the All-In Labour Rates:

- Construction's vehicles; small tools and consumables;
- Office trailers and lunch rooms;
- All sizes crane rental.

The following items are excluded from the All-In Labour Rates and need to be estimated separately. These items are covered in the Indirect Costs under Common Construction Equipment.

- Mobilisation and demobilisation of crane equipment to site;
- Project management of complex and/or facilities;
- Construction's sanitary facilities;
- Construction's guard house and security personnel;
- First-aid room.

21.1.4.3 **Site infrastructure**

The amount related to site infrastructures direct cost is equal to and it includes the following items:

- Bulk earthworks, landscaping and fencing
- Water intakes and distribution system
- Sanitary system
- Roads, overpasses and parking
- Safety
- Fuel distribution and prevention
- Fire detection and prevention
- Auxiliary buildings (Non-Process)
- Rail diversion
- High voltage substation
- Automation, instrumentation and communications

SGS has added three sub-sections in addition to the FS report that are:

- Waste Dump
- Noise Barrier
- Ore Stockpiles

The costs associated to the three above sub-sections, are included in the rehabilitation items.

The rail diversion Area 460 was estimated as described in the following section.

Mass Earthworks

Mass earthworks are assumed to contain no rock. For new track construction ballast is estimated to be 1,950 m^3 per kilometre and 2,510 m^3 per kilometre of sub-ballast.

Rail Diversion Area

Track unit costs are primarily based upon the budget quote provided by PNR Railworks on May 5, 2011. They include an additional Contractor construction management amount of 10%.

The estimate accounts for the entire rail work. It includes the re-alignment of 244 metres of existing track with the supply of new ballast and sub-ballast. The installation and removal of one additional 'temporary' turnout is included as to allow traffic to run uninterrupted on the existing track.

Unit costs are based upon Vendor supplied budget prices and supplement with in-house historical information.

Track

The following track design parameters where utilized for the purpose of costing:

- Rail: 136RE; 23.8m [78'] long segments;
- Track Gauge: 1,435 mm [4'-8.1/2"];
- Rail Joints: Bolted c/w Bolts, Nuts and Spring Washers;
- Axle Loads: 32.75 tonnes;
- Ties: Timber; 7" x 9" x 8'-6" long spaced at 500 mm [20"] centres;
- Tie Plate: AREMA 14"; Cut Spikes: 5/8" Sq. X 6" long;
- 2 per plate on tangent and 3 per tie plate on curved track;
- Rail Anchors: box anchored every second tie;
- Ballast: 300 mm [12"] deep with 300 mm [12"] shoulders;
- Sub-Ballast: 300 mm [12"] deep with a 7,750 mm [305"] wide roadbed shoulder;
- Turnouts: No. 12 - 136RE rail; on Timber Ties with RBM frogs, elastic fasteners and a switch stand;
- Culverts: 900 mm [36"] diameter Corrugated Metal Pipe (CMP).

21.1.4.4 Apatite transportation to port

The amount related to apatite transportation to port direct cost is equal to and it includes the following items:

- Rolling equipment

In the 2012 FS, the cost of forty-eight (48) 3,281 cu. ft. bulk covered hoppers (including spares) is included in the estimate. The 2013 maximum annual estimated apatite production is below the one of the 2012 study by a factor of ~8%, therefore the quantity of hoppers has been reduced to 44.

21.1.4.5 Port facilities

The amount related to port facilities direct cost is equal to and it includes the following items:

- Site and material handling
- Marine structures (separate estimate to the Port of Sept-Iles)

21.1.4.6 Exclusions

The capital cost estimate excludes the following:

- Environmental or legislative permits;
- Geotechnical investigation;
- Environmental impact studies;
- Legal and financing costs;
- Property acquisitions;
- At-Grade road crossings;
- Removal of existing track;
- Bridges and Tunnels (not required);
- Telecommunication & Train Control System (not required).

21.1.5 Indirect Costs

The indirect costs are composed of the following items:

21.1.5.1 Engineering, Procurement and Construction Management (EPCM)

As agreed with Mine Arnaud, the CAPEX Estimate for the Mine Arnaud Project is based on an EPCM type mode of construction. An EPC type mode of construction would result into a higher cost than for an EPCM.

For the purpose of this Feasibility Study, a percentage of Direct Costs is used to cover the cost of the EPCM services. The percentages are as per client's request for a total of 16% of Direct Costs. The breakdown is as follows:

- Project Management and Home Office Services – 3 %;
- Detailed Engineering – 6.75 %;
- Procurement – 0.75 %;
- Construction Management – 5.5 %.

The following table provides a general description of those services.

Table 21-5: Definition of EPCM services

Detailed Engineering	Includes the drawings and documents for the complete engineering package necessary to construct the intended facilities. In addition to drawings, the work includes preparation of construction and equipment specifications, bills of materials and data sheets, technical bid evaluations, vendor drawing review and checking, field vendor drawing review and checking, and field support from the office.
Procurement	Includes both local and foreign purchasing. Procurement encompasses requests for quotations, commercial evaluations and recommendations, assistance to Mine Arnaud for the terms and conditions negotiation, purchase order placement and maintenance, logistics and traffic, expediting, site inspection, but not foreign source inspection. Sub-contracts for installation and other services are also included.
Project Management and Home Office Services	Includes specialist personnel necessary to support the engineering and construction plans. These services involve project management, cost control, scheduling, estimating, project accounting, administration, etc.
Construction Management	Includes construction management services at site such the required field expenses for construction management personnel including travel and

21.1.5.2 Temporary Site Installations and Services

For the purpose of the previous Feasibility Study, a detailed evaluation was made to cover the cost of the Temporary Site Facilities. It includes the project management team's site installation requirements for the project execution.

21.1.5.3 **Commissioning**

For the purpose of the previous Feasibility Study, an evaluation was made to cover the cost of commissioning. This portion of the estimate includes for the project management personnel and related temporary installation required for the commissioning & start-up period. Costs corresponding to a period of three (3) months after mechanical completion have been included, including a provision for ten (10) tradesmen on a normal work week schedule for the potential assistance requirement during start-up & commissioning.

Also included within the estimate is a provision for engineering personnel to perform pre-operational checks.

Vendors' representative costs are estimated separately.

21.1.5.4 **Common Construction Equipment**

For the purpose of the previous Feasibility Study, an evaluation was made to cover the cost of common construction equipment. This item is to account for the items that are excluded from the All-In Labour Rates and need to be estimated separately such as mobilisation and demobilisation of crane equipment to site. In this item the minimum requirement of lifting equipment for the laydown area has been included and an allowance for the initial mobilization cost of the major cranes.

21.1.5.5 **First Fills**

For the purpose of the previous Feasibility Study, an evaluation was made to cover the cost of first fills. The evaluation is based on data from the OPEX and includes items like consumables, chemicals, fuels and lubricants. The total mill processing capacity has been higher by 3.7%, but as the equipments remain the same, there is no impact on the first fills.

21.1.5.6 **Others**

Vendors Costs

This item covers the cost for the vendors' representative for:

- Erection assistance;
- Start-up & Commissioning assistance;
- Training.

The above services are usually quoted separately from the equipment and are based on hourly or daily rates. A percentage of 2% of Direct Purchase was used to cover for those costs.

Premium Time

A percentage of 0.5% of Direct Costs was used to cover for the cost of premium time. SGS recommends that in the future studies, the mass salaries should be use instead of Direct Costs to estimate the premium time (overtime).

Construction Insurance

A percentage of 0.5% of Direct Costs was used to cover for the cost of construction insurance.

21.1.5.7 Third Party Services, Testing & Inspection (Hydro-Québec Pre-Project Study)

This item is to cover for the cost of Hydro-Québec Pre-Project Study to tie the mine site to the Hydro-Québec network. The cost is based on Hydro-Québec Planning Study dated September 19, 2011. It has a precision level of ±30%.

21.1.5.8 Third Party Services, Testing & Inspection

No costs are included in the estimate for other third party services than Hydro-Québec Pre-Project Study.

21.1.5.9 Hydro-Québec Tie-In to Network

This item is to cover for the cost provided by Hydro-Québec to tie-in the mine site to the Hydro-Québec network. The cost is based on Hydro-Québec Planning Study dated September 19, 2011 and includes the cost of a pre-project study. It has a precision level of ± 30%. An additional amount of 15% has been added to cover for operating and maintenance charges as per discussion with Hydro-Québec in October 2011.

After the pre-project study is completed, Mine Arnaud will have to provide a financial guarantee for the full amount. The amount owing will be reduced each month by a credit given by Hydro-Québec as a function of electrical power consumption. With the estimated power loads and the credit currently given by Hydro-Québec, the financial guarantee will need to be in place for a period of approximately about 10 years.

The additional optional cost for firm tie-in to the network of is not included in the CAPEX estimate.

Hydro-Québec Planning Study includes (2) important conditions:

Note 1: The cost is based on a thermal upgrade of the transmission line 161.7 KV at 50% of the cost of reconstruction. If the existing structures cannot be reused, the line will need to be rebuilt over 8.5 km. The cost of the tie-in to the network could then reach

Note 2: Hydro-Québec is currently reviewing the charges to apply for tie-in to the network. If Mine Arnaud does not get the pre-project study quickly, the cost may increase.

Note: The Tie-In to the Hydro-Quebec Network should not be mislead with the 161 kV line relocation estimated at a cost of in the previous FS under item 16.14, that has been deleted following recent information's supplied by Mine Arnaud.

21.1.5.10 **Owner's Costs**

Pre-Execution Phase Costs 2011-2012 up to DG4.

Over the course of the Feasibility Study, many items were identified as material for further study between the Feasibility Study and EPCM stages of the Project to either further define and confirm the scope of the Project, or reduce the CAPEX or OPEX of the Project, or for other reasons. A preliminary list of these items was identified and an approximate cost required to cover those items determined.

Mine Arnaud Costs

The Owner's costs included in the Capital Cost Estimate cover the Owner's incurred costs for the period starting from the Project approval date and up to the Project completion date. Where costs incurred by the Owner prior to the Project approval date are to be capitalised on the Project, they should be included under Sunk Costs.

A percentage of 3% of direct costs was used to cover for Owner's cost.

Land/Property Acquisition

This amount was provided by Mine Arnaud to cover the cost of buying land and properties near the mine site.

Training

A percentage of 0.5% of Direct Costs was used to cover for the cost of training Mine Arnaud personnel. As a reminder: in Quebec, each Employer with an annual mass salary higher than has to invest a minimum of 1% of the total salaries for training, even if the training costs are not fully realized, the Employer has to pay the 1% charge to the Government.

Two Year Spare Parts

A percentage of 2% of Direct Purchase was added to cover for the purchase of Two Year Spare Parts as requested by the client.

Capital Spares

A percentage of 5 % of Direct Purchase was used to cover for the cost of Capital Spares.

21.1.5.11 Taxes and Duties

There is no amount included for taxes and duties.

21.1.6 Contingency

The contingency reflects the potential growth in capital costs within the same scope of work. The contingency includes variations in quantities, differences between estimated and actual equipment and material prices, labour cost and site specific conditions. It also accounts for variation resulting from uncertainties that are clarified during detail engineering, when designs and specifications of the basic engineering scope are finalized.

Contingency is an amount of money allowed in an estimate for costs which, based on past experience, are likely to be encountered, but are difficult or impossible to identify at the time the estimate is prepared. It is an amount which is expected to be expended during the course of the Project. Contingency does not include scope changes, force majeure, labour strikes/wobbles or labour availability.

An allowance of 16.2% of Direct Costs and Indirect Costs excluding the mining equipment and rail work is used for this Feasibility Study level estimate. An allowance of 20.0% of the mining equipment costs has been included as well as an allowance of 15% of the rail diversion work costs.

21.1.7 Escalation

An escalation provision of Direct Costs, Indirect Costs and Contingency is normally included in the estimate to cover for inflation over the course of the Project. A significant portion of the Project expenses for the Mine Arnaud Project will be in Years 3 and 4 of the Project. As inflation will be taken into account in the Financial Model, Mine Arnaud requested that no amount for escalation be included in the CAPEX Estimate.

21.1.8 Exclusions

The following items are not included in the initial capital cost estimate:

Labour Relations

- Loss of efficiency due to negotiations in the construction industry;
- Allowances for industrial dispute or lost time arising from industrial actions;
- No allowances are made for special incentives (schedule, safety or others).

Environmental

- Provisions for the cost of remedial actions with respect to contaminated soil, lead contaminants and archaeological historical findings;
- Allowance for future designation of hazardous classification areas;
- Asbestos, lead paint and any other hazardous material removal – not expected to be required for a Greenfield project;
- Cost for removal of sheet metal with lead paint – not expected to be required for a Greenfield project;
- Plant closure and rehabilitation costs

Community and Native Relations

- Costs for Community and native relations and services

Permits, Fees, Taxes and Legal Costs

- Cost of Permits;
- License and Royalty fees;
- All Owner payable taxes;
- Legal costs;
- Force majeure issues.

Financing Costs

- Owner's Cost prior to Project approval;
- Project interest and financing cost during construction;
- Cost changes due to currency fluctuation (a sensitivity analysis should be included in the financial model);
- Any requirements related to project financing;
- Financing Fees;
- Resettlement / Relocation costs;
- Other Owner's costs;
- Sunk cost.

Operating and Maintenance Costs

- Operating and Maintenance Costs (OPEX) are provided separately in the OPEX Estimate Section;
- Any operational insurance such as business interruption insurance and machinery breakdown.

21.2 Sustaining capital

A provision of been applied in the development of the study for sustaining capital during the mine life for the three following sectors:

- Mining sector, including production fleet and service equipments.
- Processing equipment.
- Increase capacity of tailings ponds.

The summary of this sustaining capital, which is spread over the first 25 years of production, is shown in the following Table.

Table 21-6: Sustaining capital

Description	Sustaining Capital ($'000s)	Contingency 15% ($'000s)	Total Sustaining ($'000s)

Please refer to the cash flow Table in Section 22 to view the full spreading of the sustaining capital through the mine life.

21.3 Working Capital

The working capital represents the current assets (money owned) less current liabilities (entries on a balance sheet), or the cash available for day-to-day operations. A provision of has been estimated for this working capital. It is expected that Year (-2) and Year (-1) will require a total of and Year 1 and Year 2 will need an additional In the developed cash flow model, the working capital recovery is assumed to be done entirely on Year 30. During operations, a portion of this working capital may be recovered earlier, but leaving the complete recovery at the end of the operations is a kind of safety for the economical results.

21.4 Rehabilitation and Mine Closure Costs

Costs estimation for closure and rehabilitation has been produced according to the requirements of Québec's Ministry of Natural Resources and Fauna presented in the document entitled Guidelines for Preparing a Mining Site Rehabilitation Plan and General Mining Site Requirements, and to the Mining Act. The costs have been estimated based on prices given by contractors as well as costs observed in other rehabilitation works or presented in various rehabilitation plans.

The current Mining Act stipulates that a financial guarantee must be paid by the proponent of a mining project. The guarantee corresponds to 70% of the estimated cost for the rehabilitation of the accumulation areas (tailings accumulation cells, waste rock piles and top soil/overburden piles). Table III of the Guidelines provides the schedule of the annual payments of the guarantee. The Mining Act is however under review and subject to major modifications. For example, Bill 14 (An Act respecting the development of mineral resources in keeping with the principles of sustainable development) prescribes that the financial guarantee corresponds to 100% of the total estimated costs, and that this guarantee be paid within the first three years of commercial operations. Even though Bill 14 is not yet adopted, the above prescription is currently operational, as recently (in October 2011) observed, when Osisko deposited the first tranche of its financial guarantee covering the entire cost for rehabilitating the Canadian Malartic site. It is assumed the Arnaud Project will also be subject to this new prescription.

The total cost of closure and rehabilitation is estimated at . This amount includes the rehabilitation of the accumulation areas and the cost of other closure and rehabilitation activities.

The value of recycled equipment and heavy machinery has not been considered. The steel value is expected to be close to the cost of loading and transporting the steel off the mine site.

Table 21-7: Restoration Program – Costs Estimate

Item	Activity	Quantity	Unit	Unit cost ($)	Total cost ($)
1. Tailings accumulation areas					
Surface area of accumulation areas		9 451 400	m²		
Surface area of the dikes surrounding the accumulation areas		1 099 000	m²		
Volume of OB/top soil to cover accumulation areas and dikes with a 15 cm layer		1 582 560	m³		
	Cover accumulation areas and dikes with a 15 cm layer of OB/top soil	1 582 560	m³	5	7 913 000
	Hydraulic seeding over accumulation areas and dikes	10 550 400	m²	0,8	8 441 000
	Improve drainage pattern (using geotextile and coarse granular material in the ditches)	15 827	m	31	491 000
	Emergency spillway	2	unit	25 000	50 000
Sub-total 1					**16 895 000**
2. Polishing pond					
Surface area of the pond		104 100	m²		
Surface area of the dike		9 900	m²		
Volume of OB/top soil to cover accumulation area and dike with a 15 cm layer		17 100	m³		
	Pump water out and create a breach in the dike	1		20 000	20 000
	Cover the pond and dike with a 15 cm layer of OB/top soil	5 000	m³	5	25 000
	Hydraulic seeding over accumulation the pond and dike	114 000	m²	0,8	92 000
Sub-total 2					**117 000**
3. Settling pond					
Surface area of the pond		12 100	m²		
Surface area of the dike		9 900	m²		
Volume of OB/top soil to cover accumulation area and dike with a 15 cm layer		3 300	m³		
	Pump water out and create a breach in the dike	1		20 000	20 000
	Cover the pond and dike with a 15 cm layer of OB/top soil	2 000	m³	5	10 000
	Hydraulic seeding over accumulation the pond and dike	22 000	m²	0,8	18 000
Sub-total 3					**28 000**
4. Waste rock accumulation areas (#1 and #2)					
Surface area of accumulation areas		1 047 000	m²		
Volume of OB/top soil to cover accumulation areas with a 40 cm layer		418 800	m³		
	Smoothen pile slopes	1		200 000	200 000
	Cover accumulation areas with a 40 cm layer of OB/top soil	418 800	m³	5	2 094 000
	Hydraulic seeding over accumulation areas	1 047 000	m²	0,8	838 000
Sub-total 4					**3 132 000**
5. Low grade ore stockpile					
Surface area of the accumulation area		412 000	m²		
Volume of OB/top soil to cover accumulation area with a 15 cm layer		61 800	m³		
	Cover accumulation area with a 15 cm layer of OB/top soil	61 800	m³	5	309 000
	Hydraulic seeding over accumulation area	412 000	m²	0,8	330 000
Sub-total 5					**639 000**
6. O/B piles					
Surface area of the accumulation area		402 000	m²		
	Hydraulic seeding over accumulation areas	402 000	m²	0,8	322 000
	Planting of seedlings (over O/B piles and other accumulation areas, where required)				492 500
Sub-total 6					**814 500**
7. Other items					
	Revegetation of other areas (roads, building sites, etc.)	1		500 000	500 000
	Building dismantling	1		2 500 000	2 500 000
	Solid waste disposal	1		100 000	100 000
	Soil characterization, dangerous goods and contaminated soil disposal	1		350 000	350 000
	Partial reestablishment of drainage pattern and setting of open pit safety bunds (2 m high, top width of 1 m, slopes 1H:1V)	1		350 000	350 000
Sub-total 7					**3 800 000**
GRAND TOTAL					**25 500 000**

Notes

1. This cost estimate considers the general layout of the tailings management facilities as shown on the map entitled « General layout of project infrastructures » sent on October 3, 2011 through YouSendIt. Rehabilitation cost of accumulation areas includes the placement of overburden/topsoil, revegetation and monitoring.
2. The value of recycled equipment and heavy machinery has not been considered.
3. The steel value is expected to be close to the cost of loading and transporting the steel off the mine site.
4. Payment of the financial guarantee expected as per current Bill 14

Next Table shows the payment schedule required by the Provincial Government for the first three years of production. This amount represents the total estimated site rehabilitation and closure cost expense schedule over the life of mine based on progressive site rehabilitation. This amount represents the guarantee required by the Government to avoid that mining operators in economical difficulties goes bankrupt and leave mining sites and tailings areas without rehabilitation. This amount is added to the Cash Flow, and is part of the grand total capital cost of the project.

Table 21-8: Rehabilitation Guarantee Schedule

Year	Site Rehabilitation Guarantee ($'000s)
1	
2	
3	
Total	

21.5 Operating Costs Estimate

21.5.1 Summary

In the purpose of this exercise, SGS did not recalculate all the operating costs, but reviewed and compared them from in-house and heavy equipment suppliers cost data. SGS also discussed with Mine Arnaud staff to select the operating costs that appear reasonable in the context of this preliminary feasibility study (PFS).

At this stage, the estimated operating costs used in the financial model are:

Table 21-9: Summary of operating costs

Operating cost	unit	$*
Mining cost**	*$/t mined*	
Mining cost increment	*$/5m vertical*	
G&A cost	*$/t of ore treated*	
Transport cost	*$/t of ore treated*	
Processing cost	*$/t of ore treated*	
Rehandling cost (end of mine life)	*$/t rehandled*	

**Based on a concentrate production of 1.2 Mt per year*

***Mining cost of first mining bench*

The operating cost estimates for specific areas were estimated by the responsible consultants and compiled by Roche in the FS. The operating cost for the Mine Arnaud operation covers mining, ore processing, concentrate ship loading and transportation, tailings and water management on site, general and administration fees as well as infrastructure and services. The PFS project operating costs estimate is based on the following parameters:

Maximum of ore and waste (including overburden) mined at year 8:	29,500,000
Tonnes of ore milled per year:	11,201,120
Tonnes of apatite concentrate produced per year:	1,195,159
Total average manpower resources for the operation (payroll):	342
Operating days per year:	365

The transportation of concentrates to the port will be sub-contracted to local contractors.

The overall average operating cost for the Mine Arnaud project is estimates at per year or per tonne of apatite concentrate. In addition of to the previous Table, a more detailed summary of the operating costs for the Project is shown in next Table. All costs presented in this section are in Canadian Dollars (CAD) per year and CAD per tonne of apatite concentrate.

Table 21-10: Operating Cost Summary

COST ITEMS	ANNUAL COST ($/y)	ANNUAL COST PER TONNE MINED ($/t)	ANNUAL COST PER TONNE MILLED ($/t)	ANNUAL COST PER TONNE OF APATITE CONCENTRATE* ($/t)
GENERAL & ADMINISTRATION				
MANPOWER (INCL. ENG, ENV, & ADMIN.)				
GENERAL				
CONTRACTS				
MARKETING				
MUNICIPAL TAXES				
TRANSPORT AND SHIP LOADING				
MINING				
MINING (AVG Y2 - Y27, Incl Manpower)				
PROCESS				
MANPOWER				
STOCKPILE REHANDLE				
ENERGY (Plant + Garrage + Admin.Building + Pick up trucks				
CONMSUMABLES				
RREAGENTS				
OTHER PROCESSING				
FRESH WATER				
TAILINGS POND				
TAILINGS POND				
TOTAL				
TOTAL				

Flash Dryer estimation is /tonne produced if electric. The oporating cost would be /tonne produced for Bunker 'C'.

* Based on 1,195,159 tonnes of Apatite Concentrate

Note: The previous mining cost is based on the average costs of years 2 to 27.

21.5.2 Mining Costs

21.5.2.1 Assumptions for Costs Estimate

The following is a list of general assumptions in place for the open pit operating cost estimate:

-

-

Since the 2012 FS, some of the above assumptions have been slightly modified, mainly: verification of prices from equipment suppliers are showing that the actual tendency for prices is going down.

In spite of these minor changes, SGS is very comfortable with the mining costs that are within the accuracy of a pre-feasibility study (PFS).

21.5.2.2 General Mining Cost

The base mining cost used in this PFS is amounting to and it composed of the following sub-activities:

- Drilling
- Blasting
- Loading
- Hauling
- Mine Support and auxiliary
- Mine G&A and technical services
- Mine production and maintenance

In addition to this mining cost, an increment of tonne mined is added when mining activities deepened by a depth of 5 meters. This cost increment is used to reflect the augmentation of fuel consumption per tonne delivered to the mill and the lower productivity due to a longer cycle time.

SGS reviewed this mining cost for the main components which are in order of importance the hauling, and loading ones, representing more than 50% of all mining costs. On the average these two mining costs retained in this study are slightly above our verifications, done on a portion of the project operations. SGS is confident that the base mining cost of is safe.

The open pit mine operating cost estimate has an intended level of accuracy of -25% to +25%. Although budgetary quotes were obtained for all major equipment and first principles applied for estimation, the level of accuracy is dictated by the basis of the production schedule which is optimized pit shells versus detailed design pits.

The open pit will be operated by four crews working 12 hours shifts day and night, year round. Some crew positions will only be covered by day shift as required. Technical services staff will work a standard 40 hour work week on a seven day cycle.

21.5.2.3 Hourly Costs of Major Equipments

Next Table presents a summary of hourly operating costs retained in the FS for the major equipment with 85% mechanical availability and average to more demanding operating conditions. These costs have not been modified for the actual PFS. Operator labour cost is excluded from this Table.

Table 21-11: Major Open Pit Mine Equipment Hourly Operations Costs ($/hour)

Equipment Description	O'head	Repair		Maintenance		Fuel	Lube	Tires	Wear	Total
		Parts	Labour	Parts	Labour				Parts	
Sandvik D55SP Tower Drill	3.37	20.80	30.17	17.02	23.22	108.40	25.81		17.34	246.13
Atlas Copco ROC L* 30 LF Drill	2.29	14.12	20.48	11.55	15.76	73.59	17.52		11.77	167.08
Komatsu PC3000-6 Excavator	12.64	44.25	64.11	66.37	90.50	179.92	34.91		13.15	505.84
CAT 374D Backhoe	1.85	6.50	9.41	9.74	13.29	45.52	5.57		3.86	95.76
CAT 993 Wheel Loader	5.70	12.35	14.90	22.94	26.02	134.35	20.35	38.37	2.12	277.11
CAT 785D Haul Truck	2.43	8.99	11.16	16.70	19.52	116.43	20.04	39.53		234.80
36t Articulated 6x6 Haul Truck	0.53	2.16	3.57	4.02	6.23	26.52	4.11	9.33		56.47
CAT D9T Track Dozer	2.29	5.58	8.08	8.36	11.39	61.76	7.21		19.49	124.15
CAT 16M Motor Grader	1.87	6.13	8.41	11.40	14.70	36.89	5.37	2.39	1.82	88.98
Western Star 6900XD Truck	0.40	2.03	3.07	3.82	5.30	18.62	1.73	5.97		40.94

Fuel Costs

From the above Table, one can observe that the fuel consumption is the most important cost, representing an average of 43% of these operating costs (excluding salaries). A comparison Table, shown below, was prepared to evaluate the fuel consumption from the above Table, with estimations from Heavy Equipment Manufacturers, Caterpillar & Komatsu. To the exception of the CAT 993 Wheel loader, all equipments consumptions of the study are within the given estimations from suppliers, with most of them on the High Side estimation.

Table 21-12: Major Equipments Fuel Consumption Comparison

Hourly Fuel Consumption from Operating Costs Table 17.9 of the FS		Hourly Fuel Consumption from Equipment Suppliers		
		Low	Medium	High
Equipment Description	ltr./hr	ltr./hr	ltr./hr	ltr./hr
KOMATSU PC3000-6 Excavator	198	161.0	172	184 - 208
CAT 993K Wheel Loader	148	61.3 - 87.4	87.4 - 113.6	113.6 -140
CAT 785D Off-Road Truck	128	45 - 81.4	81.4 - 108.6	108.6 - 135.9
CAT D9T Crawler-Dozer	68	30.3 - 43.1	43.1 - 56.4	56.4 - 69.3
CAT 374D Backhoe Excavator	50	18 - 35.5	35.5 - 53.6	53.6 - 71.5
CAT 16M Motor Grader	41	20.4 - 29.1	29.1 - 37.9	37.9 - 46.6
CAT 740 Articulated Truck	29	16.7 - 23.2	23.2- 32.6	32.6 - 47.1

21.5.2.4 **Other Operating Costs**

All other equipments costs were compared with in-house available data and found to be in general above the average reference costs. This comparison is indicative that the FS study equipment costing, was prepared with a safety margin that contribute to make the economical results on the safe side.

21.5.2.5 **Mining Manpower**

Open pit mining operations are scheduled for 365 days per year. The mine manpower estimate is based on the year to year production requirements. All equipment on-site will be owned, operated, and maintained by Mine Arnaud, except that relating to the delivery of explosives to the blast holes and pre-strip activity in year -1 and the first part of year 1 which are performed by contractor.

Three work schedules are estimated for the various open pit operations.

Schedule 1 assumes four work crews working 12 hr shifts two shifts per day (both day and night shift). Each crew will work two day shifts, followed by two night shifts, followed by four days off. The majority of Schedule 1 employees are involved with the direct operations of the open pit. This includes production equipment operators, maintenance personnel, and the direct supervision of said employees.

Schedule 2 assumes two work crews working 12 hr day shifts only, on a four days on four days off rotation. Schedule 2 employees are involved with direct operations that do not require around the clock coverage. This includes additional operators for day time only activities and additional maintenance personnel.

Schedule 3 assumes a single work crew only working a 40 hr work week each seven day cycle. Schedule 3 personnel include management, supervision, and technical services personnel.

The explosive services contractor will operate a day shift only similar to Schedule 2. At full production three persons are expected on-site for a total of six persons.

First Table of section 21.5.3.2 is showing the summary of the Mine Arnaud open pit operations manpower requirements during the open pit operations. Listed are the total persons on payroll only. The average of yearly mine employees over the full production years is 245; as the average tonnage of total material to move in this PFS remains almost the same as the one of the FS, the average mine employees total remain the same. The only difference will be the maximum total mine employees that was 323 at Year-7, will drop to 290 on Year-8, which is the maximum production year of this revised PFS schedule. We recall that the number of employees is only indicative as the costs of these employees are included in the mining costs.

21.5.3 General and Administration Costs

21.5.3.1 General and Administration Costs

G&A operational costs has been estimated at of concentrate, or lower than the FS, explains by a larger production of apatite concentrate. This includes all costs which are not related to the mining and processing parts of the project's operation. This includes: administration manpower, general costs, contracts, marketing, taxes and transport & ship loading activities. Next Table depicts the total cost for each item. Additional details are provided in the following sections below.

Table 21-13: General and Administration Cost

Cost Items	Annual Cost ($CA)	Cost per Tonne of Apatite Concentrate* ($/t)
Administration Manpower		
General		
Contracts		
Marketing		
Taxes		
Transport & Ship loading		
Total		

* Based on 1,195,159 tonnes of Apatite Concentrate

21.5.3.2 Manpower

The total estimated annual manpower costs, excluding mine employees are shown in next Table below. The mine site work group has been divided into 3 sub-groups: Mine employees, Mill employees, and Administration.

An average total of 342 employees were estimated for the Mine Arnaud Project in the FS. This is considering a plant running 24 hours a day, 7 days a week, 52 weeks per year. The working schedule for most yearly compensated employees will be a standard 40 hours a week, 8 hours a days, 5 days a week, Monday to Friday.

The largest numbers of employees is in the mining group with 245, or 71% of all employees. Following a tonnage variation, the number of employees of the mining group might slightly vary in total, but as their salaries are included in the unit mining production cost, this variation of employees is taken in consideration by the production costs. The number of employees for the other two groups, administration and processing, shall stay the same. The hourly workers will be working shifts of 12 hours per day, 7 days a week, Sunday to Saturday. The shifts will be split in 4 groups, each working a 12 hour shift for one week (7 days on), and off the next week (7 days off).

Table 21-14: Manpower Cost

Area		Manpower	Annual Cost ($)	Cost per Tonne of Apatite Concentrate* ($/t)
Administration				
Engineering Services		Incl. In Mine Employees - Schedule 3		
Environment		4		
Administration		24		
	Sub-Total	28		
Mine Employees**				
Schedule 1		17	Incl. In Mining	Incl. In Mining
Schedule 2		139	Incl. In Mining	Incl. In Mining
Schedule 3		89	Incl. In Mining	Incl. In Mining
	Sub-Total	245	Incl. In Mining	Incl. In Mining
Mill Employees				
Mill Operation		42		
Mill Maintenance		21		
Mill Supervision		6		
	Sub-Total	69		
Total Excl. Mine		97		
Total Incl. Mine		**342**	N/A	N/A

* Based on 1,195,159 tonnes per year

** Mine Man Employees description

Schedule 1 : *Production equipment operators, maintenance personnel,* and the direct supervision of said employees.

Schedule 2 : *Additional operators for day time only activities, and additional maintenance personnel.*

Schedule 3 : *Management, supervision, and technical services personnel.*

The Administration group consists mostly of yearly salaried employees, which includes the Engineering Services, Environment and Administration Departments. The engineering services group is composed of mining and mechanical engineers, geologists and surveyors and is included in the Mine employees under Schedule 3. The environmental group (4 employees) is composed of a coordinator and technicians. The administration group (24 employees) is composed of the management team and the support functions (accounting, financing, marketing, information technology, human resources, health and safety, first nation coordinator, clerical, storekeeper and secretary). These employees will be working a normal work schedule during day shifts.

The mine's workforce and the mill's workforce are each divided in three sub-groups. The mine workforce is split in groups working on different schedules (Schedule 1, 2 & 3). The mill workforce is split in the following groups: Operation, Maintenance and Supervision.

The mine operation group (245 employees) is composed of mining equipment operators, maintenance, management, supervision and technical services personnel. Please refer to previous Table for more detail on the mine employees.

The mill's operation group (42 employees), is composed of foremen, process equipment operators, tailings equipment operators, load-out operators, lab technicians and labourers. The mill's maintenance group (21 employees) is composed of mechanics, electro-mechanics, electricians, welders, foremen and one planner. The mill supervision group (6 employees) is composed of the mill superintendent, a metallurgist and a metallurgist technician, a chief analyst, a planner and a clerk.

Annual salaries are based on information coming from locally established mining and metallurgy related companies. Complete details are further described in FS Appendix 17.1.

21.5.3.3 **General Costs**

Additional costs have to be accounted for the following recurrent items: Consultants, travel and seminars, communications, office equipment, insurance, legal fees, training expenses, computer IT services, recruiting fees, summer students and grants, associations' membership, rent for a downtown Montreal office and other miscellaneous costs. Details are presented in

Table 21-15: General Cost Details

Items	Annual Cost ($)	Cost per Tonne of Apatite Concentrate* ($/t)
Consultants		
Travel & Seminars		
Cmmunications		
Office Equipment		
Insurance		
Legal Fees		
Training Expenses		
Computer Services		
Recruiting		
Summer Students, Grants, etc.		
Associations		
Office Rent		
Miscellaneous		
Total		

* Based on 1,195,159 Apatite Concentrate

21.5.3.4 **Contracts**

Overall, contracts will be given for the following peripheral operations: security agents, janitor services, maintenance agreement, light vehicle mechanical maintenance service and snow removal for the site access road & parking. As part of Mine Arnaud's desire to attract qualified workforce, catering and day care services have been added to the contracts as incentives for recruiting and keeping future employees. The estimates for the costs of contracts are based on information provided by locally established companies as well as information provided by Mine Arnaud.

Table 21-16: Contractors costs

Items	Quantity	Annual Cost ($CA)	Cost per Tonne of Apatite Concentrate* ($/t)
Total			

* Based on 1,195,159 tonnes of Apatite Concentrate

21.5.3.5 Marketing

A tab has been created for marketing costs. This cost is for promoting the new apatite mine operation to the world, potential clients, population and governments. Since the mine operator and the product user will likely be the same entity, cost for marketing has not been estimated; however a tab in the OPEX spreadsheet has been kept in case the marketing strategy changes at some point in the future.

21.5.3.6 Municipal Taxes, School Taxes and Mining Right

Municipal taxes have been estimated by a third party and are composed of Sept-Îles municipal and school taxes. Mining rights yearly cost has also been estimated and has been included in this section. The breakdown is shown in following Table.

Table 21-17: Municipal taxes and mining right details

Items	Annual Cost ($CA)	Cost per Tonne of Apatite Concentrate* ($/t)
Total		

* Based on 1,195,159 tonnes of Apatite Concentrate

21.5.3.7 Transport and Ship Loading

The Transport and Ship Loading activities costs are summarised in next Table.

The transport of concentrate from the mine site to the port will be provided by Wabush Mine, at a rate of $3.00 per tonne of apatite product as per Mine Arnaud. The Port Handling operation cost is split in two cost sources. The first source is the operational cost related to the unloading of the cars and ship load-out operations, which will be sub-contracted to a specialized company. The annual operational cost of $280,000 has been estimated based on Mine Arnaud's past experience. The second source of operational cost at the port is the annual fee paid to the Port of Sept-Iles for using their infrastructure. This cost is determined to be per year plus tonne of apatite concentrate being loaded on the ships, which adds up to per year.

Table 21-18: Transport and ship loading costs

Items	Annual Cost ($CA)	Cost per Tonne of Apatite Concentrate* ($/t)
	$3 600 000	$3.01
Total		

* Based on 1,195,159 tonnes of Apatite Concentrate

21.5.4 Processing Costs

Processing operational costs include all costs related to the processing and site operations which include: mill manpower, energy for mine site and port facilities, fresh water, reagents, consumables and other processing costs. Next Table depicts the total cost for each item. Additional details are provided in the relevant sections below.

Table 21-19: Processing Operating Cost Summary

Costs Items	Annual Cost ($CA)	Cost per Tonne of Apatite Concentrate* ($/t)
Total		

* Based on 1,195,159 tonnes of Apatite Concentrate

21.5.4.1 Stockpile Re-Handle

In order to simplify the mill operation, the stockpile next to the gyratory crusher will require some re-handling to manage ore grades prior sending the ore to the mill. The cost for stockpile re-handling has been estimated to per year or per tonne of apatite concentrate. It is important not to confuse this cost with the re-handling cost of t attributable to mining stockpiles totaling 47 Mt, from which 27 Mt will be processed at the end of the open-pit activities.

21.5.4.2 Energy

Mine Arnaud's operations will be powered by the following three sources of energy: Electricity, Diesel Fuel and Gasoline. Most fixed equipment will be powered by electricity. Generators and most mobile equipment will be powered by diesel fuel. Gasoline will be kept for small pick-up trucks, and for the eventuality of having other equipment running on gasoline such as small generators and hand tools. The consumption of each energy source and the basis for operational cost evaluation is described below,

Table 21-20: Energy Cost Summary

Cost Items	Annual Cost ($CA)	Cost per Tonne of Apatite Concentrate* ($/t)

* Based on 1,195,159 tonnes of Apatite Concentrate

21.5.4.3 Electricity Consumption

For practical reasons, all electricity consumptions are considered as a processing operating cost which represents nearly 96% of the energy consumption. Electricity consumption is based on the connected and running kW for the entire site. The total electricity requirement has been based on the power requirement for the equipment shown on the load list, Appendix 10.3. Mine Arnaud's estimated electricity consumption, which is above 5,000 kW, qualifies for Hydro-Québec's Tarif L. Tarif L states that Hydro-Québec charges per subscribed kW agreed with HQ per month and consumed.

To calculate the cost of demand load in relation to the per subscribed kW agreed with HQ per month, the best estimate is to use the running load. This is explained by the fact that the cost of the demand load is not based on average power but is based on the peak load taken by the plant during the monthly billing period (even if this is for 1 hour of 1 day per month). Therefore, for billing purposes, this peak load cannot be less than the subscribed power agreed in the contract with HQ.

To calculate the cost of consumption in relation to the , it is correct to use the subscribed power which is a product of the Running load and utilization rate in percentage, which has been partly calculated based on the projected utilization of equipment in different areas and partly estimated based on experience. Next Table shows the details of the estimated annual electrical cost for the Mine Arnaud Project.

Table 21-21: Mine Site and Port Facilities Electrical Power Cost

Areas	Connected (kW)	Running (kW)	Utilization (%)	Suscribed Power	Consumption (kWh/y)	Annual Cost ($)	Cost per Tonne of Apatite Concentrate* ($/t)
	47 873	35 858	92	32 989	288 986 794	13 823 913	11.57
	20 202	15 152	86	13 031	114 149 107	5 604 845	4.69
	2 110	1 721	67	1 153	10 100 893	551 538	0.46

* Based on 1,195, 159 tonnes of Aspatite Concentrate

Note: The Flash Dryer consumption utilization rate has been lowererd from 02% to 86% to account for the by-pass of moist apatite concentrate which will not go through the Flash Dryer

21.5.4.4 Diesel Consumption

Diesel cost per litre is established at and corresponds to the cost after tax credit/refund from the government. Diesel for mine mobile equipment is not included in this section. It was calculated as a component of the mining operating cost. The annual diesel for personnel transportation and maintenance cost, which is the same as in the FS, is per tonne of apatite concentrate.

21.5.4.5 Gasoline Consumption

Gasoline cost per litre is established at and corresponds to the cost after tax credit/refund from the government. Gasoline consumption has not been accounted at this stage due to insufficient information on gasoline equipment. Gasoline will be used for potential gasoline powered pick-up trucks, small generators, chain saw, pumps, vibrating plates, etc… At this stage of the study, it is assumed that most equipment will be running on diesel fuel. Therefore, no gasoline consumption has been accounted for in the OPEX.

21.5.4.6 **Make-Up Water**

Make-up water is estimated at 668 cubic meters per hour. The anticipated annual cost for the water recirculation is per tonne of concentrate. Costs for pumping energy and pumps maintenance are include in energy cost and mechanical maintenance supplies cost.

21.5.4.7 **Reagents**

The annual consumption of reagents has been based on pilot plant and laboratory testing done throughout the feasibility study. The quantities have been scaled up to reflect the full scale process plant mass and water balances. Reagents unit prices came from various manufacturers and reflect annual quantities required as well as actual market price. Next Table details the cost of reagents.

Table 21-22: Reagents Cost Summary

Reagents	Description	Reagents Dosage (g/t)	Annual Consumption (tpy)	Cost per Tonne ($/t)	Cost per year ($)	Cost per tonne of concentrate* ($/t)
Lime as a setting agent						

* Based on 1,195,159 tonnes of Apatite Concentrate

21.5.4.8 **Consumables**

Consumables are divided in four sub-groups: liners, grinding media, mechanical maintenance supplies and lubricants. Details are shown in next Table.

Table 21-23: Consumables Summary Cost

Items	Annual Cost ($CA)	Cost per Tonne of Apatite Concentrate* ($/t)
Total		

* Based on 1,195,159 tonnes of Apatite Concentrate

The liners are for the crusher, SAG mill and Ball mills. The crusher steel liners unit cost comes from the Crusher manufacturer quotations' spare parts lists. SAG mill chrome-moly steel liners and Ball mill rubber liners unit prices come from mills manufacturer quotations. As indicated in next Table, the consumption for liners has been evaluated based on the ore's abrasion index laboratory results. It represents per year or of apatite concentrate.

Table 21-24: Steel Liners Cost Details

Steel Liners	Consumption					Price ($/set)	Annual Cost ($)	Cost per Tonne of Apatite Concentrate* ($/t)
	kg/kWh	kWh/t	kg/t	t/y	# of sets per year			
Total Cost of Liners								

* Based on 1,195,159 tonnes of Apatite Concentrate

Chrome-moly mill liners offer a potential for cost savings in an abrasive and impact environment. Various suppliers and quality of liners can be found on the market. It is impossible to predict the benefit of using chrome-moly liners and tests with suppliers would be required to do so. By using the abrasions indexes determined by SGS on ore samples, it was possible to estimate the liner consumption.

Next Table indicates that the chrome-moly mill liners are 20% to 35% more expensive than the normal steel liners. Unless the chrome-moly liners producer can guaranty a liner life 20% to 35% higher than the normal Steel liners, Roche-Ausenco would not recommended their use. The benefit of having a longer liner life would be the increase of the mechanical availability.

Table 21-25: Steel Liners Consumption

Description	Steel Liners ($/set)	Cr-Mo Liners ($/set)	Variations

Roche-Ausenco considers that using normal steel liners with a combination of forged steel grinding balls at the SAG mill and high chrome steel balls at the ball mills is the most efficient way to operate the grinding mills at the beginning of the operation. The optimization of the consumption of those consumable will have an important impact on the OPEX and it will have to be conducted during the course of the operation. Nevertheless, Roche-Ausenco recommends to initiate discussion with liner and grinding media suppliers shortly after the purchasing of the grinding mills.

The grinding media consumption comes from calculation based on the ores abrasion index laboratory result, developed by Allis-Chalmers/Bond F.C. These formulas give consumption in kg/kWh. Next Table shows the consumption conversion from kg/kWh to tonnes per year, the unit rate and the annual cost for each application.

The grinding media and the mill liners consumption comes from formulas developed by Allis-Chalmers/Bond F.C. and are based on the ores abrasion index laboratory result. These formulas give grinding media and liners consumptions in kg/kWh for normal steel.

The used of forged steel grinding media to liberate valuable minerals prior flotation is a common practice due to the relative low cost of this media compared to other form. However, even if grinding media made of steel alloy or high chromium steel are more expensive, there are potential cost savings if the consumption is reduced to the point of off setting the higher unit price.

The relative performance of high chrome balls compared to forged steel balls is dependant of the milling environment:

- In high wear rate application like SAG and primary grinding, the high chrome balls has 0 to 30% lower wear rate.
- In low wear rate application, like secondary ball mills, the high chrome relative performance will depend on the corrosiveness and abrasiveness of the environment. In high corrosive environments, the high chrome balls can have less than half the wear of the forged steel balls.
- The three recognized components to wear are: impact, abrasion and corrosion. Impact wear is proportional to the ball's weight, while abrasion and corrosion wear is proportional to the ball's surface area. In addition to ball speed, ball size and recharge practice, charge volume and feed rate are significant factors affecting the actual production wear rate.

Also based on literature, the grinding media could have an impact on pulp chemistry. This in turn may affect the recovery in a flotation circuit.

The optimization and final selection of the grinding media is normally done during the course of the operation. The selection is based on consumption, unit cost and impact on flotation recovery using plant data. But for preliminary comparison, we can assume the following:

- High chrome (15% Cr) grinding media unit cost:
- SAG Mill grinding media consumption: 5% lower with high chrome than forged steel;
- Ball Mill grinding media consumption: 25% lower with high chrome than forged steel;

Considering the higher cost of the high chrome balls and the low potential for a reduced consumption, Roche do not recommend to the operation with high chrome grinding media for the SAG mill operation. For the ball mill operation there is a true potential to make saving by the use of high chrome grinding media. Next Table presents the estimated cost difference related to the consumption of the grinding media.

Table 21-26: Cost Comparison between High Chrome and Forged Steel Grinding Consumtion

Description		Consumption				Price	Cost
		(kg/kWh)	(kWh/t)	(kg/t)	(t/y)	($/t)	($/y)
High Chrome steel grinding media							
Forged steel grinding media							
Proposed selection for grinding media							

*This Table was not updated from the last FS

Table 26 shows clearly that high chrome grinding media has a great potential for cost saving for the ball mill operation. High chrome-grinding media can become profitable for the SAG mill operation, if their consumption can be decreased by more than 5% when compared to the predicted consumption of conventional forged steel balls.

The mechanical maintenance supplies have been estimated using 8% of the direct cost of the concentrator equipment, which is per year or of concentrate.

The instrumentation supplies have been estimated using 8% of the direct cost of the instrumentation for the concentrator, which is per year or of concentrate.

Next Table shows lubricants' consumption and cost were estimated with a mining estimation guide based on hourly cost and estimated hourly operational hours for each major equipments.

Table 21-27: Lubricant cost details

Lubricant	Hourly operating cost ($/hour)	Operating time for 12 months	Number of units	Annual Cost ($)	Cost per Tonne of Apatite Concentrate* ($/t)

**Based 1,195,159 tonnes of Apatite Concentrate*

21.5.4.9 Other Processing Costs

Other processing costs such as environmental services, R&D costs, and process building maintenance have been estimated based on Roche's experience and database or factorized from similar mining operations. Laboratory supplies have been estimated based on inputs from mineral process laboratories.

Table 21-28: Others processing cost details

Items	Annual Cost ($)	Cost per Tonne of Apatite Concentrate ($)

**Based 1,195,159 tonnes of Apatite Concentrate*

21.5.4.10 Tailings Costs

Tailings costs have been estimated based on Roche's experience and database or factorized from similar mining operations. The costs are for annual maintenance of tailings ditches and dikes' base as well as for the water treatment unit annual reagent consumption.

Table 21-29: Tailings cost details

Items	Annual Cost ($)	Cost per Tonne of Apatite Concentrate ($)

**Based 1,195,159 tonnes of Apatite Concentrate*

22. Economic Analysis

- The economic evaluation of the Arnaud Project was performed using the discounted cash flow model on a before-tax basis. The capital and operating cost estimates input into the financial analysis model were based on the mine plan developed in this study to process 11,200,000 tonnes of ore per year. The internal rate of return (IRR) on total investment was calculated based on 100% equity financing. The net present value (NPV), based on a discounting rate of 5% resulting from the net cash flow generated by the Project was also calculated. The payback period is also indicated as a financial measure. A sensitivity analysis was also performed and presented. The following assumptions were made for the financial analysis:
- Phosphate market price of per tonne of P_2O_5 concentrate FOB (*Freight On Board)*;
- Marketing and concentrate transportation costs (from port) are included in the P_2O_5 price provided by Mine Arnaud;
- Initial capital (capital expenditure) spent over the 3 first years of construction (10, 40 and 50 % spent for years -3, -2 and -1 respectively);
- Constant exchange rate of $1.00 (US$:CDN$);
- Economical analysis is presented as pre-finance and pre-tax;
- Sunk costs and owner's costs are not included in the model;
- Salvage value of was considered.

22.1 Cash Flow Forecast

Based on the mine plan presented in Section 16, a cash flow forecast had been prepared. A summary of the base case results is given in Table 22-1, while the cash flow statement related to the base case scenario is presented by Table 22-2.

Table 22-1: Project Cash Flow Summary

ITEMS	UNITS	VALUE
Total revenue	M$	
Total operating costs	M$	
Pre-production capital costs	M$	
Sustaining capital costs	M$	
Rehabilitation costs	M$	
Others expenses	M$	
Undiscounted benefits	M$	
NPV discounted at 5.00 %	M$	
Internal rate of return (IRR)	%	
Payback period*	years	

**After start of production*

Table 22-2: Cash flow model

	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14
O																	
R																	
M																	
C																	

YEAR		
tonnes		
% P_2O_5		
%Wrec		
tonnes		
$/t. conc.		
$		
tonnes		
tonnes		
tonnes		
tonnes		
tonnes		
$		
$		
$		
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$		
$		
$		
$		
$		
$		

22.2 Net Present Value, Internal Rate of Return and Payback Period

The financial analysis results of the Arnaud Project for the base case scenario are calculated as:

- net present value (NPV) at 5.00 % discount rate;
- internal rate of return (IRR)
- years payback (after start of production)

22.3 Taxes, Royalties and Interests

Taxes

This economic analysis of the Arnaud Project was done without taking into consideration any applicable taxes. However, Mine Arnaud will be subject to current and planned Federal and Quebec tax rates and related tax rules. At the date of this report, the applicable tax rates are:

Federal income tax rate

Provincial income tax rate:

Quebec mining tax

A mining corporation in Quebec will be subject to mining taxes on the annual profit earned on its property that is reasonably attributable to the mine and that can be reasonably be attributable to the operations of the mine. For the purpose of the Mining Tax Act, annual profit is determined by subtracting from gross revenue the operating expenses and allowances directly related to the mine, including:

- exploration and development expenses;
- depreciation;
- a processing allowance;
- an additional allowance for a mine located in the North or mid North (not applicable).

Important note:

On May 6th 2013, the Quebec Government has tabled legislation that modifies the royalty/tax regime associated to the mining industry. According to documents submitted by the Government, the new mining royalty regime will ensure that all operators (mining companies producing cash flow) have to pay a minimum tax based on revenue. The new regime proposes the imposition of an additional minimum royalty based on the value of the ore extracted by a mining company from

the Quebec soil (run-of-mine ore). If the production value is less than the royalty will amount to 1% and if the production value is greater than , the royalty will be of the value of the ore mined. In addition, the Quebec Government proposes to implement a progressive mining tax rate (instead of actual flat rate of) for companies that generate profits and that will not be subject to the minimum royalty. The mining tax will be if the profit margin of the company is between and If profit is between and the tax will be and if the profit is between and the tax will be This represents tabled legislation by the Government which will need to be voted into law before it can be implemented. If accepted, the new tax regime is expected to be implemented in January 2014.

<u>Royalties</u>

No royalties were considered in this study.

<u>Interests</u>

All the economical analysis presented in this study is calculated as pre-financed, so no interest attributable to capital financing was considered.

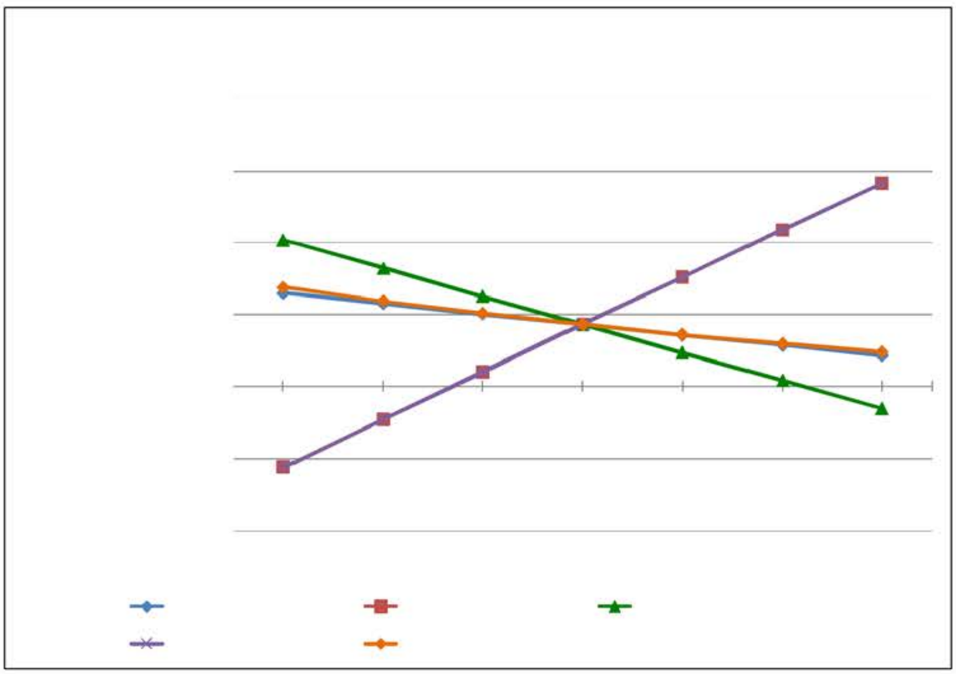
22.4 Sensitivity Analysis

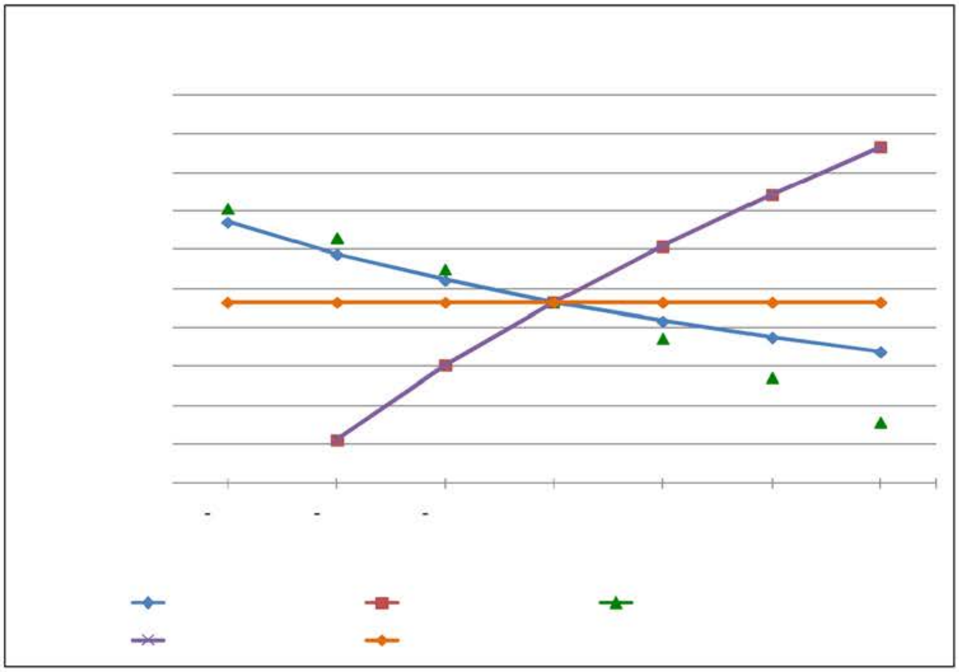
The sensitivity of the pre-tax and pre-financed Net Present Value (NPV) and the internal rate of return (IRR) were evaluated for changes in key variables and parameters such as:

- Capital investment (CAPEX);
- Concentrate value;
- Operating cost;
- Head grade, calculated as % of weight recovery;
- Discount rate.

The result of the sensitivity is presented by the next Table and Figures:

Parameters	Units	-30%	-20%	-10%	0%	+10%	+20%	+30%





It can be seen that the concentrate value and the head grade cost have the greatest impact on project NPV and IRR. The project becomes uneconomic when the concentrate price drop by about and when the overall cost per tonne treated increase by about Overall, the project is sensitive to each of the major variables. This sensitivity analysis clearly demonstrates that concentrate value needs to remain over /tonne (at an exchange rate of 1.00 US$:CDN$); in order to keep the project economically viable.

23. Adjacent Properties

Using GESTIM's database from the Ministère des Ressources Naturelles du Québec, a number of properties are found around and in the vicinity of Mine Arnaud's project in Sept-Îles (Figure 23-1). Only 3 properties are contiguous to the Mine Arnaud Property with one being located inside the Mine Arnaud Property. Two of the contiguous properties are Surface Mineral Rights and the other property is registered under Philippe Tremblay, on which no information is available.

A search of the Metals Economics Group's database for projects within 50km of LONG 66°31'38" W and LAT 50°16'13" N does not provide a significant amount of results. Two (2) active projects were identified and neither are phosphate projects. The closes of the two projects is located about 15km Southwest of Mine Arnaud and is owned by Corporation Éléments Critiques which have reported a Rare Earth/Tantalum showing named Reine on their website (http://www.cecorp.ca/fr/projets/quebec-terres-rares.html).

Other claim owners are listed around the property (Figure 23-1), but no significant information could be found regarding the declared work and/or type on mineral exploration ongoing on their mineral claims.

As of July 11th 2013, three companies had declared work on their claims (GESTIM), one of them being Mine Arnaud. The other two companies are Corporation Éléments Critiques, who declared technical work and Carrière Lalancette who declared striping and excavating on their Pointe-Noire Property.

According to the Ministère des Ressources Naturelles du Québec, no historic mining activity is documented on the Mine Arnaud property or in the vicinity of the property other that Surface Mineral Extraction (peat, sand, gravel and aggregates).

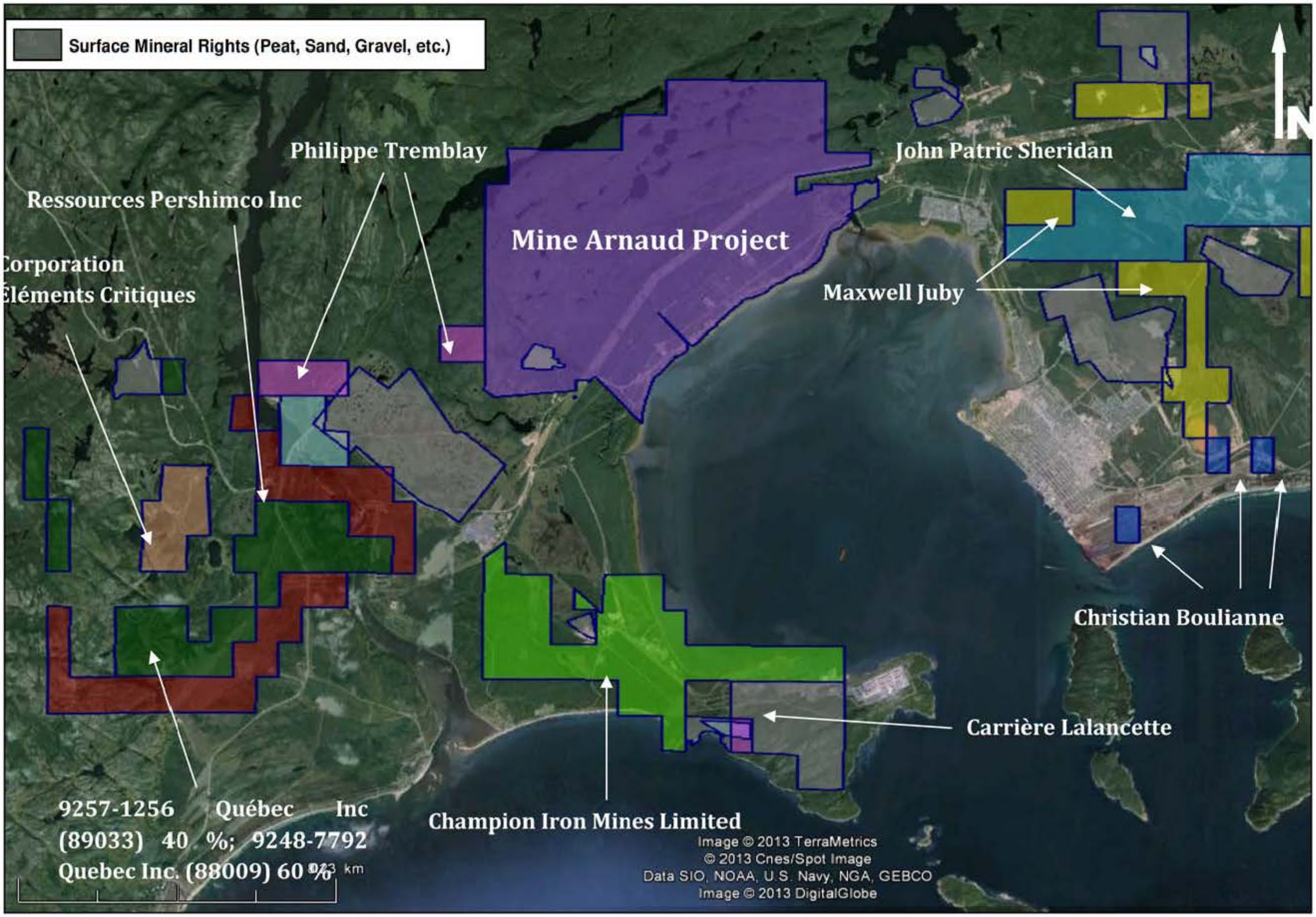


Figure 23-1: Adjacent Properties in the Vicinity of Sept-Îles

24. Other Relevant Data and Information

This pre-feasibility study is performed only on a portion of the Arnaud deposit. The entire mineral resource is still open at depth and on both East and West extensions. Future discoveries in these areas could materially affect the profitability of this Project.

To the author's knowledge, there are no other relevant data and information.

25. Interpretation and Conclusion

Following the last study realized on Mine Arnaud deposit, numerous progresses have been made. Drilling significantly increased to level of confidence of the mineral resources and it is now possible to estimate the quality of the produced concentrate using different models. Increasing geological knowledge will enable to limit unpredicted variation of topics like ore grade, concentrate produced and mining sequence.

25.1 Deposit

The deposit's geology is now well understood and increasing drilling will not change significantly the geometry and interpretation of the mineral deposit. However, information related to structures and displacements due to faulty could be better define and taken into account when conducting further geological modeling.

Work done on chlorine behavior and prediction of the values for the concentrate; enable SGS Geostat to re-assure, both the buying party (Yara) and Mine Arnaud on the potential chlorine problematic. For the moment, chlorine is under acceptable threshold (Cl% Conc <0.14%), established by Yara, for the 28 first year of mining (see Report in Appendix 4). According the different models, the Mineral Resources also have values for Cl in the concentrate under acceptable threshold except for the inferred resources.

During interpolation process, it was noted that a single hole from the database (1166-10-83) shows abnormal chlorine results (average of 0.1069% Cl) over 38.2m. This hole only accounts for an increase of 15% Cl of the California block model when tested by SGS Geostat. Hence, it would benefit Mine Arnaud to re-assay hole 1166-10-83 for chlorine values to verify these abnormal results.

25.2 Drilling and Resources Estimation

SGS Geostat verified the work done before September 2012, principally by RPA, and is comfortable with what has been accepted by the other parties who worked on the past mineral resources estimates. Drilling and sampling programs respect the industry's standards and the acquired data is reliable. Most drill holes show a core recovery over 90% and holes are placed in order to cross cut the mineralization perpendicularly.

QA/QC programs were not conducted before 2010 but upon verification of the laboratories internal QA/QC data, SGS Geostat is satisfied with the quality and duplicability of the data. QA/QC programs instigated following 2010 are acceptable and the values from blanks and standards show only minor failures. Furthermore, independent control sampling done by SGS Geostat shows a good correlation between original samples and duplicates values analyzed at a third party laboratory. Implementation of systematic re-assaying of QA/QC failure would increase the reliability of the data, but the data provided by Mine Arnaud is judged suitable for resources estimation by SGS Geostat.

Mineralized solids were modeled on vertical sections with the projection of the 430 drill holes using the assay values for P_2O_5 at a model minimal value of 2.0%. Numerous intercalated assays below this lower model value were still incorporated in the mineralized solids in order to respect the general geometry of the mineralization, but are always surrounded (top and bottom) by assay higher than the modeling value. Upon modeling the four recognized mineral zones (California, Upper, RailRoad and Nelsonite), variographic study show that statistically the RailRoad and Upper zone could be combine together and this new zone was named Combine. A block model was generated for the whole deposit (block size of 5m x 10m x 5m) and blocks were tagged according to the zones they belong to (Nelsonite, Combine, California or Waste). The block model was also limited at surface by the overburden surface, which was modeled using lithological information form drill holes.

Density measurements were conducted on drill core samples over the years and the values were used to generate a statistical model for the density. Fe_2O_3% and TiO_2% values from the assays can be used to calculate a density for each assay. This calculated density based on the two heaviest mineral of the deposit shows a good correlation with the measured density; hence it was used to assign a density value to each assay interval.

Variographic studies were conducted for each of the three mineralized zones for P_2O_5, K_2O and Cl variables. The variograms were used in the kriging process of the block interpolation but also to establish search ellipsoid and classification of the mineral resources. Each zone was domained differently and interpolated using its own set of 3m composite and parameters. Upon interpolation of the variables, metallurgical variables were calculated from interpolated values and added to the block model.

25.3 Mineral Resources

Mineral resources of the Mine Arnaud deposit are limited at depth by an optimized pit shell in order to account for the "reasonable prospect of economic extraction" of reported mineral resources under the NI 43-101 regulation. The pit shell represent in this section outlines an open-pit shell that generates the maximum economic value. However, this value does not take into account mine planning and time value of money (discounting rate). It is for this reason that there are no guaranty that this shell shall be selected as the base case scenario to develop the mining scenario; and thus, to calculate the in-pit reserves.

Table 25-1: Mineral Resources for the Mine Arnaud Sept-Îles deposit

Category	Cut Off (P2O5%)	Zones	Tonnage (Mt)	Average P2O5 (%)	Average WRec (%)	Average P2O5 Conc (%)	Average Cl Conc (%)
Measured	1.91	Nelsonite	38.48	5.91	13.62	38.98	0.082
Measured	1.76	Others	332.39	3.95	9.25	38.25	0.124
Measured		**TOTAL**	**370.87**	**4.16**	**9.70**	**38.69**	**0.106**
Indicated	1.91	Nelsonite	9.39	6.22	14.31	39.06	0.087
Indicated	1.76	Others	101.48	4.06	9.48	38.27	0.127
Indicated		**TOTAL**	**110.87**	**4.24**	**9.89**	**38.75**	**0.113**
Inferred	1.91	Nelsonite	-	-	-	-	-
Inferred	1.76	Others	42.76	3.52	8.29	38.11	0.158
Inferred		**TOTAL**	**42.76**	**3.52**	**8.29**	**38.11**	**0.158**

Notes: -The mineral resource estimate has been calculated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definitions Standards for mineral resources in concordance with National Instrument 43-101, Standards of Disclosure for Mineral Projects.
-Mineral resources which are not mineral reserves do not have demonstrated economic viability.
-Inferred mineral resources are exclusive of the Measured and Indicated resources.
-Resources are constrained by Pit Shell and under the bottom surface of the California zone.
-Effective date 10-07-13.
-Others are referring to the material categorize as Combine and Surrounding mineralization.

25.4 Mining

The Arnaud Project is a very straight forward mining operation, wide and long open pit, low stripping ratio, etc. The resources will be mined by a single open pit, which will have 28 years of production life following a three year construction and pre-production period. The mine plan is based on probable and proven reserves contained in the pit design, which was based on a Lerchs-Grossmann optimized pit shell. Open-pit mining will be done by the Project operator from the beginning to the end of the operation. Surface mining will follow the standard practice of an open-pit operation; with conventional drill and blast, load and haul cycle using a drill/truck/shovel mining fleet. The overburden and waste rock material will be hauled to the overburden and waste disposal areas near the pit. The run-of-mine mineralization will be delivered by large mining trucks to the primary crusher or stockpiles near the crusher area.

The main challenge will be to mine the overburden layer, especially in the East of the deposit. There are many possibilities to optmize the developped mine plan using new mining approachs, new highgrading scenarios, different mill feed, etc. Those optimizations could greatly impact the Project NPV.

As presented in Section 16, the overall in-pit reserves are defined as:

Table 25-2: Mine Arnaud Project Reserves (presented as mill feed)

Material Type		Cut-off (%P2O5)	Tonnes*	Grade* (%P2O5)	Grade (%Wrec)	Concentrate tonnes	Chlorine** (%Cl)
Ore (Probable Reserves)	Combine	1.76%	54 400 000	4.67%	10.85%	5 900 000	0.112%
	Surrounding material	1.76%	1 900 000	2.39%	5.75%	110 000	0.136%
	Nelsonite	1.91%	8 100 000	5.79%	13.34%	1 080 000	0.081%
	Total		**64 400 000**	**4.74%**	**11.01%**	**7 090 000**	**0.108%**
Ore (Proven Reserves)	Combine	1.76%	227 000 000	4.26%	9.93%	22 540 000	0.110%
	Surrounding material	1.76%	7 000 000	2.50%	6.00%	420 000	0.109%
	Nelsonite	1.91%	26 000 000	5.62%	12.97%	3 370 000	0.073%
	Total		**260 000 000**	**4.34%**	**10.13%**	**26 330 000**	**0.105%**
Ore (Total Reserves)	Combine	1.76%	281 400 000	4.34%	10.11%	28 440 000	0.111%
	Surrounding material	1.76%	8 900 000	2.47%	5.95%	530 000	0.114%
	Nelsonite	1.91%	34 100 000	5.66%	13.06%	4 450 000	0.075%
	Total		**324 400 000**	**4.42%**	**10.30%**	**33 420 000**	**0.106%**

Note: This reserve includes 2% dilution and 98% mining recovery for Railroad, Upper and surrounding material ore types and a 10% dilution and 90% mining recovery for the Nelsonite ore type. Surrounding material is referring to mineralized material between zones but excluding the sterilized material as defined in Figure 16-2.

25.5 Process

Except for few issues, SGS Geostat is in complete accordance with most parts of chapter 19 of the Roche-Ausenco Feasibility Study. However, because in the opinion of SGS Geostat these issues are very important, the whole chapter 19.1 is reproduced here with the appropriate changes SGS considers significant.

There are a few areas that still need further investigations prior assessing that no major technical threats related to the beneficiation are to be expected during the course of the operation of Mine Arnaud. This section addresses these threats.

Ore variability

The various samples studied indicated important variations with the mineralogy, particularly in regards to magnetite content. Important variations with the chlorine content and P_2O_5 grade could also be encountered. These variations could be important depending on the location and the depth of the ore. In order to minimise the effect of these upsets during the ore beneficiation, it will be important to insure that mine planning includes proper blending prior to the crushing operation. The actual dome stockpile design does not allow for ore blending prior to grinding. The risk associated with important variations of ore feeding the plant may impact the recovery, the consistency of the concentrate grade and eventually the operating cost.

Another threat associated with ore variability concerns the pumping problems that may occur if grade varies considerably. A 10% overdesign factor is applied on pump flows, but it may not be sufficient. A 20

% margin on pump flow may be required to account for the ore variability mostly related to the magnetite content.

From the beginning of the work, SGS was told that there was no need for conducting the pilot plant with the individual ore types because the material would be blended prior processing with constant ratios. The mine planning finally showed that there will be variations through the mine life and the proportion of Upper (36.1%), Nelsonite (19.4%) and Railroad (44.5%), may not correspond to any of the annual planning. It is now understood that each ore type responds differently to the process. The various ore types have not been tested independently at the pilot plant.

Dewatering cyclone

Even if Roche-Ausenco added dewatering cyclones to the circuit to enhance the efficiency of the conditioning based on the principle that a high pulp density of 50% solids should facilitate the contact between the reagents and the pulp and therefore enhance the flotation process, we at SGS Geostat believes that it is not necessary. If the contact time between the reagents and the pulp is a problem, it is much simpler to have bigger conditioners than to deal with all this water out and then back in the circuit. On the other hand if the problem comes from the production of an excessive amount of fine particles by over grinding, here again the solution is to solve the problem at its source and not further down the circuit.

Flotation (column vs mechanic cells)

Roche-Ausenco designed the whole flotation circuit with column flotation cells. It seems that there was quite a debate between the proponents of the column flotation cells and the proponents of the mechanical cells. Based on some consensus between mill metallurgists, that column flotation cells are better for the concentrate grade while mechanical cells usually permit better recoveries, SGS Geostat believes that the answer lies in a somewhat compromise between the two proponents : column flotation cells at the rougher and cleaner stages and mechanical cells at the scavenger and cleaner/scavenger stages. After all, best results were always obtained from LCT's made in mechanical cells.

Cleaner stage

In Roche-Ausenco mass and water balance, SGS Geostat could not find the justification for the addition of 500 m^3/h of water at the cleaner feed thus reducing the cleaner feed % solid to around 18%. It was proven at the laboratory level that the best condition for apatite flotation was around 35% solid. This is the reason why at the laboratory level, a much smaller flotation cell is used at the cleaner stage in order not to float at a too low percentage solid.

Not only all this water will have to be dealt further down the process, but it forces to use much bigger cleaner and cleaner/scavenger cells than necessary.

High intensity magnetic separation

In all cases, it was possible to produce apatite concentrate meeting the specifications of Yara without the use of Wet High Intensity Magnetic Separation (WHIMS). The used of WHIMS in apatite beneficiation is quite common for quality control but it is felt that it will not be necessary for Mine Arnaud. Nevertheless, the design provides some space in the beneficiation plant if deemed necessary at some point in time during the mine life. All tests conducted with the WHIMS showed that an important proportion of the apatite concentrate reports to the magnetic product. Microscopic observations indicated that some apatite grains contain a fine line of iron bearing mineral. By submitting these grains to a high magnetic field, they report to the magnetic product and therefore, it would reduce considerably the P_2O_5 recovery.

If for some reason, it is required one day to use WHIMS to achieve the quality specifications, Mine Arnaud will have to seek a new market for the magnetic by-product or to regrind and reprocess it.

Filtration

Roche-Ausenco selected a belt filter as the most efficient and low capital cost piece of equipment to de-hydrate the apatite concentrate prior final drying with the use of heat. Two (2) buffer tanks, ahead of the filter and having 8 hours retention time should allow for maintenance planning without major production disturbance. A problem remains with belt filtering with the particular case of Mine Arnaud Project: the moisture content in final concentrate for shipping should be between 0.5 to 1.5% moisture content. Therefore, an important quantity of water remaining in the filtered concentrates needs to be evaporated which implies important energy cost. The risk is that on a long run the selected option of using a belt filter may become more expensive than investing initially on pressure filters, which are more costly in terms of CAPEX and OPEX. However, it would allow reducing the CAPEX and OPEX of the flash dryer.

Flash dryer

Roche-Ausenco upon a suggestion from both FLSmidth and GEA Barr Rosin, in order to mitigate the difficulty of obtaining proper moisture content in the final dried concentrate, proposed a mixing device that will allow by-passing some of the flash dryer feed and remixing it with the dried product to obtain a final concentrate having proper moisture content as specified by Yara.

We at SGS Geostat fail to see the rationale of this procedure. Usually, the problem is never over drying. Even if the dryer would dry the apatite concentrate to 0% moisture, as soon as it gets out of the dryer and cools down, it attracts some 0.5% moisture from the surrounding air by the dew point phenomenon (especially in colder humid climate). On the other hand, it is always difficult to mix in the right proportion of damp material to a dry one in order to have the right % moisture.

Finally this question of using a flash dryer for environment purpose is questionable. We understand that a flash dryer has zero CO_2 emission but flash dryers are very expensive and the CO_2 emission from the stack of a propane burner rotary kiln will probably be a fraction of the CO_2 emissions coming from the mining machinery (trucks, shovels, loaders, drills, etc).

25.6 Project Continuity

SGS recommend to Mine Arnaud to move forward to the next step in the development of its phosphate project with completion of the recommendation of additional works.

26. Recommendations

SGS Geostat emits the following recommendations:

Geological

- Use present drilling data in order to establish and position the structural breaks of the deposit and includes them in the next resources modeling of the mineralized envelops;
- Continue to work towards developing a prediction model for Cl in the concentrate that can be interpolated in the block model. This could be achieved through detailed mineralogical and geochemical study (chemical analysis, mineralogical mapping and mineralogical chemistry) of feed, reject and concentrate material;
- Analysis for chlorine in the wedge area in order to increase the level of confidence in the resources of this zone to the measured category. Re-assay chlorine in hole 1166-10-83;
- Conduct further metallurgical testing in order to solidify the WRec equation used to calculate the WRec of each block of the model, therefore increase reliability of the pit optimization, which in turn is based on WRec values.

Processing

- SGS Geostat still has some concerns about the effect of the ore variations on the mill performance. Since it is out of question to do any blending after the crushing stage, SGS Geostat recommends mitigating the effect of the ore variations by planning for proper blending directly at the mine and leaving sufficient equipment and space for material handling of the ore before crushing.
- SGS backed by Roche-Ausenco recommends that an extensive laboratory program be put in place in order to study the mineralogy and liberation grain size of the apatite for various location and depth. At the same time, flotation tests should also be scheduled to study the impact of these variations on reagent consumptions and quality of the concentrate. The program should allow for studying some other unstudied environmentally friendly reagents. It will be important to study how the variability of minerals other than the magnetite has an impact on flotation kinetics.
- Because all the pilot plant campaigns done so far on the Sept-Îles ore were from surface weathered samples, SGS Geostat recommends conducting at least one additional pilot plant run with fresh large diameter drill core samples. Last laboratory testwork (COREM's Project T1518) has shown that flotation wise, surface medium grade weathered ore is quite different from fresh medium grade ore.
- This additional pilot plant run would also permit to test both columns and mechanical cells at the scavenger and scavenger/cleaner stages and close this debate once and for all.

- Filter press tests by manufacturers should be conducted in order to make a final judgment on this matter. A trade of study comparing OPEX and CAPEX for the belt filter and the pressure filter options is required. This trade of study should also consider the impact of both options, on size and energy consumption for the operation of the flash dryer.
- The option of using a propane burner rotary kiln instead of an electric flash dryer should also be better investigated. The CO_2 Solutions **"Enzymatic Carbon Capture Technology"** may provide a cheap answer to the propane dryer exhaust problem.
- A better understanding of the magnetite distribution will have to be done for two purposes: to fully assess its potential economical value and to evaluate the variability of the magnetite distribution within the deposit to see how blending requirement could be achieved.

Mining and economical

- Re-do a detailed pit design using the latest optimization results;
- Assess various mine plans in order to maximise the Project profitability;
- Assess various high-grading scenarios in order to maximise the Project profitability;
- Analyse the pros and cons of using bigger mining equipment in order to lower mining cost versus increasing mining dilution;
- Once the final mine plan will be developed, use specific software to precisely calculate the required mining fleet;
- Review the mining advance into the South-East section of the pit in the overburden area to assess any possible geotechnical problematic attributable to the developed mining phases;
- Precisely recalculate mining costs attributable to the variation of the height of the bench faces (5m and 10m) and include the results into the economical analysis;
- With future investigations, lower the amount attributable to the contingency by detailing more precisely the Project capital expenditure, sustaining capital, required working capital, etc.

Others

It should be understood that the Arnaud Project, as almost all mining projects, is subject to a series of risks. Habitually, the majors risks associated to a mining project are divided in terms of the following categories:

- Political risk;
- Social acceptability risk;
- Technology and technical risk;
- Risk related to meeting the targeted project schedule;

- Risk affecting CAPEX;
- Risk affecting OPEX;
- Risk affecting Profitability;
- Risk affecting Closure Costs.

It is in SGS opinion that the next Feasibility Study should include a section discussing the major risks associated with this Project accompanied by well-defined mitigation plans or actions.

Estimation of budget for further work

Mine Arnaud provided information concerning their budget for the next phases of development of the Sept-Îles deposit. SGS, using the provided budget, established to future work cost estimate as following:

Table 26-1: Future Work Cost Estimate

	Item	Amount	Source
Geology	Condemnation drilling		*Mine Arnaud , SGS Canada In. &Ccontractors*
	Define structures in pit		*SGS Canada Inc.*
	Cl prediction tool		*SGS Canada Inc.*
	Re-assay Cl on hole 1166-10-83		*Commercial lab*
	Solidify Wrec		*SGS Canada Inc.*
	Magnetite distribution	NA	*Mine Arnaud in house project*
	Pit Slope and OVB stability		*SGS Canada Inc. & Golminds*
	OVB Instrumentation		*S Canada Inc. & Golminds*
Processing	In pit blending parameters	NA	*Mine Arnaud in house project*
	45 tonnes pilot plant		*SGS Canada Inc.*
	Lock Cycle Test on blends		*SGS Canada Inc.*
	Mineralogy and liberation size sutdy		*SGS Canada Inc.*
	Filter press test		*SGS Canada Inc.*
	Kiln vs flash dryer	NA	*Will be evaluated in EPCM*
	SPI and BWI caracterization		*SGS Canada Inc.*
Environmental	Investigation of propane burner		*SGS Canada Inc.*
Mining & economical	Optimise design & planning	NA	*Mine Arnaud in house project*
	Review equipment fleet	NA	*Will be evaluated in EPCM*
	Confirm wall stability of Ovb	NA	*Mine Arnaud in house project with consultant*
Feasibility	Technical Report		
Sub-total			
Contingencies	Contingencies 25%		
TOTAL			

27. References

- All appendix and annexes related to: DUCHESNE, R. et Al., Mine Arnaud – Feasibility Study Final Report, Roche and Ausenco, Project no 121848/59858, Document no 1848-01-RE-GE-001, February 2012.
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- COREM, *Mine Arnaud - Laboratory Testing Program for Apatite Flotation, Project T1224*, March 27,2012
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- COTNOIR, D., *Bench Scale Flottation Locked Cycle Tests on Selected Samples and Blends*, T-1518, COREM, July 12th 2013.
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- G MINING SERVICES INC., *Highlight of Mine planning and financial modeling exercise*, December 2011.
- PRESSACCO, R., *Chlorine Status Report, Mine Arnaud Project, RPA, Mémoramdum*, Document no 1848-02-ME-GM-024, September 4th, 2011.

CERTIFICATE OF QUALIFICATION

JEAN-PHILIPPE PAIEMENT, M.Sc., P.Geo
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I, Jean-Philippe Paiement, M.Sc., P.Geo., do hereby certify:

a) I am a Geologist with SGS Canada Inc. - Geostat with an office at 10 Boul. de la Seigneurie Est, Suite 203, Blainville, Quebec, Canada, J7C 3V5.

b) This certificate applies to the Technical Report entitled Pre-Feasibility Study (PFS) Sept-Îles Deposit, Mine Arnaud Inc., Quebec, dated July 24, 2013, (the "Technical Report")

c) I am a graduate of the Université du Québec à Montréal (B.Sc. Resources Geology, in 2006) and Université Laval (M.Sc. Economic Geology, in 2009). I am a member of good standing, No. 1410, of the l'Ordre des Géologues du Québec. My relevant experience includes working as a mine geologist for a base metal mining company and working as a consulting geologist to estimate the resources of various mining projects. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").

d) I have visited the Chibougamau facilities in November of 2012.

e) I am responsible for Sections 4, 5, 6, 7, 8, 9, 10, 11, 12, 14 and I have participated in the development of Sections: 1, 2, 3, 25, 26 and 27 of the Technical Report.

f) I am independent of Mine Arnaud Inc., as defined by Section 1.5 of the Instrument.

g) I have no prior involvement with the property that is the subject of the Technical Report.

h) I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.

i) As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report that I am responsible for, contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 15th day of August 2013, at Blainville, Quebec.

"Original document signed and sealed
by Jean-Philippe Paiement, M.Sc., P.Geo."
Jean-Philippe Paiement, M.Sc., P.Geo.
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CERTIFICATE OF QUALIFICATION

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I, Jonathan Gagné, Eng., do hereby certify:

a) I am a Mining Engineer with SGS Canada Inc. - Geostat with an office at 10 Boul. de la Seigneurie Est, Suite 203, Blainville, Quebec, Canada, J7C 3V5.

b) This certificate applies to the Technical Report entitled Pre-Feasibility Study (PFS) Sept-Îles Deposit, Mine Arnaud Inc, Quebec, dated July 24, 2013, (the "Technical Report")

c) I am a graduate of the École Polytechnique de Montréal (B.Sc. Mining Engineer, in 2007). I am a member of good standing, No. 146075, of the l'Ordre des Ingénieurs du Québec (Order of Engineers of Quebec). My relevant experience includes working as a mine planning engineer for a gold mining company and working as a consulting engineer to evaluate the potential of various mining projects. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").

d) I have visited the property in October of 2012.

e) I am responsible for Sections 15, 16, 18, 19, 21, 22, 25 and 26, and I have supervised the development of Sections: 1, 2 and 3 of the Technical Report.

f) I am independent of Mine Arnaud Inc., as defined by Section 1.5 of the Instrument.

g) I have no prior involvement with the property that is the subject of the Technical Report.

h) I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.

i) As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report that I am responsible for, contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 15th day of August 2013, at Blainville, Quebec.

"Original document signed and sealed by Jonathan Gagné, Eng., MBA"
Jonathan Gagné, Eng., MBA
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CERTIFICATE OF QUALIFICATION

GILBERT ROUSSEAU, Eng.

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I, Gilbert Rousseau B Sc.A, Eng., of Ville de Saguenay, Province of Quebec, do hereby certify:

a) I am a Senior Mining-Metallurgical Engineer with SGS Canada Inc., with a business address at 10 Boul. de la Seigneurie, Blainville, Quebec, J7C 3V5.

b) This certificate applies to the Technical Report entitled Pre-Feasibility Study (PFS) Sept-Îles Deposit, Quebec, Mine Arnaud Inc., dated July 24, 2013, (the "Technical Report")

c) I graduated from The Ecole Polytechnique of the University of Montreal (B.Sc.A, Mining Engineer in 1969). I am a member in good standing of the "l'Ordre des Ingénieurs du Québec" #20288). My relevant experience includes more than 40 years of experience in the mining and milling of minerals including iron, copper, lead, zinc, silver, gold, asbestos, graphite, nickel, silica, etc. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").

d) I did not visit the property.

e) I am responsible for Sections 13 and 17 of the Technical Report, and I have collaborated for the preparation of Sections 1, 2, 3, 25 and 26 of the Technical Report.

f) I am independent of Mine Arnaud Inc., as defined by Section 1.5 of the Instrument.

g) I have no prior involvement with the property that is the subject of the Technical Report.

h) I have read the Instrument and the sections of the report that I am responsible. These sections have been prepared in compliance with the Instrument.

i) As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 15th day of August 2013 at Blainville, Quebec.

"Original document signed and sealed
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CERTIFICATE OF QUALIFICATION

CLAUDE DUPLESSIS, Eng.
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I, Claude Duplessis, Eng., do hereby certify:

a) I am a Consultant in Geological Engineer with SGS Canada Inc. - Geostat with an office at 10 Boul. de la Seigneurie Est, Suite 203, Blainville, Quebec, Canada, J7C 3V5.

b) This certificate applies to the Technical Report entitled Pre-Feasibility Study (PFS) Sept-Îles Deposit, Mine Arnaud Inc., Quebec, dated July 24, 2013, (the "Technical Report")

c) I am a graduate of the University of Chicoutimi, Quebec, (B.Sc. Geological Engineering, in 2007). I am a member of good standing, No. 45523, of the l'Ordre des Ingénieurs du Québec (Order of Engineers of Quebec). I am also a registered engineer in the province of Alberta and Newfoundland & Labrador. My relevant experience includes 19 years in the field of Mineral Resource estimation, orebody modelling, mineral resource auditing and geotechnical engineering. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").

d) I have visited the property in October of 2012.

e) I am responsible for Sections 2, 3, 5, 6, 7, 8, 9, 10, 11, and I have supervised the development of Sections: 1, 4, 12, 14, 23, 25 and 26 of the Technical Report.

f) I am independent of Mine Arnaud Inc., as defined by Section 1.5 of the Instrument.

g) I have no prior involvement with the property that is the subject of the Technical Report.

h) I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.

i) As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report that I am responsible for, contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 15th day of August 2013, at Blainville, Quebec.

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Claude Duplessis, Eng.
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CERTIFICATE OF QUALIFICATION

Gaston Gagnon, Eng.
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I, Gaston Gagnon, Eng. of Saint-Eustache, Quebec, do hereby certify:

a) I am Senior Mining Engineer with SGS Canada Inc. - Geostat with an office at 10 Boul. de la Seigneurie Est, Suite 203, Blainville, Quebec, Canada, J7C 3V5.

b) This certificate applies to the technical report entitled Pre-Feasibility Study (PFS) Sept-Îles Deposit, Mine Arnaud Inc., Quebec, dated July 24, 2013, (the “Technical Report”)

c) I am a graduate of the University of Laval in Quebec City (B.Sc. Mining Engineering, 1964). I am a member of good standing (#15918) of the l’Ordre des Ingénieurs du Québec (Order of Engineers of Quebec). My relevant experience includes over 40 years of experience in mining minerals in underground and surface producers, processing mainly gold, silver, copper, zinc, aggregates and niobium. Experience also includes 5 years of consulting for several mining projects under development. EPCM experience covers scoping (now PEA) studies and prefeasibility studies, detailed economic estimation and construction management in Canada, Africa, Mexico, South America and Saudi Arabia. I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”).

d) I have not visited the property.

e) I have collaborated to the preparation of Sections 1, 2, 3, 16, 18, 21, 22, 25 and 26 of the Technical Report.

f) I am independent of Commerce Resources Corporation as defined by Section 1.5 of the Instrument.

g) I have no prior involvement with the property that is the subject of the Technical Report.

h) I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.

i) As of the effective date of the Technical Report, to the best of the qualified person’s knowledge, information, and belief, the Technical Report, or part that the qualified person is responsible for, contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 15th day of August 2013 at Blainville, Quebec.

“Original document signed and sealed
by Gaston Gagnon, Eng”
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CERTIFICATE OF QUALIFICATION

MICHEL DAGBERT, Eng.
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I, Michel Dagbert, Eng., do hereby certify:

a) I am Senior Geostatistician with SGS Canada Inc. - Geostat with an office at 10 Boul. de la Seigneurie Est, Suite 203, Blainville, Quebec, Canada, J7C 3V5.

b) This certificate applies to the Technical Report entitled Pre-Feasibility Study (PFS) Sept-Îles Deposit, Quebec, Mine Arnaud Inc., dated July 24, 2013, (the "Technical Report")

c) I am a graduate from Paris School of Mines in 1971, and McGill University in 1972. I have worked as a geostatistician continuously since my graduation from University. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").

d) I have not visited the property.

e) I have collaborated to the preparation of Sections 12 and 14 of the Technical Report.

f) I am independent of Mine Arnaud Inc., as defined by Section 1.5 of the Instrument.

g) I have no prior involvement with the property that is the subject of the Technical Report.

h) I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.

i) As of the effective date of the Technical Report, to the best of the qualified person's knowledge, information, and belief, the Technical Report parts that I am responsible for, contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 15th day of August 2013, at Blainville, Quebec.

"Original document signed and sealed
by Michel Dagbert, Eng"
Michel Dagbert. Eng
Senior Geostatistician Engineer
SGS Canada Inc. - Geostat

APPENDIX 1: CLAIM LIST

APPENDIX 2: YARA CONCENTRATE CHARACTERISTIC LETTER

APPENDIX 3: CHLORINE REPORT

APPENDIX 4: PROCESS AND METALLURGICAL BALANCE

APPENDIX 5: PROCESSING FACILITIES

APPENDIX 6: FLOWSHEET