Investissement Québec

Marc Paquet, *LL. M., MBA*
Vice-président
Affaires juridiques et secrétaire de la Société

Montréal, le 28 juillet 2016

Objet : Votre demande d'accès du 28 juin 2016 (Projet Mine Arnaud – Décret du 1er août 2014 - «MINE ARNAUD Feasibility Study – NI 43-101 – Mine Arnaud inc. – Sept-Îles Deposit, Québec – Appendices A to P, par SGS Canada inc., 1er août 2014, totalisant environ 314 pages, and Appendices Q to AA, totalisant environ 291 pages»)

Nous faisons suite à votre demande d'accès formulée en vertu de la *Loi sur l'accès aux documents des organismes publics et sur la protection des renseignements personnels* (RLRQ, chapitre A-2.1) (ci-après, la «Loi sur l'accès») datée du 28 juin 2016, reçue, par courriel, à nos bureaux le même jour, dont copie est jointe en annexe, et à notre accusé de réception et avis de prolongation de délai de traitement datés du 11 juillet 2016.

Après vérification, nous ne pouvons vous divulguer les documents demandés et invoquons à cet égard, comme applicables en l'espèce, les articles 20, 21, 22, 23, 24, 27, 37 et 39 de la Loi sur l'accès.

En terminant, à titre d'information, nous vous référons à l'article 135 de la Loi

> *«135. Une personne dont la demande écrite a été refusée en tout ou en partie par le responsable de l'accès aux documents ou de la protection des renseignements personnels peut demander à la Commission de réviser cette décision.*
>
> *Une personne qui a fait une demande en vertu de la présente loi peut demander à la Commission de réviser toute décision du responsable sur le délai de traitement de la demande, sur le mode d'accès à un document ou à un renseignement, sur l'application de l'article 9 ou sur les frais exigibles.*
>
> *Ces demandes doivent être faites dans les trente jours qui suivent la date de la décision ou de l'expiration du délai accordé par la présente loi au responsable pour répondre à une demande. La Commission peut toutefois, pour un motif raisonnable, relever le requérant du défaut de respecter ce délai.»*

…/2

600, rue de La Gauchetière Ouest, bureau 1500, Montréal (Québec) H3B 4L8 CANADA
514 876-9339 1 866 870-0437 Télécopieur : 514 876-9306
marc.paquet@invest-quebec.com investquebec.com

Nous vous prions d'agréer l'expression de nos sentiments les meilleurs.

Le responsable de l'accès aux documents,

ORIGINAL SIGNÉ

Marc Paquet, avocat
Vice-président, Affaires juridiques et secrétaire de la Société

p.j. Votre demande d'accès; et articles 20, 21, 22, 23, 24, 27, 37 et 39 de la Loi sur l'accès.

AVIS AU LECTEUR

Expéditeur : **Le responsable de l'accès aux documents et de la protection des renseignements personnels**

Date : **Décembre 2018**

Objet : **Étude de faisabilité Mine Arnaud**

Le document ci-joint a été remis à la demanderesse d'accès dans le cadre d'une audience tenue devant la Commission d'accès à l'information en 2018.

Il représente certaines parties des documents demandés.

Feasibility Study
NI 43-101
Mine Arnaud Inc.
Sept-Îles Deposit, Québec
<u>Final Report</u>

Respectfully submitted to:
Mine Arnaud Inc.

Effective Date:
August 1st, 2014

Prepared by:

Jean-Philippe Paiement, M.Sc., P.Geo — *SGS Canada Inc*
Jonathan Gagné, Eng, MBA — *SGS Canada Inc*
Gaston Gagnon, Eng. — *SGS Canada Inc*
Simon Latulippe, Eng. — *Groupe WSP Global Inc*
Alain Dorval, Eng. — *Roche ltée*
Noel Journeaux, Eng. — *Journeaux Associates*
Rémi Duchesne, Eng. — *Ausenco*

DATE AND SIGNATURE PAGE

The effective date of the Technical Report NI 43-101 on the Mine Arnaud project, Sept-Iles, Quebec, Canada is August 01, 2014.

Prepared by:

August 15, 2014

Jean-Philippe Paiement, M.Sc., P.Geo.

August 15, 2014

Jonathan Gagné, Eng., MBA

August 15, 2014

Gaston Gagnon, Eng.

August 15, 2014

Simon Latulippe, Eng.

August 15, 2014

Alain Dorval, Eng.

August 15, 2014

Noel Journeaux, Eng.

August 15, 2014

Rémi Duchesne, Eng.

CERTIFICATE OF QUALIFICATION

Jean-Philippe Paiement, M.Sc. P.Geo

Jp.paiement@sgs.com

I, Jean-Philippe Paiement, M.Sc., P.Geo, Québec, Québec, do hereby certify:

a) I am a resources geologist at SGS Canada Inc., 10 boul de la Seigneurie, suite 203, Blainville, Qc, J7C 3V5.

b) This certificate applies to the technical report entitled Feasibility Study (FS) Sept-Îles Deposit, Mine Arnaud Inc., Quebec, with an effective date of August 01, 2014, (the "Technical Report")

c) I have a B.Sc. degree in resources geology from the Université du Québec à Montréal and I received a M.Sc. in 2009 from Université Laval. I have worked on several technical reports regarding resource estimations and exploration. I am a registered member of the Ordre des Géologues du Québec (#1410). I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument");

d) I have visited the Chibougamau store facilities, but not the Sept-Îles site.

e) I am responsible of Sections 4, 5, 6, 7, 8, 9, 10, 11, 12, 14 and 23. I have collaborated to the preparation of the 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 prior involvement in the preparation of the 2013 PFS study of this 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 2014 at Québec, Québec.

"Original document signed and sealed

By Jean-Philippe Paiement, M.Sc., P.Geo"

Jean-Philippe Paiement, M.Sc., P.Geo

Resources Geologist

SGS Canada Inc.

CERTIFICATE OF QUALIFICATION

Jonathan Gagné, Eng., MBA

jonathan.gagne@sgs.com

I, Jonathan Gagné, Eng., MBA, Blainville, Quebec, do hereby certify:

a) I am a Mining Engineer at SGS Canada Inc., 10 boul de la Seigneurie, suite 203, Blainville, Qc, J7C 3V5.

b) This certificate applies to the Technical Report entitled Feasibility Study (FS) Sept-Îles Deposit, Mine Arnaud Inc., Quebec, with an effective date of August 01, 2014, (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 of Sections 15, 16,and 22. I have collaborated to the preparation of Sections 1, 18, 21, 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 prior involvement in the preparation of the 2013 PFS study of this 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 2014 at Blainville, Quebec.

“Original document signed and sealed

By Jonathan Gagné, Eng., MBA”

Jonathan Gagné, Eng., MBA

Mining Engineer

SGS Canada Inc.

CERTIFICATE OF QUALIFICATION

Gaston Gagnon, Eng.

Gaston.Gagnon@sgs.com

I, Gaston Gagnon, Eng. of Saint-Eustache, Quebec, do hereby certify:

a) I am a 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 Feasibility Study (FS) Sept-Îles Deposit, Mine Arnaud Inc., Quebec, with an effective date of August 01, 2014, (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 am responsible of the coordination of the whole Technical Report and responsible for the preparation of Section 19, 21.2.1, 21.2.2 and 21.4. I have collaborated to the preparation of Sections 1, 2, 3, 21, 22, 24, 25, 26 and 27 of the Technical Report.

f) I am independent of Mines Arnaud Inc. as defined by Section 1.5 of the Instrument.

g) I have had prior involvement in collaborating to the preparation of Sections 1, 2, 3, 16, 18, 21, 22, 25 and 26 of the 2013 Technical Report entitled Pre-Feasibility Study (PFS) Sept-Îles Deposit, Mine Arnaud Inc.

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 Technical Report, or part 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 2014 at Blainville, Quebec.

“Original document signed and sealed

by Gaston Gagnon, Eng.”

Gaston Gagnon, Eng.

Senior Mining Engineer

SGS Canada Inc. - Geostat

CERTIFICATE OF QUALIFICATION

Simon Latulippe, Eng.

Simon.Latulippe@wspgroup.com

I, Simon Latulippe, Eng, Quebec, Quebec, do hereby certify:

a) I am Project manager at WSP, 5355, boul. des Gradins, Québec, (Québec) G2J 1C8.

b) This certificate applies to the technical report entitled Feasibility Study (FS) Sept-Îles Deposit, Mine Arnaud Inc., Quebec, with an effective date of August 01, 2014, (the “Technical Report”).

c) I graduated with a Bachelor’s degree in Geological Engineering from Laval University, Quebec, Canada in 1998. I am a member of good standing (#121692) of the l’Ordre des Ingénieurs du Québec (Order of Engineers of Quebec). My relevant experience includes over I have worked as a geological engineer on a continuous basis for 15 years since my graduation from university, mainly providing services to the environment industry as a consultant. My relevant experience for the purpose of the Technical Report includes environmental studies as project manager, characterization and monitoring, soil rehabilitation and groundwater treatment, site tests and design, Mine site reclamation projects as water management and tailings leader, Mine Closure plans design, Mine projects permitting leader, Mine project water management and tailings design. I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”). I have read the definition of “qualified person” set out in National Instrument 43-101 (“NI 43-101”) and certify that by reason of education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a “qualified person” for the purpose of NI 43-101.

d) I have visited the property on May 22, 2014.

e) I am responsible or/and I have collaborated to the preparation of Sections 16.11.1, 18.4, 20, 21, 21.2.4, 21.2.5, 26.10, 26.11, 26.12 Of the Technical Report.

f) I am independent of Mine Arnaud Inc. as defined by Section 1.5 of the Instrument.

g) I don’t have 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 2014 at Quebec, Quebec.

“Original document signed and sealed

by Simon Latulippe, Eng.”

Simon Latulippe, Titre (Eng)

Project Manager

WSP

CERTIFICATE OF QUALIFICATION

Alain Dorval, Eng.

alain.dorval@roche.ca

I, Alain Dorval, Eng. of Montreal, Quebec, do hereby certify:

a) I am Manager Mining and Mineral Processing at Roche Ltd. Consulting Group. with an office at 33 St-Jacques, 2nd floor, Montréal, Québec, Canada, H2Y1K9.

b) This certificate applies to the technical report entitled Feasibility Study (FS) Sept-Îles Deposit, Mine Arnaud Inc., Quebec, with an effective date of August 01, 2014, (the “Technical Report”)

c) I am a graduate of the Laval University (Mining Engineering 1983). I am a member of good standing (#961229) of the l’Ordre des Ingénieurs du Québec (Order of Engineers of Quebec). My relevant experience includes over 31 years related to the mineral processing including industrial minerals, precious metals and base metals. My experience includes mainly: mining operation, research process development for various minerals, consulting and engineering. I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”).

d) I have not visited the property.

e) I am responsible of the Sections 13, 17, 21.3.1 to 21.3.3, 21.5 and 21.6. I have collaborated to the preparation of Sections 18 and 21 of the Technical Report.

f) I am independent of Mine Arnaud Inc. as defined by Section 1.5 of the Instrument.

g) I have prior involvement with the property that is the subject of the Technical Report. I was involved with the study of the project in 2010-2012. I participated in the past to various pilot plant testing programs and laboratory scale testing programs for the Mine Arnaud project

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 2014 at Montréal, Quebec.

“Original document signed and sealed

by Alain Dorval, Eng.”

Alain Dorval, Eng.

Manager, Mining and Mineral Processing,

Roche Ltd, Consulting Group.

CERTIFICATE OF QUALIFICATION

Noel L. Journeaux, Eng., P. Geo., P. Eng., M.S.C.E., F-A.S.C.E.

njourneaux@noeljourneaux.com

I, Noel L. Journeaux, Eng., resident of Beaconsfield, Quebec, do hereby certify:

a) I am currently president in the consulting firm Journeaux Assoc. Division of Lab Journeaux Inc. in an office located at 801 Bancroft, Pointe-Claire, Quebec.

b) This certificate applies to the technical report entitled Feasibility Study (FS) Sept-Îles Deposit, Mine Arnaud Inc., Quebec, with an effective date of August 01, 2014, (the “Technical Report”).

c) I am a graduate of Purdue University, West Lafayette, IN, USA with a M.S.C.E. degree in Civil Engineering in 1962. I am a member of good standing (#14341) of the l’Ordre des Ingénieurs du Québec (Order of Engineers of Quebec). My relevant experience includes over 35 years in consulting practice related to geotechnical engineering, rock mechanics and stability of rock slopes. I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”).

d) I have visited the property on March 10-17, 2011.

e) I am responsible or/and I have collaborated to the preparation of Sections 1.15.1, 1.15.2, 18.2, 18.3.1, 18.3.2, 18.3.5, 18.3.5, 18.3.7, 18.3.8, 21 and 21.2.3 of the Technical Report.

f) I am independent of Mine Arnaud Inc. as defined by Section 1.5 of the Instrument.

g) I don’t have 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 15^{th} day of August 2014 at Pointe-Claire, Quebec.

Noel L. Journeaux, Eng., P. Geo., P. Eng., M.S.C.E., F-A.S.C.E.

President

Journeaux Associates

Division of Lab. Journeaux Inc.

CERTIFICATE OF QUALIFICATION

Rémi Duchesne, Eng.

remi.duchesne@ausenco.com

I, Rémi Duchesne, Eng. of Sainte-Anne-de-Bellevue, Quebec, do hereby certify:

a) I am Client Solutions Manager with Ausenco Engineering Canada Inc. with an office at 555 René-Lévesque Blvd West, Suite 200, Montréal, Québec, Canada, H2Z 1B1.

b) This certificate applies to the technical report entitled Feasibility Study (FS) Sept-Îles Deposit, Mine Arnaud Inc., Quebec, with an effective date of August 01, 2014, (the “Technical Report”)

c) I am a graduate of the Laval University (Mechanical Engineering 1985). I am a member of good standing (#42099) of the l’Ordre des Ingénieurs du Québec (Order of Engineers of Quebec). My relevant experience includes over 28 years related to the engineering of industrial projects including over 5 years on mining projects. My experience includes mainly mining infrastructure such as materials handling systems, rail and port facilities. I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”).

d) I have visited the property for 2 days during the study of the project in June 2010.

e) I have collaborated to the preparation of Sections 18 and 21 of the Technical Report.

f) I am independent of Mine Arnaud as defined by Section 1.5 of the Instrument.

g) I have prior involvement with the property that is the subject of the Technical Report. I was involved with the study of the project in 2010-2012.

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 2014 at Montréal, Quebec.

“Original document signed and sealed

by Rémi Duchesne, Eng.”

Rémi Duchesne, Eng.

Client Solutions Manager

Ausenco Engineering Canada Inc.

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1. Summary

1.1 Introduction

In April 2014, Mine Arnaud Inc. commissioned SGS Canada Inc. – Geostat (“SGS Geostat”) to provide an independent Qualified Person’s (QP) review and NI 43-101 Technical Report for the Sept-Îles apatite deposit project.

The study was prepared by SGS Geostat with contribution from the following consulting companies: Roche ltée (Roche), Journeaux Associates (Journeaux), Groupe WSP Global Inc (WSP) and Ausenco. The economical modeling and economic analysis was prepared in collaboration with Pricewaterhouse Coopers LLC (PWC). Integer Research Limited from London was mandated by Mine Arnaud to prepare a market report for phosphate rock supply, demand and pricing outlook to 2023.

The project has an expected life of 31 years, excluding the pre-production phases and is located next to the city of Sept-Îles. The project is anticipated to process 362 million tonnes of ore and producing 33.8 million tonnes of high grade apatite concentrate (39%-40% P_2O_5).

1.2 Property Description, Location and Accessibility

Mine Arnaud’s property is located in the Québec North Shore region about 650 km Northeast of Québec City and 900 km Northeast of Montréal. The closest cities are Sept-Îles (15 km to the East with a population of 27,600) and Port-Cartier (40 km to the West). The property comprises 303 contiguous claims covering an area of 9,842 ha. The property is 100% held by Mine Arnaud, which is a partnership between two owners; Investissement Québec (IQ), a Quebec Government Corporation and Yara International, ASA (Yara) (public limited company) from Norway. The project is owned at 62% by IQ and 38% by Yara (the buyer of all the concentrate to be produced during the mine’s life). The property hosts an apatite deposit comprising significant phosphate (P_2O_5) resources. The project has excellent access to infrastructure and work force due to its proximity to Sept-Îles.

The property is limited to the Sept-Îles Bay in the St-Lawrence River and, to the North by an important Hydro-Quebec electric pylon corridor. Arnaud Railway (operated by Wabush Mines - a subsidiary of Cliffs Natural Resources), which is connecting Arnaud Junction to Pointe-Noire, runs through the property in a general East-West direction.

The site was visited in October 2012 by Jonathan Gagné, Eng, MBA, accompanied by Claude Duplessis, Eng who was then an SGS employee and is now acting as a Consultant for SGS Geostat.

1.3 History, Geological Settings, Mineralization and Deposit Types

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 within gabbroic rocks (Nelsonite Layer). Stratigraphically above the Nelsonite horizon, ilmenite-magnetite-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% (stated by 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 Exploration and Drilling

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 drilling 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.

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 10.80% P_2O_5. A two phase exploration program was undertaken on the deposit 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. SOQUEM conducted several drilling programs between 1996 and 2008, in order to estimate mineral resources.

2010 Exploration and Drilling

Recent drilling resumed in 2010, under Mine Arnaud, to support a feasibility-level study of the Sept-Îles property. Holes were proposed on a 100 m by 100 m drill centers as per Genivar Inc.'s 2008 (Genivar) 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 (in 2011 by RPA) using the ISD method for a total of 105.30 Mt of measured and indicated resources at an average grade of 5.32% P_2O_5, and 157.37 Mt of inferred resources at an average grade of 4.66% P_2O_5. The cut-off grade used to report the mineral resources was 2.60% P_2O_5. No mineral reserves were identified in this Feasibility Study (FS) due to the insufficient drilling grid.

Following historic drilling and mineral resources estimation in 2011, Mine Arnaud conducted further drilling on the property.

2012 Exploration and Drilling

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

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 m 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.

2013 Exploration and Drilling

In 2013, a 9 holes drilling program was conducted by Mine Arnaud, managed by InnovExplo. The total length of the drilling campaign was 1,041 m and focused primarily on increasing the level of confidence of the mineral resources in the "wedge" area and northern limit of the deposit. 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.

1.5 Resource Estimation Methodology

The mineral resource estimate has been conducted following the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definitions Standards for mineral resources in accordance with NI 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 refer to Item 14 of this report.

The mineral resources were estimated using kriging interpolation of blocks within Genesis© software. Each mineralized layer was domained separately, and variography studies were conducted on each layer independently. On the total 2,859,094 blocks inside the deposit limits, 2,537,166 blocks have been interpolated for the P_2O_5%, K_2O% and density values. Insitu Cl was interpolated for 2,771,847 blocks; most missing values for Cl are from 2010 Soquem DDH, 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%, as per stated by Roche. Predicted P_2O_5% value for the concentrate was calculated using the WRec values and predicted Cl% of the concentrate was calculated using a statistical model from insitu K_2O values.

1.6 Mineral Resource Estimate

An optimized pit shell scenario (comprising all categories of resources) 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 zones they are part of. The general cut off grade is of 1.65% P_2O_5 except for the blocks inside the Nelsonite solid where a cut off grade of 2.05% P_2O_5 is used due to the higher dilution factor caused by the thinner nature of the Nelsonite layer. The mineral resources are then reported following their classifications; inferred resources cannot be added to others.

Table 1-1: Mineral Resources Estimate

Category	Material Type	Cut Off ($\%P_2O_5$)	Tonnage (Mt)	Grade ($\%P_2O_5$)	WRec (%Wrec)	Conc. Grade ($\%P_2O_5$)	Conc. Grade (%Cl)
Measured	California	1.65	27.62	2.94	6.86	38.47	0.17
	Combine	1.65	319.17	4.01	9.25	38.89	0.12
	Surrounding	1.65	28.83	2.31	5.46	37.91	0.15
	Nelsonite	2.05	37.97	5.88	13.41	39.42	0.08
	TOTAL		413.58	4.00	9.21	38.84	0.12
Indicated	California	1.65	8.23	3.14	7.30	38.54	0.16
	Combine	1.65	89.47	4.29	9.87	38.98	0.12
	Surrounding	1.65	24.95	2.31	5.45	37.92	0.17
	Nelsonite	2.05	9.26	6.19	14.09	39.48	0.09
	TOTAL		131.91	3.98	9.17	38.79	0.13
Inferred	California	1.65	0.00	1.82	4.40	37.23	0.17
	Combine	1.65	-	-	-	-	-
	Surrounding	1.65	44.63	3.36	5.45	38.67	0.16
	Nelsonite	2.05	-	-	-	-	-
	TOTAL		44.63	3.36	5.45	38.66	0.16

Notes:

- The mineral resource estimate has been conducted following the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definitions Standards for mineral resources in accordance 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.
- SGS Geostat did the supposition that the diluted material will have the same %Cl that the ore blocks where the dilution is applied.
- Resources are constrained by the pit shell and the topography of the overburden layer.

1.7 Mineral Processing and Metallurgical Testing

The common comminution characterisation and small-scale continuous grinding tests via MacPherson tests were performed on weathered ore representing the various ore types which confirmed the applicability of SAG milling. In addition to the comminution characterisation, SMC tests were performed and the Bond Work index was determined for series of samples or composite from various locations in the deposit and representing the various ore types. The results added confidence that the variability in terms of grinding characteristics should not represent a problem for the operations. JK drop weight test results concluded that the samples were characterised as soft with respect to resistance to impact, but were hard with respect to abrasion breakage. The Bond Ball Mill Work Index (BWI) showed that the samples were categorized as moderately soft to medium with abrasion indices (Ai) in the medium to abrasive range. Considering that the resistance to impact of the ore was in the soft range of hardness, a pebble crusher was not included and only a SAG/Ball mill circuit was considered. Following the mineralogical observations and liberation study on the various ore types, a P_{80} of 125 µm was selected as the most appropriate grind size to obtain proper liberation. The addition of a Low Intensity Magnetic Separator (LIMS) prior to flotation has been considered based on mineralogical observations showing significant proportions of magnetite and titano-magnetite in different areas of the deposit. The use of a LIMS prior to flotation helps reduce the iron and aluminium (Fe + Al) content in the final flotation concentrate and helps reduce the depressant consumption and enhance the efficiency of the collector.

Flotation test results demonstrated that a flowsheet with one rougher, one scavenger stage, one cleaner stage, and one cleaner/scavenger stage was the most appropriate to meet the apatite concentrate specification. Testwork 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 concentrate specifications for impurity content. The addition of reagents must to be closely monitored and paired to the mineralogy of the ore being processed. The flotation parameters such as depressant dosage and sodium silicate dosage should be fine-tuned to the flotation feed and gangue matrix in order to reach the targeted grade/recovery of P_2O_5 concentrate.

Based on selected testwork results, a relation between the concentrate weight recovery and the P_2O_5 feed grade was determined. To correspond to a production of a 39% P_2O_5 concentrate grade, the relation between the P_2O_5 feed grade and flotation concentrate weight recovery was determined to be $\%Wrec = 2.2264 * \%P_2O_5 + 0.3146$. The coefficient of correlation associated with this equation is 0.9728.

The latest testwork was performed to evaluate the metallurgical performance of the ore that will feed the concentrator at the end of the mine life. It was performed on samples from deeper zone of the deposit. Chlorine (Cl) and Fe + AL specifications were met in most cases. The calcium phosphorus ratio (Ca/P < 2.2) specification was slightly exceeded but it is believed that this can be fixed by adjusting the reagent dosage. The purpose of these tests was not to optimize but to identify if any major processing issue was to be foreseen near the end of the mining operation. Therefore, no optimisation tests have been performed since nothing abnormal has been observed. The tests results confirm that there were no specific issues related to variability at depth of the deposit. Roche is convinced that if more lock cycle tests would have been performed or if a closed-circuit pilot plant would have been conducted with similar ore coming from depth, the P_2O_5 recovery would have increased and the Ca/P ratio would have been met. As previously mentioned, based on the laboratory test results, it is expected that the quality control specification could be achieved with some reagent adjustments and ore coming from the deeper zone will behave similarly to the rest of the deposit.

1.8 Mineral Reserve Estimates

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 to proven and probable reserves respectively. They are also constrained by the final pit limit and above the marginal cut-off grade. It should be noted that a small portion of the blocks that meet the criteria are excluded nevertheless. This exclusion (19.80 Mt) comes from blocks destined for the low grade stockpile that were discarded in order to lower the chlorine content of the low grade material that will be treated in the last years of operations (28 to 31).

Following the request from Mine Arnaud, the block model was adjusted for the reserves estimation. Due to the low grade nature and high chlorine 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.

The mineral reserve (with dilution and ore loss) is therefore equal to 342.60 Mt of ore at an average grade of 4.30 %P_2O_5 using cut-off grades related to the rock type (1.65 %P_2O_5 and 2.05 %P_2O_5) and represents an operation of 30.8 years from start of production. The entire reserve comprises 33.87 Mt of apatite concentrate grading 39 %P_2O_5 and having a chlorine content of 0.1136 %. Total waste, including rock, inferred resources and overburden, is 243.57 Mt. The detailed mineral reserve estimate is shown in next table.

Table 1-2: Mine Arnaud Project Reserves (presented as mill feed)

Material Type		Cut-off (%P_2O_5)	Tonnes	Grade (%P_2O_5)	Grade (%Wrec)	Concentrate tonnes	Chlorine (%Cl)
Ore (Probable Reserves)	Combine	1.65	55,980,000	4.61	10.59	5,930,000	0.1176
	Surrounding	1.65	2,780,000	2.22	5.26	150,000	0.1451
	Nelsonite	2.05	9,660,000	5.24	11.99	1,160,000	0.0861
	Total		68,420,000	4.61	10.57	7,230,000	0.1131
Ore (Proven Reserves)	Combine	1.65	234,070,000	4.20	9.66	22,620,000	0.1189
	Surrounding	1.65	10,290,000	2.31	5.46	560,000	0.1216
	Nelsonite	2.05	29,820,000	5.06	11.59	3,460,000	0.0783
	Total		274,180,000	4.22	9.71	26,640,000	0.1137
Ore (Total Reserves)	Combine	1.65	290,050,000	4.28	9.84	28,540,000	0.1186
	Surrounding	1.65	13,070,000	2.29	5.42	710,000	0.1264
	Nelsonite	2.05	39,480,000	5.11	11.69	4,610,000	0.0803
	Total		342,600,000	4.30	9.89	33,870,000	0.1136
Waste	Waste rock		179,630,000				
	Overburden		63,940,000				
In-pit Total	**All**		586,170,000				

Note: This reserve includes 3.39 % dilution at 1.28 %P_2O_5 and 2.37 % ore loss for Combine and Surrounding ore types and a 25.89 % dilution at 0.89 %P_2O_5 and 4.35 % ore loss for the Nelsonite ore type. Also, SGS Geostat did the supposition that the diluted material will have the same %Cl that the ore blocks where the dilution is applied.

1.9 Mining Methods

<u>Open-pit mining</u>

Taking into account the geometry and the depth of the mineralized deposit, only open-pit mining method has been considered in this feasibility study. The near surface reserves will be mined by a large open pit, which will have 31 years of production following two years of construction and pre-production. Mining operations will be conducted by Mine Arnaud staff. Surface mining will follow the standard practice of an open-pit operation with conventional drill and blast, load and haul cycle using drills, trucks and shovel equipment. This main mining fleet is comprised of rotary tower drills, hydraulic excavators of 15 m^3 bucket capacity, 140 tonne trucks for ore and waste, and the overburden will be removed using 40 tonne articulated trucks. Overburden and waste rock material will be hauled to the dedicated disposals areas near the pit. The ore will be delivered to the gyratory crusher or stockpiled near the crushing plant.

A life-of-mine scenario was developed over 365 days per year. The mine plan is calling for a yearly nominal capacity production of 11,282,880 tonnes processed, and an average stripping ratio of 0.71 over twenty-seven years of mining operation, plus four years of stockpile processing. The following graph is summarizing the ore and waste production and the overburden removal, over the project life-of- mine.

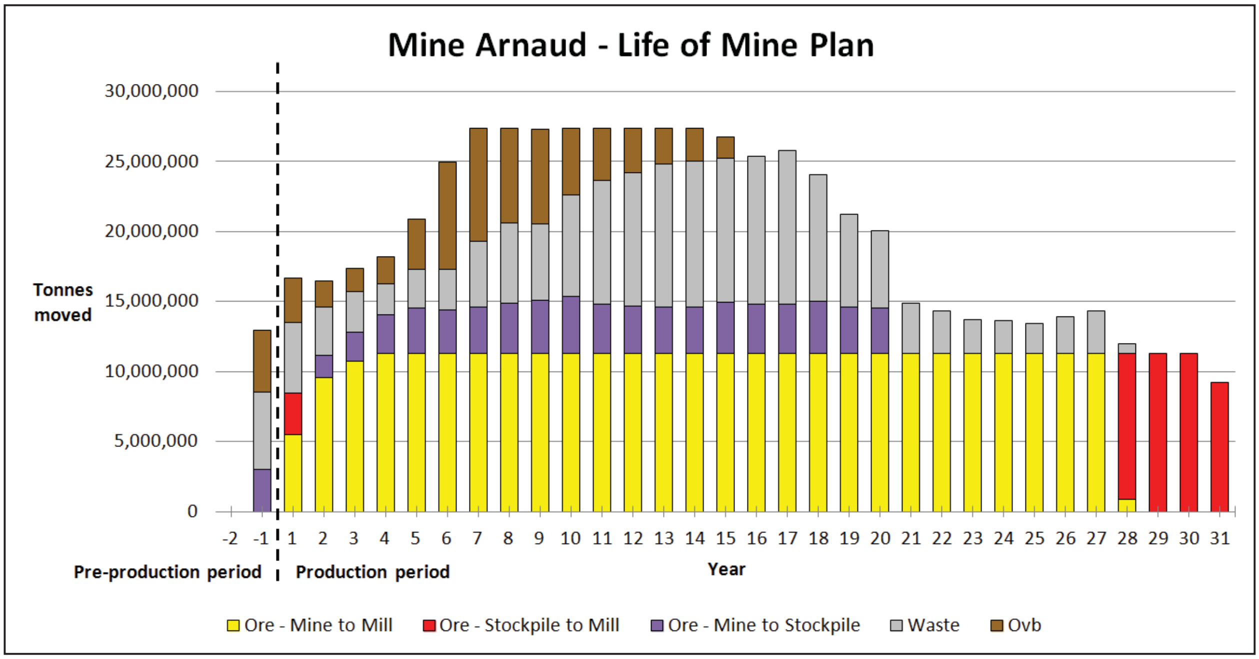


Figure 1-1: Production Schedule

1.10 Recovery Method

1.10.1 Process Plant Design Criteria

The plant capacity has been established at 11,282,880 tpy ROM, based on an ore processing rate of 1,400 tph, an overall plant availability of 92%, 24 hours per day and 365 days of operation per year. This availability has been selected based on similar existing operations having a similar type of comminution circuit consisting of a primary stage of crushing followed by a SAG mill and two (2) ball mills.

The apatite minerals are considered liberated at 125 microns with power requirements being 10.1 kWh/t for ore crushing and 12.5 kWh/t for ore grinding (Bond ball mill work index).

The concentrate weight recovery, based on the average feed grade of 4.57% P_2O_5 and grade-recovery curve, was calculated at 10.5% which is equal to 1,184,702 tpy concentrate production with 39% P_2O_5. The concentrate will be dried to between 0.5% and 1.5% moisture content for ship transportation and subsequent processing by Yara.

1.10.2 Flowsheets and Process Description

The ore will be hauled by 140-tonne mine trucks. The trucks will unload to a primary gyratory crusher, with an average feed rate of 2,576 tph. The ore will be crushed to a P_{80} of 170 mm and stored in a conical stockpile containing approximately 30,000 tonnes of live storage ahead of the grinding circuit. The ore is retrieved from the stockpile at an average rate of 1,400 tph and fed into a SAG mill.

The SAG mill will grind to a P_{80} of 2 mm. The mill discharge will flow onto double deck vibrating screens. The screen oversize, or +9.5mm material is returned to the SAG mill feed. The screen undersize will flow into the SAG mill discharge pump box and is pumped to a distributor to feed the two (2) ball mill discharge pump boxes. pump boxes. The two (2) overflow-type ball mills are configured in closed circuit with two (2) classifying cyclone clusters. The circulating load is 250%. The cyclone underflows will flow by gravity to their respective mill feed spout. The cyclone overflows will have a P_{80} of 125 µm.

The overflow from the two classifying cyclone clusters is directed to two distributors and feeds the Low Intensity Magnetic Separators (LIMS to remove the titano-magnetite). The magnetite material, referred to as magnetic tailings, is dewatered by a thickener and pumped to the magnetite tailings pond. The non-magnetic material flows by gravity to the dewatering cyclones feed pump box and is pumped to the dewatering cyclone stages. A part of the overflow of the second stage is re-used as dilution water to adjust the flotation feed percent solids following flotation reagent conditioning. The underflow from the dewatering cyclone clusters, at 50% solids, will flow by gravity to the flotation conditioners. The conditioned pulp will flow to the rougher stage #1 distributor to be diluted to 35% solids using the dewatering cyclone overflow and then distributed to rougher stage #1 flotation column cells.

Each flotation line will have two stages of rougher columns. The first stage consists of two (2) rougher column cells in parallel. The underflow from the first stage is pumped to a single second stage rougher column cell. Thus, the froth or overflow from the two rougher stages will feed two (2) cleaner column cells configured in parallel. Tailings from the cleaner column cells are pumped to a conditioner tank where starch (WW82) is metred in. The conditioned pulp then flows to a cleaner/scavenger column cell for recovery of additional apatite. Froth from the cleaner/scavenger column is recycled to the cleaner columns feed. Tailing from the cleaner scavenger column is combined with the stage 2 rougher tailings and is discarded as final tailings, and pumped to the tailings thickener.

The cleaner concentrate froth should have a grade of 39% P_2O_5 containing less than 1% Fe+Al content. The final apatite concentrate is pumped to the apatite concentrate thickener. The thickener overflow is directed to the process water reservoir for reuse in the plant. The thickener underflow is pumped to high agitated slurry storage tanks.

From the storage tanks, the concentrate slurry is pumped to a horizontal vacuum 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 dryer system discharges are transferred by belt conveyor to two 4,500-tonne capacity storage bins. From the storage bins, the apatite is loaded into a train containing 39 wagons (3,281 cubic feet capacity per wagon) for transportation to the port facilities.

Plant tailings, with the exception of the magnetic product from the LIMS, will become feed to the tailings thickener. The tailings thickener overflow is directed to the process water reservoir. The tailings thickener underflow, at 60% solids, is pumped to the tailings disposal pump box for pumping to the tailings pond.

1.10.3 Plant Design Layout

The equipment was sized according to the design criteria developed from the flowsheet drawings and the mass balance. 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.

1.10.4 Building Layout

The crusher building will house the gyratory crusher with auxiliary equipment and the tail end of the stockpile feed conveyor. The crushed ore stockpile is contained within a storage dome, 70 metres diameter footprint and a height of 30 metres. The main processing building houses the grinding, magnetic separation, flotation circuit, concentrate filtering as well as offices, mechanical and electrical shop for plant maintenance. A separate adjacent building is also required for the flash dryer. 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.

1.11 Project Infrastructure

1.11.1 Bulk Earthworks, Landscaping, Fencing

The site pad has been placed at a location showing sound properties for the concentrator and administration buildings' foundations. A standard galvanized steel fence, 1.8 m high with three rows of barbed wire will secure the site pad perimeter. A 2.45 m (8 ft) high fence to secure the open pit area at the South, East, and West ends.

1.11.2 Auxiliary Building – Non-Process

The service and administration building will contain offices on the second floor, while the laboratory, infirmary, 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 (lunch room, locker rooms, training rooms, etc.) are located on the main floor. The building's dimensions and general arrangement are based on accommodating approximately 100 workers per shift as well as a maximum of 16 trucks required for the mine operations. The garage will contain the necessary equipment to maintain and repair heavy mining trucks and other mobile equipment

The gatehouse serves several purposes, including a class room to welcome visitors to the site. There, visitors can be shown a presentation concerning the safety guidelines that are applicable to all personnel on site. Offices and a rest area are also included in the structure, as well as a lunch room, showers, locker room, and restrooms.

1.11.3 Waste rock and overburden

Waste rock will be stored in a waste rock stockpile located approximately 1.1km from the mine entrance. This stockpile will be composed of rock material that does not contain enough mineralized material to be economically processed. A volume of approximately 33 Mm^3 is expected.

An overburden material stockpile will be erected at approximately 800m from the mine entrance/exit. This stockpile will be primarily made up of top soil material that will be removed in order to reach hard rock containing mineralization. It will have a volume of approximately 1.0 Mm^3.

Screen berm will be erected south of the open pit. It will have a volume of approximately 15.8 Mm^3.

1.11.4 Surface water management

Water for startup will be provided by surface water storage in the Storage pond and tailings cell #2 at the beginning of the project. During operations, water demand will largely be met by recycling water from the TSF. Make-up water and freshwater requirements will be provided by the Storage pond.

Surface water will be collected by a network of ditches connected to sump and pumping station. Clean water will be directed to adjacent water bodies while contact water will be pumped in order to reach accumulation pond for treatment.

Two water treatment systems are included in the capital and operating costs of this study. A system is designed to treat water from the Storage Pond by a membrane system to feed the Concentrator Gland Seal Water Tank at a rate of about 260 m^3/h. The second system is designed to treat excess water from the site by a High-Density Sludge system and to discharge it to the Clet River at a mean yearly rate of 1,040 m^3/h.

1.11.5 Overpass

An overpass will allow the haul trucks to pass over the railway to reach the waste rock dumps and concentrator area. The overpass has two train tracks that allow for a main track and a Mine Arnaud siding underneath it.

1.11.6 Fuel Storage

The fuel storage consists of a diesel/gasoline distribution station.

1.11.7 Potable Water

The potable water system will be connected to the municipal water line from Sept-Îles running along route 138 and use a pump house to increase pressure allowing the water to be pumped up to the plant building.

1.11.8 Fire Protection

A robust fire protection system will be implemented including a water supply reserve of 1,250 m^3 capacity, sprinkler systems, standpipes and hydrants.

1.11.9 Plant Building

The concentrator plant is divided into two separate buildings, the concentrator and the flash dryer building. 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 and was developed later on during the original Feasibility Study 2012.

1.11.10 Electrical Infrastructure

In November 2010, Hydro-Quebec completed an exploratory study for the electrical connection of the plant into the transmission system. Consequently, in September 2011, Hydro-Quebec completed a Planning Study, which had to be revised in 2014.

The request for the new Planning Study was sent in May 2014 and the revision is expected at the end of August 2014. A meeting was held with Hydro-Quebec on 9 July 2014, in which preliminary results of that study were presented. The costs used in this study are based on that meeting.

1.11.10.1 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 which is under the responsibility of Hydro-Quebec. The mine electrical installations include a 161-13.8 kV outdoor Substation and electrical rooms located near load points around the plant.

The electrical power installed and running loads for the Concentrator-Mine site are indicated in the table below.

Table 1-3: Concentrator – Mine Site

Area	Electrical Connected Load kW	Electrical Running Load kW
Concentrator	73,335	55,554
Rail Load-Out	604	521
Garage, Warehouse & Administration	2,395	1,955
Crusher	1,849	1,576
Stockpile area	935	792
Tailings & Water Treatment	4,622	3,487
Mine Pit	324	195
TOTAL	84,064	64,081

1.11.10.2 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.

The electrical installed and running loads for the Port Facilities are indicated in the Table 1-4.

Table 1-4: Port Facilities

Area	Electrical Connected Load kW	Electrical Running Load kW
Port-Shiploader	1,122	949
Port-Receiving	708	606
Port-Storage	528	409
TOTAL	2,358	1,965

1.11.11 Control and Communication Infrastructure

Plant wide communication and integrated control system will facilitate safe and efficient operation at both the concentrator and port areas. Priority will be given to systems where local support is available.

1.11.11.1 Process Control System

A DCS based process control system including operator station located in concentrator building control room and remote I/O and controller cabinet located in different electrical room. PLC based controller and operator station supplied with package will be located in crusher control room, SAG mill, ball mill and in train load-out area. Redundant communication will link the different control system.

1.11.11.2 Communication systems

A fiber optic cable network will provide communication facility between the different buildings and areas. A redundant microwave communication links will provide communication between the concentrator and port areas. The redundant fiber optic network will serve the different communication and monitoring services such as process control, CCTV, telephone and control access systems. Reliability, safety and non-interference communication between in plant systems and external to the plant will be provided by means or dedicated server, firewall and router.

1.11.11.3 Radio communications

A radio communications system will cover both concentrator and harbor sites. Walkie-talkie and mobile units will serve operations and maintenance vocal and data communications. By means of digital capability, GPS based, equipment positions could be implemented if required.

1.11.12 Transportation

The Mine Arnaud mine site is crossed by the Chemin de Fer Arnaud (CFA), a heavy haul type railway owned by Cliffs Natural Resources. The railway is used to transport iron ore to the Wabush Mines and Cliffs Quebec Iron Mines (Bloom Lake) terminals in the Pointe Noire area of Sept-Îles.

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 possible the use of that railway for transport of apatite concentrate, to the Port of Sept-Îles.

1.11.12.1 Diversion Track

The diversion track consists of a new main line and a load out siding to be constructed to relocate the existing main line, for the purpose of accommodating the proposed mine site. The siding will allow the loading of the railcars without blocking the main line.

The silos and railcar load-out area consists mainly of two (2) concrete storage silos providing the equivalent of about one (1) day of production each.

1.11.12.2 Rail Transportation

Rail haulage must be contracted out to CFA (Cliffs) as they control the rail corridor and the crews that work on it.

The locomotives to perform rail car movement will be supplied under contract by the rail operator. These locomotives should be capable of handling the 39 loaded cars on the current grade on the CFA, each having a capacity to carry 105 tonnes of apatite concentrate.

This operation requires 39 rail cars in service. With a bad order factor of 5%, the total number of railcars to be purchased is 41. The 39-car train is expected to be able to handle 1.30 Mt/y of planned annual throughput.

1.11.12.3 Alternative Transportation Modes

Trucking is the selected alternative transportation mode. This option, evaluated by the engineering firm ‘Strudes’ at the concept level, shows a capital cost in the same range as the one with the train. The operating costs were evaluated at a slightly higher price per tonne than the train, but remained acceptable. The transportation price used in this feasibility by train is CA$3.00/t of concentrate and the alternative way with truck is estimated at CA$3.50/t of concentrate.

1.11.12.4 Port Facilities

Apatite concentrate is transported by train up to the Mine Arnaud Port Facilities (Port Facilities) located around Anse à Brochu in the Pointe Noire area of the Port of Sept-Îles, about 17 km away from the mine site.

During the course of the original FS in 2011, different concepts for the Port Facilities were considered at the La Relance Terminal of the Port of Sept-Îles. The concept developed in 2011 at the La Relance Terminal had two (2) major drawbacks: the need to build a costly new dock and a road overpass.

For this FS, Mine Arnaud requested Roche-Ausenco to develop a concept for port facilities located near the existing Berth 31 in a different location of the Pointe Noire area of the Port of Sept-Îles.

1.11.12.4.1 New Concept Considered for the Port Facilities

Mine Arnaud rail unloading and storage areas are located near the existing rail tracks within the Port of Sept-Îles property and where bedrock is anticipated near the surface. The four (4) storage silos, with provision for a future fifth silo, are located near the rail and an access road. From the silos, a series of two (2) conveyors enclosed in tubular galleries will transfer the apatite concentrate to the Berth 31. The first conveyor runs at grade in the westward direction to the first transfer tower. From the transfer tower, a second elevated conveyor oriented from south to north will cross the Bay of Sept-Îles at two (2) locations as well as over future iron ore installations for the new multi-user dock currently being built by the Port of Sept-Îles. The second elevated conveyor will terminate at a second transfer tower located at the east side of Berth 31.

1.11.12.4.2 Port Track Infrastructure

Roche Ausenco developed a new concept for the Mine Arnaud Terminal based on the addition of a new 2.28 km rail track within Cliffs property to bypass Cliffs Wabush Yard in order to reach the existing main track within the Port of Sept-Îles property.

A 20-car long storage track (siding) has also been included on the spur. This will allow CFA to split the Mine Arnaud train into two blocks (19 and 20 cars) for delivery to the unloading station.

Near Pointe-à-la-Baleine, a 771 m long unloading and storage siding will be added for the Mine Arnaud unloading operations. This new Mine Arnaud siding is set around an unloading station, which includes a dumping pit, a railcar indexer, a bottom gate opener and all required services. The siding is long enough to take half of a complete Mine Arnaud train on each side of the dumping pit.

1.11.12.4.3 Concentrate Unloading

The railcar unloading station will consist of a shed housing the indexer, the bottom gate opener, the railcar unloading equipment, the unloading pit and its ancillary systems. The station is located as close as possible to the west and to Berth 31, while allowing for 20 wagons (half a train) to be parked upstream and downstream of it on the Mine Arnaud siding. The station is also located adjacent to the access road for the new Multi-user Dock. A fence will control access to the station that will house the control room for the complete Mine Arnaud port facilities.

1.11.12.4.4 Concentrate Storage

A 1,524 mm (60”) wide, 116 m long, flexible wall belt type, Silos Feed Conveyor with a capacity of 1,325 tonnes per hour will transfer and lift concentrate at an angle of about 42 degrees to the top of the silos located about 84.5 m away for the unloading station. This angle will also allow the conveyor to be sufficiently high over the access road to the multi-user dock to clear road traffic.

Apatite concentrate storage at the port will consist of four (4) silos, 18 m in diameter by 60 m nominal height (44.2 m active height), for a total of 60,000 tonnes of concentrate storage capacity. This is equivalent to approximately 1.9 times the capacity of the largest vessels expected to handle Mine Arnaud apatite for Yara (37,000 DWT vessels with about 32,000 tonnes of product). Space will be left for a future silo that would bring total storage capacity to 75,000 tonnes.

The silos at the port will be similar in design to 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 fluidized.

1.11.12.4.5 Concentrate Ship Loading

When a vessel is ready for loading, a 1,372 mm (54") wide by 324 m long Silo Discharge Conveyor located underneath the silos will transfer, at controlled rate, the apatite concentrate from the bottom of the silo in operation to a Transfer Tower No. 1 located to the west, onshore south from Berth 31.

The Dock Feed Conveyor, which will cross the area from the onshore south to the Berth 31 on the offshore true north side, will be an elevated 1,372 mm (54") belt conveyor enclosed in an unheated no insulated tubular gallery identical to the one for the Silos Discharge Conveyor. The 738 m long conveyor will transfer apatite concentrate from the Transfer Tower No.1 near the silos to a Transfer Tower No.2 located at the east end of Berth 31.

In the Transfer Tower No. 2, the apatite concentrate will be transferred to a 1,524 mm (60") wide by 182 m long Ship loader Tripper Conveyor, a conventional enclosed belt conveyor. The Ship Loader Tripper Conveyor, installed in an elevated gallery open on one side to provide the passage of the travelling Ship Loader Shuttle Conveyor and equipped with a travelling tripper, will feed a travelling ship loader along the Berth 31. The travelling tripper will be towed by the ship loader.

The ship loader boom will have luffing capability. The 1,400 mm (55.1 inch) wide by 40 m long Ship Loader Shuttle Conveyor will be equipped with a 15 m long telescoping portion allowing the loading of vessels up to 37,000 tonne DWT. Apatite concentrate will be loaded via a Cleveland Cascade type telescopic chute to minimize dust emissions. The ship loader boom will also allow handling of mobile equipment for use in the vessel holds to trim the apatite concentrate.

1.11.12.4.6 Marine Structures for Berth 31

The Port of Sept-Îles will have to make investments at Berth 31 to adapt it for Mine Arnaud. The Port will have to reinforce the dock to suit the new travelling ship loader and tripper loads, extend it to the east, add a platform for a transfer tower and realign the access bridge. This will be indirectly charged to Mine Arnaud as an additional yearly fee.

Further study work is required to verify the dock structure and establish the cost to reinforce it for a travelling ship loader. At this stage, Mine Arnaud is considering that beyond CA$10 million, an alternative like building a new dock would be considered. Therefore, a yearly payment corresponding to a 3% interest over 30 years of CA$10 million is included in the Opex.

1.11.12.4.7 Port Facilities Services

The Mine Arnaud Facilities will be located near the new Multi-user Dock and Berth No. 31 in an area already equipped with many services. This FS is based on Mine Arnaud having access to the existing port services such as potable water, fire protection water, waste water treatment, etc.

1.12 Market studies and Contracts

A market analysis report by Integer Research Limited, London, England, was commissioned in support of this Feasibility Study. The report, dated June 2014, is comprised of two sections:

The first section is Integer's Phosphate Rock Focus Report: Phosphate rock supply, demand and pricing outlook to 2023. The report provides a better understanding of the position of the Mine Arnaud concentrate in the world market, in terms of quality and uses. It defines and describes the market for phosphate rock, its development in recent years and provides an outlook for how the market will develop over the next ten years.

1.12.1 Pricing study for Mine Arnaud's phosphate rock project in Québec

Integer has outlined the price achievable for the rock concentrate produced by the project during the course of the mine's projected operating life. It details the relationship between this price and benchmark prices for phosphate rock, and identifies where and why any differences exist.

The Mine Arnaud phosphate rock project is located in the Sept-Îles region of Québec, and has an expected average operating capacity of 1.2 million tonnes per year, producing igneous phosphate rock concentrate grading 39–40% P_2O_5. The project is 62% owned by IQ, and 38% by Yara International ASA (Yara). Yara is also a potential buyer of the phosphate concentrate, since its major motivation for participating in this project is to find an alternative supply to the Kola high-grade concentrate, which it uses as a feedstock for its nitrophosphate and calcium nitrate (CN) products manufactured at its European operations in Norway.

This section details the conditions under which the rock concentrate produced by Mine Arnaud can realize a premium over benchmark Moroccan prices, and the extent of any premium that can be achieved. First, it outlines the conditions under which premiums for high-grade rock are viable and Integer then calculates an achievable price for the rock concentrate produced by Mine Arnaud.

1.12.2 Factors affecting the price premium for Mine Arnaud's rock concentrate

For high-grade phosphate rock concentrate to realize a premium price with respect to benchmark levels, there need to be major process and cost advantages for buyers. In the section below, Integer identifies these factors and relates them to the rock concentrate from the Mine Arnaud project.

In the second section, on the basis of technical information provided by Mine Arnaud, Integer analyzed the data and provided an achievable price forecast for the project's mined phosphate rock. The Mine Arnaud concentrate is a high purity phosphate rock with little or no contaminants.

In its analysis, Integer considered two essential elements:

- The Arnaud Mine concentrate is comparable to concentrate from Russia, for which current buyers are willing to pay a premium.
- Mine Arnaud has a partner, Yara, with which an agreement exists and which states that Yara will purchase Mine Arnaud's entire production during the operation of the proposed mine.

In addition, Integer conducted a survey of buyers of igneous rock and the potential market for this kind of phosphate rock. Integer therefore provided an estimate of the premium achievable for Mine Arnaud's rock in its most likely end-use markets and an overview of general markets for igneous rock in North America today. This information enabled Integer to project likely achievable prices for phosphate rock mined by Mine Arnaud in Quebec for the projected mine life (until 2046) for key end-use markets. The phosphate rock market price forecasted for 2014–2046 (based on Morocco price in US$ per tonne, 32% P_2O_5) was estimated for three different scenarios (upside, base case and downside) and an achievable price was forecasted for Mine Arnaud's rock for the same period (FOB price) based on the premium and end-use market analysis.

The following table details the price forecast for phosphate rock on an FOB port, 39–40% P_2O_5 basis for 2014–2046.

Table 1-5: Phosphate rock FOB Morocco nominal price forecast, US$ per tonne, 32% P_2O_5, and Phosphate rock FOB Mine Arnaud price forecast, US$ per tonne , 39-40% P_2O_5, 2014-2046

1.12.3 Contract

Yara, one of Mine Arnaud's two partners, is the leading supplier of mineral-based fertilizers in the world. The company's annual production, which exceeds 20 million tonnes, is sold in more than 150 countries.

Yara and IQ have an agreement whereby Yara agrees to purchase the entire production of phosphate rock from Mine Arnaud's proposed mine. SGS Geostat has had access to this agreement.

1.13 Environmental

All information presented in the environmental section was derived from the Environmental Impact Study (EIS) dated March 2012 and its complements as well as all commitments from the proponent made during public hearings, including the Bureau d'audiences publiques sur l'environnement hearings (BAPE).

The Mine Arnaud project is subject to both provincial and federal legislations and all studies, designs and environmental management are in accordance with these legal framework.

This section also presents a summary of the information and consultation activities of stakeholders that have been implemented by Arnaud Mine since 2008.

1.14 Capital and Operating Costs

1.14.1 Capital Expenditure (Capex)

The capital expenditure (Capex) planned to be spent during the construction and pre-production period (years -3, -2 and -1) is amounting to CA$854 M, as shown in the next table, this amount is the reference for the economic analysis shown in Section 22 and does not include an amount of amount of CA$70 M attributable to the pre-mining period. The base date of the cost estimate is June 2014.

The estimate is expressed in Canadian Dollars.

For reference, the currency conversions rates used during the estimate preparation are as per instructions from Mine Arnaud:

- 1 CA$ = US$;
- 1 CA$ = Euro.

Table 1-6: Capital expenditure (Capex)

DIRECT COSTS		Amount (CA$)
200	MINING	
210	Mining Equipment	
220	Open Pit & Auxiliary Services	
280	Tailings Management Facilities including Water Treatment Plant	
290	Rehabilitation & Mine Closure Costs	
300	PROCESS FACILITIES	
320	Crusher, Storage & Conveying	
340	Concentrator	
370	Silos & Load-Out	
400	SITE INFRASTRUCTURE	
421	Bulk Earthworks, Landscaping & Fencing	
423	Water Intakes & Distribution System	
424	Sanitary System	
425	Roads, Overpasses & Parking	
426	Safety	
427	Fuel Distribution and Prevention	
428	Fire Detection and Prevention	
440	Auxiliary Buildings - Non-Process	
460	Rail Diversion	
480	High Voltage Substation	
490	Automation, Instrumentation and Communications	
500	APATITE TRANSPORTATION TO PORT	
540	Rail Transportation	
800	PORT FACILITIES	
820	Site & Material Handling	
840	Marine Structures	
TOTAL DIRECT COSTS		
INDIRECTS COSTS		
	EPCM COSTS (per Mine Arnaud)	
	FIELD INDIRECTS COSTS:	
	TEMPORARY SITE INSTALLATIONS AND SERVICES	
	COMMISSIONING	
	COMMON CONSTRUCTION EQUIPMENT	
	FIRST FILLS	
	VENDORS COSTS	
	THIRD PARTY SERVICES, TESTING & INSPECTION(0.5% of directs costs)	
	OWNER'S COSTS:	
	Mine Arnaud Project Construction Team	
	Training (0.5 % of direct costs)	
	Two years spare parts (per Mine Arnaud based on consignment)	
	Capital spares	
	Construction Insurance (0.5% of directs costs)	
	TAXES & DUTIES (Not included)	
TOTAL INDIRECTS COSTS		
CONTINGENCY		
	CONTINGENCY (15 % on all Total Direct and Indirect Costs per Mine Arnaud)	
TOTAL CONTINGENCY		
TOTAL DIRECT AND INDIRECT COSTS WITH CONTINGENCY		
	OTHER COSTS:	
	HYDRO-QUEBEC TIE-IN TO NETWORK (excluding back-up & costs at Arnaud Sub-Station)	
	HYDRO-QUEBEC - STABILIZATION OF 161 kV TRANSMISSION LINE L1619	
	LAND / PROPERTY ACQUISITION (per Mine Arnaud)	
	ESCALATION (Not included)	
TOTAL COSTS (CA$):		

1.14.2 Sustaining Capex

Sustaining capital contains costs that will be capitalized, i.e. added to the carrying amount of the asset; money spent to acquire or upgrade physical assets such as: tailings facilities, tailings cells, replacements and new purchases of the mining equipments and the mine closure costs and rehabilitation. A summary of these costs with afferent contingencies is presented below.

Table 1-7: Summary of Sustaining Capital Expenditures

The total cost of closure and rehabilitation is estimated at CA$ M. This amount includes the rehabilitation of the accumulation areas and the cost of other closure and restoration activities.

1.14.3 Operating Cost (OPEX)

The Opex estimate 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 project Opex estimate is based on the following parameters:

- Tonnes of ore & waste mined per year: 21,463,174;
- Tonnes of ore milled per year: 11,282,880;
- Average stripping ratio in the mine: 0.71
  (tonnes of waste & overburden per tonnes of ore)
- Tonnes of apatite concentrates produced per year (dry): 1,184,702;
- Total manpower resources required for operation: 276 employees (payroll).

Transportation of concentrates to the port will be sub-contracted to local contractors.

The overall Opex for the Mine Arnaud project is estimated at CA$ M per year or CA$ per tonne of apatite concentrate. A summary of the operating costs for the project is presented below. All costs presented in this section are in Canadian Dollars (CA$) per year and CA$ per tonne of apatite concentrate. Please note that the Summary Process Opex is considering an annual tonnage milled of 11,282,880 tpy which correspond to the design plant throughput.

Table 1-8: Operating Cost Summary

1.15 Economic Analysis

The economic evaluation of the Mine Arnaud project was performed using the discounted cash flow model. The Capex and Opex estimates input into the financial analysis model were based on the mine plan developed in this study to process 11,282,880 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 % resulting from the net cash flow generated by the project was also calculated, as the payback period as an additional financial measure. Finally, a sensitivity analysis on key variables and parameters was performed.

The following assumptions were made for the financial analysis:

- Phosphate market price of CA$ (US$) per dry tonne of P_2O_5 concentrate "Free On Board";
- Concentrate transportation costs (from port) are included in the P_2O_5 market price;
- The pre-production period was estimated at 30 months divided into three years (-3, -2, -1) which represent duration periods of 6, 12 and 12 months respectively;
- For simplification purposes, the costs related to the first 6 months, representing year -3, were transferred to year -2;
- Constant exchange rate of $ (US$:CA$);
- Salvage value of CA$ was considered;

- The reclamation costs that will occur once the project will be completed, i.e., after year 31, were discounted at year 31 (at %) and then treated as a normal charge.

The economic analysis of the Arnaud project was done with and without taxes consideration. An accountant consulting firm, PWC based out of Montreal, reviewed the entire economic model. PWC did a detailed review of all the project costs in order to calculate depreciation, applicable credits, etc., to come up with an after taxes result.

1.15.1 Cash Flow Summary

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 below.

Table 1-9: Project Cash Flow Summary

1.15.2 Net present value, internal rate of return and payback period

The financial analysis results of the Mine Arnaud project for the base case scenario are calculated as:

- CA$ M net present value (after tax) at % discount rate;
- % internal rate of return (after tax);
- years payback period (after tax) after start of production.

1.15.3 Sensitivity analysis

The sensitivity of the pre-tax and pre-financed NPV and the IRR were evaluated for changes in key variables and parameters such as,

- Capital investment (Capex);
- Concentrate value;
- Operating cost (Opex);

- Head grade, calculated as % of weight recovery.

The results of the sensitivity analysis are presented here.

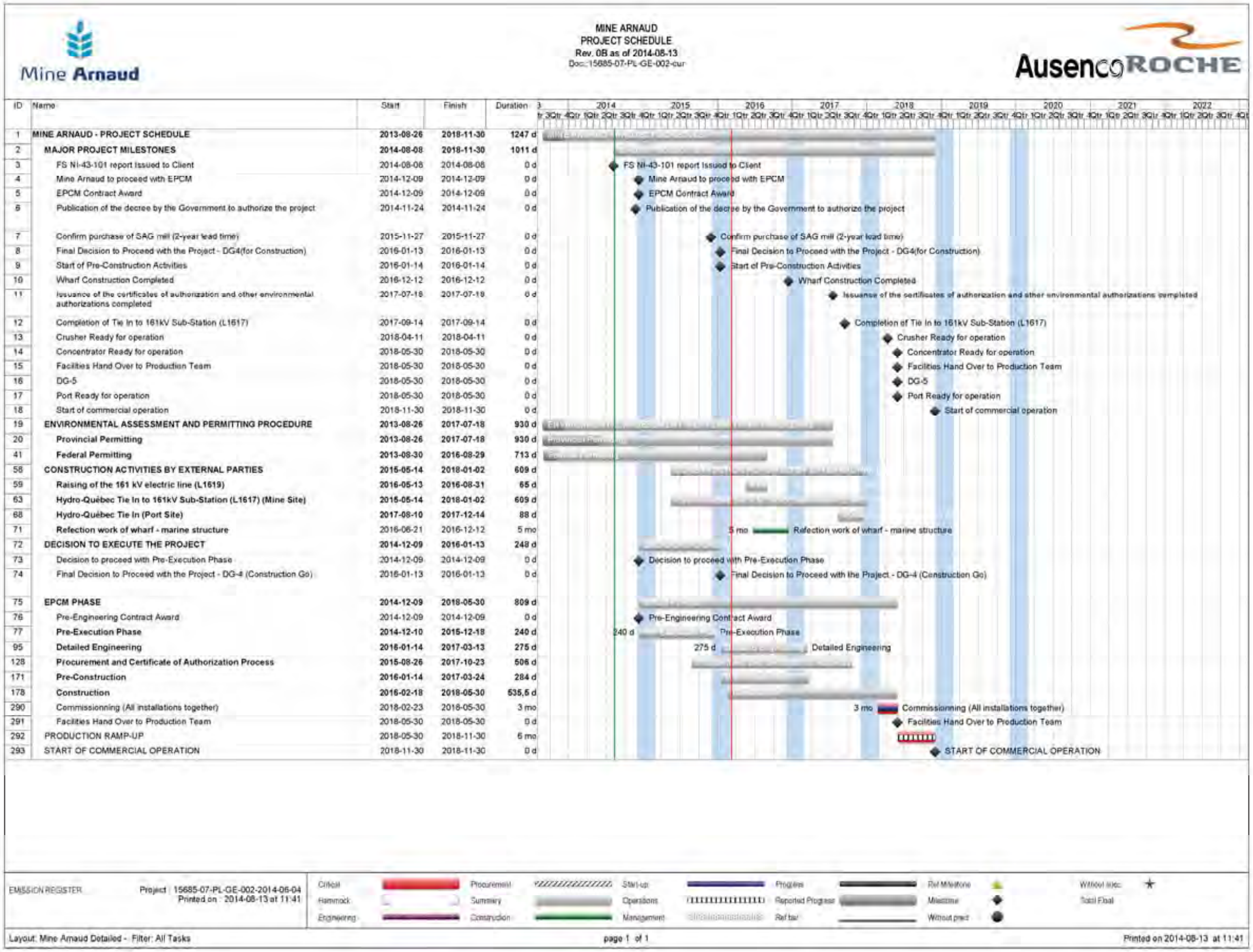
Table 1-10: Sensitivity analysis (NPV and IRR)

1.16 Other relevant Data and Information

1.16.1 Project Schedule

The Project Development schedule has been developed by Roche-Ausenco in close collaboration with Mine Arnaud. This schedule covers the entire time period from the BAPE mandate and hearing to the beginning of commercial operation of the concentrator plant. A summary of the project schedule is shown in Figure 1-2.

Figure 1-2: Production Schedule Summary



The overall project development is divided in three distinct periods:

- Pre-Execution Phase;
- Execution / Construction Phase;
- Ramp-up and Operational Phase.

The schedule considers that the project will be developed as an Engineering, Procurement and Construction Management (EPCM) type project.

1.16.1.1 Pre-Execution Phase

Once the decree from the Quebec's Government is received, Mine Arnaud will begin a one year pre-execution phase. The objective during that phase is to minimize the risks associated with the project by:

- Securing agreements with main stakeholders of the project such as Hydro-Quebec and the Port of Sept-Îles;
- Ordering the long lead items that could affect the project's schedule and preparing bidding packages to start the site preparation work shortly after the final decision to proceed with the project (Decision Gate 4, "DG4");
- Value engineering: the optimization of the engineering of the project in order to reduce costs;
- Progressing with the permitting process.

The pre-execution phase deliverable will be a report including a revised Capex, Opex, and project development schedule which will serve as a basis for the final decision to proceed with the project (DG4). The Pre-Execution Phase spans a twelve (12) months period.

1.16.1.2 Execution/ Construction Phase

Construction

Once the permits are in place, the main construction activities will start with the construction of the new railway line and the removal of the existing one, the erection of the crusher, storage dome, concentrator, and silo, as well as all related infrastructures. In parallel, construction activities at the Port will be conducted. The Port Authority of Sept-Îles is to refurbish an existing wharf at Pointe-Noire. The EPCM phase is spread over a period of 30 months, which include 27 months of construction.

Commissioning

Once the commissioning of all areas is completed, an additional period of three (3) months has been allocated for the overall commissioning of the operations.

Production Ramp-Up

Once construction and commissioning are finalized, the six (6) months ramp-up period will start in order to transition the project to full scale commercial operation.

1.16.2 Risks Analysis

A risk assessment on the project was performed to measure the variation potential in the different cost elements that make up the capital costs.

The risk was assessed on the major cost areas, as contained in the capital cost estimate. They are summarized in the table below.

Table 1-11: Cost elements of the capital cost estimate

200	Mining Equipment
210	Open Pit & Auxiliary Services
220	Tailings Management & Facilities
230	Crusher, Storage & Conveying
240	Concentrator
250	Silos & Load-Out
260	Bulk Earthworks, Landscaping & Fencing
270	Water Intakes & Distribution Systems
280	Sanitary Systems
290	Roads, Overpasses & Parking
300	Safety
310	Fuel Distribution and Storage
320	Fire Detection and Prevention
330	Auxiliary Buildings - Non Process
340	Rail Diversion
350	High Voltage Substation
360	Automation, Instrumentation and Communications
370	Rail Transportation
380	Site & Material Handling
390	EPCM Costs
400	Indirect Costs
410	Owner Costs
420	Transfer From Operating Costs

Furthermore, other risk factors that can affect costs were evaluated, they are listed below.

Table 1-12: Other risk factors that can affect project costs

450	Bad EPCM contractor
460	Need for construction camp
470	Flooding of area
480	Mine Arnaud affecting Hydro Quebec lines
490	Strike, lock out
500	Delays in the permitting
510	Decision process for DG4
520	Delivery time of grinding mills
530	Harbour not completed in time
540	Hydro Quebec not completed in time
550	Environmental issues
560	Delayed agreement with rail owner
570	Accident on the mine site, during and after construction
580	Derailment
590	Mine Arnaud cannot find human resources for the project
600	Lack of continuity of personnel on the project
610	Extreme winter weather
620	Managing of thick layer of clay
630	Shortage of manpower
640	Shortage of construction equipment
650	Lack of delivery of consumables
660	Bad productivity of construction labour
670	Change of ship size

The risks to the project were evaluated according to their probability of occurrence, and its potential cost impact in the event of an occurrence. The resulting data was then incorporated in a Monte Carlo simulation. A total of 10,000 iterations were performed by the simulation, resulting in a risk profile that warrants a 15% contingency, based on the P_{50} results.

1.17 Interpretation & Conclusions

1.17.1 Mining Method

The Mine Arnaud project is a very straight forward mining operation, wide and long open pit, low stripping ratio, etc. The material will be mined by a single open pit, which will have 28 years of production life following a two 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 is expected to be done by the project owners 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. To facilitate the hauling operation two ramps are proposed. The overburden and waste rock material will be hauled to the overburden and waste disposal areas near the pit. Some of the waste rock will be returned to the shallow western part of the open pit to lower the waste left on surface. The run-of-mine mineralization will be delivered by large mining trucks to the primary crusher or stockpiles near the crusher area.

The production mining fleet for ore and waste, of 140 tonne trucks and 15 m^3 excavators is in line with heavy equipment manufacturer's standards. The overburden will be transported by 40 tonne articulated trucks that were selected because of their low environmental impact. There are possibilities to optimize the developed mine plan using different mining approaches, new high grading scenarios, different mill feed, etc. Those optimizations could impact the project NPV.

1.17.2 Mineral Processing and Recovery Method

1.17.2.1 Ore Variability

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 is the impact that it will have on the recovery, the consistency of the concentrate grade and eventually the operating cost. Another risk associated with ore variability concerns the pumping problems that may occur if grade varies considerably. A 10% overdesign factor applies on pump flows, but it may not be sufficient.

Recent testwork program conducted at Corem during winter and spring 2014 with samples taken at depth showed that no major issues are to be expected toward the end of the mine life because the ore at depth responds similarly to the various ore types previously studied.

1.17.2.2 Dewatering cyclone and Flotation

The risks associated with the dewatering cyclone is that, if very fine grinding of the apatite occurs for some reasons in the grinding mills, there is a potential of additional losses of P_2O_5.

1.17.2.3 Wet High Intensity Magnetic Separation (WHIMS)

All testwork programs indicated that there will be no major problem achieving concentrate meeting the specifications of Yara without the use of Wet High Intensity Magnetic Separation (WHIMS). 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 by generating a high grade P_2O_5 product that do not meet Yara's specifications for their apatite concentrate.

1.17.2.4 Filtration

A belt filter was selected as the most efficient and low capital cost piece of equipment to dewater 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 the final concentrate for shipping should be between 0.5 to 1.5%. 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. Filter press tests by manufacturers should be conducted in order to make a final judgment on this matter.

1.17.3 Project Infrastructure

1.17.3.1 Electrical Infrastructure

A request for a new Planning Study was sent to Hydro-Quebec in May 2014. The study is expected to be issued at the end of August 2014. A meeting was held on 9 July 2014, in which preliminary results of that study were presented by Hydro-Quebec. The costs used in this study are based on that meeting. In general, the solutions found by Hydro-Quebec for tie-in to the network and stabilization of the 161 kV transmission line L1619 are adequate and less expensive than in 2011.

1.17.3.2 Rail Transportation

Rail haulage must be contracted out to CFA (Cliffs) as they control the rail corridor and the crews that work on it. The locomotives to perform this movement will be supplied under contract by the rail operator. These locomotives should be capable of handling the 39 loaded cars on the grade present on the CFA subdivision, each having a capacity to carry 105 tonnes of apatite concentrate.

1.17.3.3 Port Facilities

Apatite concentrate is transported by train up to the Mine Arnaud Port Facilities located around Anse à Brochu in the Pointe Noire area of the Port of Sept-Îles, about 17 km away from the mine site. For this Feasibility Study, Mine Arnaud requested Roche-Ausenco to develop a concept for port facilities located near the existing Berth 31, in a different location of the Pointe Noire area than in 2011.

1.18 Recommendations

1.18.1 Geology

1. Use present drilling data in order to establish and position the structural breaks of the deposit and include them in the next resources modeling of the mineralized envelops;

2. 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;

3. Analysis for chlorine in the wedge area was biased by hole 1166-10-83. In order to increase the level of in situ Cl continuity hole 1166-10-83 should be re-assay for chlorine. Still, this will have no impact as mill feed ore is control by in situ K2O to assess Cl in concentrate.

1.18.2 Mining Method

1. Assess various mine plans and high-grading scenarios in order to maximize the project profitability;
2. Analyze the pros and cons of using different suppliers and larger size mining fleet in order to lower mining cost, while taking in consideration the impact on the mining dilution;
3. Once the final mine plan will be developed, use specific software (for example TALPAC from Runge Inc.) to precisely estimate the required mining fleet;
4. 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;
5. 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;
6. With future investigations, lower the amount attributable to the contingency by detailing more precisely the project capital expenditure.
7. The sustaining capital should be reviewed by allowing a salvage value for heavy equipment renewals; the present study was done with salvage value only at the closure of the operations.

1.18.3 Mineral Processing and Recovery Method

1. To reduce the process operating cost, additional testwork is recommended with similar reagents from different suppliers. With these test results, it will be possible to evaluate the circuit response and reagent consumptions and to optimize the reagent selection.

2. Perform a trade-off study to save energy by using filter press instead of belt filter to reduce the concentrate moisture content prior to feeding the flash dryer. Filter press filtration would involve increased Capex and Opex for the filtration stage, however much less energy is required to reduce the moisture content by filtering versus vaporization; thus there could be savings in the long term. Pressure filtration tests are required to perform the trade-off study between the two technologies.

3. Results received from SGS in July 2011 show that additional grindability testworks should be conducted on the material in order to confirm the proposed SAG-Ball Mill circuit. Additional grindability tests on the newest drill core sample would help assess the variability in terms of grindability and abrasiveness and reconfirm the sizing of the grinding mills.

4. A study to better understand the magnetite distribution will have to be performed with the following objectives:

- To evaluate the variability of the magnetite distribution within the deposit and its impact on the grinding circuit and on P_2O_5 recovery;
- To observe how blending requirements could be achieved using a unique stockpile;
- To fully assess if magnetite can offer an economical value by having a clear evaluation of the magnetite quality and content over the deposit. . Also assess if magnetite lims process can be improved to create a magnetite concentrate more attractive to potential buyers.

5. Building size optimization could result in Capex savings with the following activities:

- Replace the two (2) ball mills by one (1) of a bigger size:
    - Reduction of Capex for the purchase of equipment;
    - Reduction of Capex by reducing the size of the building: eventually the building could have one less bay (7 m wide) over 24 bays (7 m long each) for a potential reduction of the footprint of the building of 1176 m² or approximately 10%;
- Eliminate the space for future columns and Wet High Intensity Magnetic Separator;
    - Reduction of Capex by reducing the size of the building: eliminating the space for additional columns in prevision of future expansion as well the space for WHIMS can help reducing significantly the foot print of the building. The empty floor space planned for the future equipment corresponds to 588 m² or approximately 5% of the beneficiation plant footprint.
- Optimization of the space requirements for offices, maintenance around the equipment, and equipment installations can potentially help reduce the beneficiation plant footprint by approximately 3 to 5 %.
- Reduction of the height of the Flash Dryer building if possible after discussion with Flash Dryer suppliers.
- Other minor optimization should be considered including reduction of the building height by approximately 0.5 m to 1 m for certain portion of the beneficiation plant building.

1.18.4 Project Infrastructure

1.18.4.1 Electrical Infrastructure

When Hydro-Quebec will issue their study at the end of August 2014, it should be verified for any changes from the meeting held on 9 July 2014. The investments required, if any, at Hydro-Quebec Arnaud sub-station remained to be evaluated.

1.18.4.2 Rail Transportation

The rail alternative is the preferred mode of transportation retained for the Mine Arnaud project providing Mine Arnaud can reach a reasonable agreement with Cliffs. During this study, there were no negotiations with Cliffs to firm up an agreement. This will have to take place as the project moves forward. The transportation price to be paid to Cliffs used in this feasibility is CA$ 3.00/t of concentrate. It is NOT based on an agreement or recent discussions with Cliffs. Beyond this transportation cost, Mine Arnaud would consider an alternative mode of transportation.

1.18.5 Port Facilities

Further study work is required to verify the dock structure and establish the cost to reinforce it for a travelling ship loader. At this stage, Mine Arnaud is considering that beyond CA$10 millions, an alternative would be used.

2. Introduction

This FS is for the developpement of the Sept-îles apatite deposit, operated by Mine Arnaud Inc. located near Sept- Îles, Quebec. This FS has been based solely on the development of the Sept-îles deposit. Mine Arnaud Inc. commissioned the engineering consulting group SGS Geostat to perform this Study which was realized in collaboration with Roche, Journeaux, WSP and Ausenco. This Technical 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 Geostat are Jonathan Gagné, Eng., MBA, Jean-Philippe Paiement, M.Sc., P.Geo and Gaston Gagnon, Eng. 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 Hugo Latulippe, Eng. and Bruno Perron, Eng. A complete list of the reports available to the authors is found in the references section of this report.

The purpose of the Study is to develop all the aspects of the the project in sufficient details to enable Mine Arnaud to use the report for financing and permitting purposes. The Study was also prepared to identify the risks and critical issues and to propose mitigation recommendations.

2.2 The Technical Report

This technical report is prepared to support the same geological, engineering, and economic factors as reported in previous studies, but with more detail and precision. It is assumed that the level of accuracy of this study Report is in accordance with the NI 43-101 Standards that are expected to be in the range of up to ± 15%. The Report describes the basis and methodology used for modeling and estimating the Sept-Îles 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, interpretation 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 estimated minerals resources and mineral reserves. This updated analysis was conducted using a combination of parameters derived from the previous studies and established by SGS Geostat in collaboration with the other Consultants.

The reader must be advised that the content of this Technical Report is based on the previously published studies, 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.

2.4 Site Visit

A site visit was conducted by Jonathan Gagné, Eng., MBA from SGS, at the Sept-Îles project site in October 2012. Jean-Philippe Paiement, M.Sc., P.Geo, 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 test. The property was also visited by Simon Latulippe, Eng. and QP, from Group Global WSP, on May 22, 2014. Rémi Duchesne, Eng. and QP from Ausenco Engineering Inc., visited the property on June 2010, and Noël Journeaux, Eng. and QP, from Journeaux Associates, visited the property from March 10 to March 17 in 2011.

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
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
CA$	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

Abbreviation	Description
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
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 tonne
GEA	GEA Barr-Rosin
Genivar	GENIVAR Société en Commandite currently named WSP
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-Quebec
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 tonne
km	Kilometer
km/h	Kilometer per hour
kPa	Kilopascal
kV	kilovolt
kW	Kilowatt
kWh	Killowatt hour
kWh/t	Kilowatt hour per metric tonne
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
LRS	Liquid Resistance Starter
m	Meter
m/s	Meter per second
m^2	Square meter
m^3	Cubic meter
m^3/h	Cubic meter per hour

Abbreviation	Description
Ma	million years
masl	Meters Above Sea Level
Mbps MC	Megabits Per Second Moisture Content
MCC	Motor control centre
MDDELCC	Ministère du Développement durable, de l'Environnement, et Lutte contre les changements climatiques (formerly known as MDDEFP and MDDEP)
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 and Parks
MSDS	Material safety data sheets
Mtpa	Million tonnes per annum
Mtpy	Million tonnes per year
MVA	Megavolt-ampere
MW	Megawatt
N	North
N/mm^2 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
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

Abbreviation	Description
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
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 tonnes
t/d	Dry metric tonnes per day
t/h	Dry metric tonnes per hour
$t/h/m^2$	Dry metric tonnes per hour per square meter
$t/m^2/h$	Dry metric tonnes per square meter per hour
t/m^3	Dry metric tonnes per cubic meter
t/y	Dry metric tonnes per year
ta	Abrasion breakage
TCLP	Toxicity Characteristic Leachate Procedure
TDMA	Time division multiple access

Abbreviation	Description
TJCM	Table Jamésienne de Concertation Minière
TLs	Threshold limits
TMF	Tailings management facility
tpd	Tonnes per day
tph	Tonnes per hour
tph/m^2	Tonnes per hour per meter square
tpy	Tonnes per year
TSS	Total suspended solids
U/F	Underflow
UPS	Uninterruptable power supply
US$	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
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 Experts

SGS Geostat prepared this Study Report using documents and reports as noted in Section 27 "References" of this Report. The current Report has been written by the following main authors: Mr. Jonathan Gagné, Eng., MBA, Mr. Gaston Gagnon, Eng. and Mr. Jean-Philippe Paiement, M.Sc., P.Geo, who are responsible for sections 1 to 23 and its content, with the exception of sections 13, 17, 18, 19 (Market Studies and Contracts) and 20 (Environmental Studies, Permitting and Social or Community Impact). The Sections 24, 25, 26 and 27 were prepared by SGS Geostat in collaboration with the other Consultants involved in this Study Report: Roche, Journeaux, WSP and Ausenco.

Section 19 (Market Studies and Contracts) relies on a report prepared by Integer Research Limited, United Kingdom, for the phosphate rock supply, demand, and pricing outlook to 2023. SGS Geostat QP's have reviewed the work done in the FS and are satisfied that the information presented is accurate and sufficiently detailed for the current technical report and disclosure.

Section 20 (Environmental Studies, Permitting and Social or Community Impact), relies on all information contained in this section was drawn from the Environmental Impact Statement (EIS) dated March 2012 and its supplements, as well as all commitments made by the proponent during public hearings, including those held by the BAPE [Board of Public Environmental Hearings]. The EIS and its supplements can be viewed on Mine Arnaud's Website:

(http://www.minearnaud.com/fr/étude-impact).

Section 22 (Economic Analysis) was prepared in collaboration with PWC from Montreal. PWC prepared a financial model in which the production data were validated by SGS Geostat. SGS Geostat QP's have reviewed the work done in the FS 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 Mine Arnaud with the required Study Report in the permitting purpose.

Table 3-1: Responsibilities by QP

Item Description	Lead	Item №	Organization	Responsible Qualified Person
Study Coordination	All	1,2,3	All	G. Gagnon
Property Description and History	JP. Paiement	4,5,6	SGS	JP. Paiement
Geology, Exploration and Database	JP. Paiement	7, 8, 9, 10, 11	SGS	JP. Paiement
Data Verification	JP. Paiement	12	SGS	JP. Paiement
Mineral Processing and Metallurgical Testing	A. Dorval	13	Roche	A. Dorval
Mineral Resource Estimate	JP. Paiement	14	SGS	JP. Paiement
Mineral Reserve Estimate	J. Gagné	15	SGS	J. Gagné
Mining Methods	J. Gagné	16	SGS	J. Gagné
Hydrology and Hydrogeology	S. Latulippe	16.11.1	WSP	S. Latulippe
Recovery Method	A. Dorval	17	Roche	A. Dorval
Infrastructure	A. Dorval	18 Except 18.1.2, 18.2 to 18.9	Roche	A. Dorval
Infrastructure	R. Duchesne	18.1.2, 18.5 to 18.9	Ausenco	R. Duchesne
Dykes and Geotechnical	N.Skiadas	18.2 & 18.3	Journeaux	N.Journeaux
Stockpiles and Water Management	S. Latulippe	18,4	WSP	S. Latulippe
Market Study	G. Gagnon	19	SGS	G. Gagnon
Environmental Studies	S. Latulippe	20	WSP	S. Latulippe
Capital and Sustaining Costs	G. Gagnon N.Skiadas S. Latulippe A. Dorval R. Duchesne	21,1	SGS Journeaux WSP Roche Ausenco	G. Gagnon N.Journeaux S. Latulippe A. Dorval R. Duchesne
	G. Gagnon	21.2.1 to 21.2.2	SGS	G. Gagnon
	N.Skiadas	21.2.3	Journeaux	N.Journeaux
	S. Latulippe	21.2.4 to 21.2.5	WSP	S. Latulippe
Operating Costs	G. Gagnon	21,4	SGS	G. Gagnon
	A. Dorval	21.3.1 to 21.3.3, 21.5 and 21.6	Roche	A. Dorval
Economic Analysis	J. Gagné	22	SGS	J. Gagné
Adjacent Properties	JP. Paiement	23	SGS	JP. Paiement
Other Relevant Data and Information	G. Gagnon	24	SGS	G. Gagnon
Schedule	W. Leclerc	24,1	Roche	A. Dorval
Risk Analysis		24,2	YARA/Arnaud	G.Gagnon
Interpretations and Conclusions	All	25	All	All
Recommendations	All	26	All	All
References	All	27	All	All

4. Property Description and Location

4.1 Location and Access

The property is located in the Québec North Shore region about 650 km East of Québec City and 900 km East of Montréal (Figure 4-1). 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. Lawrence River in the Central Northern part of the Sept-Îles Bay. It is easily accessible through Highway 138, which runs approximatively one kilometre to the south of the deposit limit, along the Sept-Îles Bay shoreline. Highway 138 is a provincial road linking the communities of Québec's North Shore along the St-Lawrence River to the rest of Québec province. The immediate vicinity of the project is scarcely populated and the settlements are mainly concentrated along Highway 138 with relatively limited local traffic. The site is accessible from Highway 138 via a network of bush trails which run more or less north-south. The project is located within the boundaries of the city of Sept-Îles and is zoned as agricultural land, mainly used for the harvesting of firewood (Figure 4-2).

4.2 Property Description

The property is 100% held by Mine Arnaud which is, in turn, a wholly owned subsidiary of IQ and Yara, a Quebec Government Corporation. The project is owned at 62% by IQ and 38% by Yara International, ASA (public limited company) from Norway.

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

4.3 Ownership

The property is composed of 303 contiguous claims covering an area of 9,842 ha. The claim package comprises 123 active titles with expiry dates ranging from June 27, 2015 to November 7, 2016. The other 180 claims are in the process of being converted to mining lease by Mine Arnaud.

Exploration on the property is a joint venture between IQ and Yara. SGS Geostat verified the property title and mineral rights on the *Ministère de l'Énergie et des Ressources naturelles*'s (MERN) web site. The 303 claims, as registered with the MERN, are 100% owned by Mine Arnaud and are in good standing.

The area underlying the property is composed of Crown Land (6,227 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 A – 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 A – Claim List, there are no other encumbrances on the project.

4.4 Restrictions

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).

4.5 Terms and agreements of First Nations peoples

The property of Mine Arnaud is located on the land claimed (Nistassinan) by the Innu community Uashat mak Mani-Utenam. To the knowledge of Mine Arnaud no other community is affected by the mining project. The Innu Uashat mak Mani-Utenam was not involved in the agreement signed in 2004 with the Government of Quebec.

Mine Arnaud must obtain approval from the band council prior to interventions on the territory claimed by the community in question. The authorization request was sent through the government's representative during the request of the intervention permit, required for the exploration work.

As part of the EIS of the Mine Arnaud project, the developer has done a description of the land uses by the Innu communities of Uashat mak Mani-Utenam. This description is based on information gathered during several meetings with representatives of the community and land users (trappers) and other studies describing the land uses (environmental assessment of the Romaine project and the expansion project of the transmission network in the Mingan (Hydro-Quebec)). These studies therefore indicate that no evidence of any trapline, family plot or permanent installations belonging to an Innu of Uashat community is located on the proposed project site.

Mine Arnaud has held three information sessions in the community to present the different aspects of the project. The objective of Mine Arnaud is to sign an agreement on the impacts and benefits with the Uashat mak Mani-Utenam community.

Figure 4-1: Project Location

Figure 4-2: Project Location Map

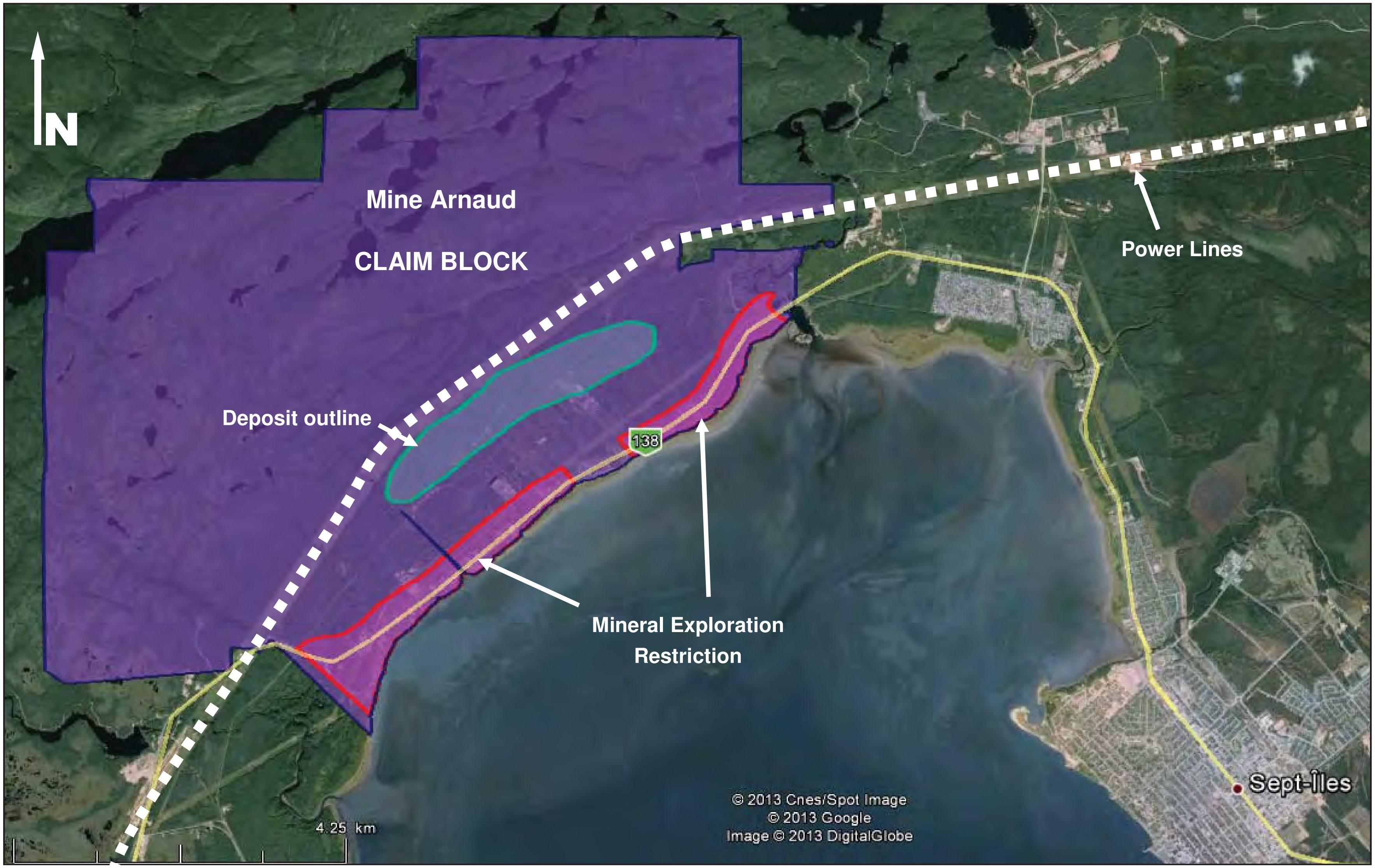

Figure 4-3: Claim block and mineral exploration restrictions

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

Most of the information contained in this item was taken from the Feasibility Study (prepared by Roche-Ausenco) dated February 2012 that was updated by SGS Canada Inc. to a NI 43-101 compliant Pre-Feasibility Technical Report with an effective date of July 24th, 2013.

5.1 Access

The property is located about 15 km west of Sept-Îles and is easily accessible through Highway 138, which runs approximately 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 accessed 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 the Pointe Noire Area. This road to Pointe-Noire is the access point to the port facilities (Quai de la Relance), and also to major industrial sites (such as Aluminerie Alouette, Wabush Mines and Consolidated Thompson).

The northern part of the deposit is limited by a large and important Hydro-Quebec corridor (Figure 4-3) which already contains three 735 kV power lines coming from Churchill Falls. A fourth line (from La Romaine Hydro project) has been 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 connects Arnaud Junction and Pointe-Noire, also runs through the deposit in an East-West direction (Figure 4-2). Rail connection to the 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). Additionally, by use of the Québec North Shore and Labrador (QNSL) and Arnaud rail carriers, there is easy access to Sept-Îles and Labrador City.

5.2 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 annually precipitates 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.

5.2.1 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 respectively.

5.2.2 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 north and north-northeast, but they also often come from the northwest, the west-northwest, and the west. The northwest north and west-northwest winds are frequent and relatively strong. Although less frequent, east winds are the strongest (4.4% of the time over 20.5 km/h).

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**

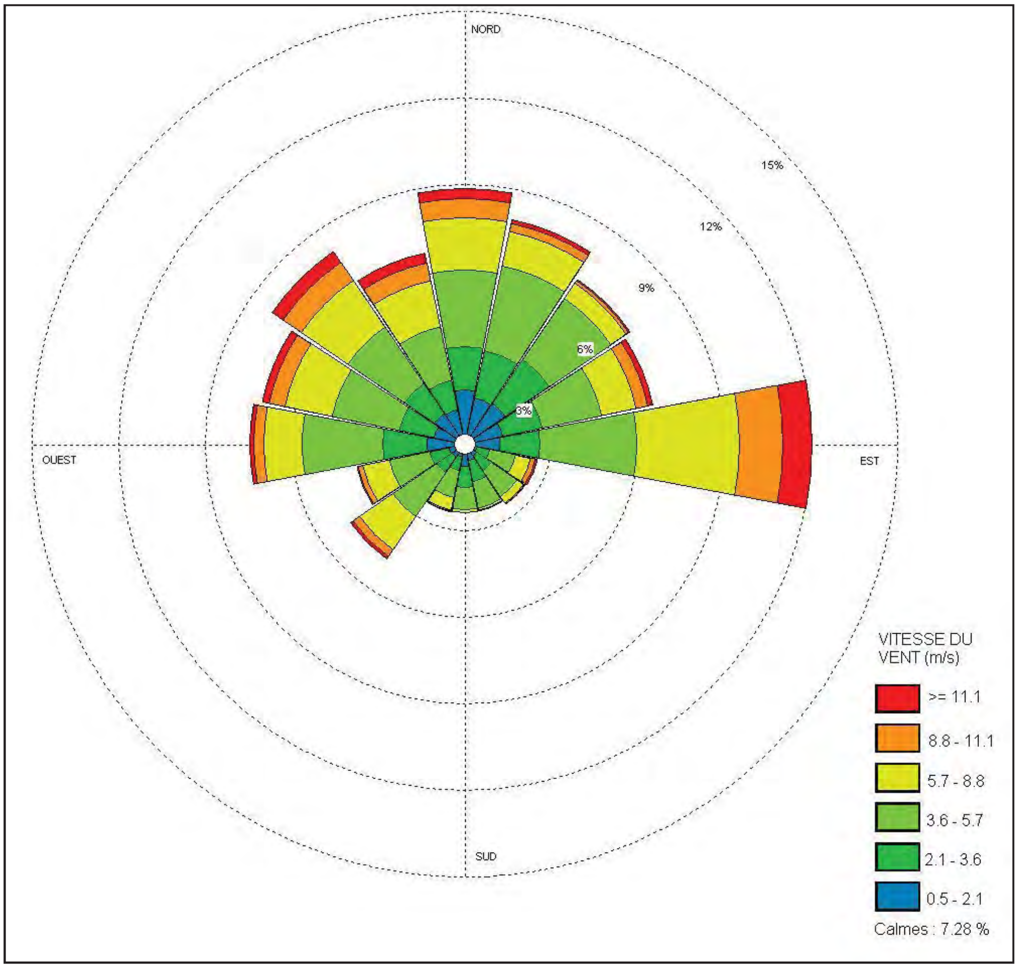

Figure 5-1: Wind Rose – Sept-Îles Airport Meteorological Station – Annual Average

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).

5.3 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 2011 census, the population of Sept-Îles was 28,487, 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-Quebec.

5.4 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.

5.5 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.

5.6 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 et 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. Till is composed of a a mix of boulders, gravel, 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 (AMEC).

Moving forward, the region is covered by a glacial-marine deposit as the elevation decreases going toward St.Lawrence 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 (AMEC).

6. History

The information contained in this item was taken and modified from the Feasibility Study (prepared by Roche-Ausenco) dated February 2012 that was updated by SGS Canada Inc. to a NI 43-101 compliant PreFeasibility Technical Report with an effective date of July 24th, 2013.

6.1 Discovery and exploration until 2005

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 (Roche, 1996), feasibility study (Met-chem, 2002) and an updated feasibility study (SNS-Lavalin, 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 10.80% 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 FS that would be based on the production of apatite, ilmenite, and potentially magnetite as a secondary product. A Pre-Feasibility study (PFS) 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 6.60% P_2O_5 and 9.20% TiO_2 (Genivar, 2008) and was used for pilot plant scale test work at Lakefield Research Laboratories Inc. (Met-Chem, 2002).

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

In 2002, Met-Chem completed an updated FS 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).

6.2 Fieldworks from 2005 to 2013

In 2008, field work 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 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 FS in 2012 by Roche-Ausenco. The drilling programs were conducted by SOQUEM from 2010 to 2011, and then Axor conducted resource definition dirlling in 2012, followed by further drilling in 2013 under the supervision of InnovExplo.

6.3 2012 Feasability Study

A FS and Capex 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: 105 Mt Indicated and Measured resources at an average grade of 5.32% P_2O_5 and an additional 157 Mt inferred resources at an average grade of 4.66% P_2O_5.

6.4 2013 Pre-Feasability Study

In 2013, Mine Arnaud had mandated SGS Geostat to complete a PFS on the Sept-Îles deposit. The study was completed using new metallurgical testing (Corem and SGS Lakefield), information from the 2012 study from Roche-Ausenco and new data from exploration programs leading to a new Mineral Resources estimation.

The estimated Mineral Resources of this study comprised 370.87Mt of measured resources at an average grade of 4.16% P_2O_5, 110.87Mt of indicated resources at an average grade of 4.24% P_2O_5 and 42.76Mt of inferred resources at an average grade of 3.52% P_2O_5. The work also enabled to estimate the quality of the produced concentrate, especially the Cl content in order to ensure that the required specification (Yara) where met.

The Mineral Resources were converted to 64.Mt of proven reserves (4.74% P_2O_5) and 260Mt of probable reserves (4.34% P_2O_5) over 28 years LoM. These reserves enabled SGS to estimate an NPV (discounted at 5%) of CA$ M with an IRR of %.

7. Geological Setting and Mineralization

The information contained in this item was taken and modified from the Feasibility Study (prepared by Roche-Ausenco) dated February 2012 that was updated by SGS Canada Inc. to a NI 43-101 compliant Pre-Feasibility Technical Report with an effective date of July 24th, 2013.

7.1 Regional Geology

The St. Lawrence valley formed through rifting 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 Complexes, 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 in 2010 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.

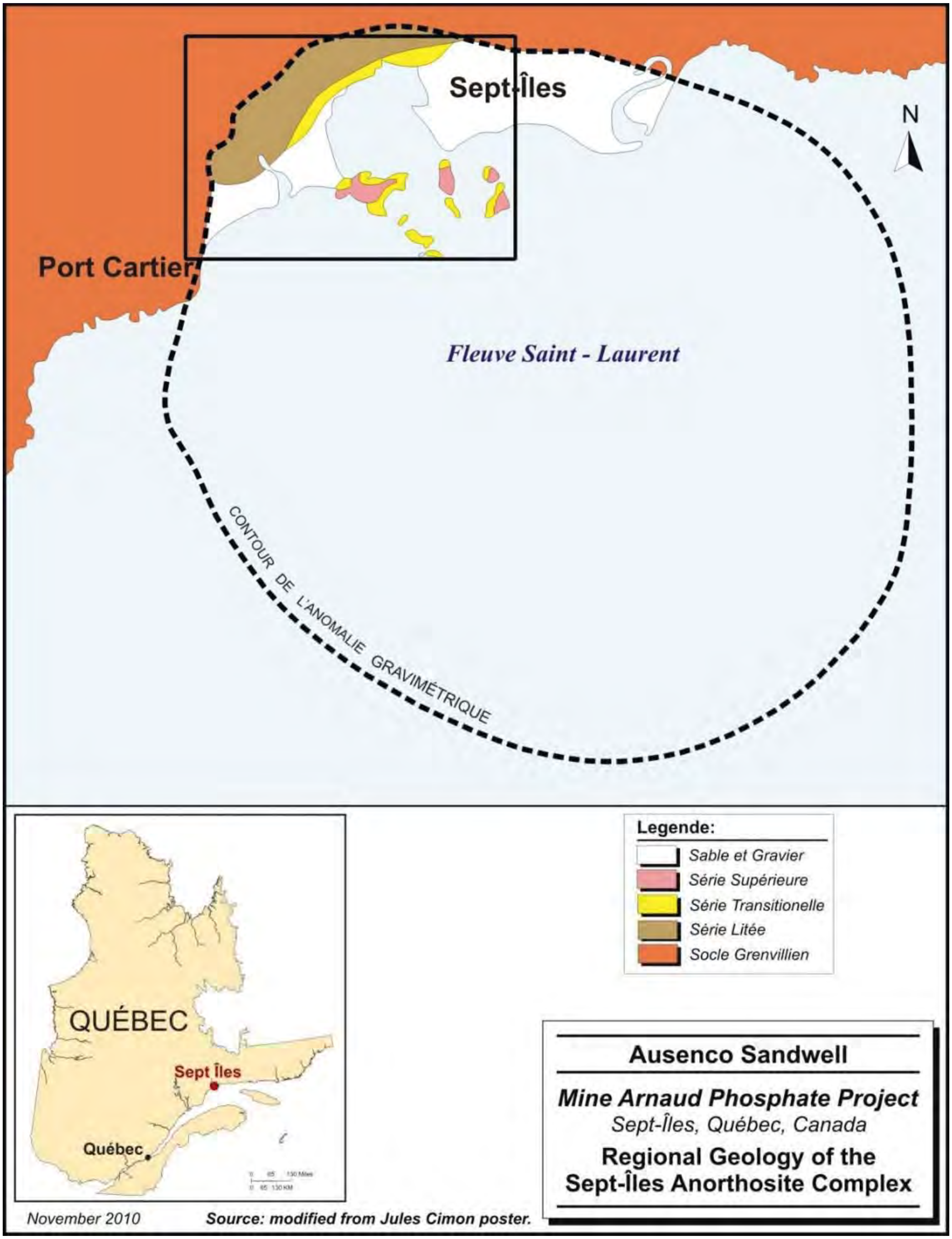

Figure 7-1: Outline of the Gravimetric Anomaly Associated with the Sept-Îles Anorthosite Complex (Cimon,1998)

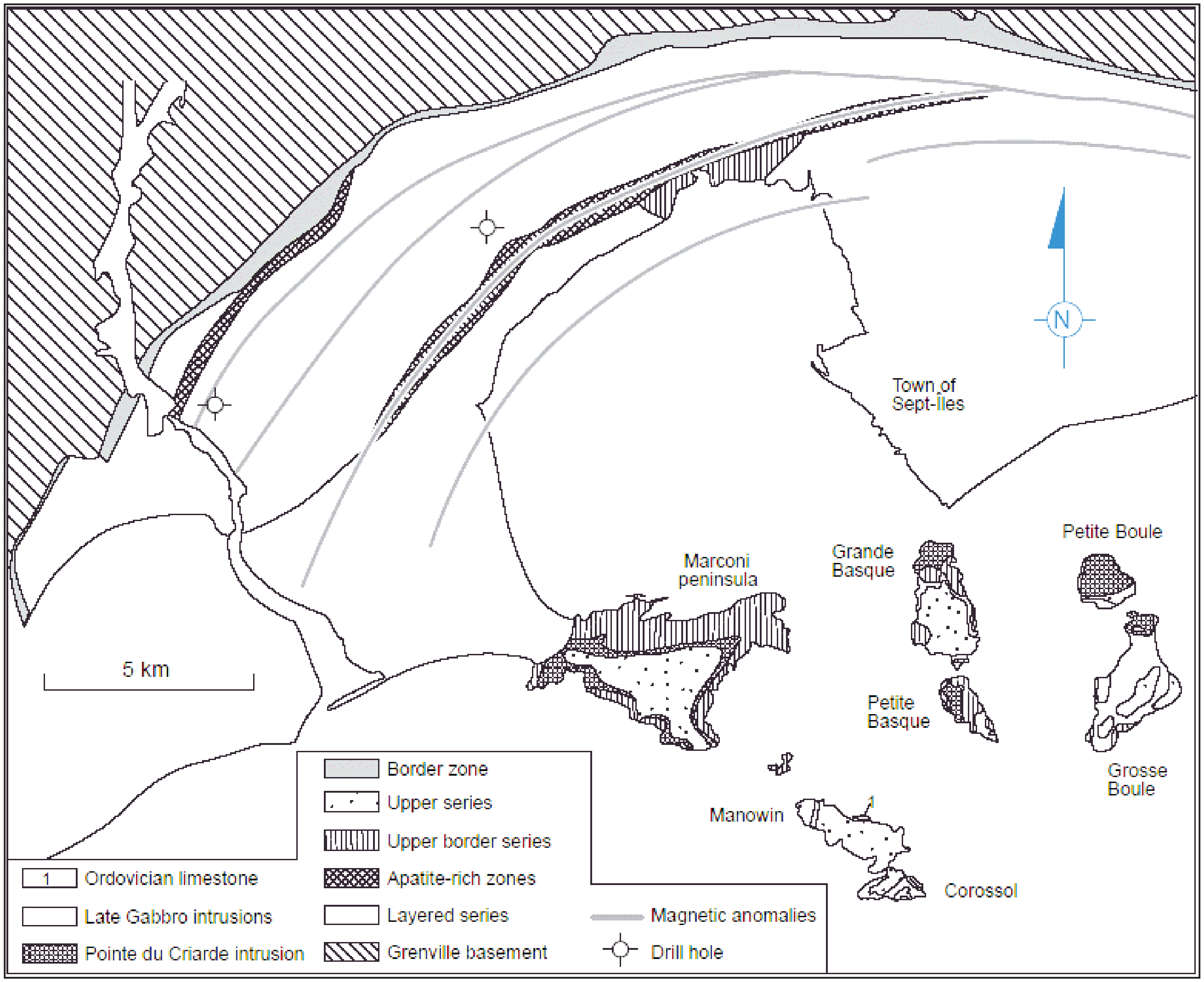

Figure 7-2: Simplified Geology of the N-E Sector of the Sept-Îles Anorthosite Complex (Higgins, 2005)

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, decimeter 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, centimeter 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 centimeter 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% P_2O_5 and 18% TiO_2 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 centimeter 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 lays 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.

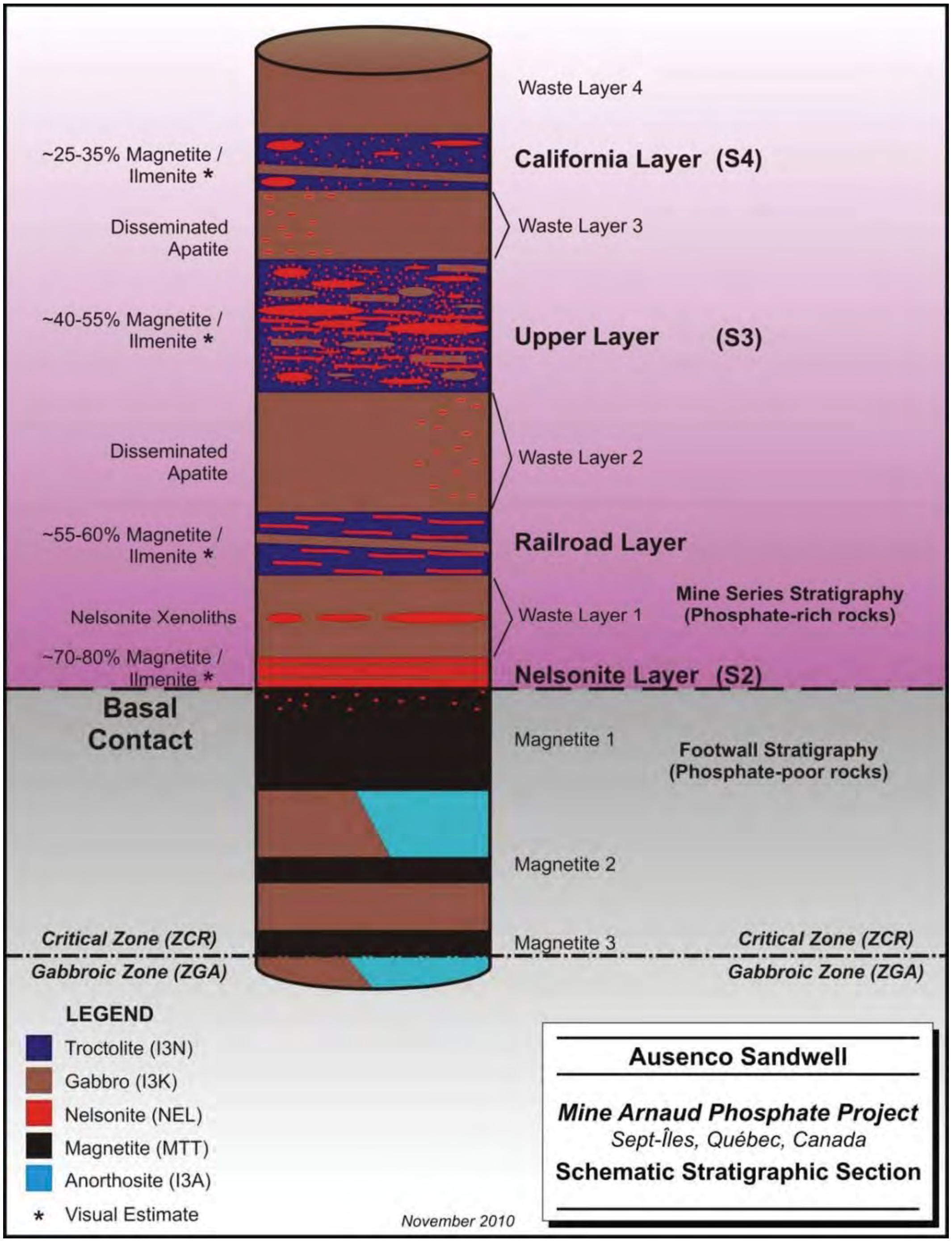

Figure 7-3: Schematic Stratigraphic Section, Mine Arnaud Phosphate Project

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 differentiation which enhances P_2O_5, F and Cl 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

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

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 and Figure 8-2) 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. 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 cross-cutting 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 in drilling has been referred to as the Railroad Layer. 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 in the Nelsonite Layer (visually estimated to range from 55% to 60% magnetite/Ilmenite) and the mode of occurrence changes from massive layeres 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 relatively silicate-rich matrix. As observed with the Nelsonite Layer, narrow dikes of gabbroic composition can be seen cross-cutting 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. This unit is recognized by its thickness, the presence of black iron-oxide minerals, the massive texture of the rock, the presence of discontinuous fragments (xenoliths), and layers of Nelsonite. The concentration of iron-oxide minerals 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 are present in association with the Nelsonite xenoliths, and as disseminated grains throughout the unit. The assay information suggests that the Upper Layer can often be portioned into three sub-layers by the phosphate content. A central, slightly higher grade, sub-layer is observed and seems 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 at all levels. In general terms, the phosphate grade within this unit will decrease, as the quantity of these barren/low-grade xenoliths increases.

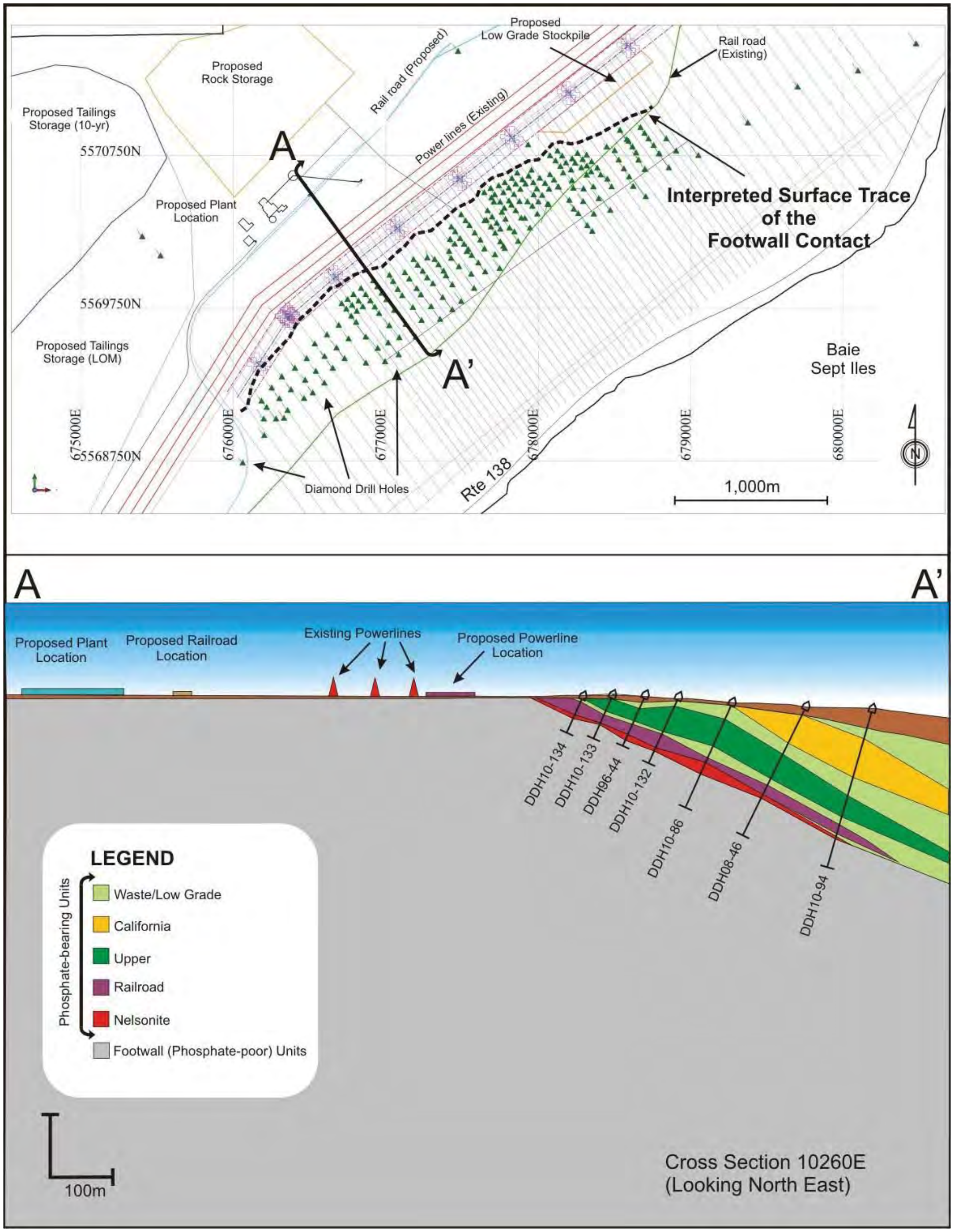


Figure 8-1: Generalized Cross Section 10260E (Looking Northeast)

(Saucier et al., 2012)

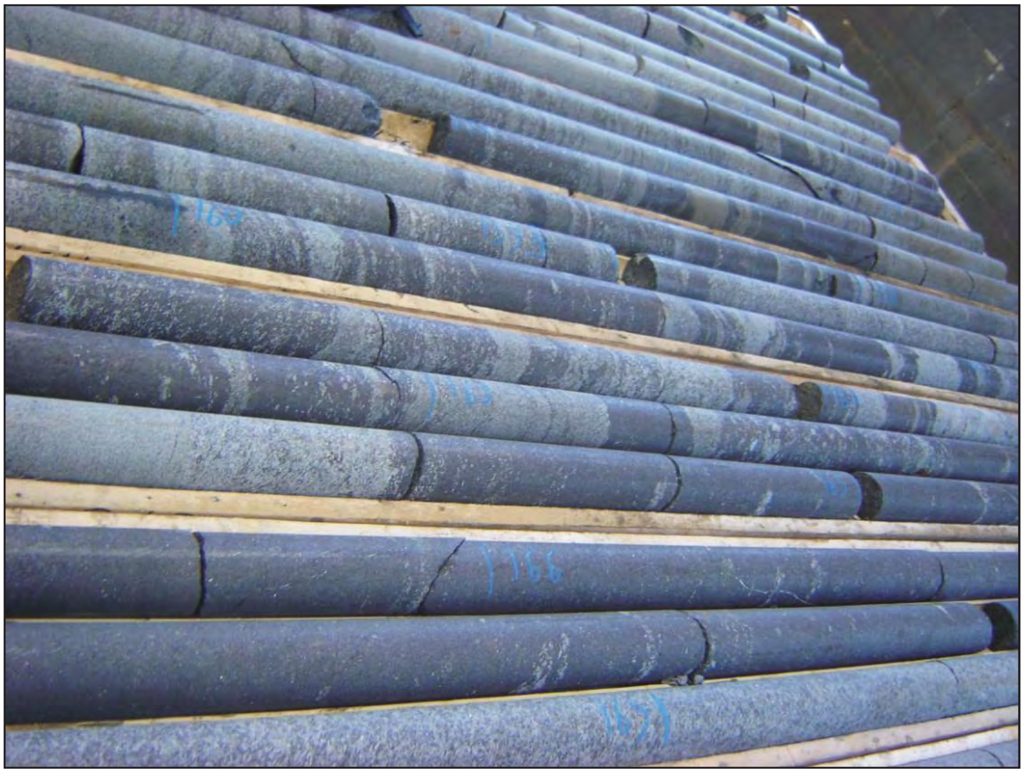


Figure 8-2: View of the Footwall Massive Magnetite Units

Figure 8-3: View of the Nelsonite in DDH1166-10-85 - (129.0 m to 129.5 m Assayed 5.88% P_2O_5)

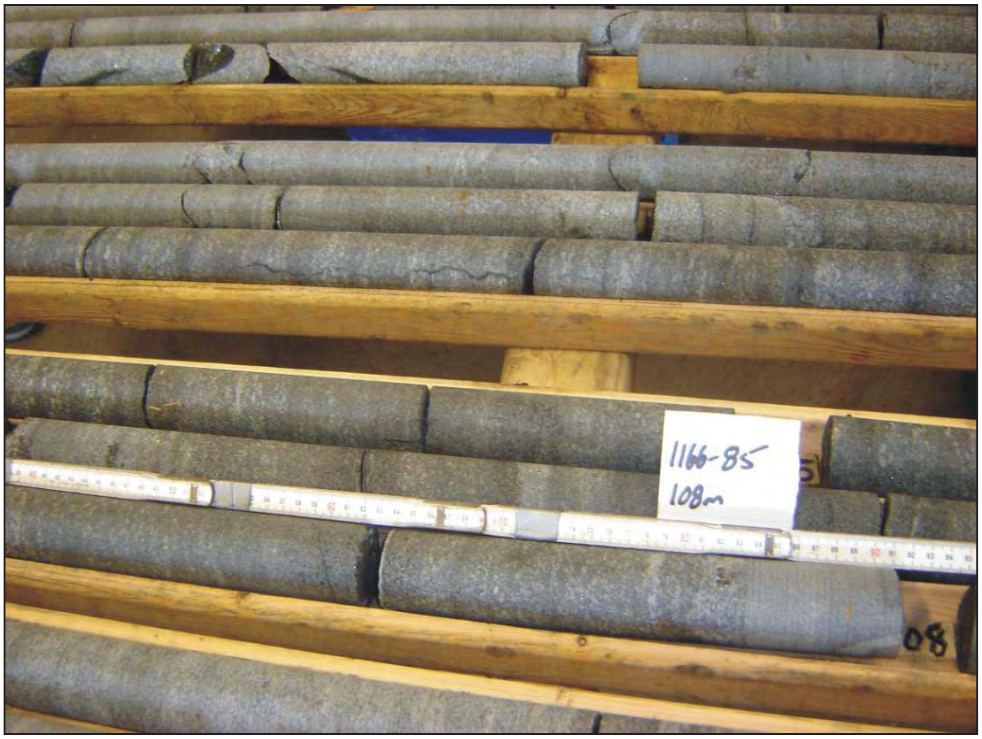


Figure 8-4: View of the Railroad unit in DDH1166-10-85 - (103.0 m to 106.0 m Assayed 10.16% P_2O_5, 106.0 m to 108.0 m Assayed 8.28% P_2O_5)

8.5 California Layer

This layer forms the upper limit of the phosphate-bearing sequence as presently understood. It has not been intersected in drilling as often as the three other layers, as it is located relatively high up in the stratigraphy. The California Layeris only intersected in drill holes that are targeting 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 dykes of gabbroic composition can be seen cross-cutting 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 different layers dominated by units of gabbroic composition. For the most part, these layers are barren of phosphate mineralization, or contain only low values. However, the assay information available shows 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 is commonly observed in association with xenoliths of Nelsonite in Waste Layer 1, while the apatite occurs more often 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 only on occasion and was not correlated from hole-to-hole or from section-to-section.

9. Exploration

The information contained in this item was taken and modified from the Feasibility Study (prepared by Roche-Ausenco) dated February 2012 that was updated by SGS Canada Inc. to a NI 43-101 compliant Pre-Feasibility Technical Report with an effective date of July 24th, 2013.

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 (1977-2010)

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 10.80% 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 from 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 PFS 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 6.60% P_2O_5 and 9.20% 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 600,000 t of apatite and 425,000 t of 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 400,000 t 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 600,000 t of apatite concentrate and 243,000 t 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 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 6.60% 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 4.50% P_2O_5 and 6.70% TiO_2, diluted Mineral Resources were estimated to be 185.6 Mt at 6.20% P_2O_5 and 8.40% 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. The modeling was 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 4.00% P_2O_5 cut-off grade, Indicated Resources were calculated to be 148 Mt at 6.20% P_2O_5, 8.07% TiO_2, 30.70% Fe_2O_3 and Inferred Resources of 86 Mt at 6.30% P_2O_5, 9.10% TiO_2 and 31.20% Fe_2O_3.

9.3 Mine Arnaud Exploration (2010-2012)

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 tonnes of Measured and Indicated Resources at an average grade of 5.32% P_2O_5 and 157,365,106 tonnes of Inferred Resources at an average grade of 4.66% P_2O_5. The cut-off grade used to report the mineral resources was 2.60% 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 and increase the level of confidence of the mineral resources in the expected LoM pit shell.

There has been no production at the project to date.

10. Drilling

The information contained in this item was taken and modified from the Feasibility Study (prepared by Roche-Ausenco) dated February 2012 that was updated by SGS Canada Inc. to a NI 43-101 compliant Pre-Feasibility Technical Report with an effective date of July 24th, 2013.

10.1 Drilling Procedures

Drillholes 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, using DGPS. In addition to the phase one holes, Group Cadorette surveyed the 2008 collars. For phase two, drillhole 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. SGS Geostat agrees with the RPA's statement that, due to relatively shallow depths of the drillholes, the current spacing of the drillhole 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, was 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 2,065.5 m 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. In 2013, core located in Québec warehouse was transported back to Sept-Îles warehouse (543 Perreault, Sept-Îles). 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 N325° 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 azimuth and dip 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 increasing the level of confidence of the mineral resources. The drilling campaign focused primarily on the lateral extension of the deposit and the eastern potential. The drilling campaign followed the recommendation of RPA (Pressacco, 2011) following the resources estimation (Pressacco, 2011).

The drilling was done using “NQ” size drill rods and down hole 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 m 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 hole drilling program was conducted by Mine Arnaud, under InnovExplo’s management. The total length of the drilling campaign was 1,041 m 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.

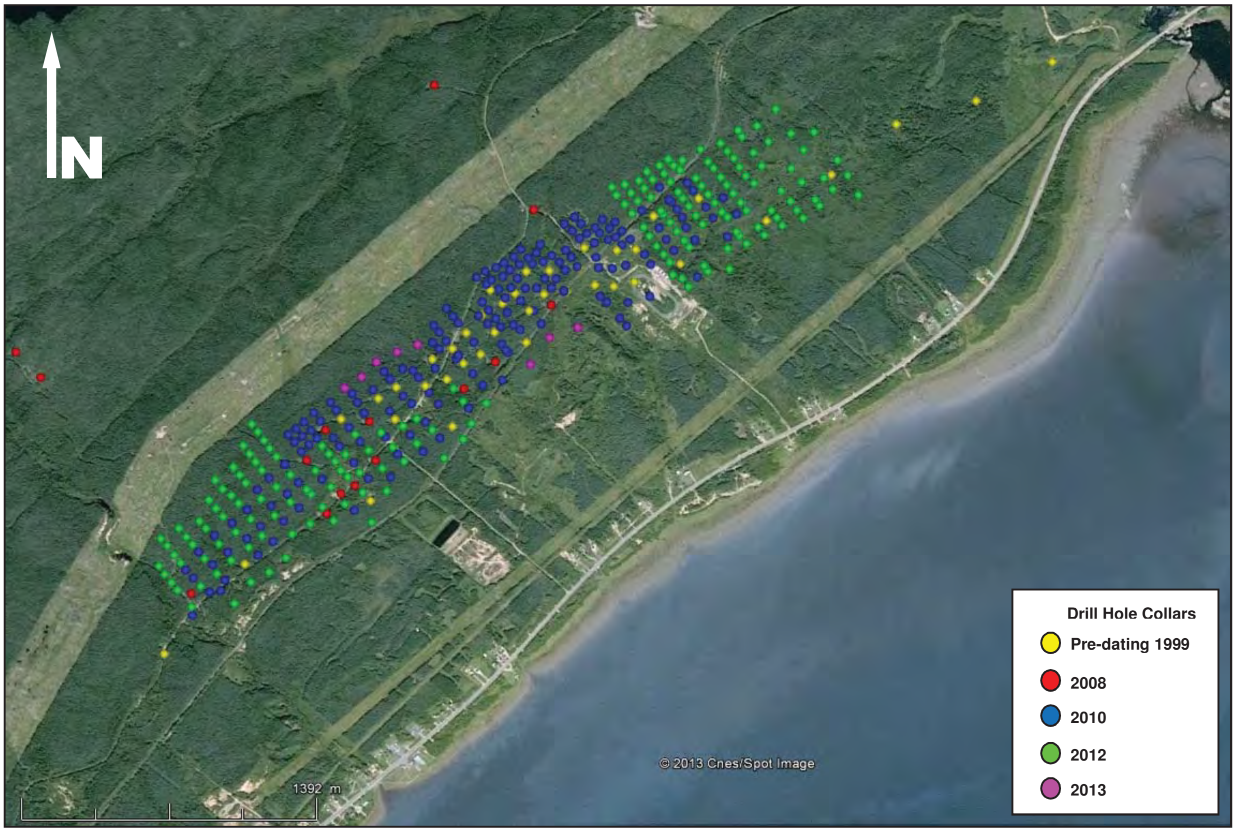


Figure 10-1: Drill Hole Positions

11. Sample preparation, Analyses and Security

The information contained in this item was taken and modified from the Feasibility Study (prepared by Roche-Ausenco) dated February 2012 that was updated by SGS Canada Inc. to a NI 43-101 compliant Pre-Feasibility Technical Report with an effective date of July 24th, 2013.

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 at 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-Îles.

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 Material (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

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 and then its weight 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 (Pressacco, 2011) 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

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 had 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 were 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

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**

For the 2010 drill program a total of 298 lab duplicate samples were taken. As with the 2008 data, RPA reviewed the results for the three economic minerals on scatter and Thompson-Howarth plots. The scatter plots indicated that pulp reproducibility was 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 was noted that 78 results did exceed the detection limit for apatite (0.001% P_2O_5).

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 results 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 (cation) of the minerals so those values were converted by RPA to the oxide value for reporting consistency.

For SCH-1, the accuracy of results was 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 only 210 laboratory duplicates results was available. A total of 8 lab duplicates were from holes newer than 1166-13-189 which was the cut-off for SGS Geostat’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 Geostat for statistical analysis. SGS Geostat plotted the data on scatter plots, Thompson-Howarth plots, and for selected components HARD plots. SGS Geostat 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 P_2O_5 (summary statisticsTable 11-1) there is less than 1% difference between the means of two populations with a slightly higher mean 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)				Bivariate Statistics	
	Original	Duplicate	Relative Difference		Σ(μ₁·μ₂)	2187.31
	P_2O_5	P_2O_5			count	208
Count	208	208	0.00%		Covariance	10.57
Min	0.01	0.02	7.14%		R	0.9867
Max	11.91	12.12	1.77%		R^2	0.9735
μ	2.90	2.89	-0.61%			
median	1.24	1.24	0.00%		SIGN TEST	
skewness	1.17	1.14	-3.00%		Σ [+]	99
σ	3.30	3.25	-1.63%		Min	0.43
kurtosis	0.12	0.07	-39.50%		Max	0.57
range	11.90	12.11	1.77%		Result	0.47
variance	10.89	10.53	-3.24%		Count	208

Table 11-2: Summary Statistics TiO_2 duplicates

	Summary Statistics (univariate)				Bivariate Statistics	
	Original	Duplicate	Relative Difference		Σ(μ₁·μ₂)	7485.50
	TiO_2	TiO_2			count	208
Count	208	208	0.00%		Covariance	36.16
Min	0.95	0.95	0.00%		R	0.9772
Max	26.26	26.66	1.52%		R^2	0.9548
μ	8.03	8.02	-0.13%			
median	5.73	5.82	1.48%		SIGN TEST	
skewness	1.23	1.25	1.90%		Σ [+]	102
σ	6.08	6.09	0.12%		Min	0.43
kurtosis	0.56	0.68	22.09%		Max	0.57
range	25.31	25.71	1.58%		Result	0.49
variance	36.96	37.05	0.25%		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

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

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 the 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)				Bivariate Statistics	
	Original	Duplicate	Relative Difference		$\Sigma(\mu_1 \cdot \mu_2)$	39742.15
	orig	dup			count	208
Count	208	208	0.00%		Covariance	191.99
Min	5.42	5.33	-1.66%		R	0.9617
Max	68.11	68.00	-0.16%		R^2	0.9249
µ	28.89	29.08	0.64%			
median	24.69	24.57	-0.51%		SIGN TEST	
skewness	1.11	1.14	2.53%		Σ [+]	110
σ	13.99	14.27	1.95%		Min	0.43
kurtosis	0.63	0.68	7.82%		Max	0.57
range	62.69	62.67	-0.03%		Result	0.53
variance	195.80	203.53	3.95%		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)				Bivariate Statistics	
	Original	Duplicate	Relative Difference		$\Sigma(\mu_1 \cdot \mu_2)$	38766.91
	orig	dup			count	208
Count	208	208	0.00%		Covariance	187.28
Min	3.06	2.52	-17.65%		R	0.9707
Max	55.08	54.69	-0.71%		R^2	0.9423
µ	31.51	31.54	0.08%			
median	36.54	36.03	-1.40%		SIGN TEST	
skewness	-0.71	-0.72	0.91%		Σ [+]	108
σ	13.90	13.88	-0.12%		Min	0.43
kurtosis	-0.79	-0.76	-3.42%		Max	0.57
range	52.02	52.17	0.29%		Result	0.52
variance	193.16	192.69	-0.25%		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	$\%RD=\frac{\text{Duplicate Value-Original Value}}{\text{Original Value}}$
±20%	92.8%	7.2%	193	15	
±50%	99.0%	1.0%	206	2	
±100%	99.5%	0.5%	207	1	

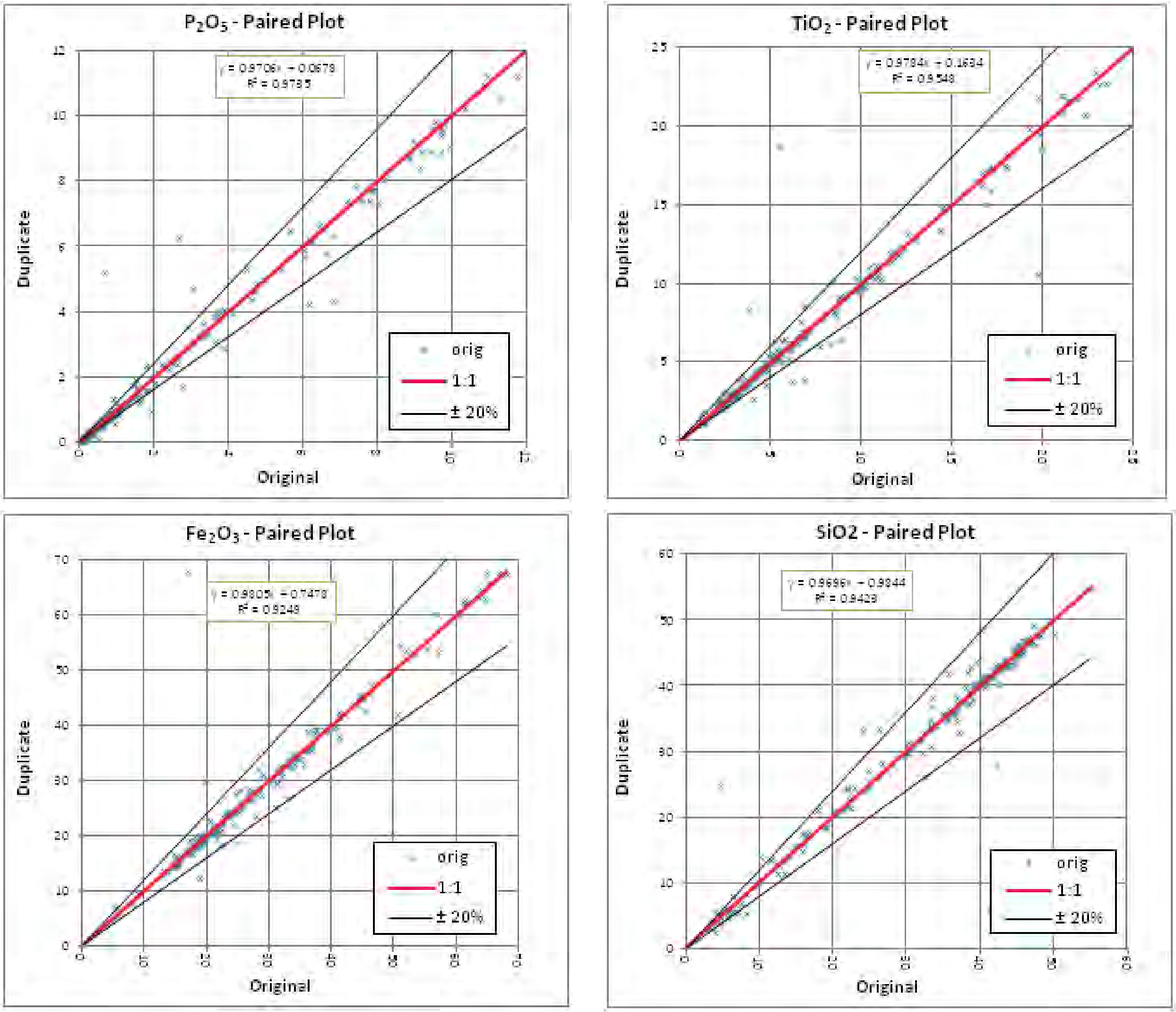


Figure 11-1: Scatterplots for Duplicate analyses

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, they 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 Geostat consideres that the data is adequate for use in the resources estimation phase of the work.

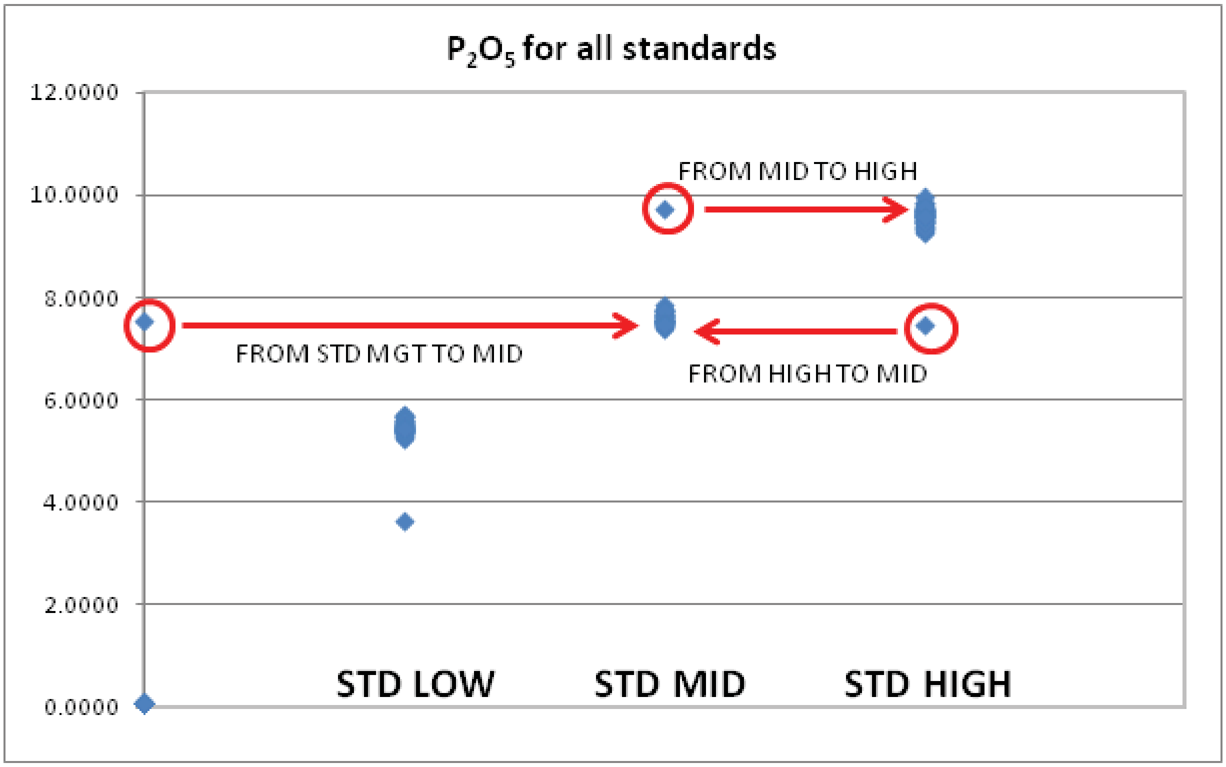


Figure 11-2: Comparison of P_2O_5 values grouped by standard type – All Standards

11.3.1.4 2013 Lab Duplicates

Since the last report, the analytical results for the 8 duplicates have been incorporated to the 2012-2013 QAQC. The duplicate samples are from holes: 1166-13-192, 1166-13-193, 1166-13-196, 1166-13-197. All these holes were drilled in 2013, and there was 8 duplicate samples taken from these 4 holes. The effect of incorporating these duplicates with the 2012 results had minimal impact on previous findings; they slightly improved the (R^2) correlation factors, and sign tests for P_2O_5, TiO_2, Fe_2O_3 and SiO_2. For P_2O_5, TiO_2, Fe_2O_3 and SiO_2 the scatter was close to the 1:1 line when paired duplicates and original assay values. Figure 11-3 illustrates that there is little change with this 8 duplicates when compared to Figure 11-1.

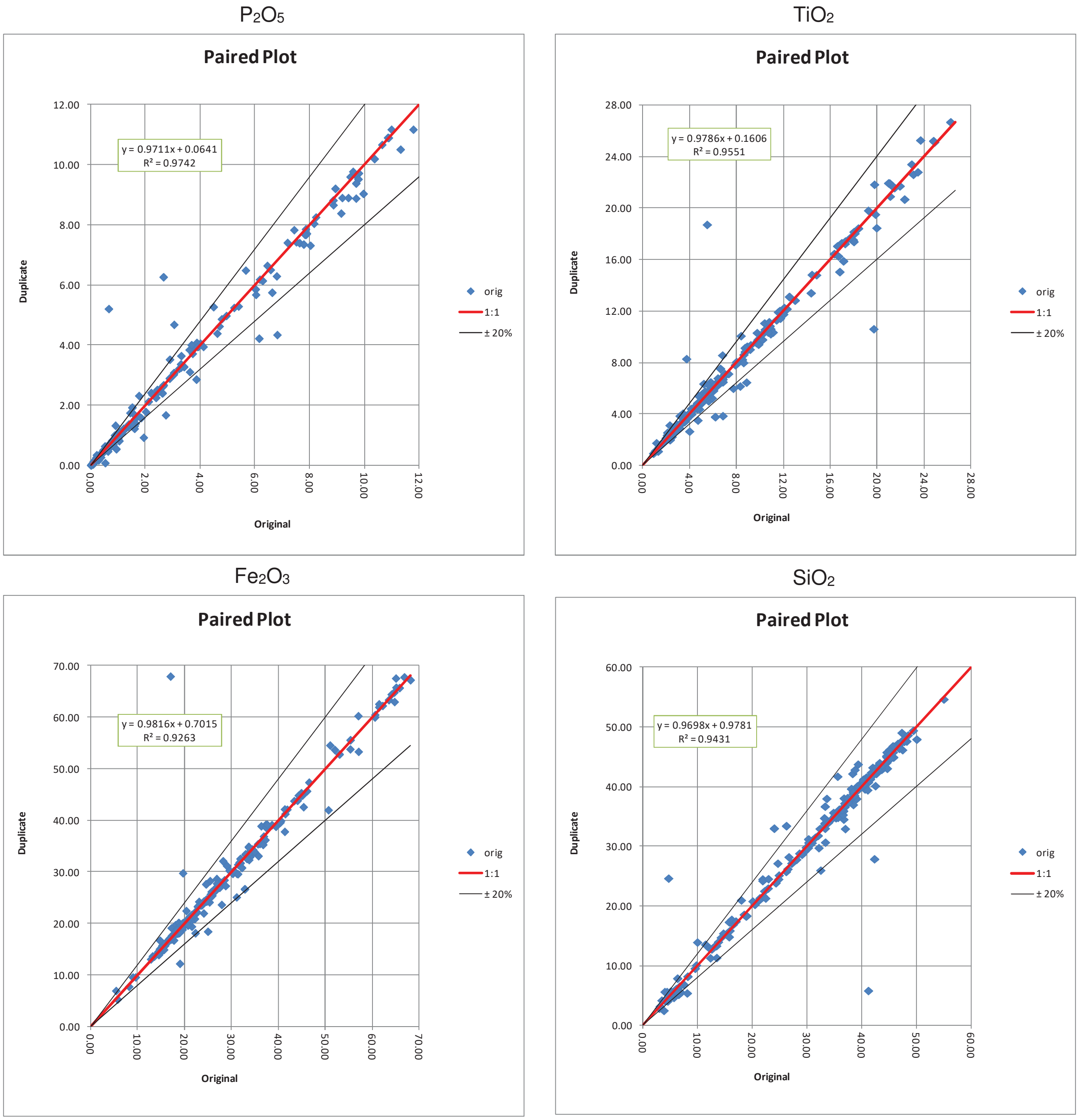


Figure 11-3: Duplicate Paired plots including new data

Figure 11-4 has the new duplicates plotted alone to investigate their individual behavior, which is well correlated.

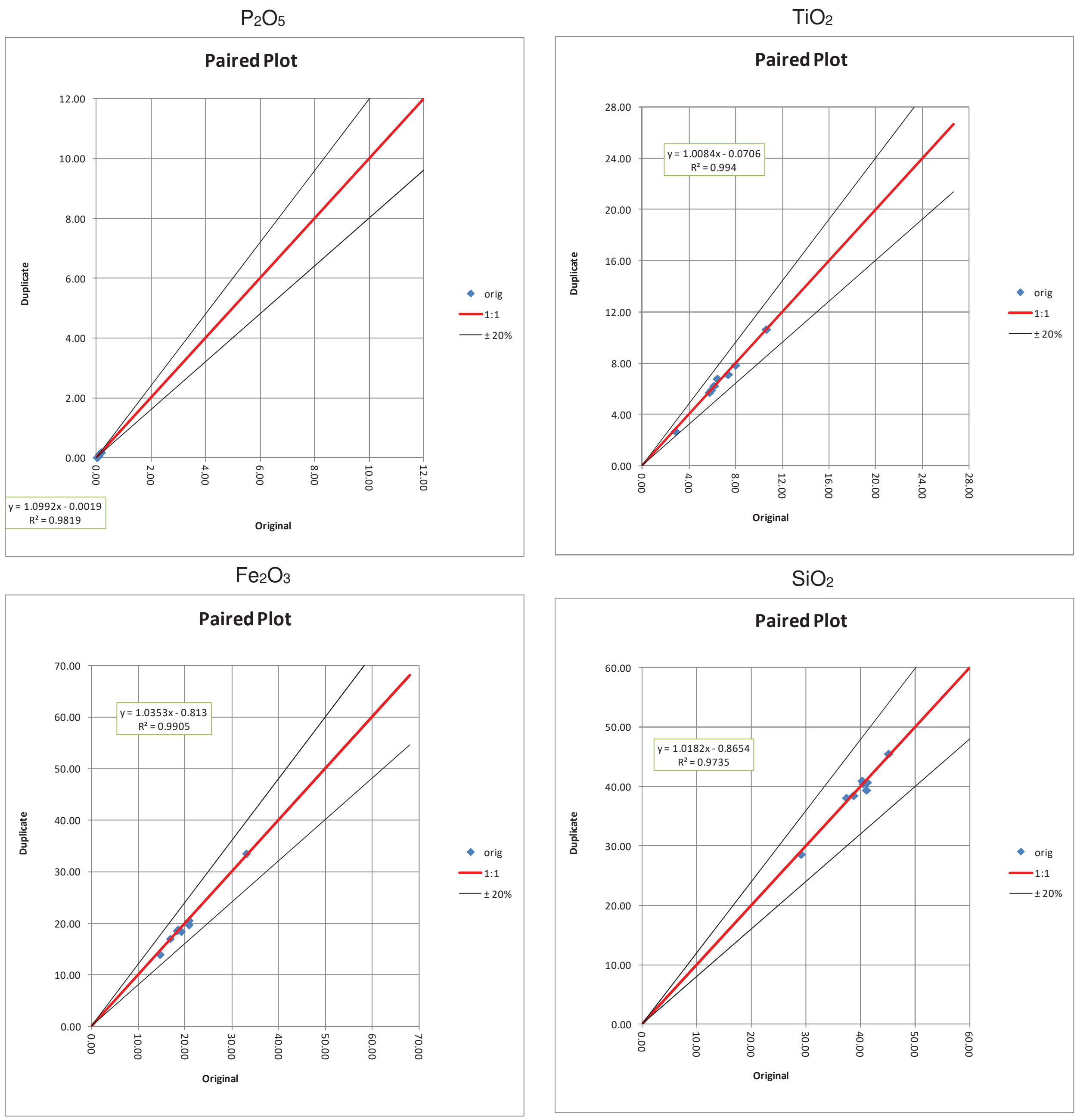


Figure 11-4: Duplicate paired plots with only the new points

To better illustrate the P_2O_5 data, the axes have been expanded in Figure 11-5. Figure 11-4 was left with the smallscale to compare like scales with Figure 11-3 and Figure 11-1.

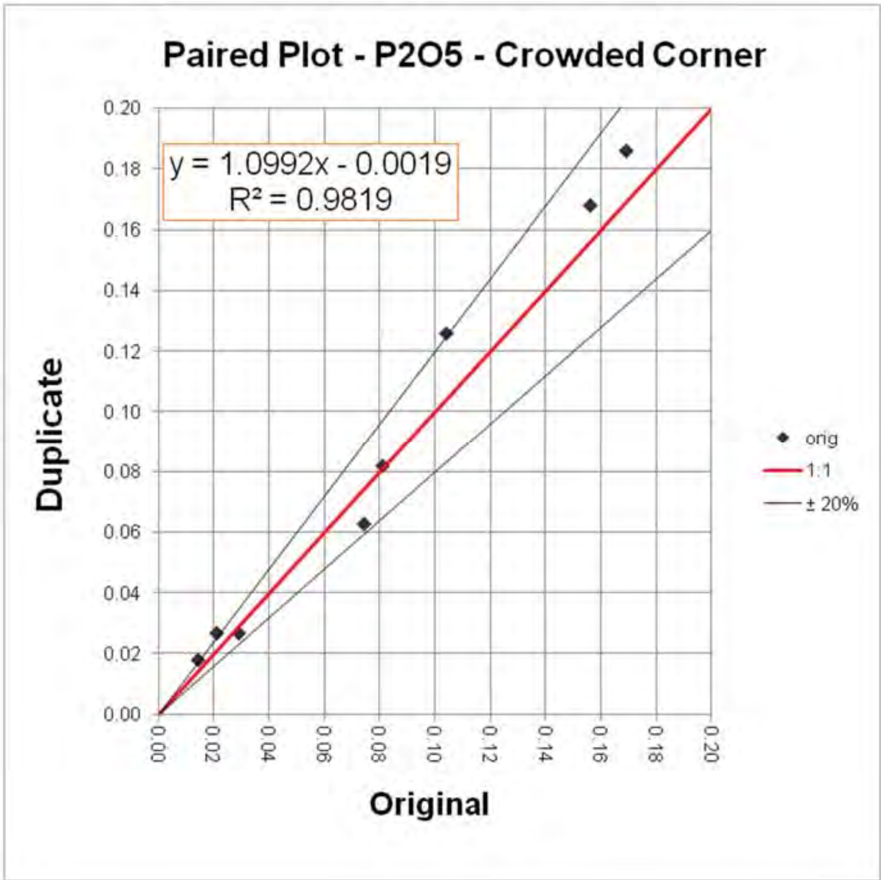


Figure 11-5: Expanded to view the crowded corner

Since the P_2O_5 values are relatively low for these 8 samples, we have an expanded plot, notice most of the duplicate values are a bit higher, regardless there is typically more variation toward the detection limits and P_2O_5 correlation is reasonably demonstrated here.

Table 11-8: Updated Summary statistics for duplicates 2013 (univariate)

	Summary Statistics (univariate)											
	P_2O_5			TiO_2			Fe_2O_3			SiO_2		
	Orig.	Dup.	Rel. Diff.	Orig.	Dup.	Rel. Diff.	Orig.	Dup.	Rel. Diff.	Orig.	Dup.	Rel. Diff.
Count	216	216	0%	216	216	0%	216	216	0%	216	216	0%
Min	0.01	0.02	7%	0.95	0.95	0%	5.42	5.33	-2%	3.06	2.52	-18%
Max	11.91	12.12	2%	26.26	26.66	2%	68.11	68.00	0%	55.08	54.69	-1%
μ	2.80	2.78	-1%	7.98	7.97	0%	28.57	28.74	1%	31.80	31.82	0%
median	1.13	1.14	1%	5.88	5.90	0%	24.11	24.23	1%	36.81	36.44	-1%
skewness	1.22	1.19	-3%	1.26	1.29	2%	1.16	1.18	2%	-0.76	-0.77	1%
σ	3.28	3.23	-2%	5.98	5.99	0%	13.87	14.14	2%	13.74	13.72	0%
kurtosis	0.24	0.19	-22%	0.69	0.82	18%	0.74	0.79	7%	-0.70	-0.67	-4%
range	11.90	12.11	2%	25.31	25.71	2%	62.69	62.67	0%	52.02	52.17	0%
variance	10.77	10.42	-3%	35.81	35.91	0%	192.24	199.95	4%	188.81	188.31	0%

Table 11-9: Updated Summary statistics for duplicates 2013 (bivariate)

Bivariate Statistics				
	P_2O_5	TiO_2	Fe_2O_3	SiO_2
$\Sigma(\mu_1 \cdot \mu_2)$	2248	7535	40570	39370
count	216	216	216	216
Covariance	10.46	35.05	188.70	183.12
R	0.9870	0.9773	0.9625	0.9711
R2	0.9742	0.9551	0.9263	0.9431

Table 11-10: Updated SignTest results including the 2013 duplicates

SIGN TEST				
	P_2O_5	TiO_2	Fe_2O_3	SiO_2
Σ [+]	105	105	114	111
Min	0.43	0.43	0.43	0.43
Max	0.57	0.57	0.57	0.57
Result	0.48	0.49	0.53	0.51
Count	216	216	216	216

By comparing the updated statistics Table 11-8 through Table 11-10 with that of 2012, the new data has not significantly changed the QAQC analysis, the sign test illustrated that there is no major bias since all the values are close to 50%, and between the minimum/maximum windows.

11.3.2 Quality Assurance/Quality Control for Chlorine

11.3.2.1 Chlorine Duplicates

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 11-6. 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.

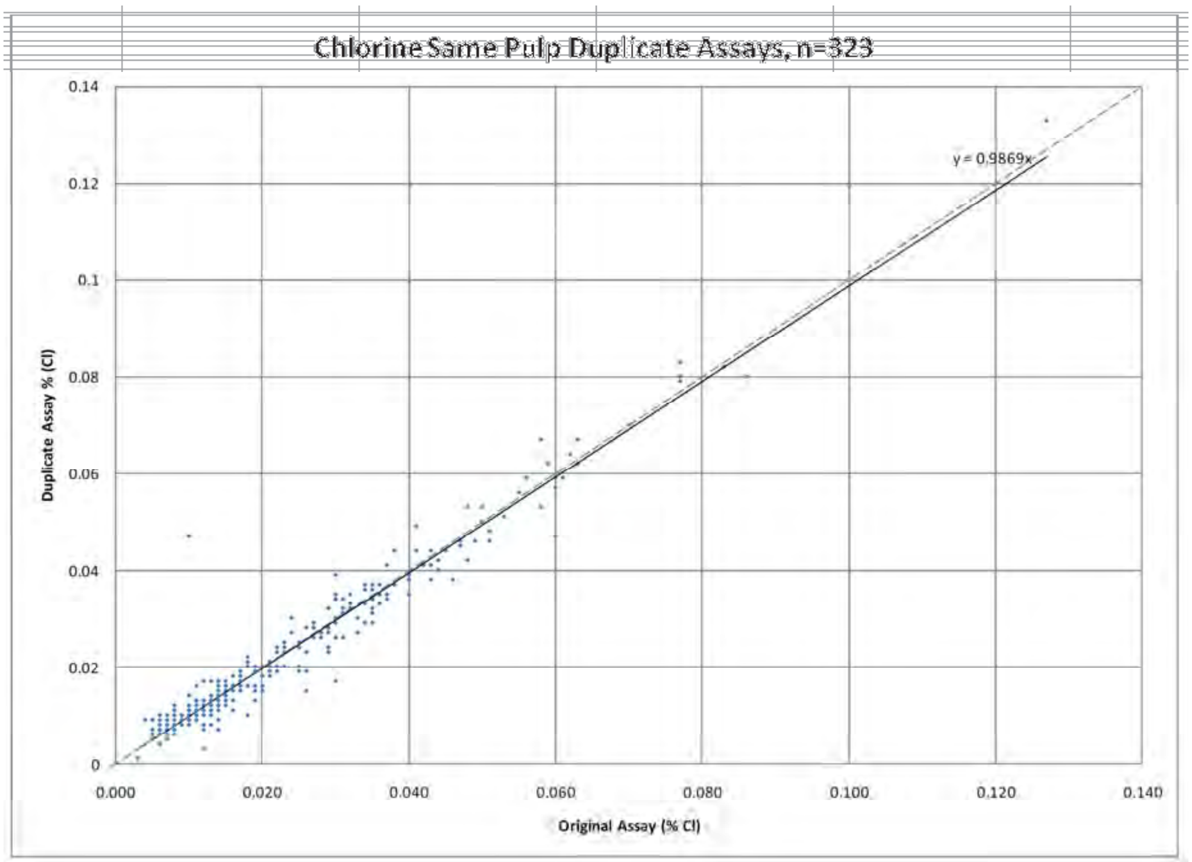


Figure 11-6: Results of Same Pulp Duplicates for Chlorine at ALS

Chlorine Certified Reference Materials

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

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 11-11.

Table 11-11: Samples Comprising IRM Composite

Mine Arnaud Inc. –Mine Arnaud 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. –Mine Arnaud 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 11-12.

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 11-12: Internal Reference Materials and Composite Proportions

Mine Arnaud Inc. –Mine Arnaud 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 11-14 to Table 11-15.

Table 11-13: XRF P_2O_5 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 11-14: XRF TiO_2 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 11-15: Fe_2O_3 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 Geostat and COREM but the dispersion was minimal;
- For magnetite, ALS returned values 2% lower than SGS Geostat and COREM but, again, the dispersion was low.

SOQUEM concluded that the internal IRMs were acceptable for apatite analysis by XRF, but recommended that these IRMs 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

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

The same quality assurance program as 2012 is in place, all additional QAQC samples were inserted into the statistical analysis since the last geological database cutoff date.

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, 2012 and 2013 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 SGS Geostat's 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. SGS Geostat 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.

SGS Geostat 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 Geostat 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 Geostat'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

The information contained in this item was taken from the Feasibility Study (prepared by Roche-Ausenco) dated February 2012 that was updated by SGS Canada Inc. to a NI 43-101 compliant Pre-Feasibility Technical Report with an effective date of July 24th, 2013.

Mine Arnaud believes that no change in this area is required to integrate commitments related to environmental process.

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 visits 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 by SGS Geostat and Roche 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 Geostat using Access and Genesis©. Deviation at 0 m depth were also removed from the deviation table and transferred to collar orientation columns in the collar table, which is database structure specification for Genesis©.

As part of past data verification, RPA had conducted spot checking of the drill hole database which still stands for SGS Geostat validation process. Approximately 10% of the drill holes that intersected the mineralized domains 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 drillhole 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 drillholes (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 drillhole 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 drillhole 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.

As a result of its data validation efforts, SGS Geostat believes that the drillhole 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, a serie 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 Geostat.

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 sets returned a R^2 = 0.9913 which illustrates a strong correlation between the original and duplicate values (Figure 12-1). Only a slight difference is observed in the mean between the two sets 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 Geostat that the P_2O_5 assay values show a strong enough reliability to be used in the process of mineral resources estimation.

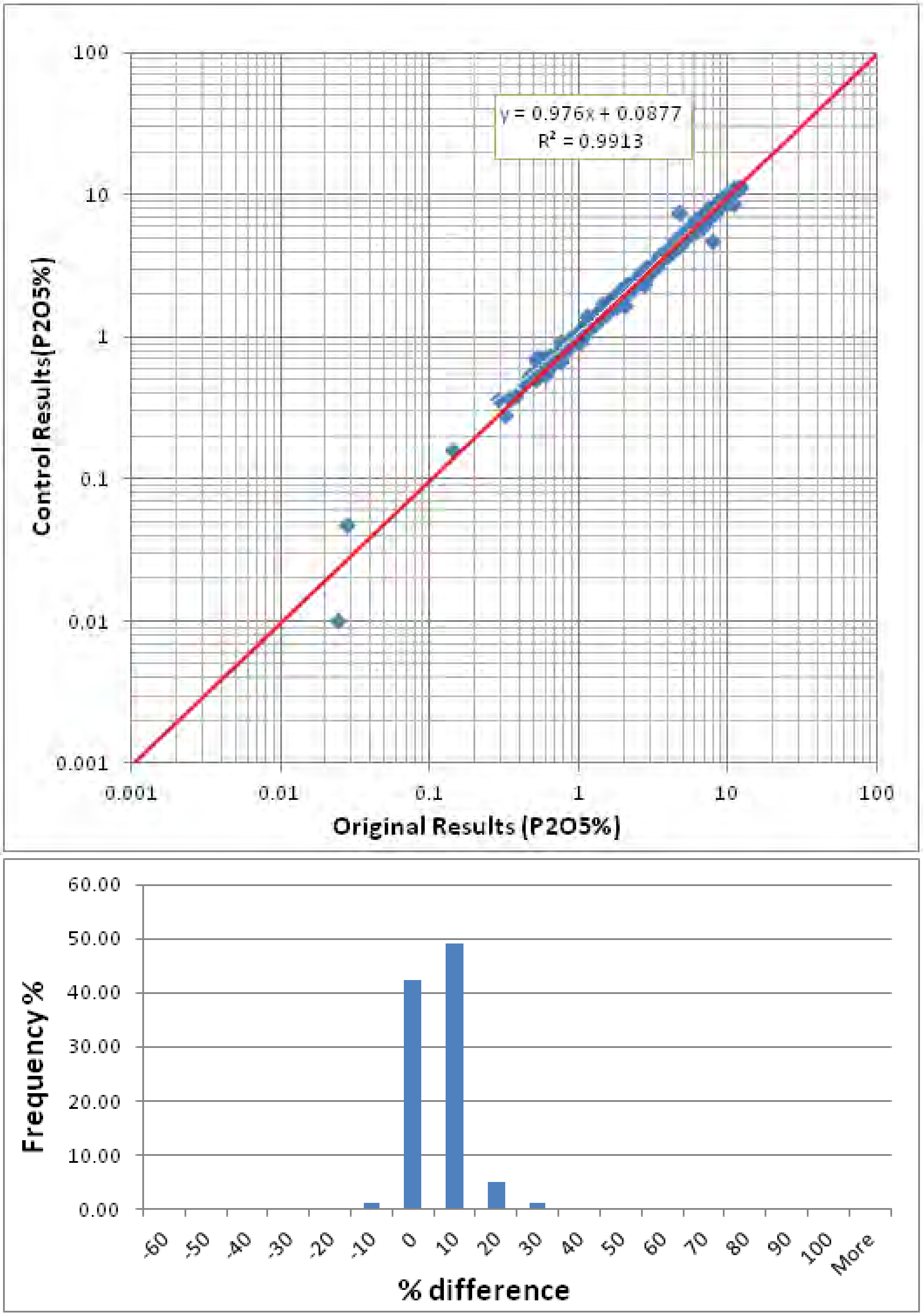


Figure 12-1: Statistical comparison between original assays and independent control samples

12.3 2013 Independent Metallurgical Testing

During the November 2012 site visit by Jean-Philippe Paiement, M.Sc. P.Geo and Floran Faiello, a serie 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 Item 13.4 of this report.

Table 12-1: List of composites used by SGS Geostat 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 2014 Independent Metallurgical Testing

In 2014, Roche proceeded to additional Metallurgical parameters testing of the Sept-Îles deposit. These tests were made on material from drillholes, representing the material mined between years 20 through 28 of the planned life of mine. SGS Geostat reviewed the sample selection for these additional tests and produced a memorandum (APPENDIX B – Memorandum Metallurgical Test Validation) signed by Jean-Philippe Paiement, M.Sc., P.Geo. Item 13 explains results from these tests (COREM, project T1654).

12.5 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 APPENDIX C – 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.

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

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 D – Yara Concentrate Characteristic Letter.

13. Mineral Processing and Metallurgical Testing

13.1 Introduction[1]

Over the years, many metallurgical testwork programs and other mineralogical studies were carried out on the Arnaud's apatite ore.

Mineralization of the Sept-Îles apatite, magnetite and ilmenite deposit was recognized early in 1992 by SOQUEM. Several metallurgical testwork programs 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 testwork. The sections which follow review the metallurgical testwork carried out and how the data have been used to establish the process flowsheets, design criteria, and process plant design for the Feasibility Study.

13.2 Grinding Characteristics and magnetic separation

The first comminution characterisation was performed on weathered ore (surface samples) representing four ore types: Nelsonite (S2), Upper (S3), California (S4), and Railroad (SRR) ore. The comminution characterisation allowed conducting small-scale continuous grinding tests via MacPherson tests, which confirmed the applicability of SAG milling for the Mine Arnaud deposit. A series of SMC tests that are considered as abbreviated Drop tests were also performed with various ore types from various locations and proportions. Bond Work index on the same samples were also performed.

In addition to the characterisation, the Bond Work index was determined for series of samples from various locations in the deposit and representing the various ore types. The resulting indices added confidence that the variability in terms of grinding characteristics should not represent a major problem for the operations, because it falls within usual acceptable limits that do not require specific mitigation measures.

Some of the samples tested with the JK drop test characterisation show strong evidence of bimodality which makes the density of the Autogenous mill charge impossible to predict without continuous testing.

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

The bimodality could imply that the ore can break with two different patterns and therefore one portion of the ore can eventually exit the mill at a different rate than the other portion. In order to eliminate this doubt, continuous MacPherson tests were conducted. These tests confirmed that the mill charge builds up with a coarse high-density material, but the throughput stability was not affected during the tests that last at least 6 hours.

The JK drop-weight test results concluded that the samples were characterised as soft with respect to resistance to impact (A x b), but were hard with respect to abrasion breakage (t_a). The Bond low-energy impact test determined the Bond Crusher Work Index (CWI), which fell in the medium to moderately hard range of hardness in the SGS database. The Bond Rod Mill Work Index (RWI) showed that the samples were categorized as soft with RWI ranging from 9.3 to 11.2 kWh/t. The Bond Ball Mill Work Index (BWI) showed that the samples were categorized as moderately soft to medium, with BWI ranging from 11.6 to 14.2 kWh/t. Finally, the Bond Abrasion Index (Ai) showed that the samples fell in the medium to abrasive range, with abrasion indices varying from 0.311 g to 0.526 g. Results are shown in Table 13-1.

Table 13-1: Grindability Characterisation Testwork Results

Sample Name	Relative Density	JK Parameters			MacPherson Test		Work Indices (kWh/t)				Ai (g)
		Axb[1]	Axb[2]	t_a	(kg/h)	(kWh/t)	AW_i	CW_i	RW_i	BW_i	
Railroad	3.61	89.5	-	0.46	18.9	5.1	8.6	-	-	11.6	-
Railroad (1997 ARM-0294)	-	-	-	-	*11.2*	*8.4*	*12.6*	-	*10.1*	*13.7*	*0.311*
Nelsonite	3.26	65.4	-	0.37	12.6	7.1	11.3	12.1	11.1	13.0	0.382
Upper	3.58	75.1	-	0.31	16.7	5.4	9.3	8.0	9.3	12.5	0.311
California	3.31	55.6	-	0.27	-	-	-	10.9	11.2	14.2	0.526
Year 11 Blend	-	-	-	-	18.8	4.8	8.2	-	-	-	-
51% Railroad - 49% Nelsonite	3.66	-	76.0	0.54	-	-	-	-	-	12.4	-
71% Upper - 29% Nelsonite	3.51	-	75.2	0.56	-	-	-	-	-	12.8	-
74% Railroad - 26% Nelsonite	3.55	-	82.1	0.60	-	-	-	-	-	12.2	-
76% Upper - 24% Nelsonite	3.69	-	64.0	0.45	-	-	-	-	-	12.0	-
80% Railroad - 20% Nelsonite	3.49	-	81.0	0.60	-	-	-	-	-	12.6	-
100% Upper	3.33	-	52.4	0.41	-	-	-	-	-	13.2	-
100% Upper	3.45	-	56.4	0.42	-	-	-	-	-	12.7	-
100% Upper	3.32	-	53.5	0.42	-	-	-	-	-	13.9	-
100% Upper	3.55	-	65.9	0.48	-	-	-	-	-	12.2	-
22% Railroad - 18% Upper - 60% Nelsonite	3.67	-	85.8	0.60	-	-	-	-	-	12.3	-
100% Upper	3.50	-	57.5	0.42	-	-	-	-	-	12.9	-
40% Railroad - 40% Upper - 20% Nelsonite	3.51	-	68.9	0.51	-	-	-	-	-	12.5	-
61% Railroad - 39% Nelsonite	3.68	-	69.8	0.49	-	-	-	-	-	11.8	-
69% Railroad - 31% Upper	3.49	-	62.8	0.46	-	-	-	-	-	12.4	-
Average	3.53	68.7		0.47	15.7	6.2	10.0	10.3	10.5	12.7	0.383

[1] A x b from Drop Weight Test

[2] A x b from SMC test

Italicized result were reported in 1997

In addition, simulations for a SAG/Ball mill circuit and an HPGR/Ball mill circuit were performed at SGS-Lakefield. In spite of the positive test results, HGPR selection for the commercial plant cannot be recommended until further investigation with bigger size equipment is completed in order to give a higher confidence to the results and the requirements for sizing. Considering that the resistance to impact value of the ore was in the soft range of hardness, a pebble crusher was not included in the simulation and only the SAB flowsheet was investigated

Following the mineralogical observations and liberation study on the various ore types, a P_{80} of 125 µm was selected as the most appropriate grind size to obtain proper liberation. The addition of a Low Intensity Magnetic Separator (LIMS) prior to flotation has been considered based on mineralogical observations showing significant proportions of magnetite and titano-magnetite in different areas of the deposit. Testwork was conducted with the objective of minimizing the Fe + Al content in the rougher concentrate according to an optimal P_2O_5 recovery. Depending on the ore types and samples, the magnetic proportion varied from 15% to 45% by weight of the ROM and the grade of P_2O_5 in the magnetic product was usually ≤ 1%. Based on those observations, it was decided to pursue the flotation development with the use of a LIMS prior to flotation to focus on the reduction of the Fe + Al content in the final flotation concentrate. It appears also that the use of LIMS prior to flotation helps to reduce the depressant consumption and enhance the efficiency of the collector.

13.3 Flotation Testwork

A more modern flowsheet for the recovery of apatite and ilmenite was developed by GTK in Finland in 2009. In 2010, 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 Lakefield 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 Lakefield. Once the SGS Lakefield 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.

The results of this additional laboratory testwork (COREM projects T1224 and T1242) served as the basis of the internal Feasibility Study prepared by Roche-Ausenco in 2012.

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 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% P_2O_5 could be produced with a P_2O_5 recovery of over 88%, while meeting the specifications of Yara (Mine Arnaud's principal buyer for the apatite concentrate) for impurity content. The grade and recovery obtained for each sample taken during the pilot plant are shown in Table

13-2. 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 the internal Feasibility Study prepared by Roche-Ausenco, it was 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 13-2: Comparative results obtained at the pilot Plant for the first and second phase (2011)

Pilot Plant	Sample*	Raw Feed	Final Concentrate		
Date - Hour	Type	P_2O_5 %	Wt Rec%	P_2O_5 %	P_2O_5 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
March 8	SRR/S2/S3	7.43	17.03	37.59	86.62
March 9	SRR/S2/S3	7.66	7.91	41.24	42.61
March 10	SRR/S2/S3	7.80	6.93	41.12	36.51
March 11	SRR/S2/S3	7.70	6.47	41.22	34.60
March 14	SRR/S2/S3	7.74	11.53	38.59	62.25
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
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
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 = Railroad
S2/S3 = 20% Nelsonite, 80% Upper
SRR/S2/S3 = 55% Railroad, 25% Upper, 20% Nelsonite

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 the capital expenditure, Mine Arnaud following recommendations from both COREM and Roche-Ausenco and advised by SGS Geostat, decided to add to the Roche-Ausenco Internal Feasibility Study by reworking the mining block model and also by performing more metallurgical testworks on individual type of ore and on the most probable mill feed blend.

Following the Roche-Ausenco Internal Feasibility Study more laboratory tests were carried out both at the bench scale and at pilot plant level.

13.4 Validation of Pilot Plant Flowsheet[2]

A meeting held in the end of June 2012 confirmed that Mine Arnaud and its partners wanted to go forward with new 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 types). Also, some concerns were raised regarding the blending of the ore zones during the previous pilot plant testing (SGS lakefield, 2011).

To do so, Mine Arnaud arranged to send to COREM: 13 040 kg of Upper ore, 16 040 kg of Railroad ore and 12 870 kg of Nelsonite ore.

13.4.1 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% Railroad, and 19.4% Nelsonite by weight.

Upper/Railroad/Nelsonite Blend Chemical Analysis

The chemical assay of the blend head sample is presented in Table 13-3

Table 13-3: Chemical analysis for selected elements of the composite

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 13-4 gives the results of the low energy impact test as well as the specific gravity of the ore blend.

[2] This section is for a great part an excerpt from the SGS-GEOSTAT Pre-Feasibility Study (July 2013).

Table 13-4: 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 13-5 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 Work Index 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 13-5: 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 WI (KWh/t)
Upper	36.1	2180.0	124.6	13.1
Railroad	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.

13.4.2 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 13.4.1 above[3]. Mine Arnaud requested to validate the final flowsheet by operating the

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

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). 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 13-6: Lock cycle test on an ore blend comprising 36.1% Upper, 44.5% Railroad, and 19.4% Nelsonite

	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

Table 13-7: Comparative results obtained in pilot plant (flotation columns) on same ore blend as for above LCT - October 2012(includes magnetic separation)

	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 supervised by Roche and were conducted after the internal Feasibility study. Continuous long-term pilot plant testing is a very challenging task compared to operating a pilot plant circuit for a short period of time (≈4-6 hrs) where all the operating and mechanical parameters are controlled and are under steady state. Maintaining steady state for this project is almost impossible because batch grinding is performed independently. Furthermore, the flotation feed tank acts as a buffer tank with a level that varies making it difficult to feed the circuit at a constant rate over a very long period of time.

Usually the sampling campaign for a pilot plant is performed during times where all parameters of the operation are under control. While continuously piloting over a long period of time, composite samples are taken at a certain frequency, which includes times where not all the parameters are necessarily perfectly controlled. This explains some of the unexpected grade and recovery results during the 96 hour continuous pilot plant test, as reported in Table 13-7. For some work-shifts, all samples composites were made up of samples taken during non-steady state operation. To achieve an optimal pilot plant operation, all equipment would need to be sized and automated correctly for a certain feed rate.

13.5 Additional Flotation Tests

Due to the fact that both pilot plant runs returned ambiguous results (see above) and flotation tests performed at SGS Lakefield failed to attain acceptable concentrate grades and recoveries, it was decided to conduct, under the supervision of SGS Geostat's metallurgist, four more lock cycle tests (LCT) at COREM, under the project number T1518 in June 2013.

These lock cycle tests were to be carried out on the LIMS non-mag material produce during COREM project T1405 that was previously experimented (see Table 13-6 above) and on three more composites of different grades - representing low, medium, and high P_2O_5 grade - 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 came from crushed rejects from drill core samples, they are more representative of the actual P_2O_5 grade and ore type 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.

13.5.1 Results of Additional Flotation Tests

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

Table 13-8: 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

The following three (3) LCT was performed on composites of different grades, representing medium, high and low P_2O_5 grade.

Table 13-9: 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 13-9) comes from drill core samples whereas the composite used for the repeat of the lock cycle test of September 2012 (Table 13-8) came from a bulk sample retrieved from the surface of the deposit.

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

Table 13-10: 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 13-11: 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, it became clear that there is no simple or single reagent suite for flotation reagent addition and that the reagents must to be closely monitored and paired 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 performed on a new sample.

Table 13-12: 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

This indicate that the flotation parameters such as depressant dosage and sodium silicate dosage should be fine-tuned to the flotation feed and gangue matrix in order to reach the targeted grade/recovery of P_2O_5 concentrate.

13.5.2 Flowsheet

Laboratory locked cycle flotation tests were conducted using the reagent scheme developed for pilot plant test in 2011 (COREM's project T1224) and confirmed in pilot plant performed in 2012 (COREM's project T1405) as baseline. The reagent scheme includes LIACID (apatite collector), wheat starch (iron-bearing depressant), sodium silicate (gangue dispersant), and caustic soda (pH control). The flotation flowsheet used is presented in Figure 13-1

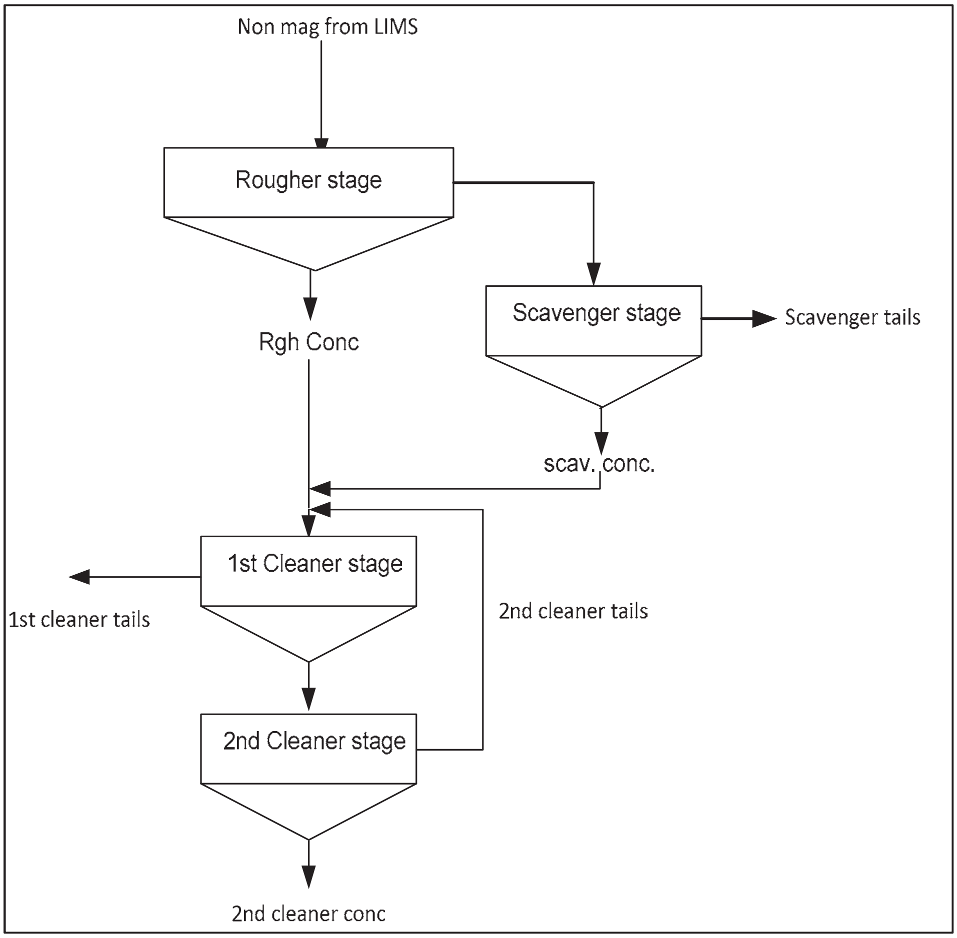


Figure 13-1: LCT's flowsheet

13.5.3 Analysis Results Validation

Upon reception of the concentrate assay results, SGS Geostat requested that the samples be analyzed in a third party laboratory, in order to ensure accuracy of the data. Flotation 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 Geostat.

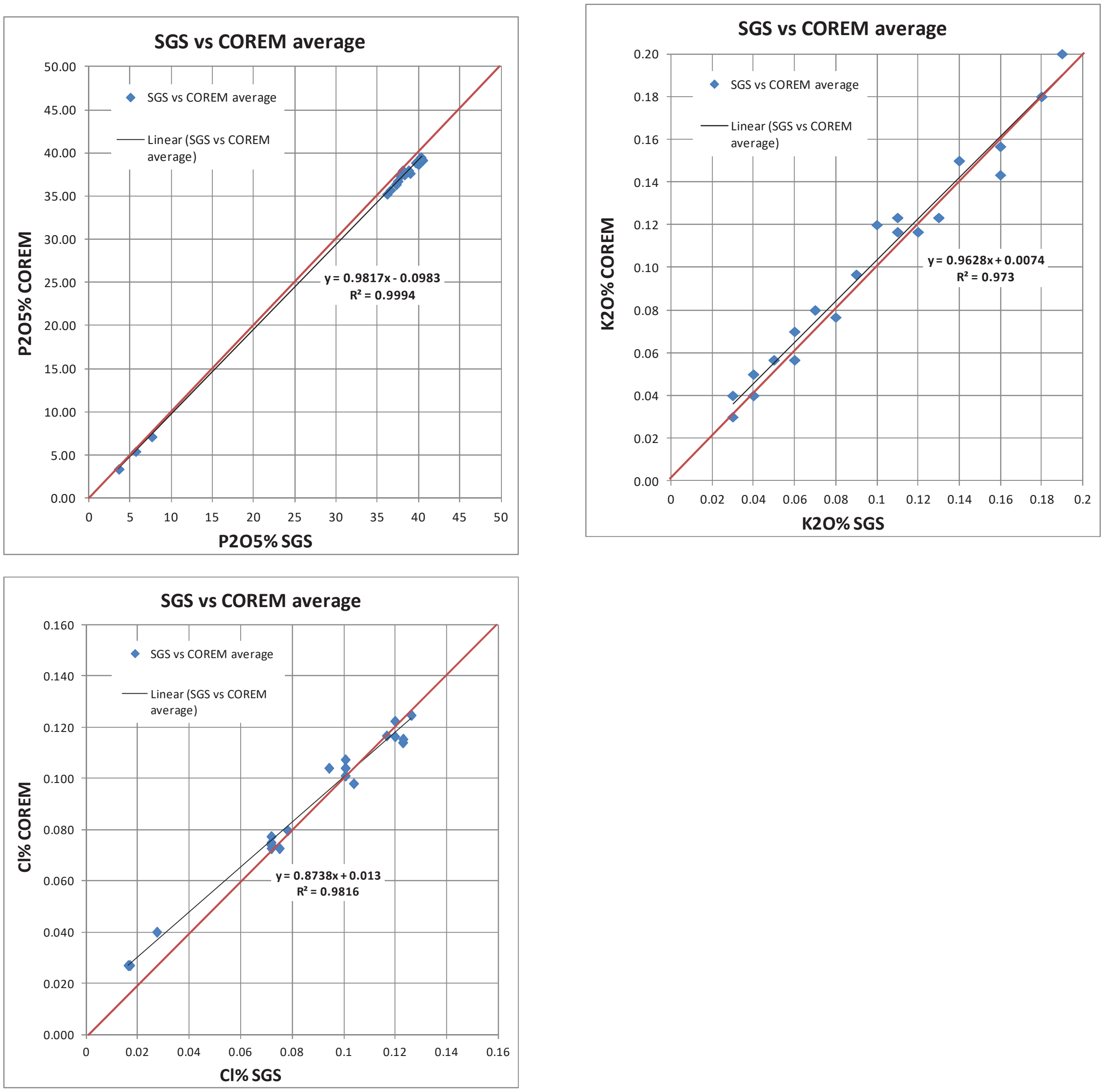


Figure 13-2: Comparison scatter plot of COREM and SGS analytical results

13.5.4 **Conclusion**

As 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 on a continuous basis, more tests will have to be done. Presently, the only way to assess if the apatite concentrate is clean enough at the laboratory level is by visual examination of a minute concentrate sample under a microscope. If not, more reagents are added to the subsequent flotation stage. A quick analyzing device (at least for iron and titanium) should be used for the next series of tests.

It is clear that there is no such thing as a unique reagent suite/dosage “recipe” for the flotation of the apatite and some “fine tuning” as a function of the P_2O_5 grade and gangue matrix of the feed will eventually have to be done since no real reagents optimization has been tried on Mine 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.

13.6 Weight Recovery Curve

Based on selected results of COREM's 2011 pilot plant (COREM’s project T1242) and the results of COREM's lock cycle test on an ore blend comprising 36.1% Upper, 44.5% Railroad, and 19.4% Nelsonite performed in September 2012 (COREM's project T1405) plus the five lock cycles tests of same ore blend but with different grade performed in June 2013 (COREM's project T1518), a relation between the concentrate weight recovery and the P_2O_5 head grade was prepared. The pilot plant work performed initially by SGS in 2010 was not considered due to differences of the process flowsheet.

There were nine (9) test results from the initial pilot plant work at COREM which were excluded from this study because they were judged as non-optimal. This is due to the difficulty of maintaining a stable operation during piloting and therefore said samples were not sampled during steady state.

There was some variability with the quality of the concentrate, with grade ranging from 36.16% to 40.69% P_2O_5 and the recovery of the apatite ranging from 81.24% to 94.14%. For all test results, the weight recovery was adjusted to correspond to a production of a 39% P_2O_5 concentrate grade which suits the new criteria of Mine Arnaud and Yara. Roche/Ausenco considered that the P_2O_5 recovery would remain similar for the adjusted weight recovery, even though this assumption is not exact. In reality, decreasing the concentrate grade would normally increase the P_2O_5 recovery, whereas increasing the concentrate grade would normally decrease the P_2O_5 recovery. Since none of the tests can be considered as optimal, it is very difficult to make a precise estimate of the effect of concentrate grade variation on the P_2O_5 recovery. This has an impact of a few hundredths of a percent on the recovery from the test. Table 13-13 show the P_2O_5 feed and concentrate grade, as well as the weight recovery from tests, and adjusted weight recovery.

Table 13-13: Pilot plant corrected weight recovery

Sample	P_2O_5 Feed grade (%)	Concentrate weight recovery (%)	P_2O_5 Concentrate grade (%)	Adjusted concentrate weight recovery (%)
PP – S2 – March 4, 2011 13h45	4,85	12,28	36,87	11,6
PP – S2 – March 4, 2011 14h30	4,79	12,29	36,16	11,4
PP – S2 – March 7, 2011 11h15	4,03	8,68	37,85	8,4
PP- SSR/S2/S3 – March 7, 2011 16h00	7.04	18.49	40.14	16.8
PP- SSR/S2/S3 – March 8, 2011	7.43	17.03	37.59	16.5
PP – S2/S3 – May 3, 2011 14h20	5,89	12,92	40,5	13,4
PP – S2/S3 – May 3, 2011 15h30	5,92	13,05	40,69	13,6
PP – S2/S3 – May 3, 2011 16h30	6,05	13,12	40,4	13,6
PP – S2/S3 – May 5, 2011 14h45	5,97	13,16	40,75	13,8
PP – S2/S3 – May 5, 2011 16h20	6,12	13,15	40,12	13,4
PP – S2/S3 – May 6, 2011 14h15	5,74	12,44	39,74	12,7
PP – S2/S3 – May 12, 2011 10h30	6,12	13,4	40,59	14.0
PP – S2/S3 – May 12, 2011 13h00	6,05	13,33	39,99	13,7
LC – SRR/S2/S3 – Sept 14, 2012	4.71	10.82	39.46	10.9
LC – SRR/S2/S3 – June 4, 2013	4,69	10,75	39,26	10.8
LC – SRR/S2/S3 – June 5, 2013 - "Medium grade"	5,56	13,61	36,96	12.9
LC – SRR/S2/S3 – June 6, 2013 - "High grade"	7,26	16,29	39,02	16.3
LC – SRR/S2/S3 – June 11, 2013 - "Low grade"	3,51	8,03	38,83	8.0
LC – SRR/S2/S3 – June 11, 2013 - "Medium-low grade"	4,18	9,85	39,97	10.1

PP: Pilot Plant
LC: Lock Cycle Test
S2: Nelsonite ore type
S2/S3: 20% Nelsonite, 80% Upper ore type
SRR/S2/S3: 36.1% Upper, 44.5 Railroad and 19.4% Nelsonite ore type

The relation between the P_2O_5 feed grade and adjusted flotation concentrate weight recovery to produce a 39.0% P_2O_5 concentrate is shown in Figure 13-3. The weight recovery formula is:

$$\%\mathrm{Wrec} = 2.2264 * \%\mathrm{P_2O_5} + 0.3146$$

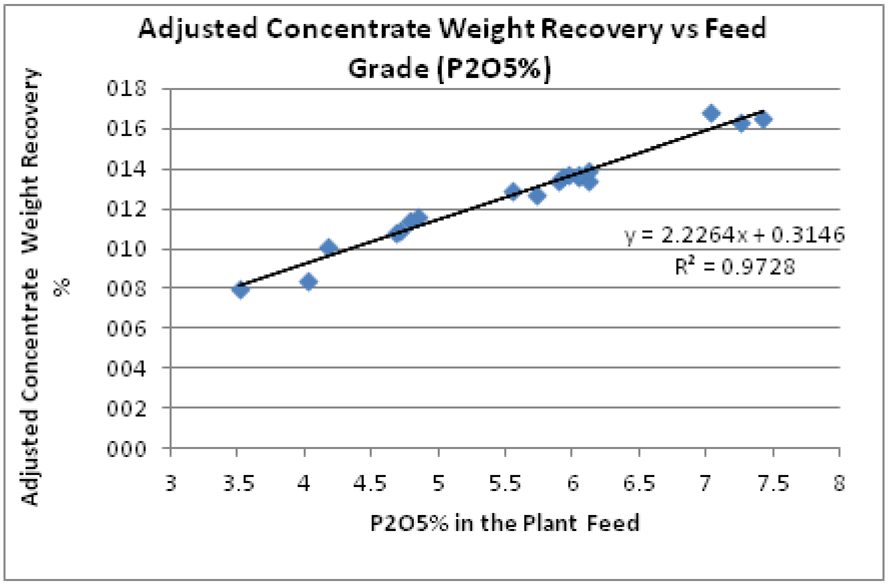


Figure 13-3: Relation between the adjusted concentrate weight recovery (%w/w) and the head grade (%P_2O_5)

13.7 Flotation Tests on Samples from Deeper Zone

In 2014, COREM was mandated to carry out bench scale flotation tests on 12 samples from a deeper zone of the deposit. The objective of this work was to verify that the process flowsheet will still be adequate for processing the ore during the late years of operations. These tests were not lock cycle tests, thus the reject of the cleaner tails were not recirculated at the rougher stage. The results may give an impression of a lower recovery while in an industrial sized processing plant, the cleaner tails would be reprocessed and the apatite be recovered. Furthermore, a full scale processing plant would allow for better recovery, reduction of reagent consumption, and better quality control over the concentrate grade.

The sample used for this program consisted of composite samples from sections of ten drill holes representing the ore that will feed the beneficiation plant from year 20 to 28 of the mine life, and of two samples from project T1405 as a baseline/control.

The quality specifications of the apatite product, supplied by Mine Arnaud, were as follows:

- P_2O_5 > 39.0% (P >17.0%)
- Ca/P < 2.2
- Fe + Al soluble in HNO_3 < 1.0%
- Mg soluble in HNO_3 < 0.3%
- Cl < 0.12%

With regards to Fe + Al and Mg specifications, since the limit specified by Mine Arnaud was measured using a procedure with HNO_3 and that COREM does not work with this method of analysis, it was agreed with Mine Arnaud that for comparison purpose the method used by COREM would correspond to an established limit of <1.3%for Fe + Al specification and <0.5% for Mg content with COREM analytical method. The following table presents the main results of the 12 samples. These tests were conducted in open circuit.

Table 13-14: Depth sample flotation testwork results

	Feed	Concentrate recovery		Concentrate grade			
	P_2O_5 (%)	Weight (%)	P_2O_5 (%)	P_2O_5 (%)	Cl (%)	Ca/P	Fe + Al
Test #1 Baseline T1405	6,04	12,57	79.49	38,20	0.123	2,238	0,896
Test # 2 Deep mine blend 4	4,63	8,39	70.47	38.90	0.127	2,272	2,885
Test # 3 Deep mine blend 5	4,56	9,56	79.73	38,00	0.135	2,228	1,089
Test # 4 Deep mine blend 1	6,79	14.05	71.48	34,54	0.078	2,265	2,586
Test # 5 Deep mine blend 6	4,73	10,23	78.77	36,40	0.310	2,254	1,722
Test # 6 Deep mine blend 3	5,55	9,14	64.07	38,90	0.113	2,223	0,737
Test # 7 Deep mine blend 2	5,95	8,95	58.60	38,60	0.096	2,215	0,983
Test # 8 Baseline T1405	5,40	12,66	81.76	34,90	0.134	2,257	2,198
Test # 9 Deep mine blend 4	4,58	9,37	78.97	38,60	0.127	2,257	0,913
Test # 10 Deep mine blend 2	6.03	8.36	53.82	38.80	0.105	2.250	0.807
Test #11 Deep mine blend 5	4,56	9,90	82.50	38,00	0,116	2,258	0,983
Test #12 Deep mine blend 4	4,48	9,64	82.23	38,20	0,119	2,259	0,896

Chlorine (Cl) specification was met in most case except for blend #6 with 0.31% Cl. In case of blends #4 and #5, the target was not met at the first attempt. The Cl grade was below the limit for the second and third tests after some reagent.

No tests met the Ca/P < 2.2 specification, but all were less than 2.275%. This is probably due to the fact that the tests were conducted in open circuit. By using close circuit tests, such as the lock cycle tests performed previously and the pilot plant, the cleaner tail will be recycling at the head of the circuit. This should increase the P_2O_5 recovery and decrease the Ca/P ratio.

Fe + AL specification was met in most case except for blend #1 and #6 with level of 2.586 and 1.722. In case of blend #4, the target was not met at the first attempt, but it was met in the second test following some reagent adjustments.

Based on the laboratory test results, it is expected that the quality control specification could be achieved with some reagent adjustments and ore coming from the deeper zone will behave similarly to the rest of the deposit.

This program conducted on representative samples of the deeper zone of the deposit increased confidence that it will be possible to process the Mine Arnaud ore from all areas of the deposit even at the end of the mine life, (year 20 to 28) without requiring major modifications to the plant.

14. Mineral Resource Estimates

The mineral resource estimate has been conducted following the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definitions Standards for mineral resources in accordance with N1 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.

The mineral resource estimation work for Mine Arnaud's Sept-Îles project was conducted by Jean-Philippe Paiement, M.Sc., P.Geo. The 3D modelling, geostatistics and grade interpolation of the block model was conducted 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 and Table 14-1) with entries for:

- Down Hole Survey (n = 5,784);
- Assays (n = 21,059);
- Lithologies (n = 7,881).

Table 14-1: Summary of data per year

YEAR	Nbre Holes	Nbre Surveys	Nbre Assays	Nbre Lithos
1995	12	31	663	323
1996	33	37	998	460
2008	15	892	729	210
2010	181	4,043	8,866	3,958
2012	180	609	9,389	2,750
2013	9	172	414	180

The database was validated upon importation in Genesis, which enabled the correction of minor discrepancies between the table entries; surveys and lithologies. Most of the errors were caused by survey and/or lithology 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 modelled on each section.

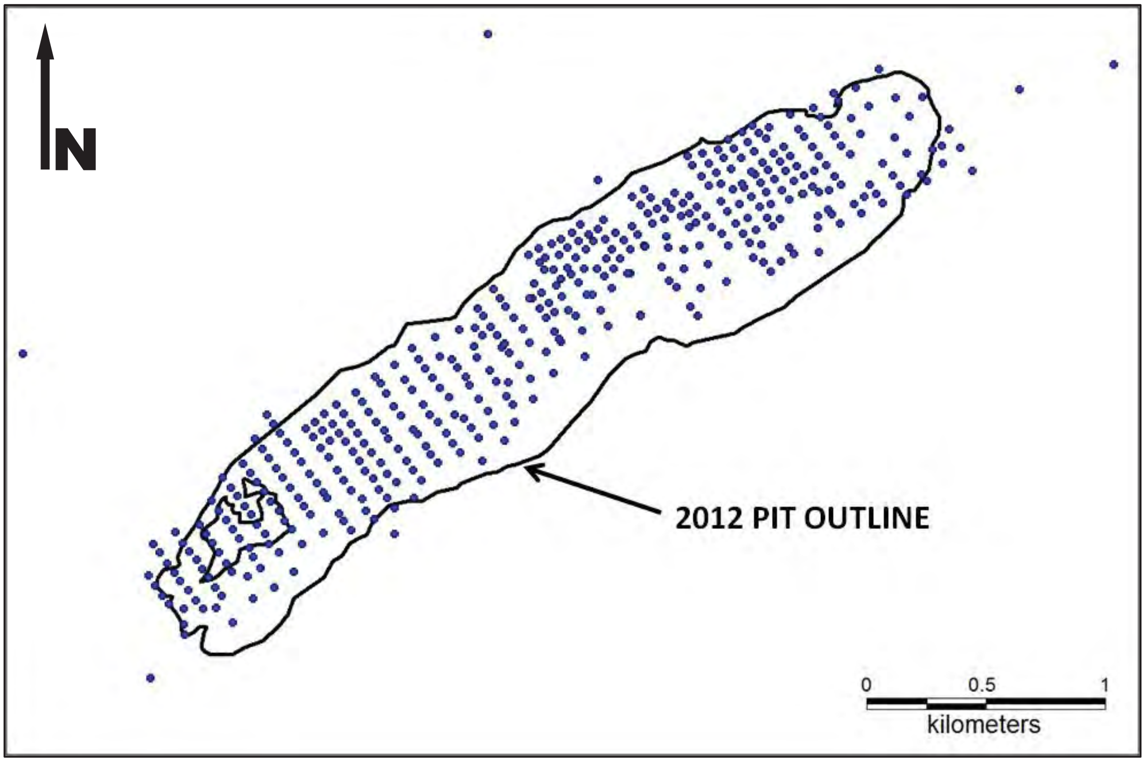


Figure 14-1: Drill holes collars positioning relative to the 2012 pit outline

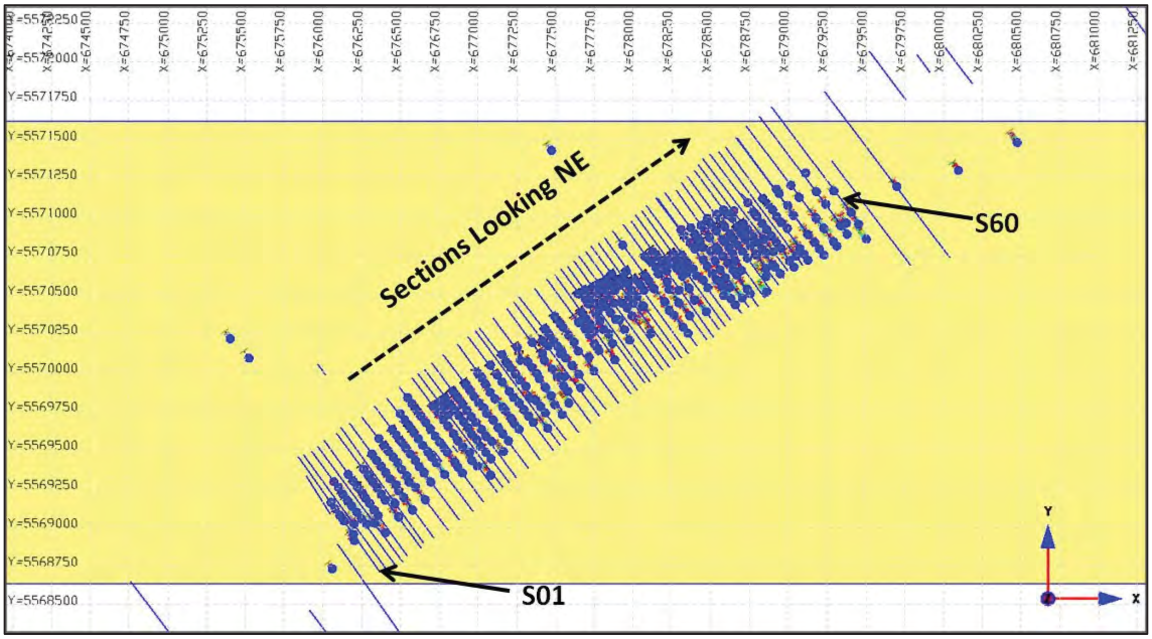


Figure 14-2: Plan view showing the trace of each vertical section

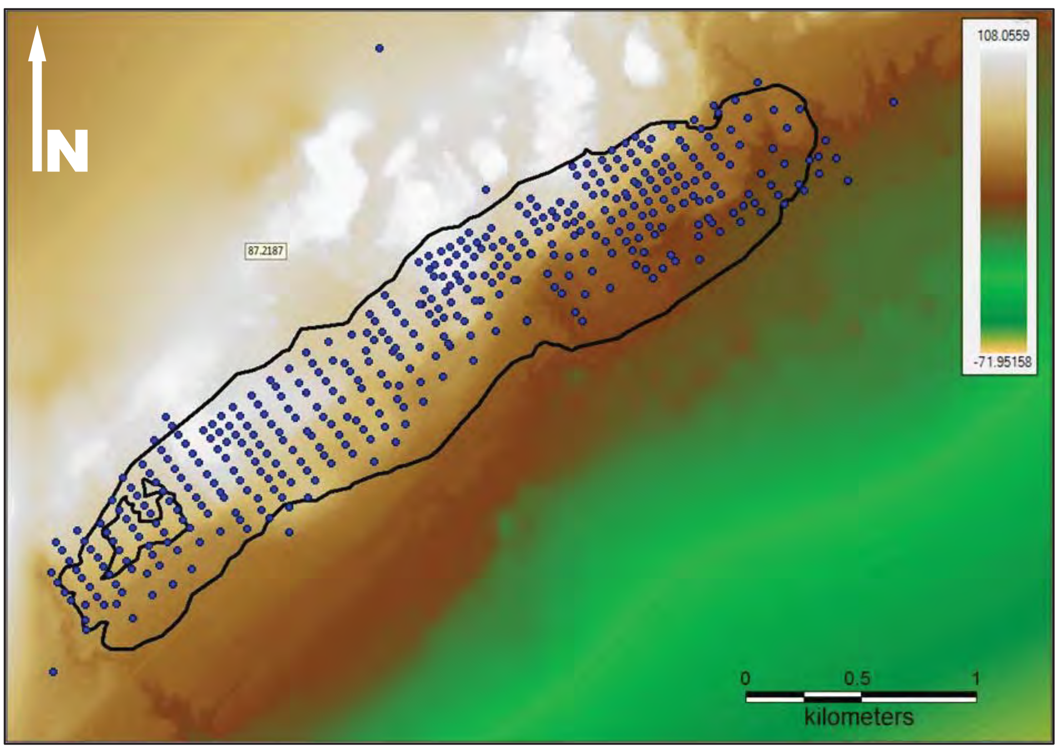


Figure 14-3: Topographic surface and 2012 pit outline

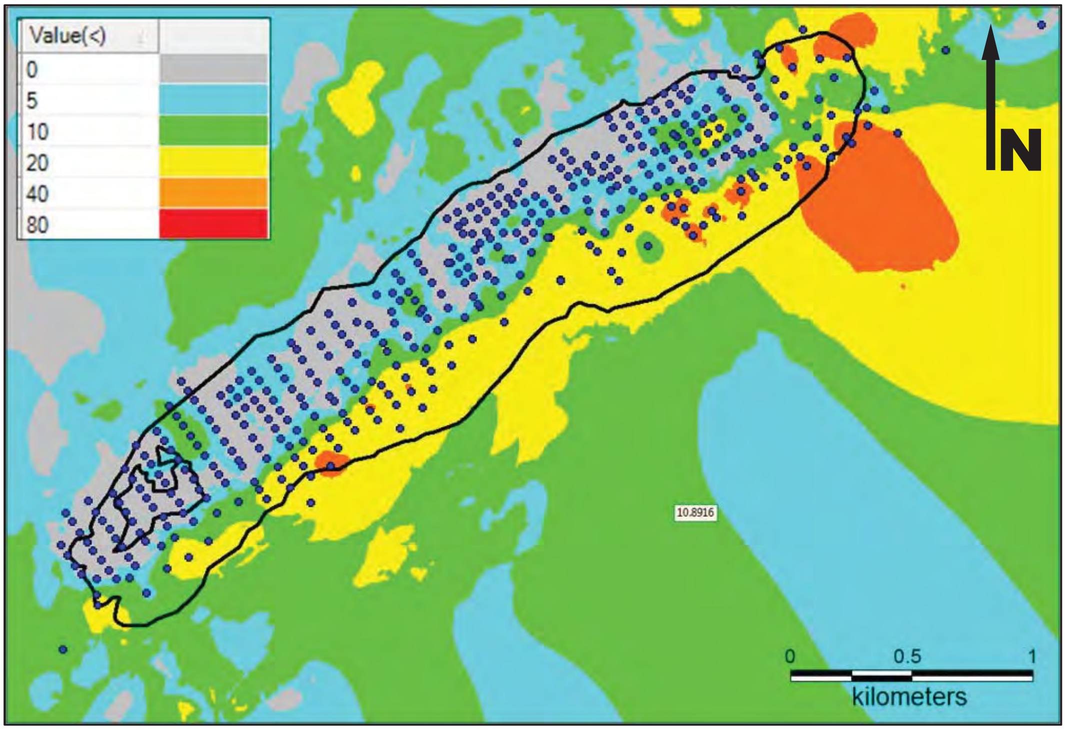


Figure 14-4: Overburden thickness grid relative to the 2012 pit outline

14.2 Geological and Structural Model

Lithological envelops were transmitted by Mine Arnaud 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 exists 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 identified: 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°.

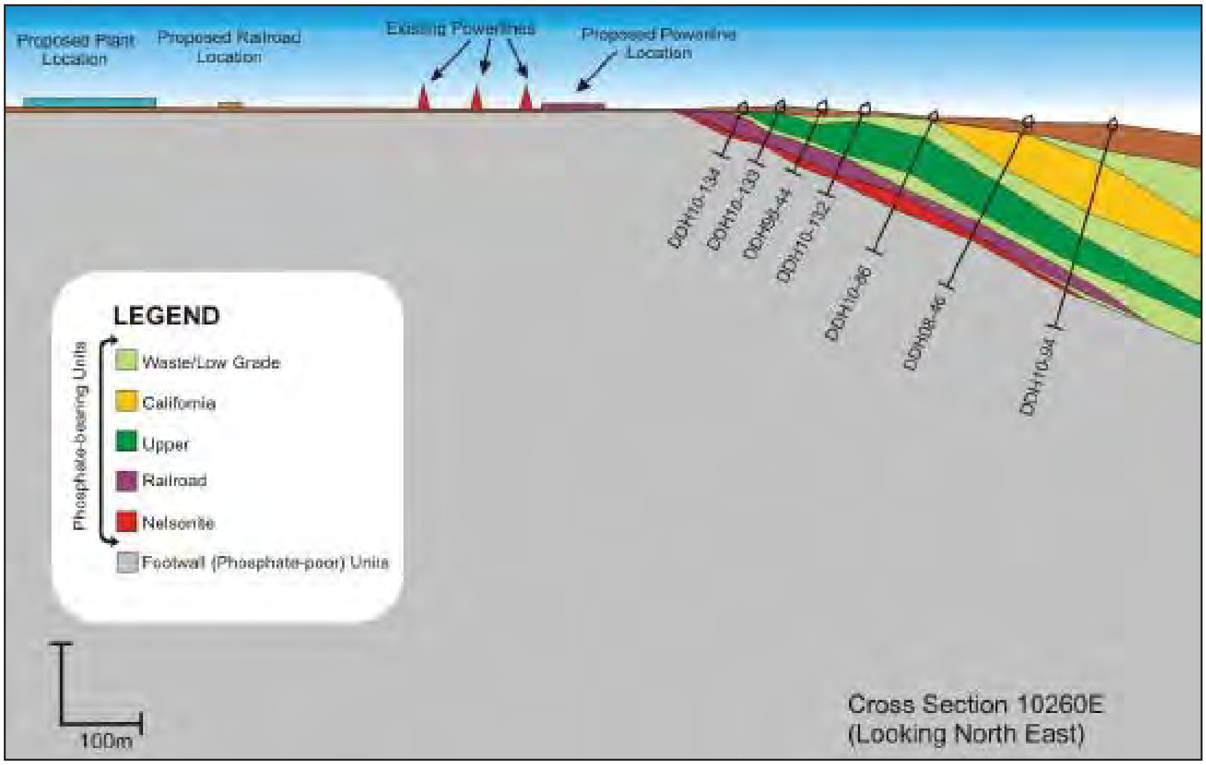


Figure 14-5: Schematic geological cross section (Duschene, 2012)

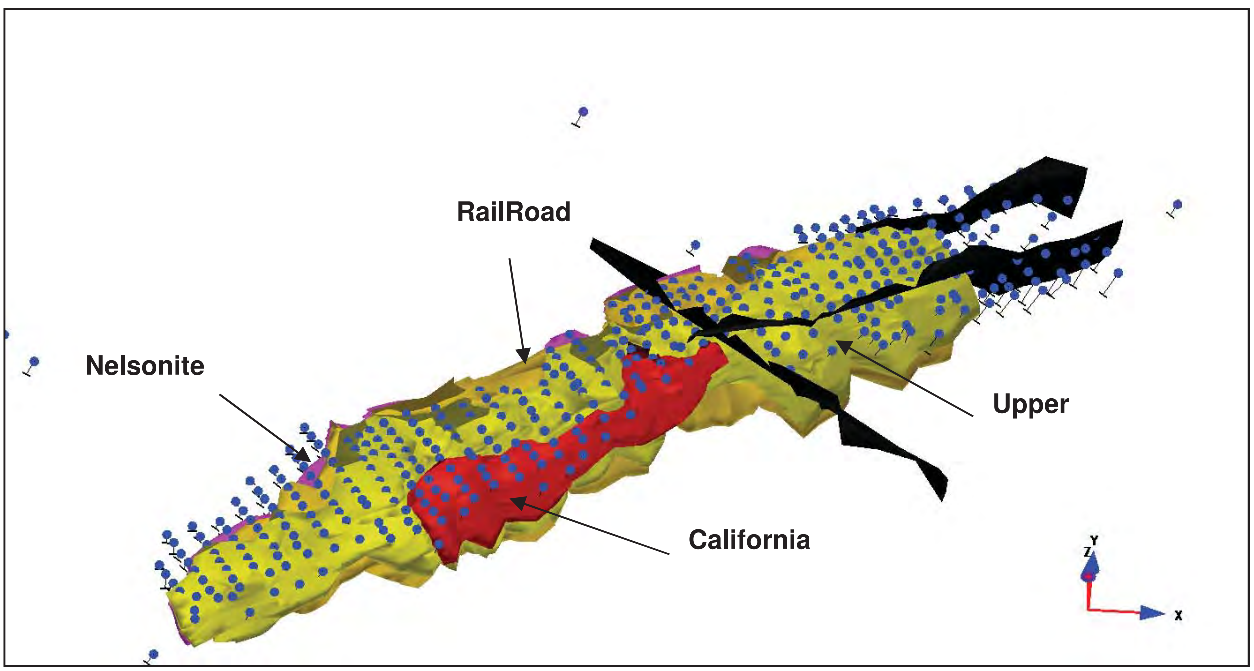


Figure 14-6: 3D view of the geological envelops provided by Mine Arnaud

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 modelling minimal grade of 2.0% P_2O_5. The modelling 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 modelling 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 section. 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 four 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).

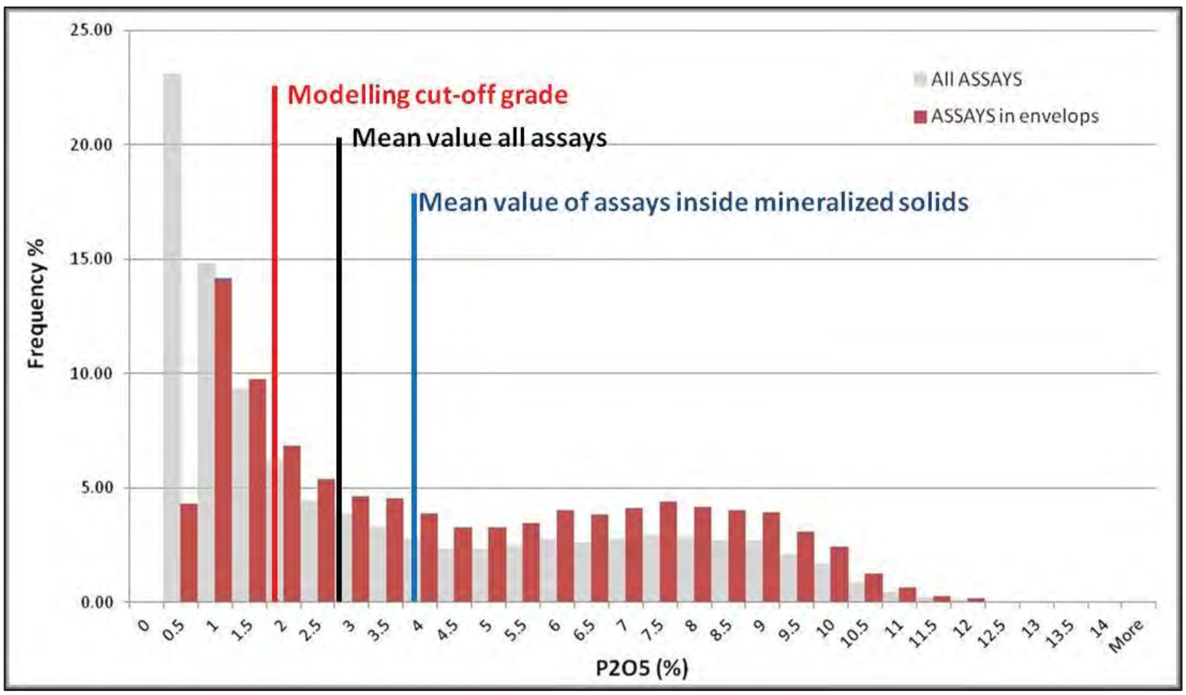


Figure 14-7: Statistic distribution of the assay data

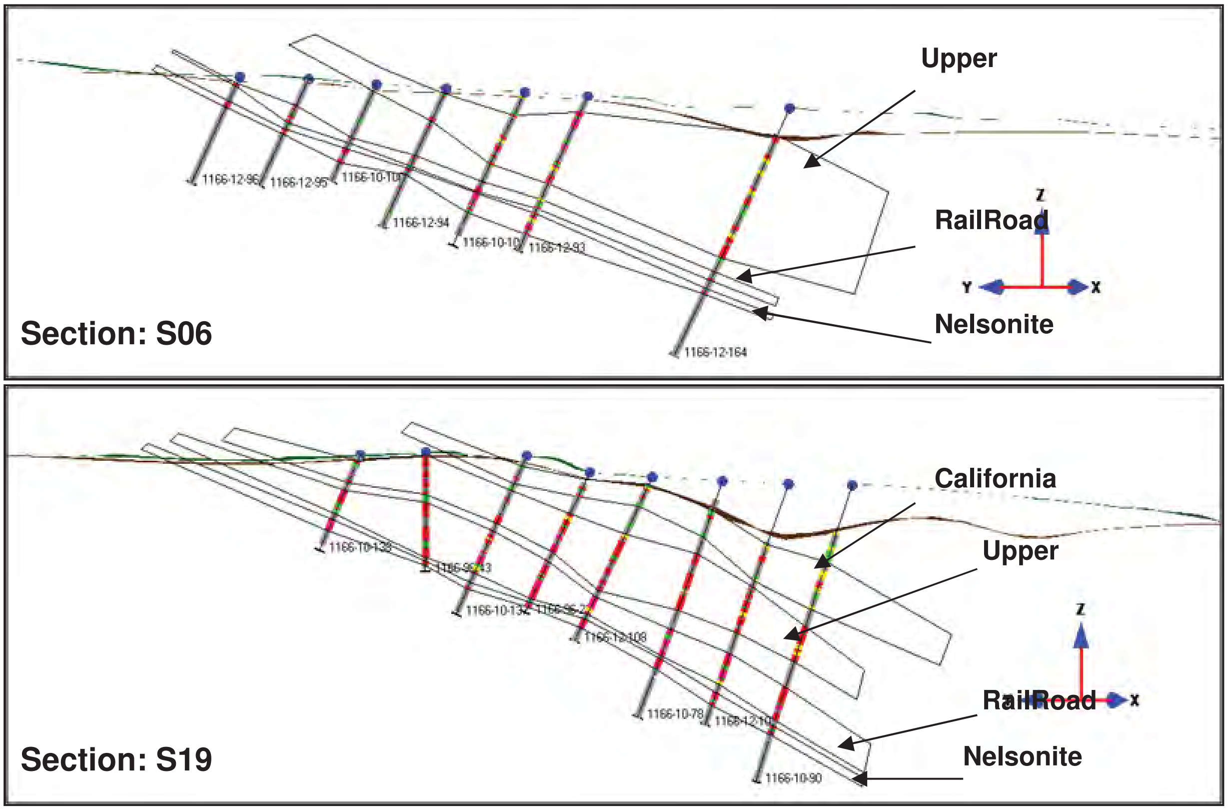


Figure 14-8: Examples of solid interpretation on sections (looking northeast)

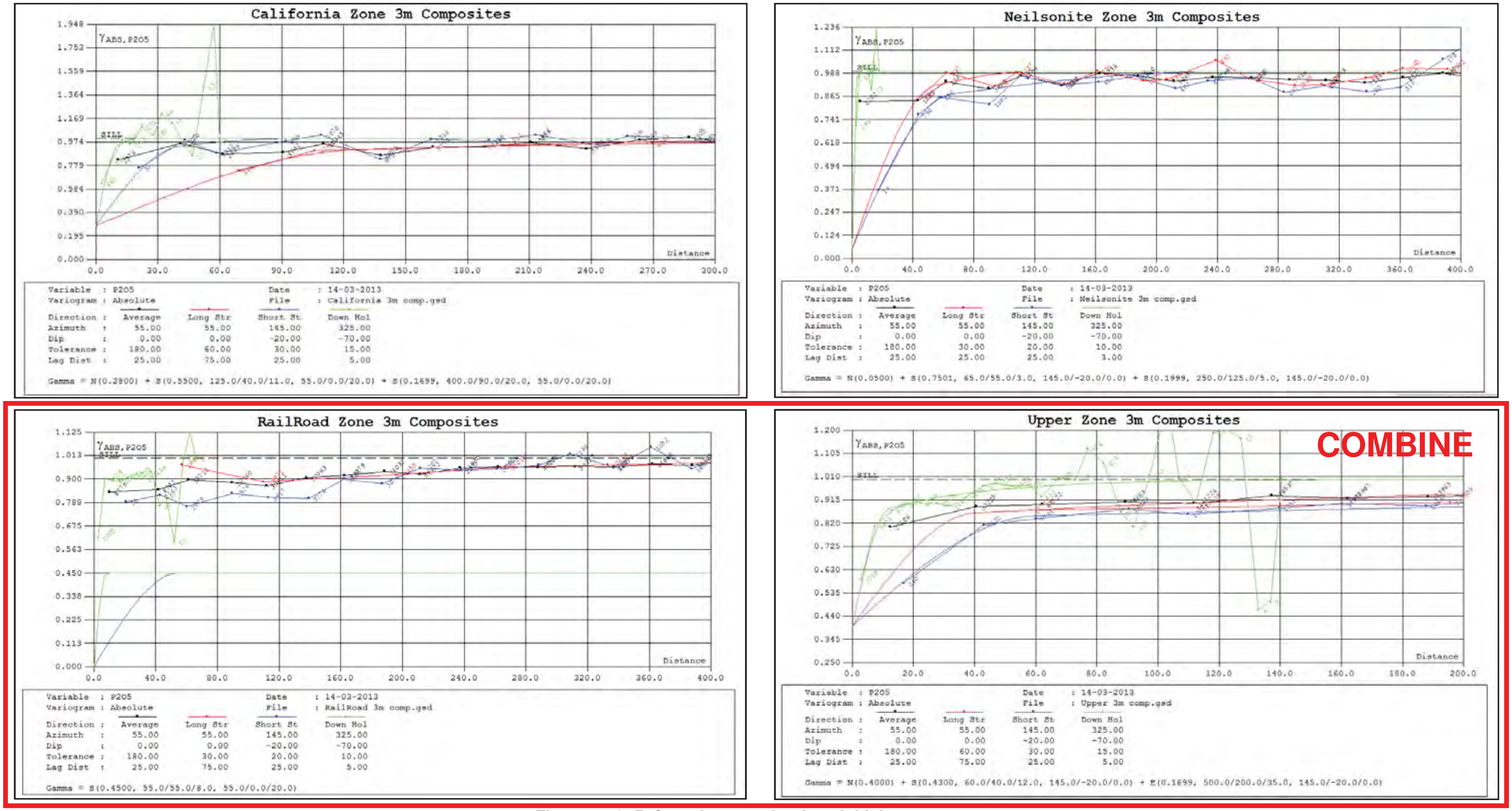


Figure 14-9: P_2O_5 variograms for the 4 initial zones

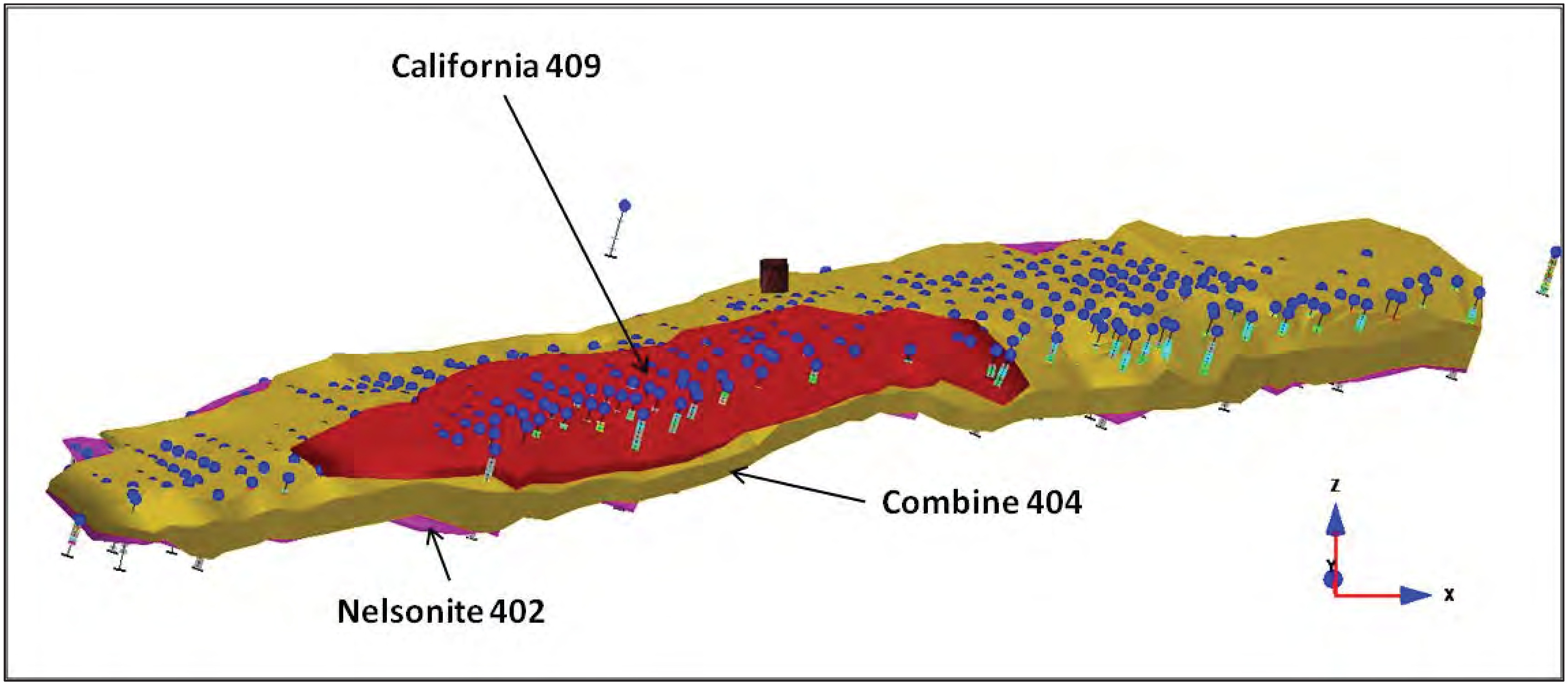


Figure 14-10: Three mineralized envelops for resource estimation

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 three mineralized solids and also for the waste between the mineralized zones. Hence, four different and independent sets of composites will be used in the estimation of the resources.

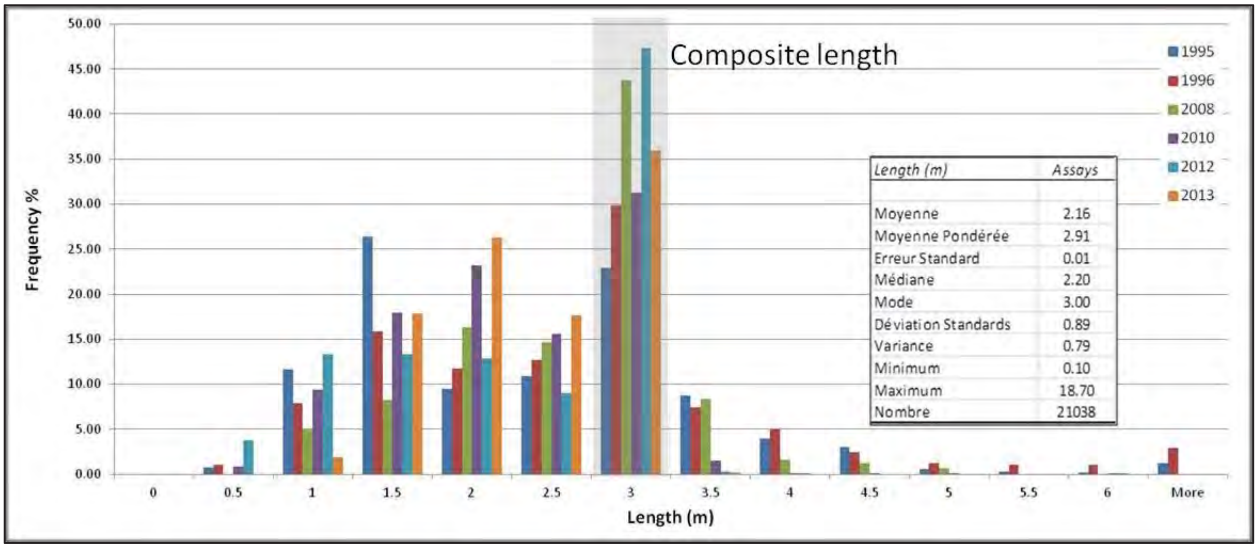


Figure 14-11: Assays length statistics

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 three 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 14-2) for each of the three 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 insitu Variable

The histograms for California and Combine zones show a single mode with a 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 modelling 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 14-2). Variography modelling 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.

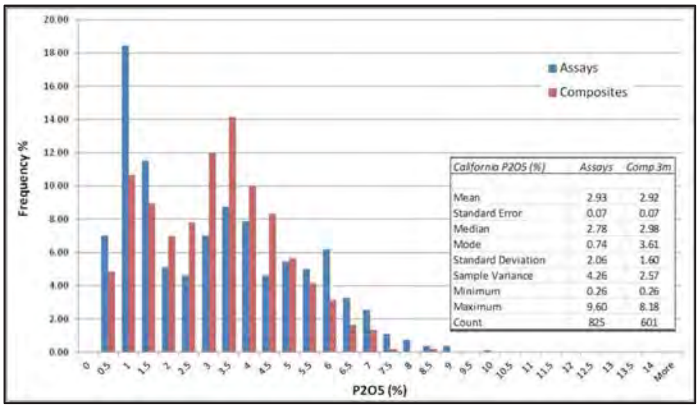
Insitu Cl values are only used as a qualitative element in the further reporting of the mmineral resources. The Cl content of the concentrate is interpolated using the insitu K_2O values as per stated in section12.5.

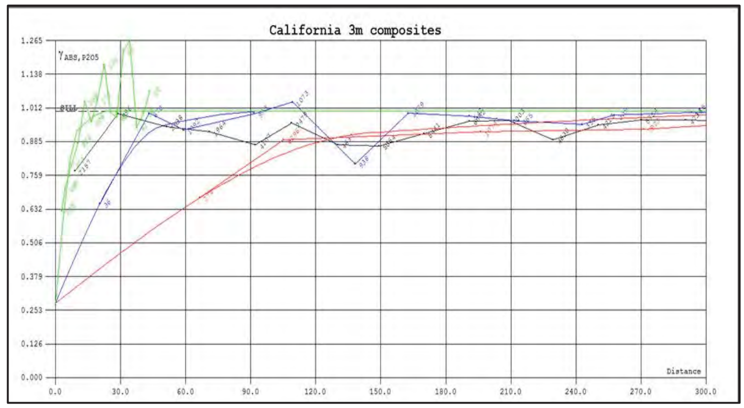
14.5.3 K_2O Variable

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.

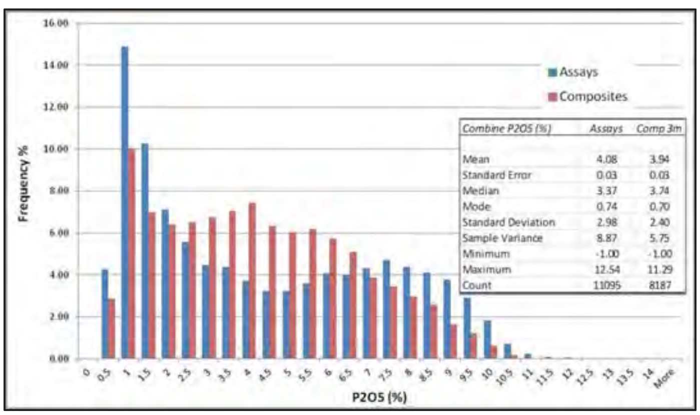
The relatively good correlation between K_2O and P_2O_5 enables the possibility of using the same base variograms for K_2O and P_2O_5. The variograms modelling was then adjusted when possible to better reflect the Cl behavior, but anisotropy and ranges where kept close to the P_2O_5 values (Table 14-2). 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 14-2).

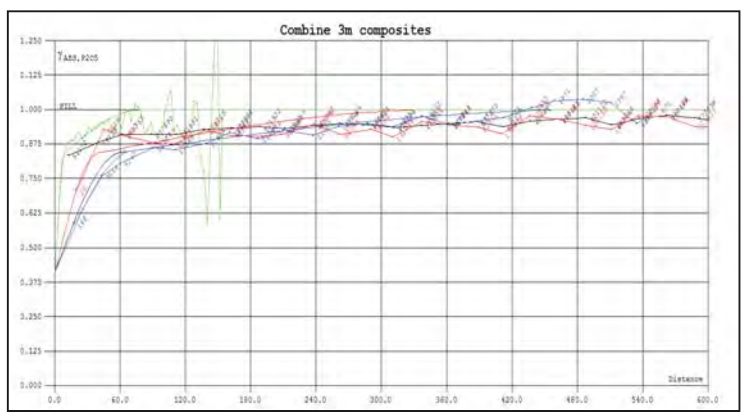
CALIFORNIA ZONE



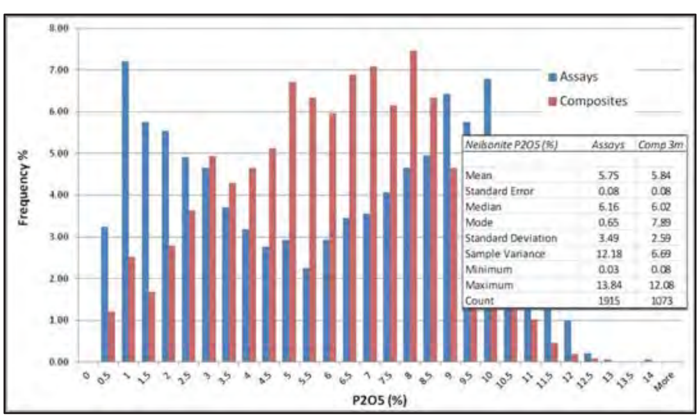


COMBINE ZONE





NELSONITE ZONE



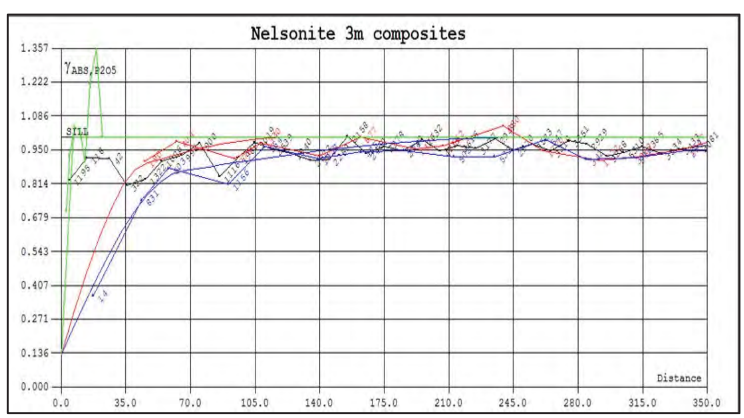
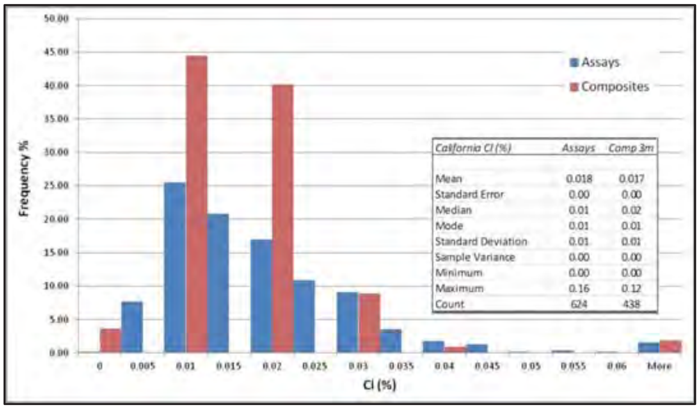
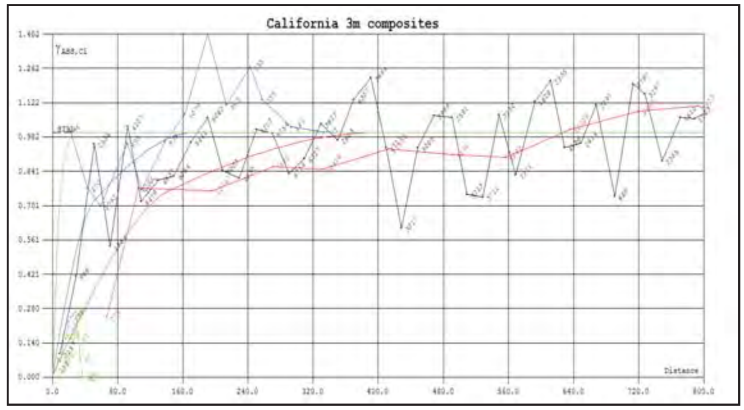


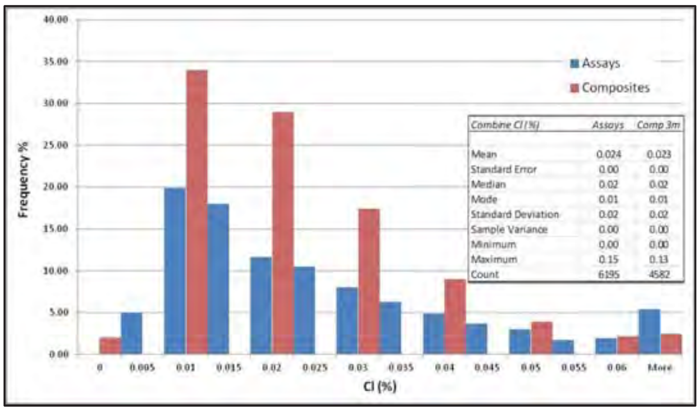
Figure 14-12: Geostatistics for P_2O_5% between different zones

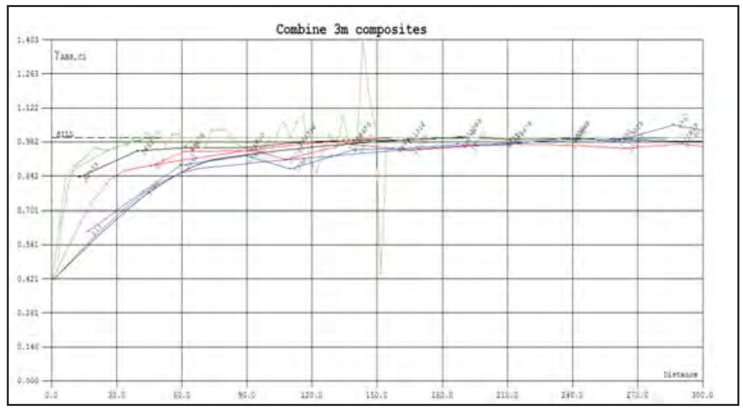
CALIFORNIA ZONE



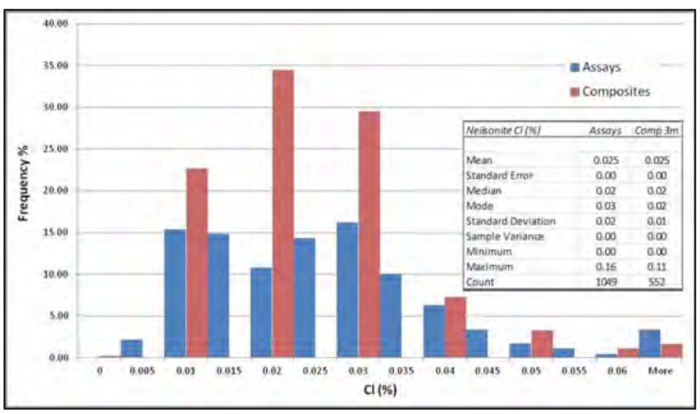


COMBINE ZONE





NELSONITE ZONE



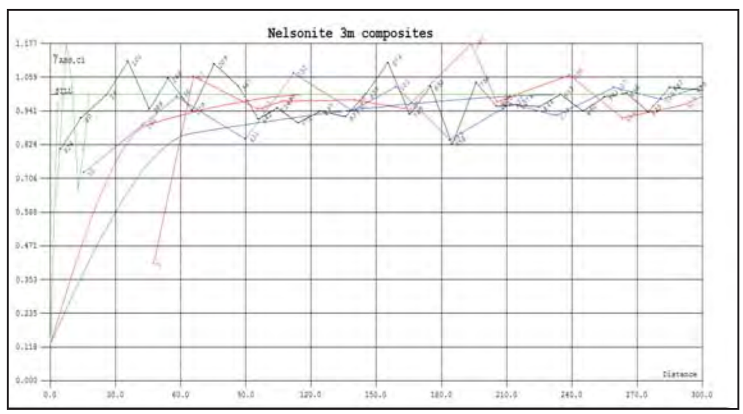
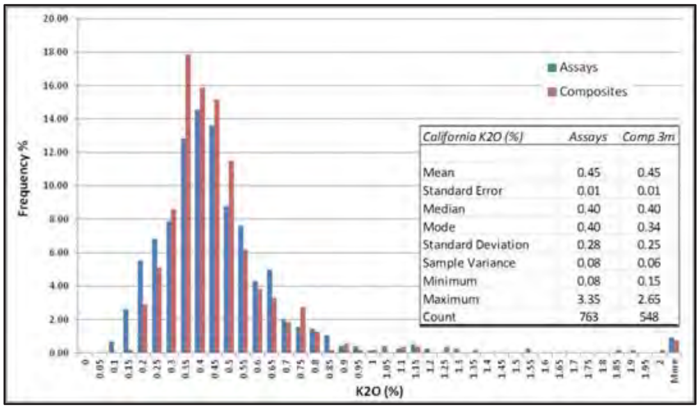
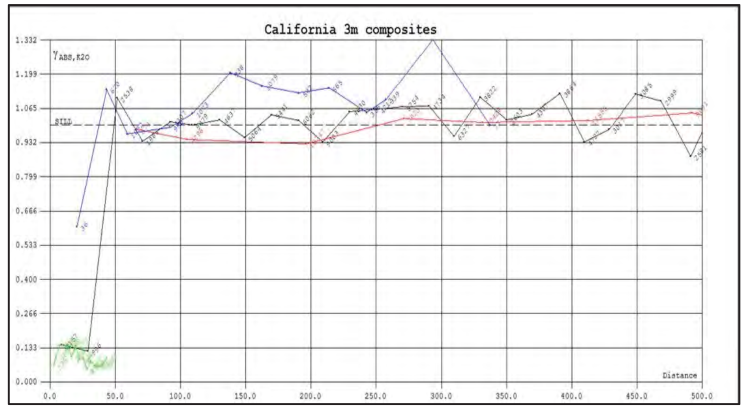


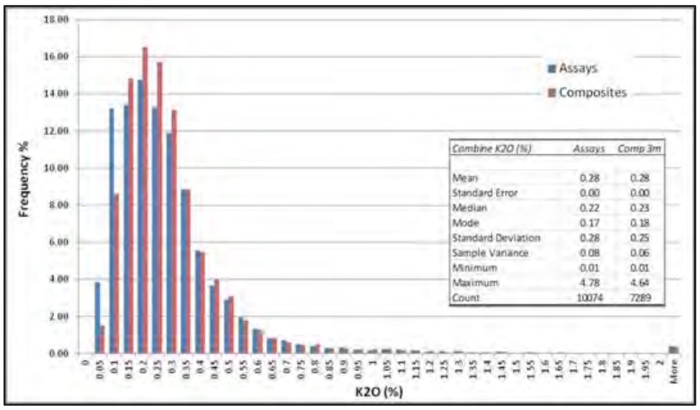
Figure 14-13: Geostatistics for Cl% between different zones

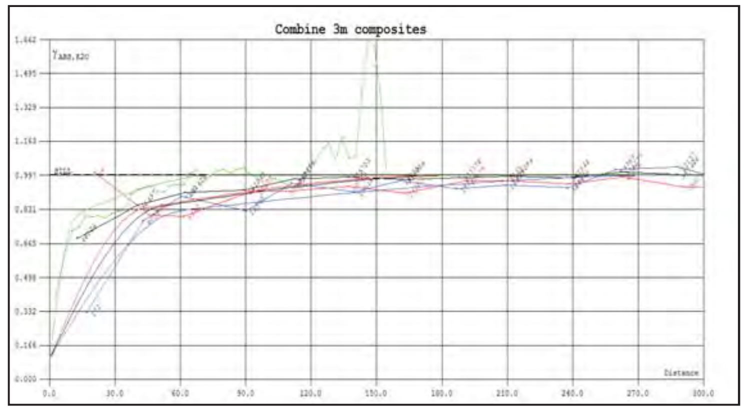
CALIFORNIA ZONE



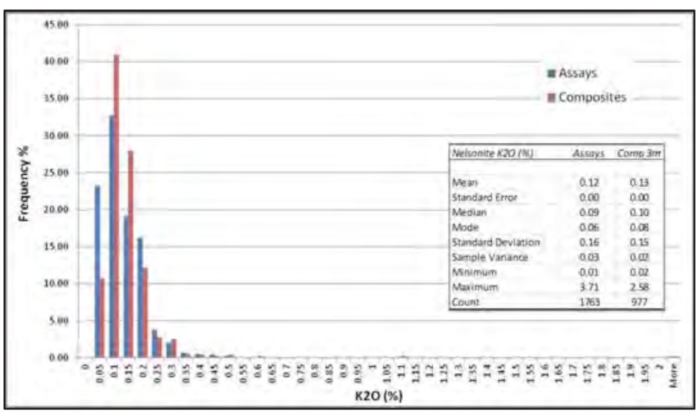


COMBINE ZONE





NELSONITE ZONE



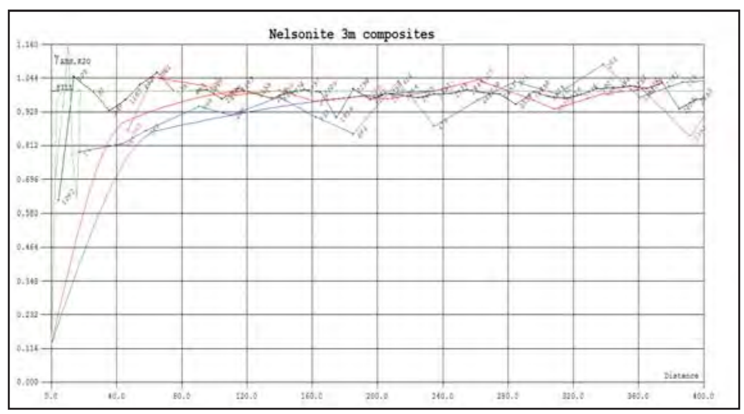
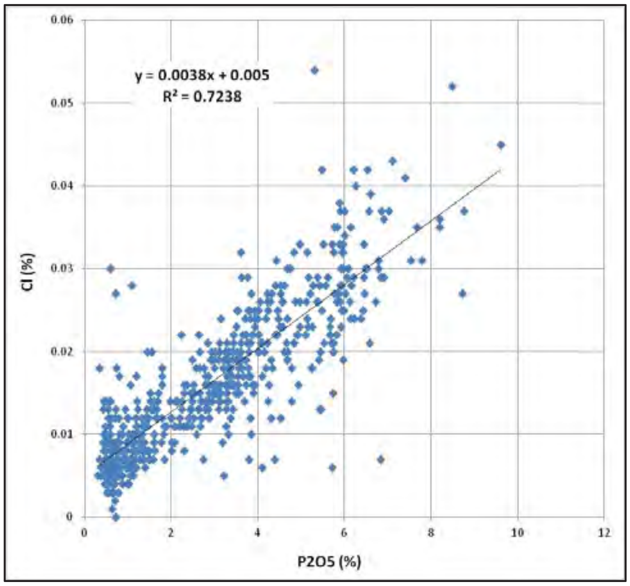
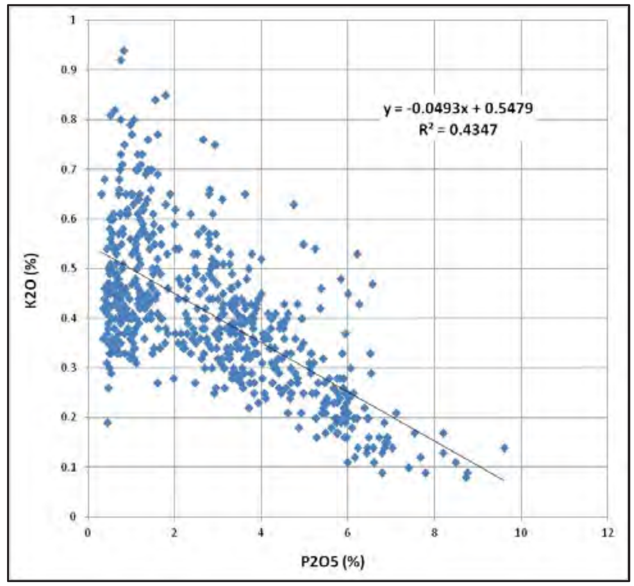


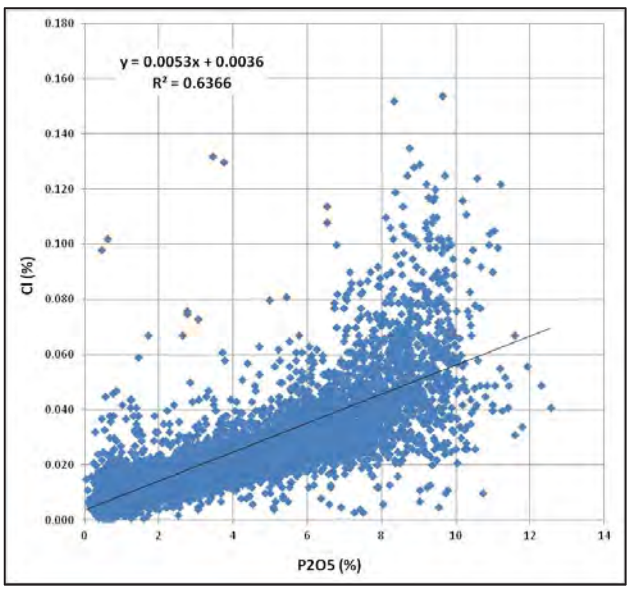
Figure 14-14: Geostatistics for K_2O% between different zones

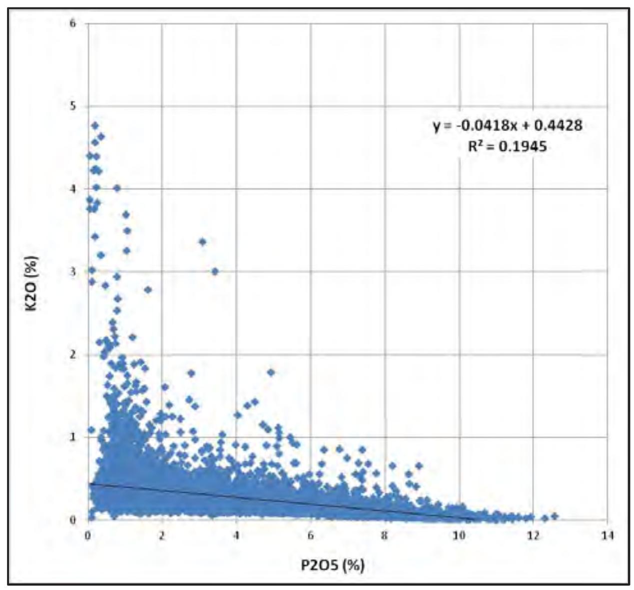
CALIFORNIA ZONE



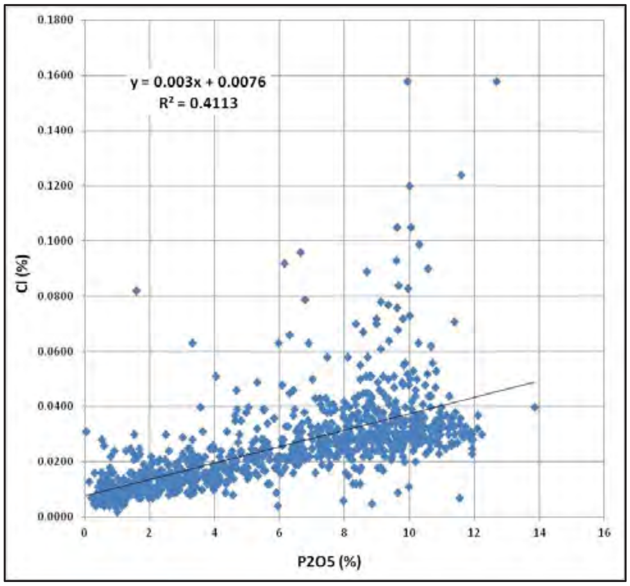


COMBINE ZONE





NELSONITE ZONE



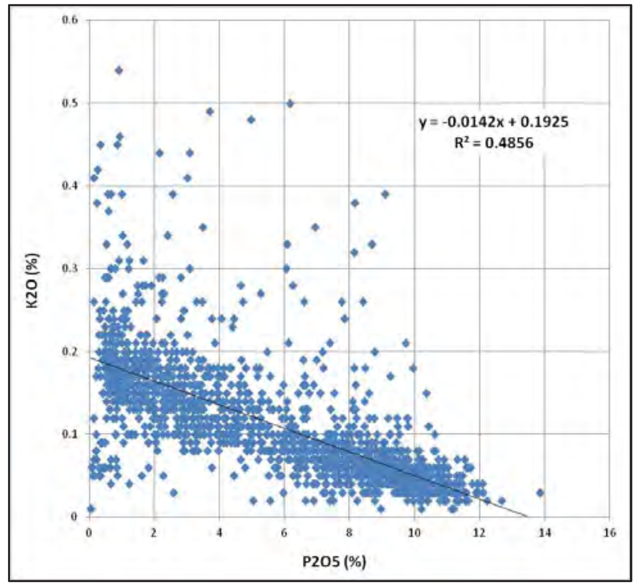


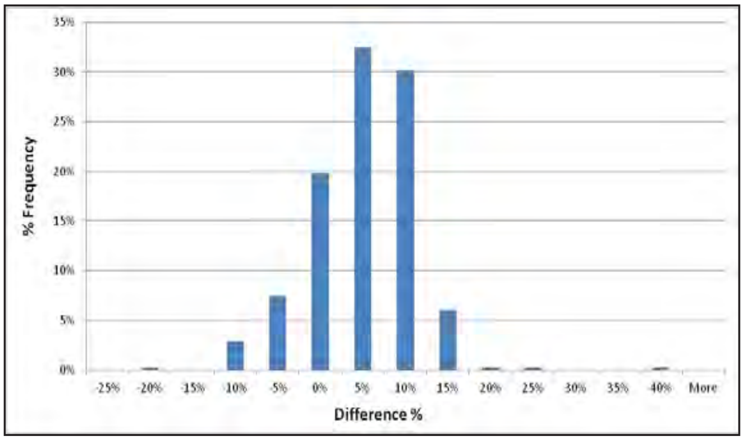
Figure 14-15: Element correlation (P_2O_5, Cl and K_2O) in the different zones

Table 14-2: 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 P_2O_5	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 K_2O	0.28	0.55	145	50	10	0.1699	400	100	25	55	0	20
Combine P_2O_5	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 K_2O	0.10	0.65	60	50	12	0.25	300	200	80	145	-25	0
Nelsonite P_2O_5	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 K_2O	0.13	0.65	65	45	3	0.2199	250	125	5	145	-20	0

14.6 Specific Gravity

The database transmitted in 2013 to SGS Geostat contained 348 density measurements made on pulp samples and 990 density measurements made on core samples. The two 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 properly for the in-situ density (especially due to the porosity changes between rock and pulp) enable the justification



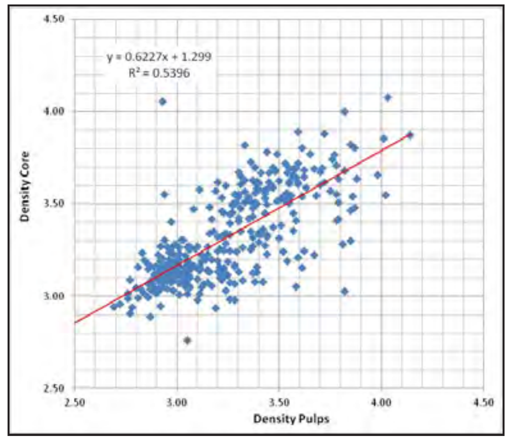


Figure 14-16: Comparison between Density measurements on Core and Pulp samples

The core sample measurements were then used to verify the existence of relationships between whole rock analysis data and density measurements, in order to calculate the missing density measurements in the database. 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 two 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.

$$Density = \frac{[100 - (Fe + Ti) \times 2.6] + (Fe - Ti) \times 5.15 + Ti \times 4.72}{100 - (Fe + Ti) + (Fe - Ti) + Ti}$$

The densities are then calculated for each of the samples in the database, creating a set of 20,486 calculated density and 990 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, and subsequently, the reserves.. The densities vary slightly from the densities used in the past (Table 14-3) where the present average densities are between 3% and 12% lower than the past densities.

Table 14-3: 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

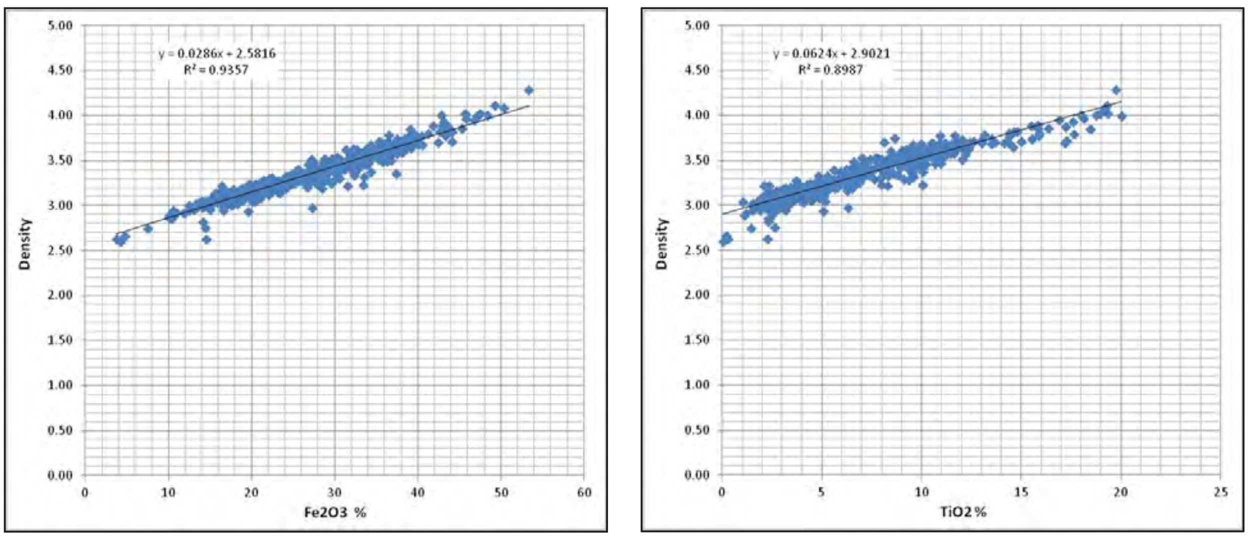


Figure 14-17: Relation between assays and density measurements

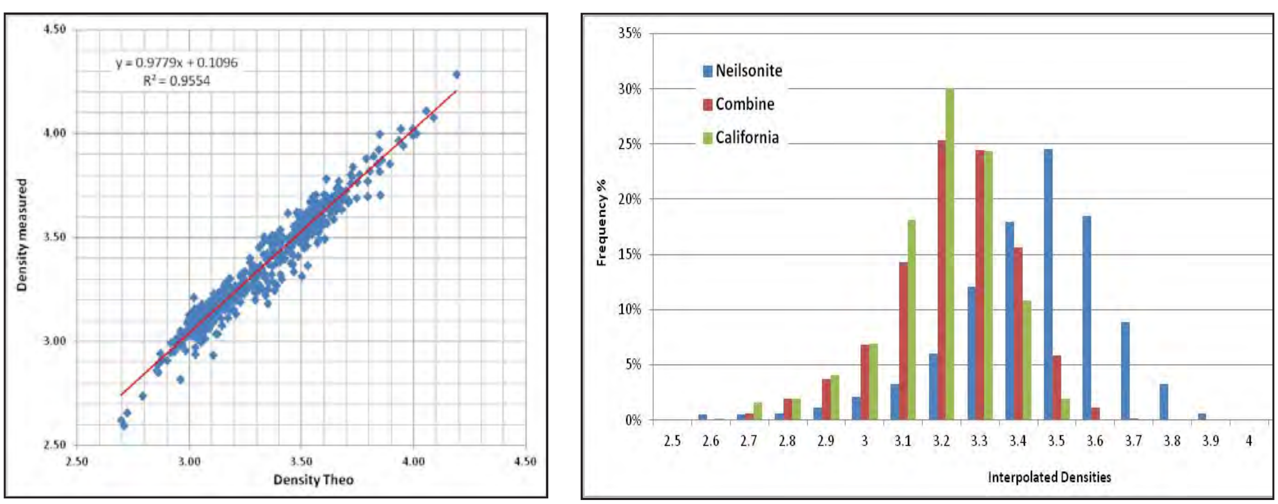


Figure 14-18: Calculated and measured density relations and Histogram of interpolated densities

14.7 Block Model

A block model was generated inside the limits stated in Table 14-4. 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 four 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 (used for resources only);
2. Combine Zone (used for resources and reserves);
3. Nelsonite Zone (used for resources and reserves) and
4. Waste, corresponding to the inter layers of waste between the zones and outside blocks (used for resources and reserves).

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

Table 14-4: 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 ellipse sizes were based on the continuities found in the variography study (Table 14-5). The mineralized zones were interpolated using kriging whereas the waste zone is interpolated using Inverse Square Distance methodology (Table 14-6). All of the zones were interpolated using three successive passes with more permissive criteria with each pass.

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). However, since insitu Cl values are not used in the predictive Cl value of the concentrate, the missing Cl insitu values should not cause significant differences in the resources estimation process. 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 drillhole 1116-10-83 for chlorine values to verify these abnormal results.

Table 14-5: 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 14-6: 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 50 m or less. The indicated category was given to blocks were the drilling grid spacing was between 50 m to 150 m. 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.

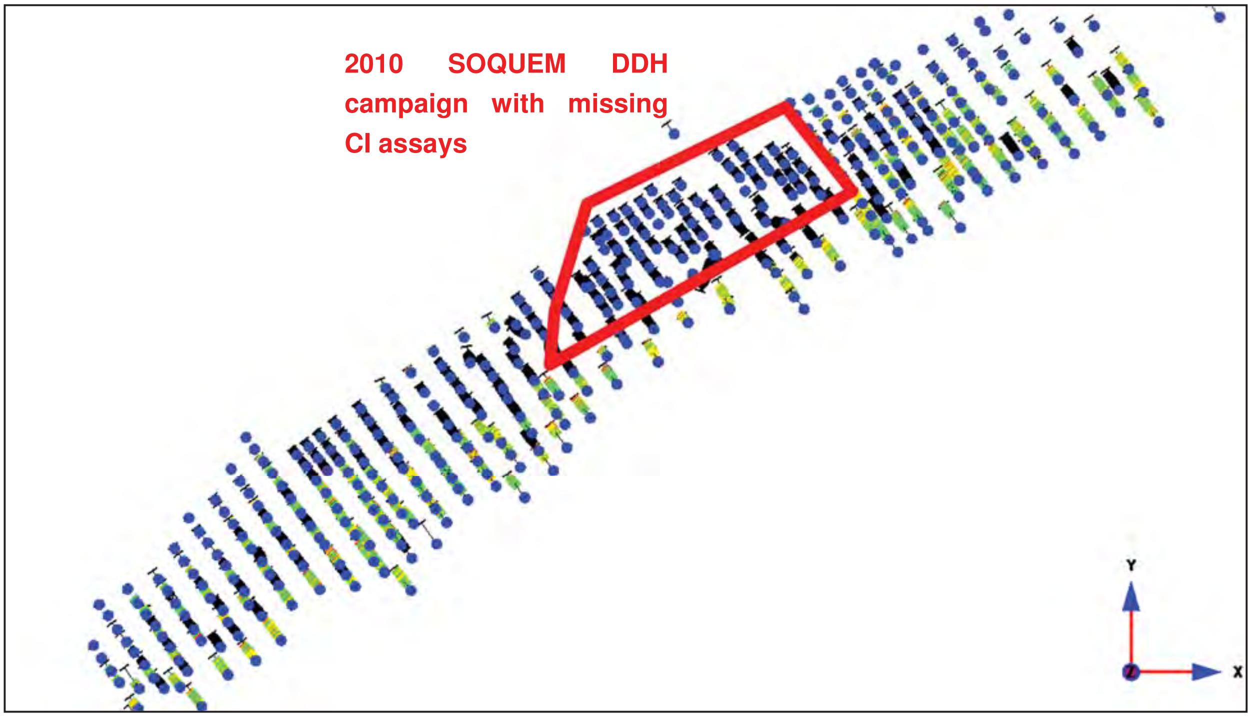


Figure 14-19: Missing Cl data in drill holes

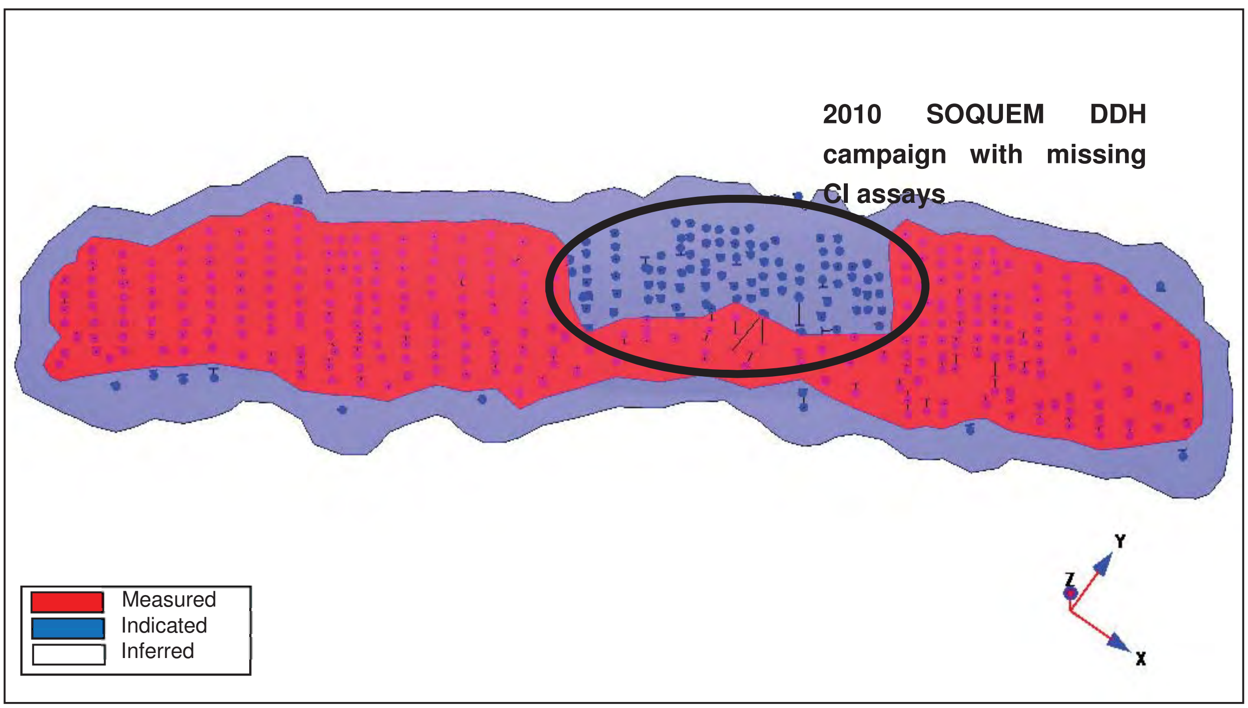


Figure 14-20: Classification envelops for the mineral resource

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 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), calculated from the WRec% and Cl grade of the concentrate (% Cl Conc), calculated from the K_2O% of the blocks.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.

Results from metallurgical testwork conducted in 2012 and 2013 (see Section 13) were used to establish the relationship between Weight recovery and P_2O_5 ROM feed grade. This relationship can be expressed as follows:

$$\text{WRec\%} = 2.2264 \times P_2O_5\text{\% feed} + 0.3146$$

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 12.45 for Cl prediction models):

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

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

14.10 Optimization Procedures and Parameters

An open pit optimization scenario 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 Table 14-7. The resulting pit shell is used solely to report the mineral resources and not the mineral reserves. This optimization scenario takes into account the Measured, Indicated and Inferred resources.

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 14-7: Open-pit optimization parameters

14.11 Mineral Resources

The optimized pit shell was used to limit the extent of the mineral resources at depth (Figure 14-21). 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 1.65% P_2O_5 except for the blocks inside the Nelsonite envelop where a cut off grade of 2.05% 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 14-8) which are exclusive categories that cannot be added to one another.

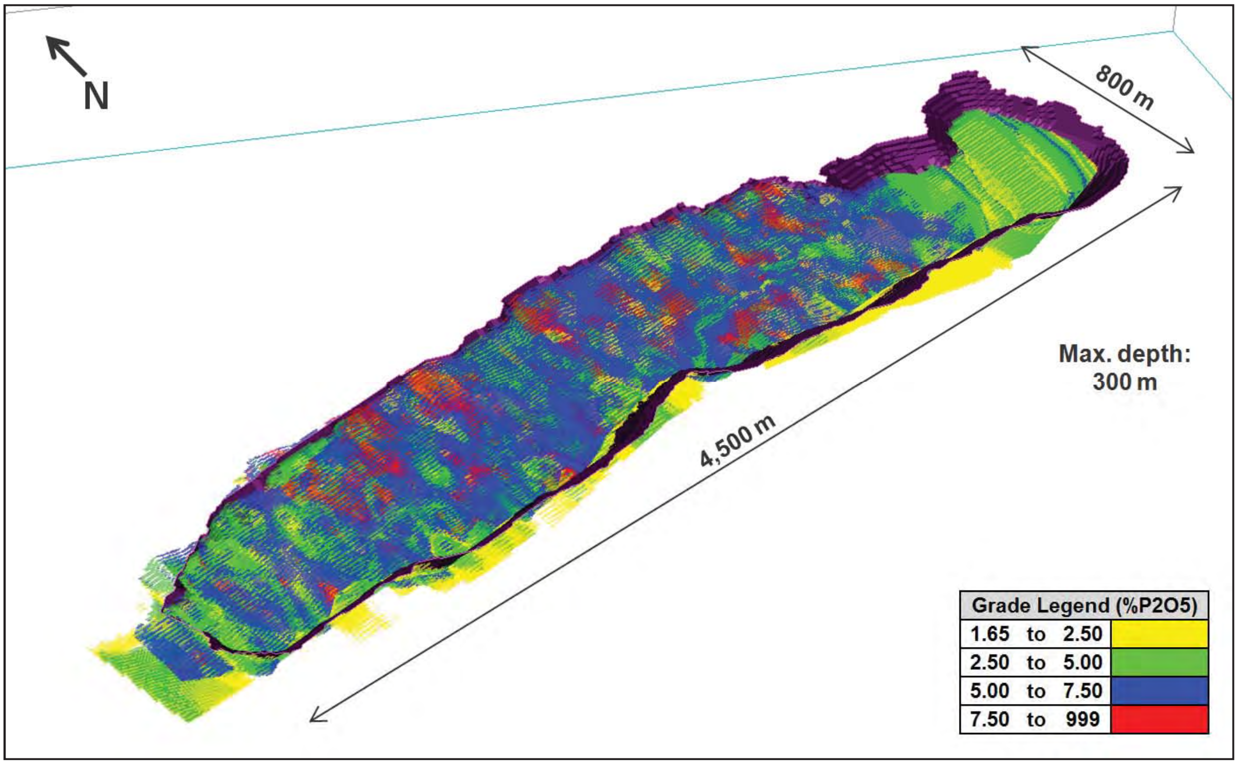


Figure 14-21: Mineral Resources Block Model with Pit Shell surface

Table 14-8: Mineral Resources Estimate

Category	Material Type	Cut Off (%P2O5)	Tonnage (Mt)	Grade (%P2O5)	WRec (%Wrec)	Conc. Grade (%P2O5)	Conc. Grade (%Cl)
Measured	California	1.65	27.615	2.94	6.86	38.47	0.1719
	Combine	1.65	319.168	4.01	9.25	38.89	0.1227
	Surrounding	1.65	28.827	2.31	5.46	37.91	0.1524
	Nelsonite	2.05	37.966	5.88	13.41	39.42	0.0823
	TOTAL		413.576	4.00	9.21	38.84	0.1243
Indicated	California	1.65	8.230	3.14	7.30	38.54	0.1556
	Combine	1.65	89.467	4.29	9.87	38.98	0.1213
	Surrounding	1.65	24.951	2.31	5.45	37.92	0.1699
	Nelsonite	2.05	9.264	6.19	14.09	39.48	0.0867
	TOTAL		131.911	3.98	9.17	38.79	0.1302
Inferred	California	1.65	0.001	1.82	4.40	37.23	0.1651
	Combine	1.65	-	-	-	-	-
	Surrounding	1.65	44.634	3.36	5.45	38.67	0.1603
	Nelsonite	2.05	-	-	-	-	-
	TOTAL		44.635	3.36	5.45	38.66	0.1603

Notes:

- The mineral resource estimate has been conducted using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definitions Standards for mineral resources in accordance 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.
- SGS did the supposition that the diluted material will have the same %Cl that the ore blocks where the dilution is applied.
- Resources are constrained by the pit shell and the topography of the overburden layer.

15. Mineral Reserve Estimates

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 to proven and probable reserves respectively. They are also constrained within the final pit limit and above the marginal cut-off grade. It should be noted that a small portion of the blocks that meet the criteria are excluded nevertheless. This exclusion (19.80 Mt) comes from blocks destined for the low grade stockpile that were discarded in order to lower the chlorine content of the low grade material that will be treated in the last years of operations (28 to 31).

Following the request from Mine Arnaud, the block model was adjusted for the reserves calculation. Due to the low grade nature and high chlorine 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.

The mineral reserves (with dilution and ore loss) is therefore equal to 342.60 Mt of ore at an average grade of 4.30 % P_2O_5, using cut-off grades related to the rocktype (1.65 %P_2O_5 and 2.05 %P_2O_5) and represents an operation of 30.8 years from start of production. The entire reserve comprises 33.87 Mt of apatite concentrate grading 39 %P_2O_5 and having a chlorine content of 0.1136 %. Total waste, including rock, inferred resources and overburden, is 243.57 Mt. The detailed mineral reserve estimate is shown in 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	1.65	55,980,000	4.61	10.59	5,930,000	0.1176
	Surrounding	1.65	2,780,000	2.22	5.26	150,000	0.1451
	Nelsonite	2.05	9,660,000	5.24	11.99	1,160,000	0.0861
	Total		68,420,000	4.61	10.57	7,230,000	0.1131
Ore (Proven Reserves)	Combine	1.65	234,070,000	4.20	9.66	22,620,000	0.1189
	Surrounding	1.65	10,290,000	2.31	5.46	560,000	0.1216
	Nelsonite	2.05	29,820,000	5.06	11.59	3,460,000	0.0783
	Total		274,180,000	4.22	9.71	26,640,000	0.1137
Ore (Total Reserves)	Combine	1.65	290,050,000	4.28	9.84	28,540,000	0.1186
	Surrounding	1.65	13,070,000	2.29	5.42	710,000	0.1264
	Nelsonite	2.05	39,480,000	5.11	11.69	4,610,000	0.0803
	Total		342,600,000	4.30	9.89	33,870,000	0.1136
Waste	Waste rock		179,630,000				
	Overburden		63,940,000				
In-pit Total	**All**		586,170,000				

Note: This reserve includes 3.39 % dilution at 1.28 %P_2O_5 and 2.37 % ore loss for Combine and Surrounding ore types and a 25.89 % dilution at 0.89 %P_2O_5 and 4.35 % ore loss for the Nelsonite ore type. Also, SGS did the supposition that the diluted material will have the same %Cl that the ore blocks where the dilution is applied.

16. Mining Methods

16.1 Introduction

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

16.2 Sterilization of the California zone

Following the request from Mine Arnaud, the block model was adjusted for the mine plan preparation. Due to the low grade nature and high chlorine 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. Refer to the Figure 16-1 for visualization.

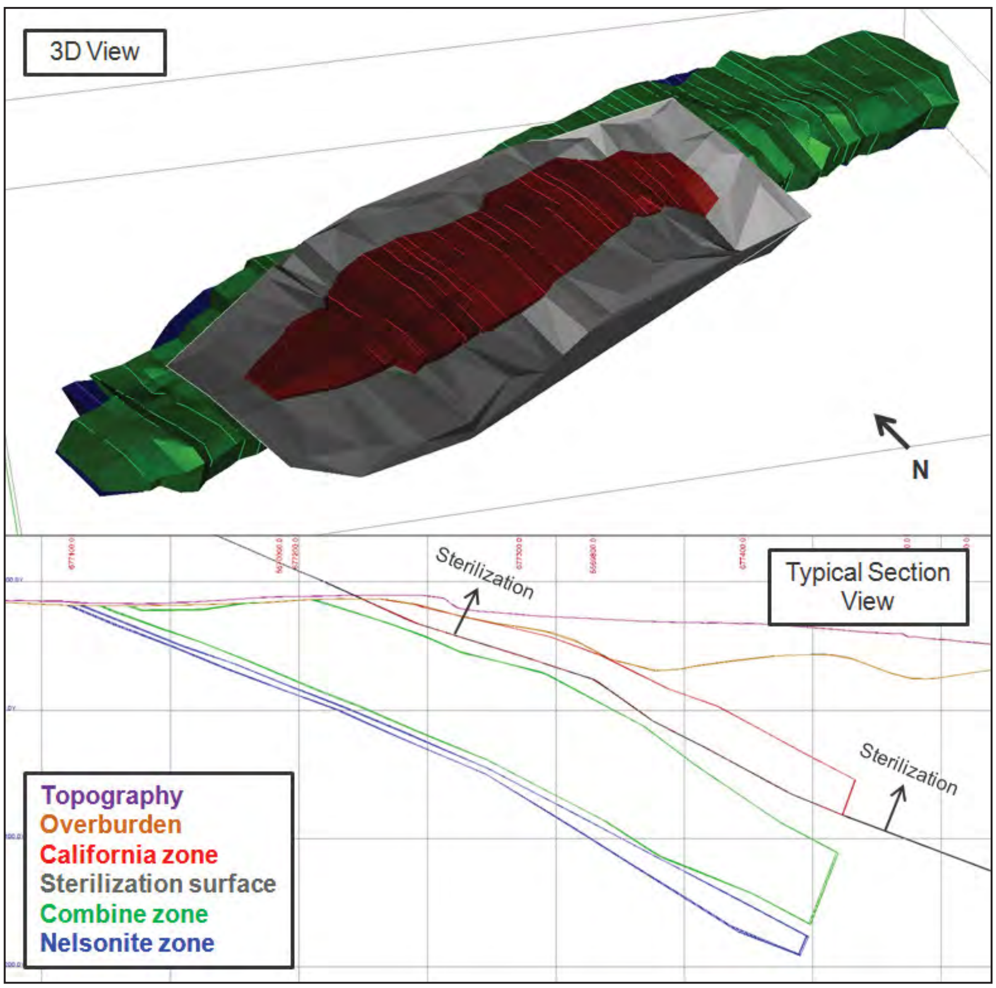


Figure 16-1: Sterilization of the California Zone

16.3 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 executed 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 or to the tailings area for on-going reclamation. The run-of-mine mineralization will be delivered by large mining trucks to the primary crusher or stockpiles near the main crusher.

16.4 Overall pit-slope angle

The selected slope angles were based on a pit slope stability study completed by Ausenco-Vector (2011). The Table 16-1 summarizes the selected slope angles. The pit was divided into six geotechnical domains and slope attributes were provided for each of them (Figure 16-2).

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] Berm width = 6.5 m

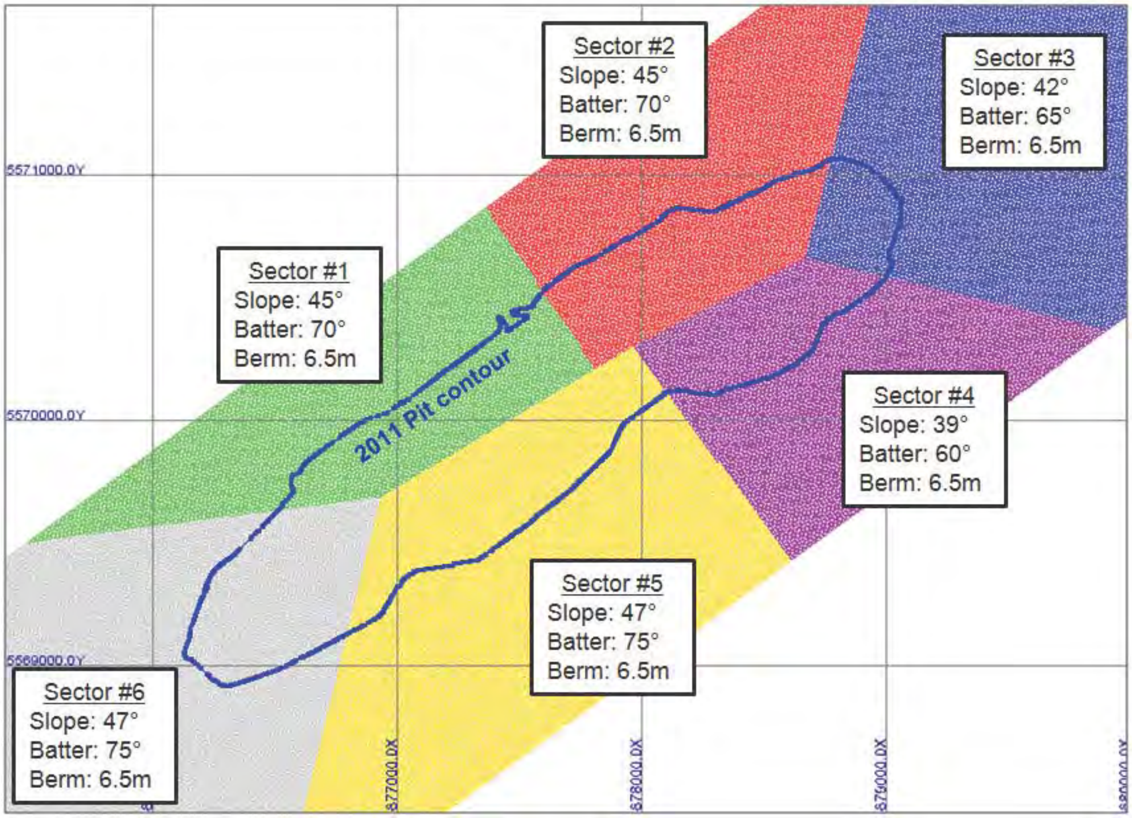


Figure 16-2: Geotechnical Domains

16.5 Pit Optimization Procedure and Parameters

The purpose of the 2014 Feasibility study was to update the previous PFS (2013) but not to considerably change the mining aspects of the project. For this reason, a simple pit optimization was carried out with updated assumptions, only to compare the size of the optimized pit shell versus the detailed mine design as presented in 2013. If the results were comparative, SGS Geostat would simply use the 2013 PFS pit design, with small adjustments, to limit the mining activities for this 2014 Feasibility study.

The optimization 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 Opex (mining, processing, and 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 located above the Upper mineralization zone should 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 had an important percentage of this deleterious element.

For the pit optimization, the required parameters were selected by SGS Geostat 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 Table 16-2.

Table 16-2: Open-pit Optimization Parameters

Note: The economic parameters used at the time of the pit optimization do not necessarily corroborate those stated further in this study, due to the iterative nature of this stage of the process. In others word, SGS had made initial assumptions in order to have a base case that would be used to perform the detailed study (the Feasibility study).

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

As anticipated, the optimization gave similar results to the 2013 PFS pit design. The fully optimized pit shell was completely overlaying the 2013 PFS pit design; as shown by the Figure 16-3. For this reason, the 2013 PFS pit design, accompanied with small modifications (see further sections) was selected as the base case in order to continue this study.

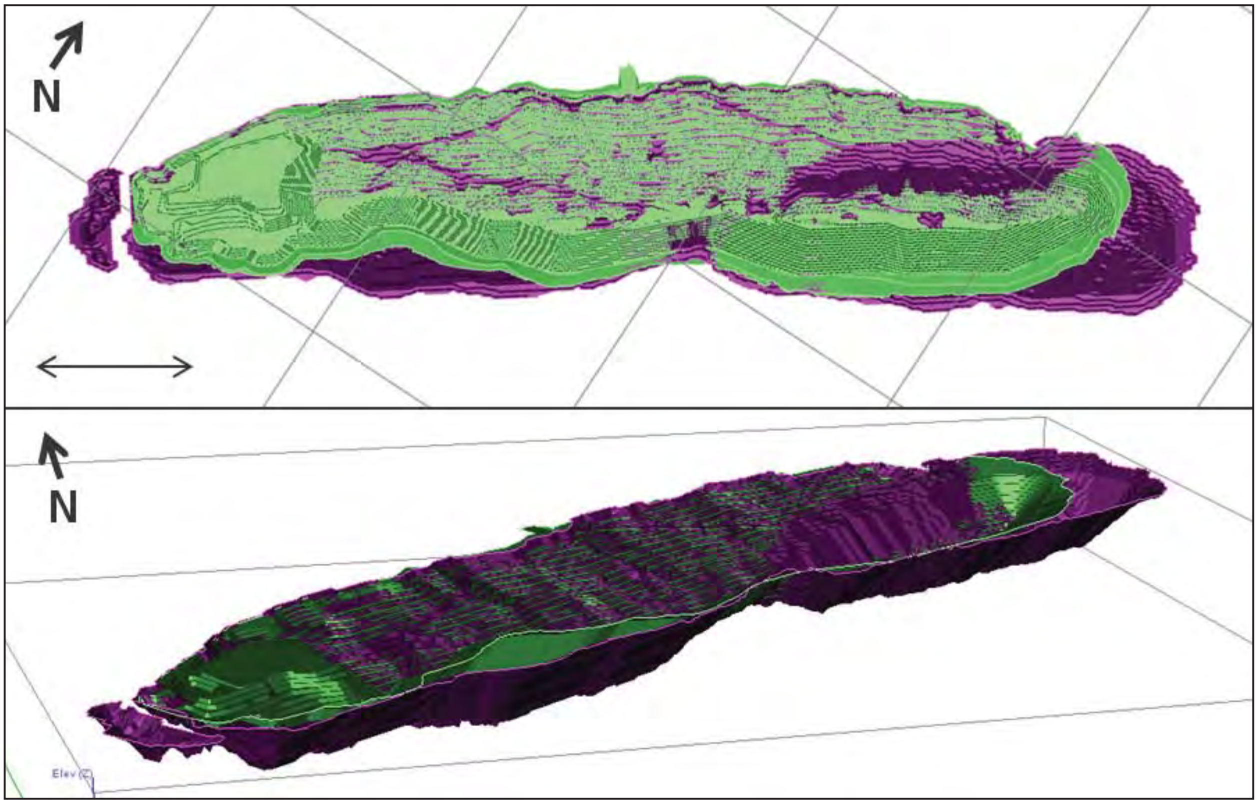


Figure 16-3: Fully optimized shell (purple) versus 2013 PFS pit design (light green)

16.6 Ore dilution and ore losses

Using the 2013 PFS pit design, SGS Geostat conducted a detailed calculation of the ore dilution by analyzing both the contact dilution and the mining dilution.

16.6.1 Contact Dilution and Contact Ore Losses

Considering that it is almost impossible to only mine ore material without including a quantity of waste, SGS Geostat conducted an exercise in order to calculate the percentage of dilution (waste material) that is introduced in the ore mining polygons. An ore mining polygon can be defined at the contour of an ore zone that the mining operation will rely on to know what has to be sent to the processing plant.

SGS Geostat used four typical benches through the 2013 PFS pit design (Elevation 45, 15, -15 and -45) and drew contours that simulate these ore mining polygons around the ore zones, see example below in Figure 16-4. These contours are based on a selected cut-off grade of 1.76 %P_2O_5 for all types of ore (Combined, Surrounding and Nelsonite).

The selective mining unit (SMU) considered in this study is 5m in all dimensions. Selection of the SMU was based on the scale of the geological unit, minimum bench height, spacing between blast holes and dimensions of the shovel used for extraction. The typical bench height for the majority of the production is 10m: this scale of blasting and mining is permitted by 90% of the ore (Combined and Surrounding). The option of using a 5m bench height was retained specifically for the Nelsonite unit which is thinner and higher grade. The dilution estimates were constructed based on these parameters. Although selective mining is possible on a 5m scale, it will likely be operated on a 10m scale for the most part (Combined and Surrounding units).

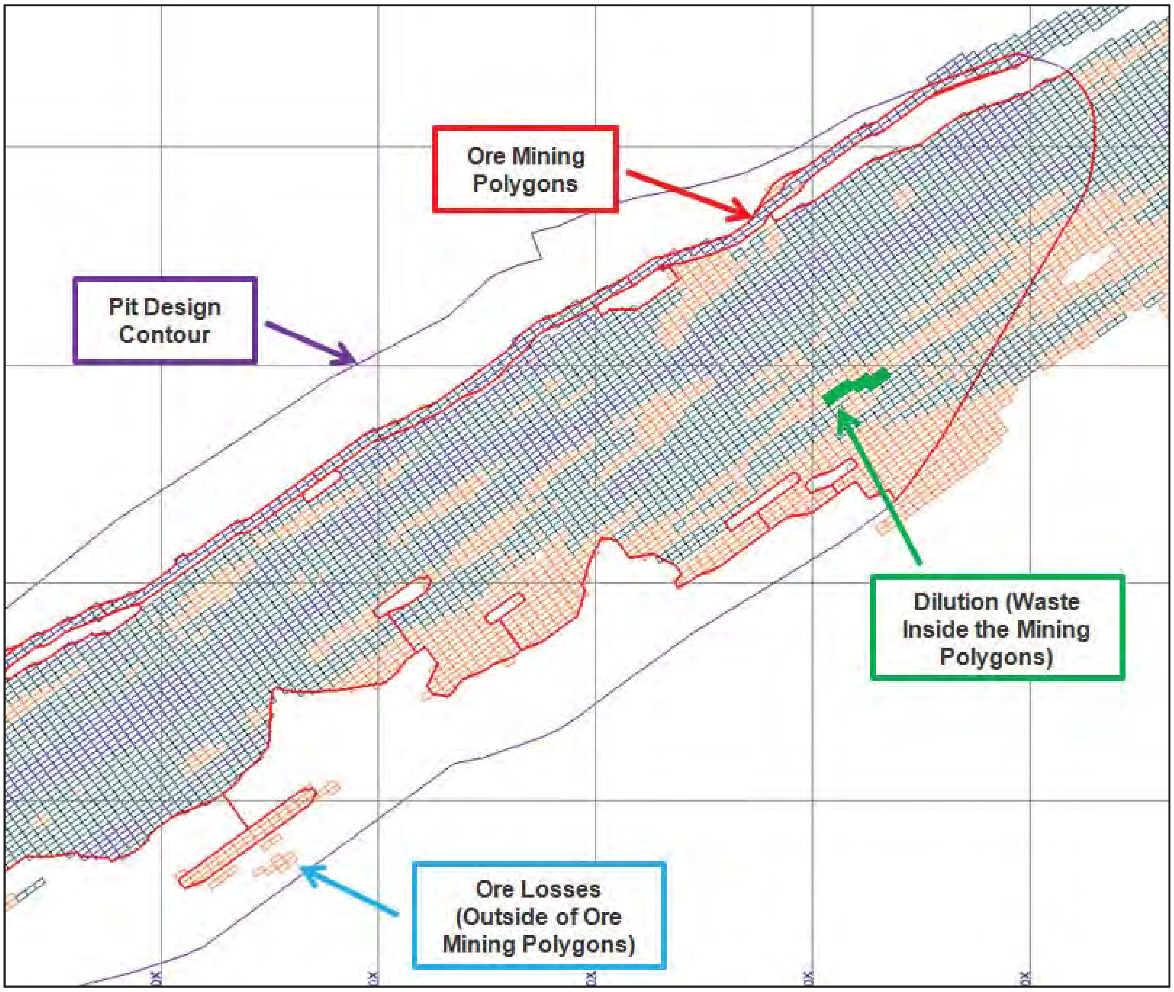


Figure 16-4: Example of Ore Mining Polygons (Bench Elevation -15)

Once completed, SGS Geostat calculated the total ore material in these polygons and compared it with the total in-situ tonnage above the cut-off grade, of the benches. This exercise allowed SGS Geostat to calculate a contact dilution and a contact ore loss of each bench. The average specific percentage values used for the entire deposit are outlined in the Table 16-3.

Table 16-3: Contact Dilution and Ore Losses

Ore Zone	Contact Dilution (%)	Dilution Grade ($\%P_2O_5$)	Contact Ore Loss (%)
Combined	2.19	1.28	1.18
Surrounding	2.19	1.28	1.18
Nelsonite	5.90	0.89	2.79

The dilution grade is attributable to the % P_2O_5 content of the waste material that contains mineralization (blocks having a grade below the 1.76 % P_2O_5 cut-off grade).

16.6.2 Mining Dilution and Mining Ore Losses

In addition to the contact dilution that is created, loading activities will also generate a mining dilution and a mining ore loss. This is mainly due to the dip of the ore zones versus the face angle that results from mining, which is normally equal to 70 degrees. For a visual explanation of the principle, refer to the Figure 16-5.

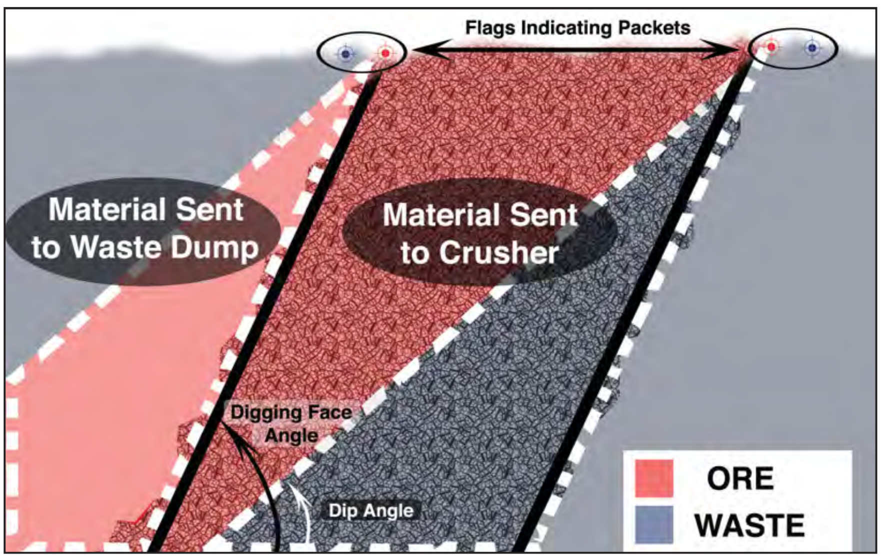


Figure 16-5: Mining Dilution Illustration (Dip Angle versus Digging Face Angle)

The Figure 16-6 and Figure 16-7 illustrate this principle in relation to Mine Arnaud's three main ore zones (Combined, Surrounding and Nelsonite). For simplification purposes, the Surrounding ore zone was treated as the Combine zone. The ore/waste contact was digitized at mid-bench.

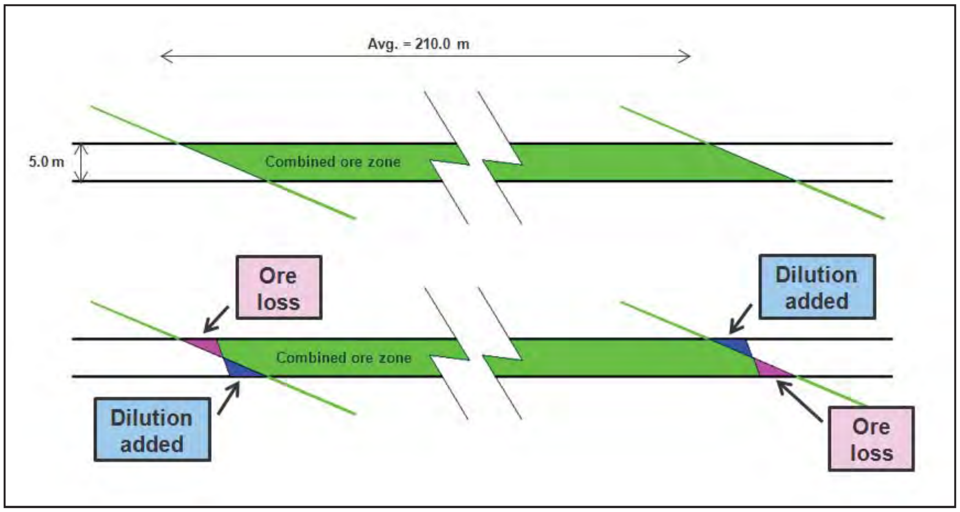


Figure 16-6: Dip Angle versus Digging Face Angle (Combined and Surrounding Ore Zone)

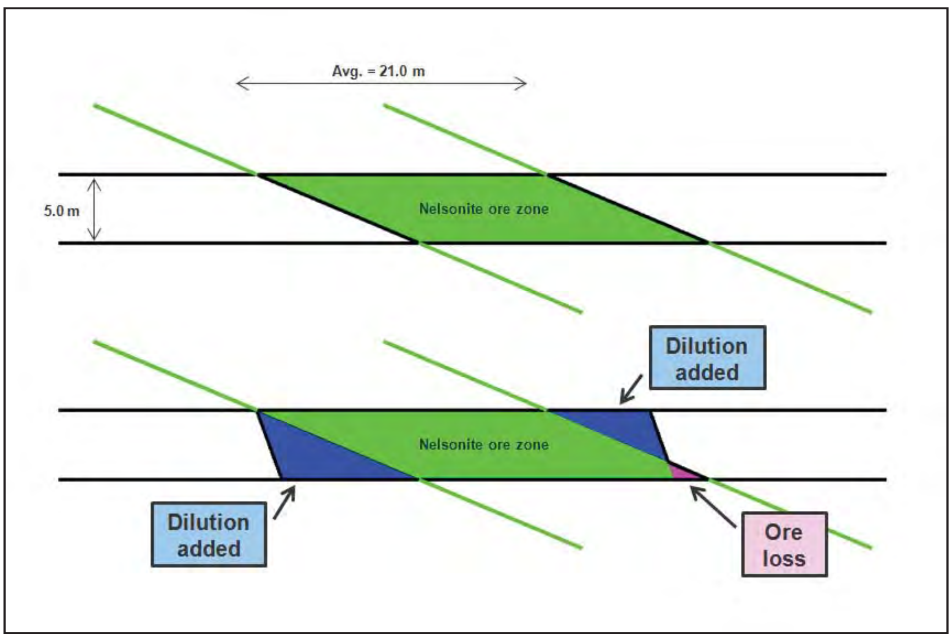


Figure 16-7: Dip Angle versus Digging Face Angle (Nelsonite Ore Zone)

Using these two previous section views, SGS Geostat calculated the resulting dilution added and the ore losses. The results in Table 16-4 are defined as follow:

Table 16-4: Mining Dilution and Ore Losses

Ore Zone	Mining Dilution (%)	Dilution Grade (%P_2O_5)	Mining Ore Loss (%)
Combined	1.20	1.28	1.19
Surrounding	1.20	1.28	1.19
Nelsonite	19.99	0.89	1.56

Note: For simplification purposes, the dilution grade was set at the same value that was calculated in the contact dilution analysis.

The selected 19.99 % mining dilution of the Nelsonite material is not entirely based on the analysis of the geometry of the ore, but was derived from the following assumptions:

- 50 % of the Nelsonite hanging wall is in contact with the Combined ore zone, so the half dilution on top of the hanging wall should be considered;
- 45 % of the diluted material on the footwall could be removed at the processing stage since this material is magnetite material, so 55 % of the dilution underneath the footwall should be considered.

16.6.3 Combined Dilution and Ore Losses

The dilution and ore losses used in this study result from the combination of the previous sections, i.e., the contact and the mining dilution and ore losses. The sums of these two are presented in the Table 16-5:

Table 16-5: Combined Dilution and Ore Losses

Ore Zone	Combined Dilution (%)	Dilution Grade (%P_2O_5)	Combined Ore Loss (%)
Combined	3.39	1.28	2.37
Surrounding	3.39	1.28	2.37
Nelsonite	25.89	0.89	4.35

16.7 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 cut-off grades were calculated using the results of the cost estimations and the selected concentrate value as presented in Sections 21 and 19 respectively. The resulting CoG for all types of mineralized material is noted in the Table 16-6.

Table 16-6: Calculated Cut-off Grades

* *Includes a CA$2.0 M year (or CA$ 0.18/t treated) contingency*

** *Based on the formula: Wrec% = 2.2264*%P_2O_5+0.3146*

16.8 Ultimate pit

16.8.1 Pit Design Parameters

As stated previously, SGS Geostat used the 2013 PFS pit design and adjusted it to reflect the updated physical constraints to be respected (the safety corridor of Hydro-Quebec power line). The 2013 PFS pit design was conducted using the following parameters (kept the same for the Feasibility study):

- Overall slope angle: Variable (refer to Section 16.3)
- Face angle: Variable (refer to Section 16.3)
- Bench height: 5.0 m
- 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 Figure 16-8

The ramp width was based on the assumption that the project operator will be using CAT 785 hauling truck. The ramp dimensions are also based on Quebec regulation (2013) 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 raods;

(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."

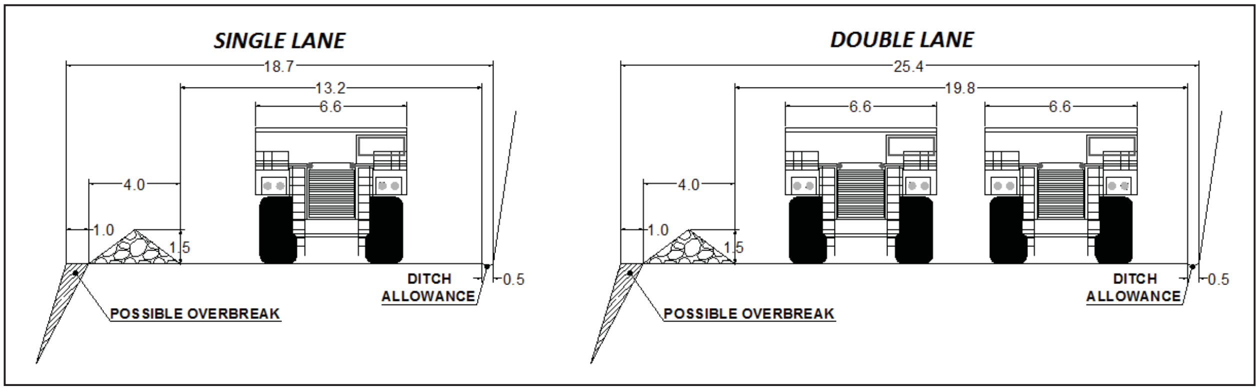


Figure 16-8: Hauling Ramps Specifications

16.8.2 Ultimate Pit Design

The Figure 16-9 shows views of the designed open-pit with its dimensions.

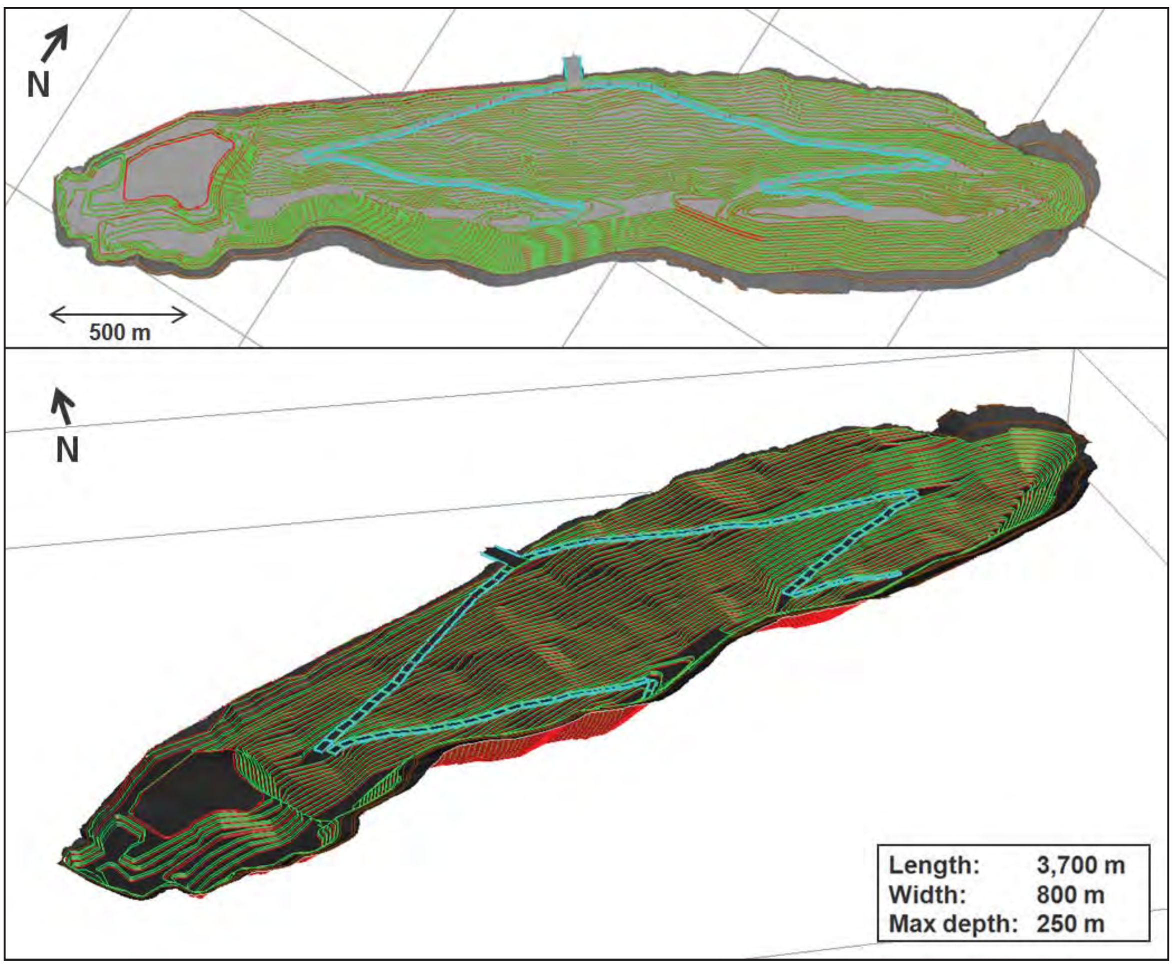


Figure 16-9: Final Pit Design

16.8.3 Material Contained within the Pit Design

The Table 16-7 denotes the tonnage contained inside the pit design.

Table 16-7: In-pit material (presented as mill feed)

Material Type		Cut-off ($\%P_2O_5$)	Tonnes	Grade ($\%P_2O_5$)	Grade (%Wrec)	Concentrate tonnes	Chlorine (%Cl)
Ore (Probable Reserves)	Combine	1.65	55,980,000	4.61	10.59	5,930,000	0.1176
	Surrounding	1.65	2,780,000	2.22	5.26	150,000	0.1451
	Nelsonite	2.05	9,660,000	5.24	11.99	1,160,000	0.0861
	Total		68,420,000	4.61	10.57	7,230,000	0.1131
Ore (Proven Reserves)	Combine	1.65	234,070,000	4.20	9.66	22,620,000	0.1189
	Surrounding	1.65	10,290,000	2.31	5.46	560,000	0.1216
	Nelsonite	2.05	29,820,000	5.06	11.59	3,460,000	0.0783
	Total		274,180,000	4.22	9.71	26,640,000	0.1137
Ore (Total Reserves)	Combine	1.65	290,050,000	4.28	9.84	28,540,000	0.1186
	Surrounding	1.65	13,070,000	2.29	5.42	710,000	0.1264
	Nelsonite	2.05	39,480,000	5.11	11.69	4,610,000	0.0803
	Total		342,600,000	4.30	9.89	33,870,000	0.1136
Waste	Waste rock		179,630,000				
	Overburden		63,940,000				
In-pit Total	**All**		586,170,000				

Note: This reserve includes 3.39 % dilution at 1.28 $\%P_2O_5$ and 2.37 % ore loss for Combine and Surrounding ore types and a 25.89 % dilution at 0.89 $\%P_2O_5$ and 4.35 % ore loss for the Nelsonite ore type.

In order to avoid any confusion, SGS preferred to present the Table 16-7 as the material that will be sent to the mill rather than all the material that was above the cut-off grades. The reader should be aware that a small portion of the blocks that meet the cut-off grade criteria are excluded nevertheless. This exclusion (19.80 Mt) was obtained from blocks destined for the low grade stockpile, that were discarded in order to lower the chlorine content of the low grade material, that will be treated in the last few years of operations (28 to 31).

16.9 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.

16.9.1 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:

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

The Figure 16-10 illustrates the proposed pushback width to be used in the design of the phase development:

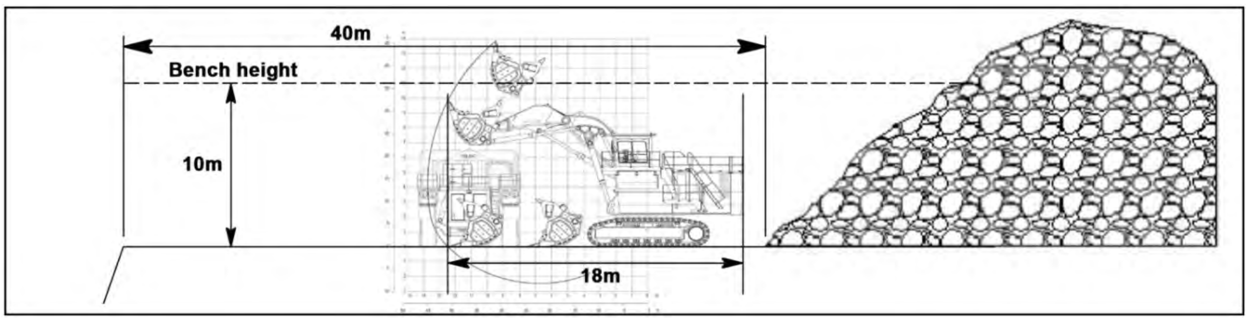


Figure 16-10: Minimum Pushback Width

16.9.2 Mine development

Four mineable 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 berm.

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 NPV. This specific Phase has an extension at the South, which was designed in order to access waste rock in the early years to fill materiel requirement attributable to the construction of the tailings dikes.

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 remaining mineralized material. The Figure 16-11 displays the four mining phases.

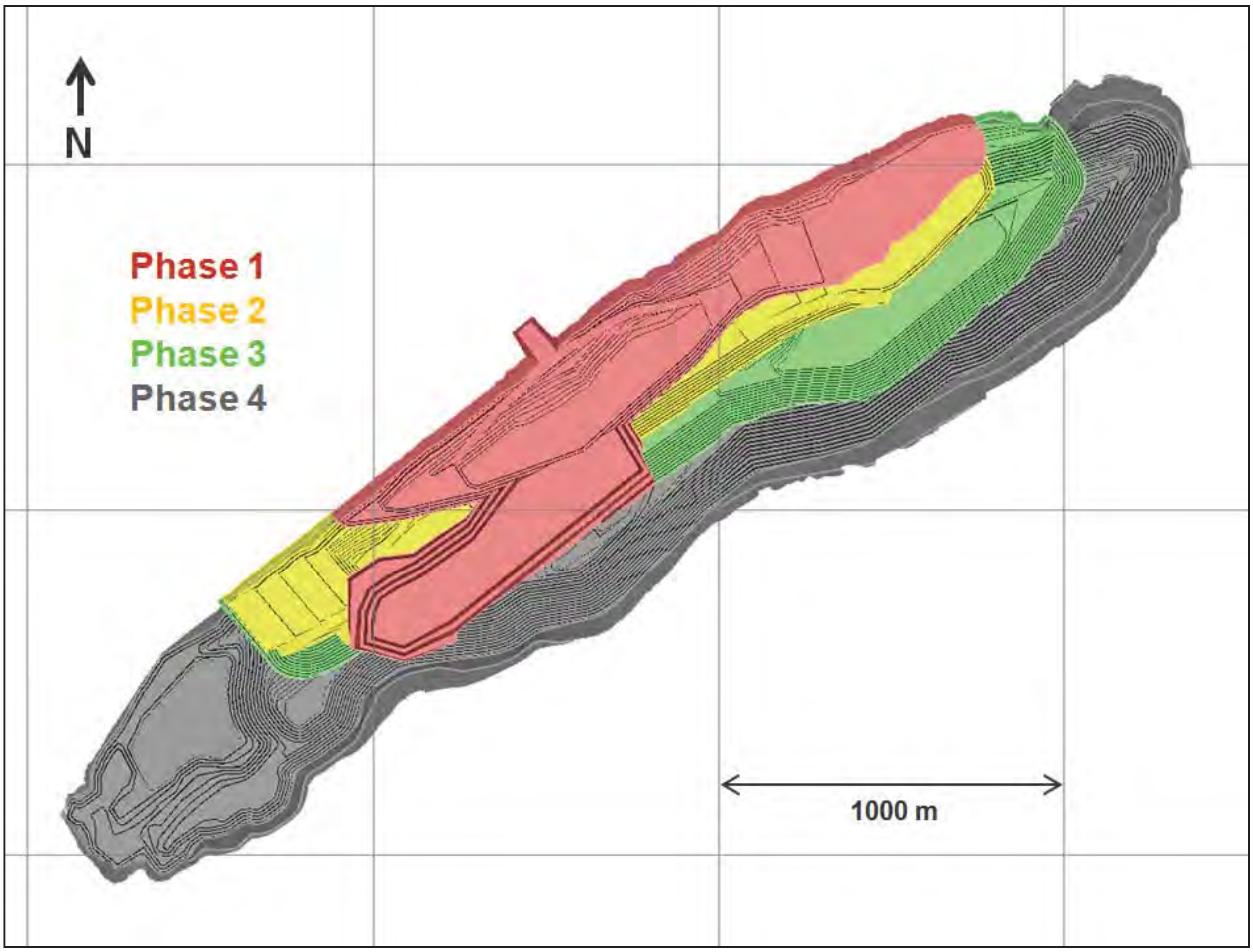


Figure 16-11: Mining Phases

16.9.3 Stockpiles

16.9.3.1 Blending Stockpiles

Blending stockpiles, of limited size, will be constructed in proximity to the main crusher. The main use of these stockpiles will be to allow blending of the material that will be sent to the processing plant to achieve any processing constraints such as P_2O_5 average feed or resulting chlorine content. They will be used when blending would not be possible to be performed directly in the open-pit from the mining activities. The sizing of these stockpiles is not fixed at this stage since their uses are on day to day and SGS only completed a yearly mine plan. However, a size ranging from 30,000 to 40,000 tonnes would be acceptable and sufficient at this stage. In this study, SGS estimated that 10 % of the daily mill feed will come from these stockpiles and that the material will be manipulated with a CAT 993K wheel loader.

It is important to not confuse these stockpiles with the low grade stockpiles since the blending stockpile will be used on a daily basis instead of storing low grade material for further use.

16.9.3.2 Low Grade Stockpiles

Two low grade ore stockpiles will be constructed. One at proximity of the main crusher and one annexed to the main waste rock dump. 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 low grade stockpiles will be constructed during years 2 to 20 of the mining operation. It is planned to send material with a grade ranging from 1.66 % to 3.00 % P_2O_5 for Combine and Surrounding ore types and ranging from 2.05 % to 3.00 % P_2O_5 for Nelsonite ore type. A total of approximately 57.00 Mt (61.95 Mt minus material that will be use for various construction activities) will be stored. From this total, an amount of 41.17 Mt will be re-handled and processed at the end of the open-pit operation, i.e: from years 28 to 31. The remaining tonnes (approximately 16 Mt) will remain in the stockpiles and will be considered as waste material.

Only a portion of the low grade stockpile was considered as treatable ore (41.17 Mt out of 61.95 Mt). This is due to SGS Geostat discarding a portion (20.78 Mt) of the total material that super exceeded the limit of 0.14% chlorine.

16.9.4 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 %P_2O_5 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 than 0.14 %;
- Mill throughput:
  - Year 1: 8,461,000 tonnes (75 % of full capacity)
  - Year 2: 9,591,000 tonnes (85 % of full capacity)
  - Year 3: 10,719,000 tonnes (95 % of full capacity
  - Year 4+: 11,283,000 tonnes (100 % of full capacity)
- High grading at 3.00 %P_2O_5 was done during years 2 to 20 in order to maximize the project NPV and IRR;
- Consequently, the usage of stockpiles was allowed (without any size limitation);
- The three interim mining phases were used as pushbacks;
- Production begins at year 1 following a two year pre-production period (-2 and -1);
- A maximum fixed lead of 20 vertical meters was allowed between mining phases.

The results of the developed schedule are summarized by the Table 16-8 and are presented by the Figure 1-1 to Figure 16-21. The key findings of the proposed mine plan include:

- Potential project life of approximately 31 years (including low grade material);
- Potential mill feed over the project life of 342.6 Mt at 4.30 %P_2O_5;
- An overall project stripping ratio of 0.71;
- Potential concentrate production of 33.87 Mt;
- Peak in mining the production at year 7 to 14;
- A well distributed stripping ratio.

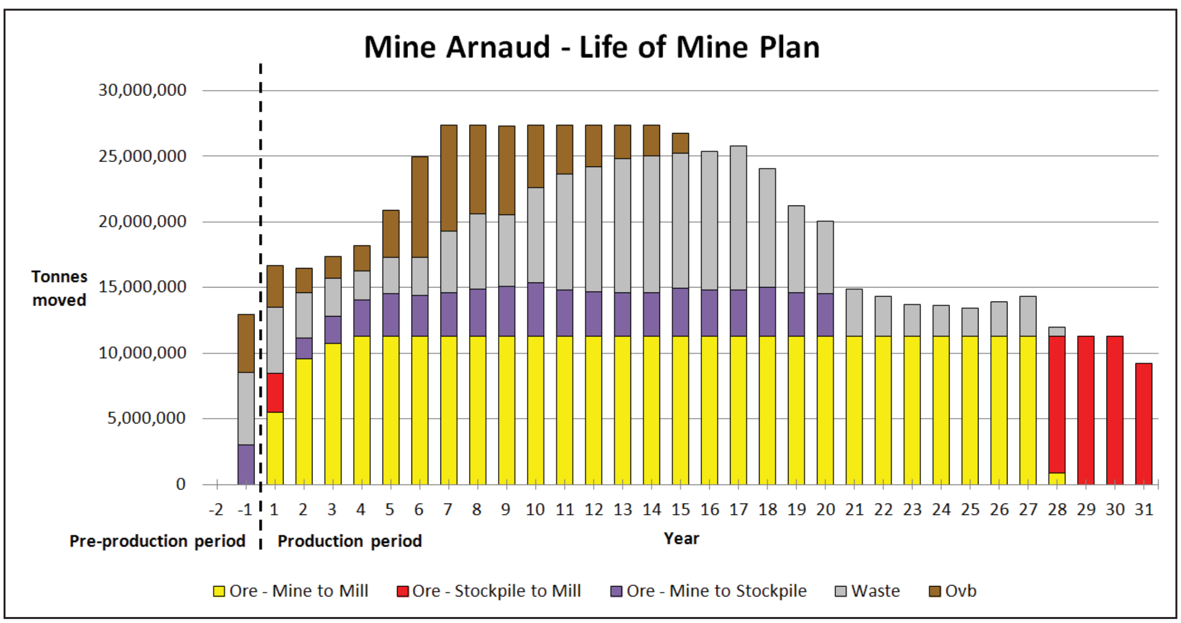


Figure 16-12: Production Schedule

Table 16-8: Production Schedule

Period	MINING																	PROCESSING				
	Ore Mine to Mill	Input Grade	Input Grade	Simulated Output	Ore Mine to Pile	Input Grade	Input Grade	Simulated Output	Ore Pile to Mill	Input Grade	Input Grade	Simulated Output	Waste	Overburden	Total mined from open-pit	Total moved	Stripping ratio	Ore treated	Input Grade	Input Grade	Concentrate production	Simulated Output
year	*tonnes*	*%Wrec*	*%P_2O_5*	*%Cl*	*tonnes*	*%Wrec*	*%P_2O_5*	*%Cl*	*tonnes*	*%Wrec*	*%P_2O_5*	*%Cl*	*tonnes*	*tonnes*	*tonnes*	*tonnes*	*t:t*	*tonnes*	*%Wrec*	*%P_2O_5*	*tonnes*	*%Cl*
-2																						
-1					3,000,000	10.69	4.66	0.1046					5,500,000	4,437,000	12,937,000	12,937,000						
1	5,461,000	10.69	4.66	0.10					3,000,000	10.69	4.66	0.1046	5,000,000	3,192,000	13,653,000	16,653,000	0.97	8,461,000	10.70	4.66	905,000	0.1046
2	9,591,000	11.18	4.88	0.11	1,540,000	5.58	2.37	0.15					3,460,000	1,849,000	16,440,000	16,440,000	0.71	9,591,000	11.18	4.88	1,072,000	0.1107
3	10,719,000	11.27	4.92	0.11	2,069,000	5.62	2.38	0.14					2,931,000	1,647,000	17,366,000	17,366,000	0.62	10,719,000	11.27	4.92	1,208,000	0.1136
4	11,283,000	11.12	4.85	0.11	2,773,000	5.59	2.37	0.15					2,227,000	1,873,000	18,156,000	18,156,000	0.61	11,283,000	11.11	4.85	1,254,000	0.1089
5	11,283,000	10.53	4.59	0.12	3,261,000	5.62	2.38	0.15					2,739,000	3,579,000	20,862,000	20,862,000	0.85	11,283,000	10.53	4.59	1,188,000	0.1158
6	11,284,000	10.64	4.64	0.11	3,085,000	5.56	2.36	0.14					2,915,000	7,665,000	24,949,000	24,949,000	1.21	11,284,000	10.64	4.64	1,201,000	0.1136
7	11,286,000	10.81	4.72	0.11	3,282,000	5.49	2.32	0.14					4,691,000	8,115,000	27,374,000	27,374,000	1.43	11,286,000	10.81	4.71	1,220,000	0.1106
8	11,283,000	10.52	4.58	0.12	3,597,000	5.59	2.37	0.15					5,734,000	6,761,000	27,375,000	27,375,000	1.43	11,283,000	10.52	4.58	1,187,000	0.1165
9	11,283,000	10.23	4.46	0.12	3,767,000	5.59	2.37	0.15					5,501,000	6,750,000	27,301,000	27,301,000	1.42	11,283,000	10.24	4.46	1,155,000	0.1200
10	11,283,000	10.10	4.39	0.12	4,046,000	5.51	2.33	0.16					7,279,000	4,767,000	27,375,000	27,375,000	1.43	11,283,000	10.09	4.39	1,139,000	0.1225
11	11,283,000	10.40	4.53	0.12	3,529,000	5.50	2.33	0.15					8,838,000	3,725,000	27,375,000	27,375,000	1.43	11,283,000	10.40	4.53	1,173,000	0.1187
12	11,283,000	10.61	4.62	0.12	3,359,000	5.52	2.34	0.15					9,573,000	3,161,000	27,376,000	27,376,000	1.43	11,283,000	10.61	4.62	1,197,000	0.1166
13	11,283,000	10.57	4.61	0.12	3,287,000	5.56	2.35	0.15					10,271,000	2,534,000	27,375,000	27,375,000	1.43	11,283,000	10.57	4.61	1,193,000	0.1156
14	11,283,000	10.54	4.59	0.11	3,337,000	5.59	2.37	0.15					10,400,000	2,355,000	27,375,000	27,375,000	1.43	11,283,000	10.54	4.59	1,189,000	0.1134
15	11,283,000	10.49	4.57	0.11	3,673,000	5.60	2.37	0.15					10,284,000	1,527,000	26,767,000	26,767,000	1.37	11,283,000	10.48	4.57	1,183,000	0.1145
16	11,282,000	10.42	4.54	0.12	3,493,000	5.58	2.36	0.16					10,590,000		25,365,000	25,365,000	1.25	11,282,000	10.42	4.54	1,176,000	0.1154
17	11,283,000	10.09	4.39	0.12	3,536,000	5.56	2.36	0.16					10,984,000		25,803,000	25,803,000	1.29	11,283,000	10.09	4.39	1,139,000	0.1197
18	11,283,000	10.19	4.43	0.12	3,736,000	5.40	2.28	0.17					9,028,000		24,047,000	24,047,000	1.13	11,283,000	10.18	4.43	1,149,000	0.1170
19	11,283,000	10.57	4.61	0.11	3,299,000	5.34	2.26	0.17					6,605,000		21,187,000	21,187,000	0.88	11,283,000	10.56	4.60	1,192,000	0.1110
20	11,283,000	10.71	4.67	0.11	3,278,000	5.32	2.25	0.17					5,484,000		20,045,000	20,045,000	0.78	11,283,000	10.71	4.67	1,208,000	0.1092
21	11,283,000	9.46	4.11	0.12									3,584,000		14,867,000	14,867,000	0.32	11,283,000	9.47	4.11	1,068,000	0.1197
22	11,283,000	9.50	4.13	0.12									3,004,000		14,287,000	14,287,000	0.27	11,283,000	9.50	4.13	1,072,000	0.1196
23	11,283,000	9.87	4.29	0.12									2,447,000		13,730,000	13,730,000	0.22	11,283,000	9.87	4.29	1,114,000	0.1158
24	11,283,000	10.38	4.52	0.11									2,322,000		13,605,000	13,605,000	0.21	11,283,000	10.38	4.52	1,171,000	0.1068
25	11,283,000	10.60	4.62	0.10									2,172,000		13,455,000	13,455,000	0.19	11,283,000	10.60	4.62	1,196,000	0.1025
26	11,283,000	10.86	4.74	0.10									2,600,000		13,883,000	13,883,000	0.23	11,283,000	10.86	4.74	1,225,000	0.0981
27	11,283,000	11.12	4.85	0.09									3,018,000		14,301,000	14,301,000	0.27	11,283,000	11.11	4.85	1,254,000	0.0892
28	864,000	10.51	4.58	0.08					10,419,000	5.58	2.36	0.1293	677,000		1,541,000	11,960,000	0.06	11,283,000	5.96	2.53	672,000	0.1227
29									11,283,000	5.58	2.36	0.1293				11,283,000		11,283,000	5.57	2.36	629,000	0.1293
30									11,283,000	5.58	2.36	0.1293				11,283,000		11,283,000	5.57	2.36	629,000	0.1293
31									9,188,000	5.58	2.36	0.1293				9,188,000		9,188,000	5.57	2.36	512,000	0.1293
Total	**297,430,000**	**10.49**	**4.57**	**0.1125**	**61,947,000**	**5.53**	**2.34**	**0.1542**	**42,172,000**	**5.58**	**2.36**	**0.1293**	**159,858,000**	**63,937,000**	**586,172,000**	**631,345,000**	**0.71**	**342,603,000**	**9.89**	**4.30**	**33,873,000**	**0.1136**

Notes

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

- The total of Mine to Pile and Pile to Mill section do not account for material coming from years -1 and 1

- Stripping ratio at years 1 to 28 is calculated according to: (Waste + Overburden + LG from Mine to Pile) / (Ore Treated)

- The Total stripping ratio is calculated according to: (Waste + Overburden + (LG from Mine to Pile - LG from Pile to Mine)) / (Ore Treated)

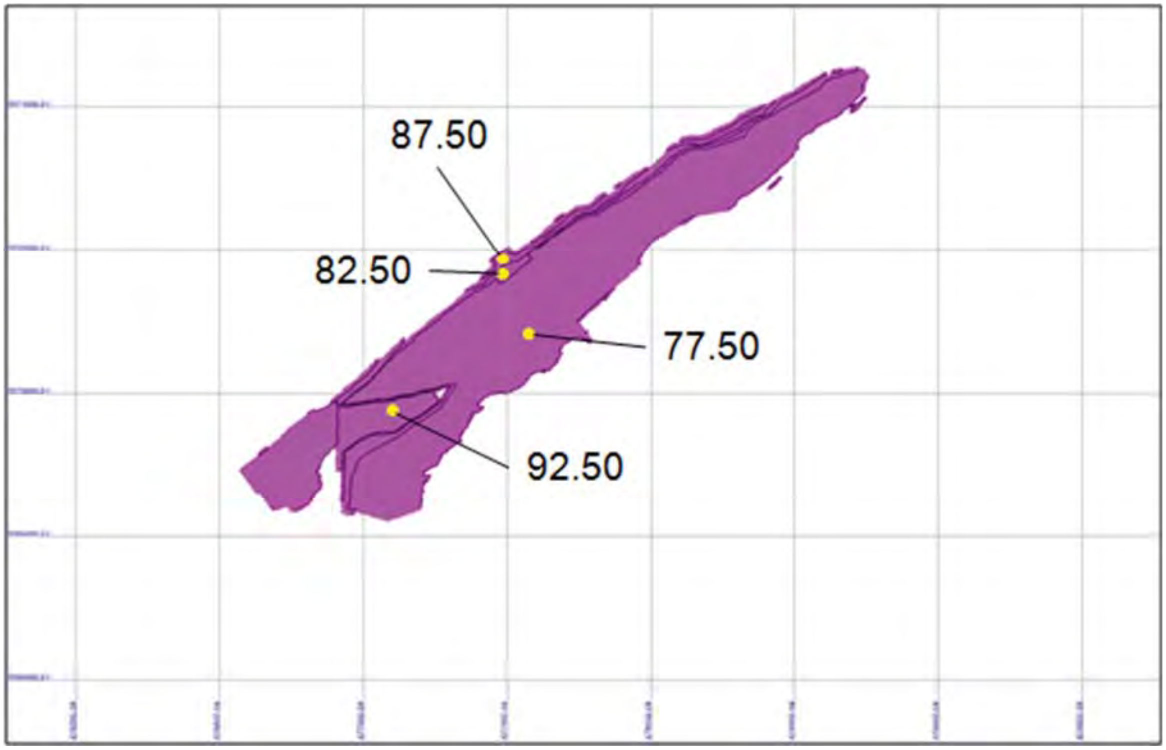


Figure 16-13: Schematic View of the Open-pit - End of Year 1

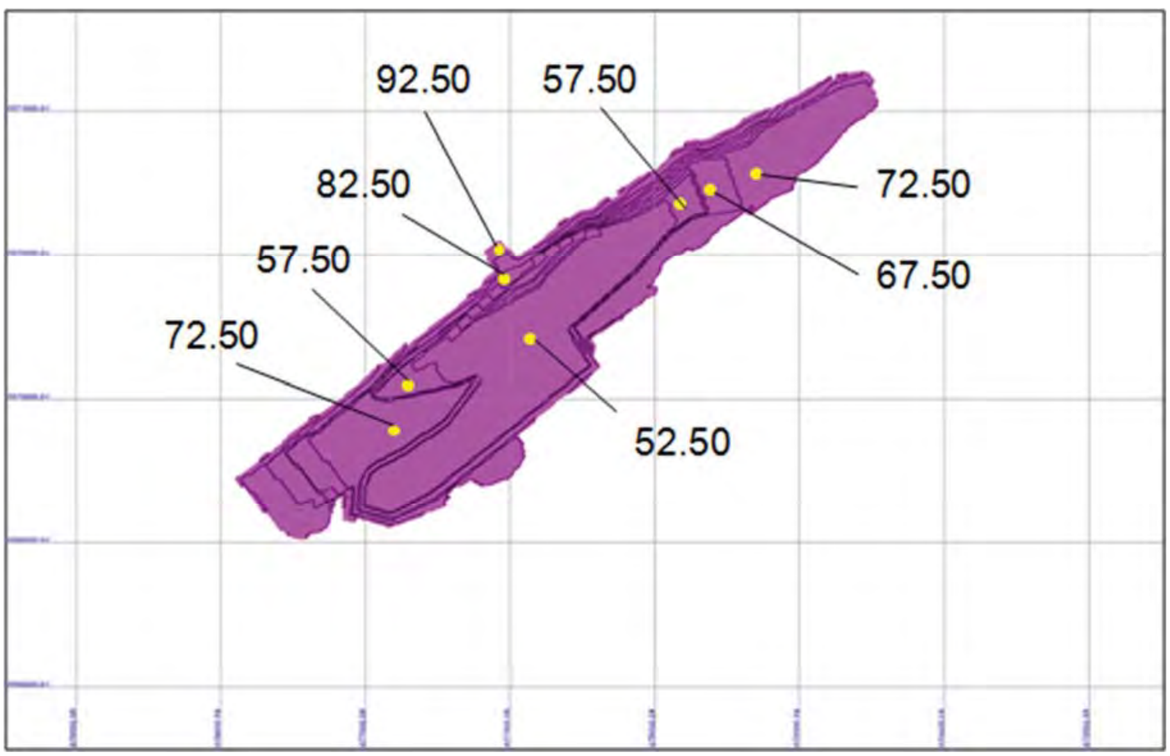


Figure 16-14: Schematic View of the Open-pit - End of Year 3

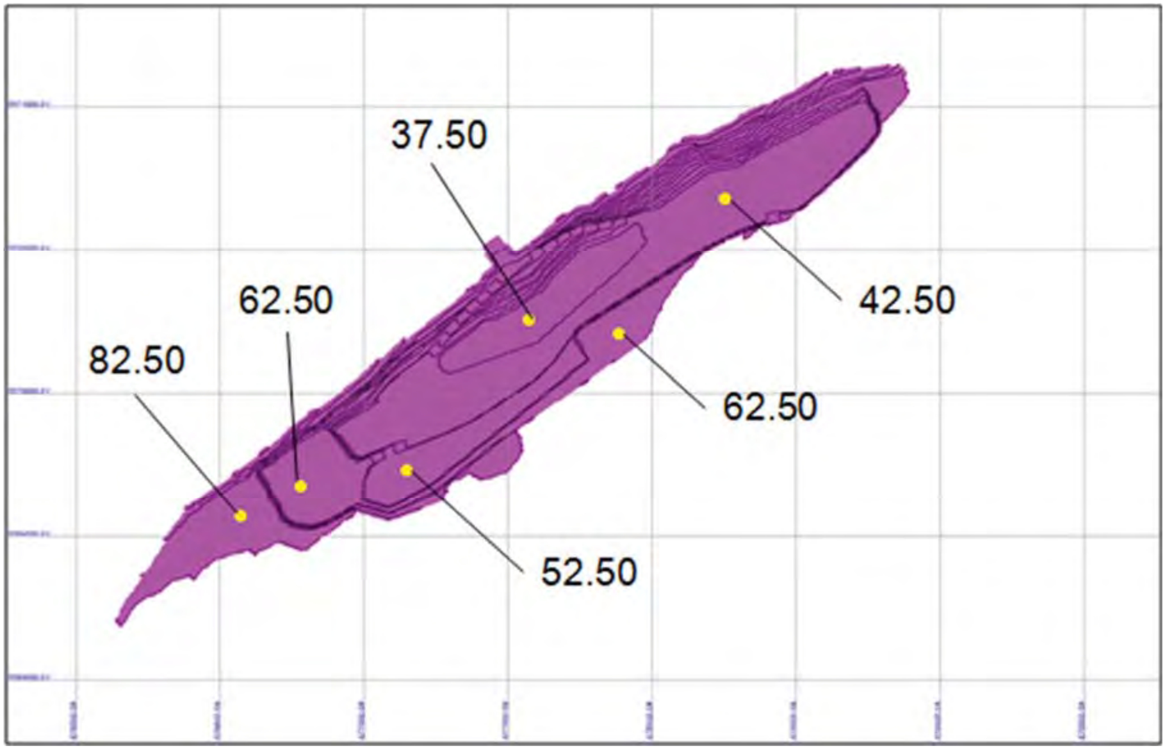


Figure 16-15: Schematic View of the Open-pit - End of Year 5

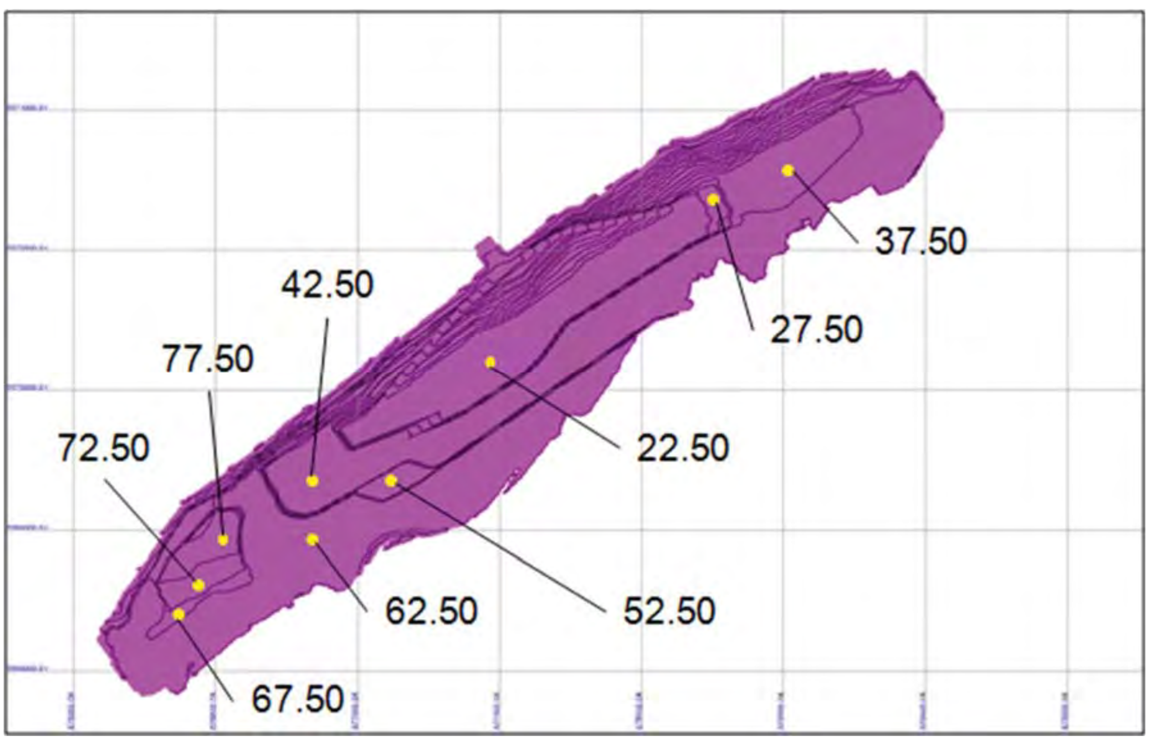


Figure 16-16: Schematic View of the Open-pit - End of Year 7

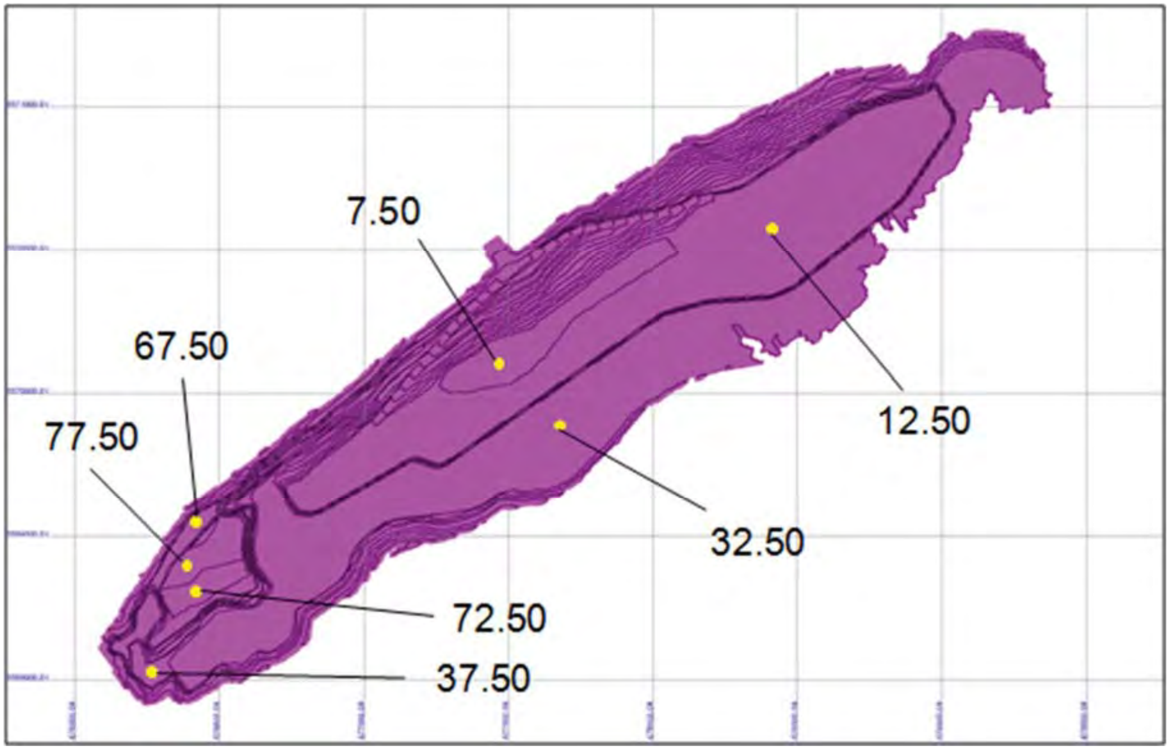


Figure 16-17: Schematic View of the Open-pit - End of Year 10

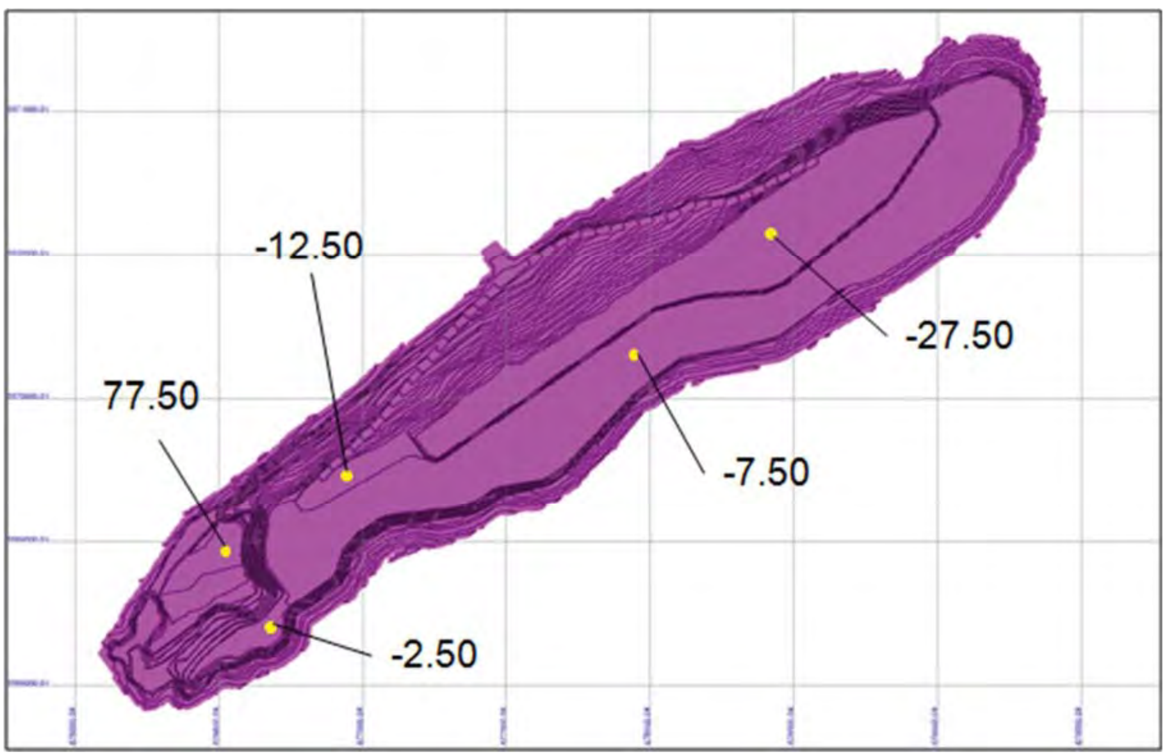


Figure 16-18: Schematic View of the Open-pit - End of Year 15

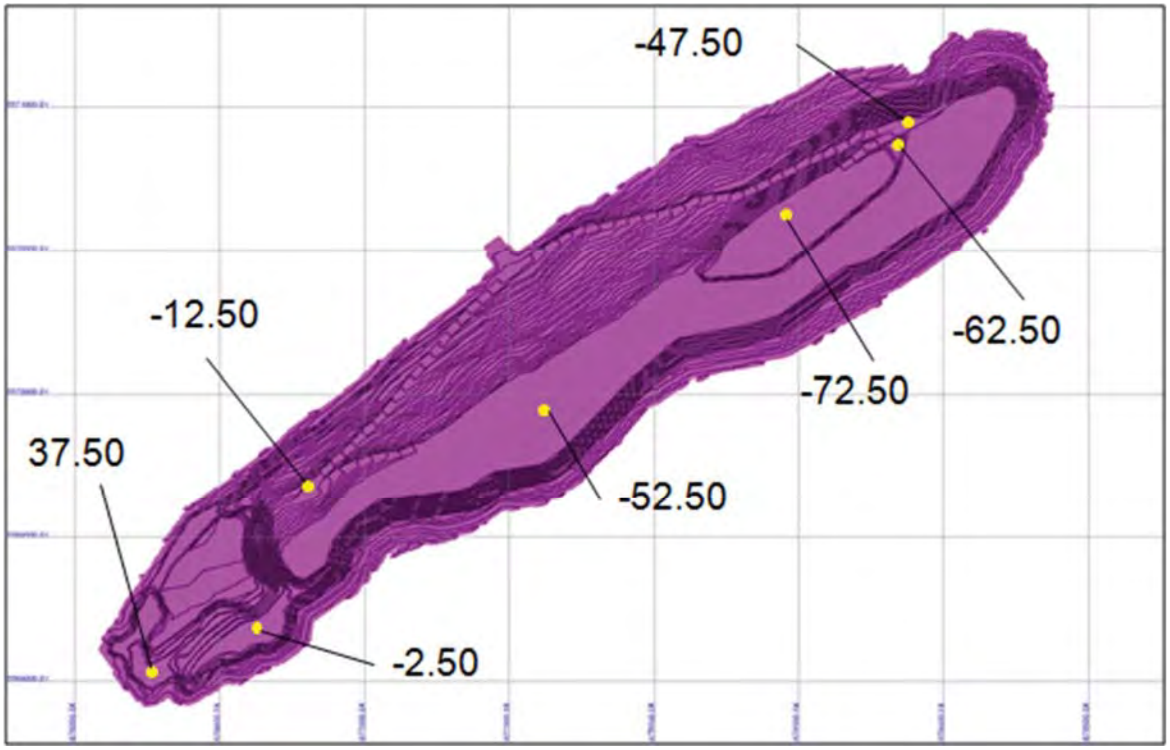


Figure 16-19: Schematic View of the Open-pit - End of Year 20

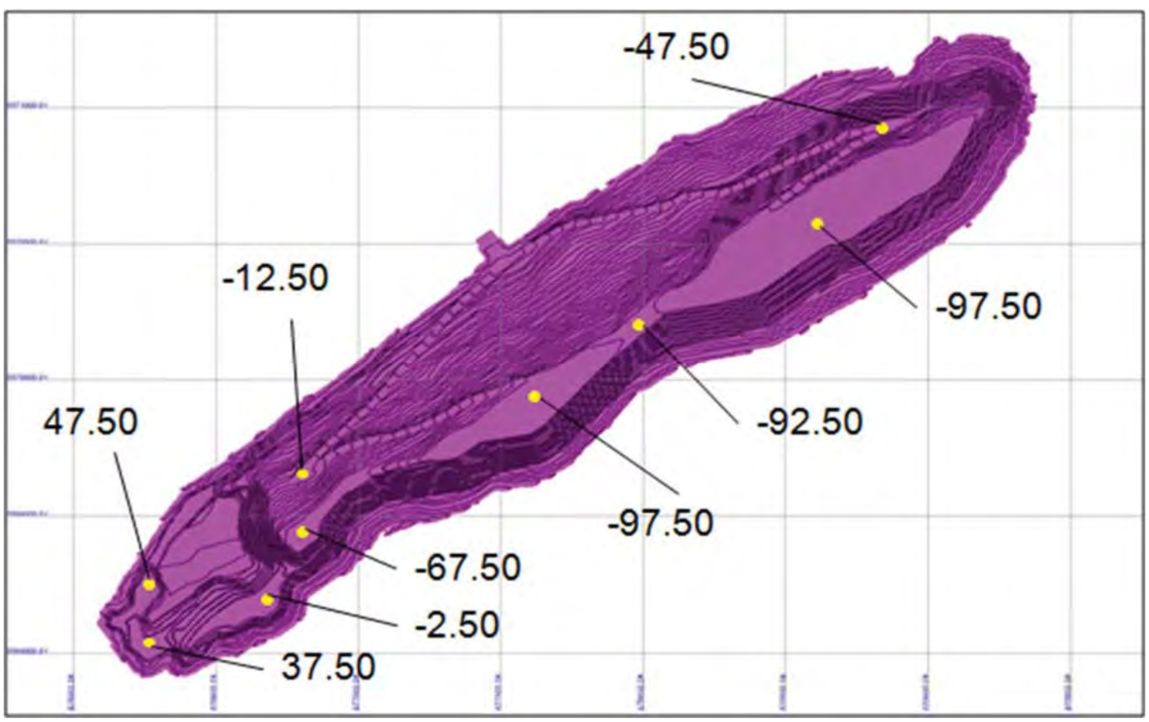


Figure 16-20: Schematic View of the Open-pit - End of Year 25

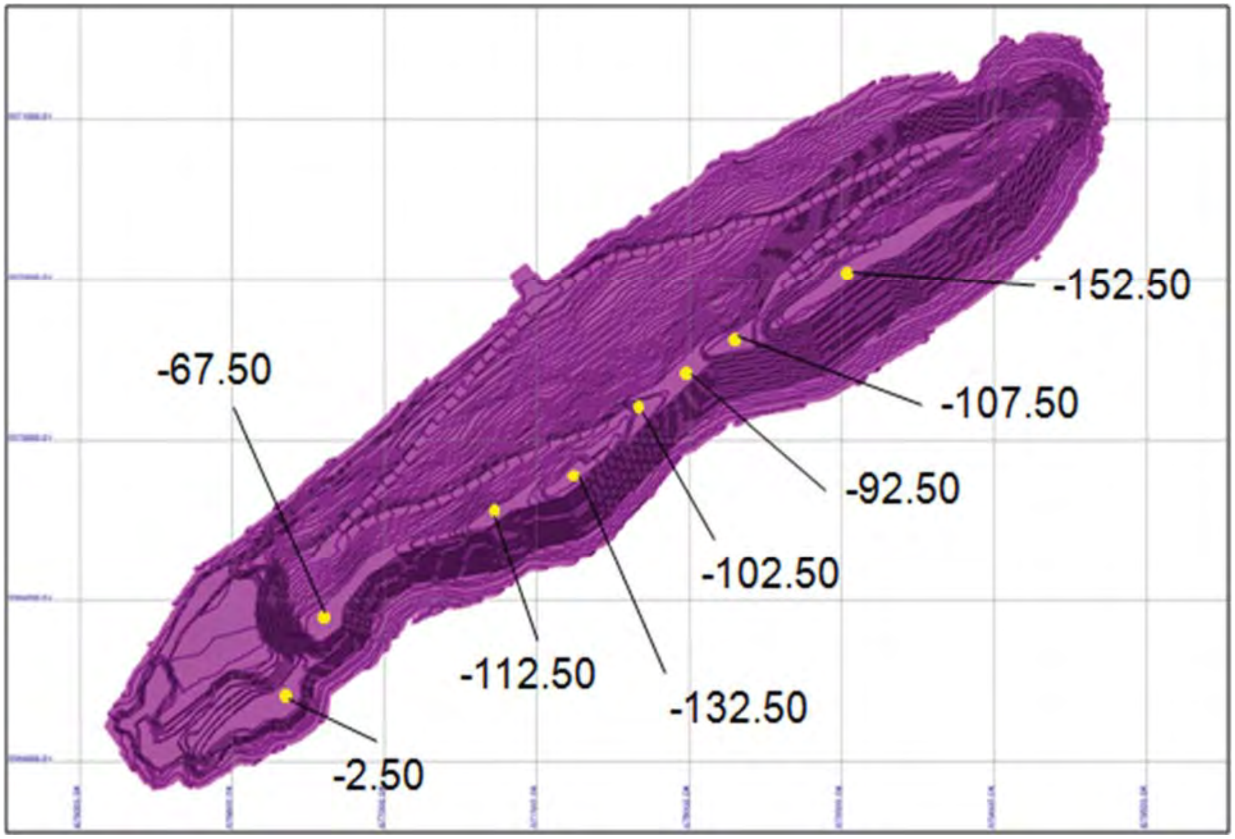


Figure 16-21: Schematic View of the Open-pit - End of Year 28

16.10 Mining Equipment Requirement and Others

16.10.1 Assumptions

The productivity of equipment has been estimated using industry standards along with information supplied by equipment manufacturers, available handbooks and in-house data. Cycle times were calculated based on average haulage distances to the respective destinations for each mining phase. On the assumption that resources are reported as dry tonnes, a moisture content of 1.0% has been assumed to convert dry tonnes to wet tonnes for the production estimate. In the present study, 90% of the tonnage is expected to be mined using 10m benches while the remainder, mainly the Nelsonite zone, will be mined using 5m benches.

16.10.2 Mining Equipment Purchase Schedule

The working cycles and productivity of equipment were prepared on an annual basis for ore, waste and overburden. The movements and impact of stockpiles were also taken in consideration for the total estimation of the equipment requirements of the life-of-mine. This information is presented below. One additional consideration in the fleet selection is the favourable location of the mine at Sept-Îles, Québec.

In the immediate area, there is a multitude of mining equipment distributors, service providers, and contractors setup primarily to service multiple large open pit iron ore mines located a further north. Table 16-9 and Table 16-10 display the purchasing schedule of major and service mine equipment.

Table 16-9: Major Open Pit Equipment Purchase Schedule

Equipment	References	Y-0	Y-1	Y-2	Y-3	Y-4	Y-5	Y-6	Y-7	Y-8	Y-9	Y-10	Y-11	Y-12	Y-13
Drilling															
Atlas Copco Viper 235	596 kW	**1**		1											
Sandvik DR560	641 kW	**1**													
Loading															
Komatsu PC3000-6	940 kW	**1**						1		1					
Wheel Loader CAT 993K	705 kW	**1**								1					
CAT 390D (overburden)	390 kW	**1**						2				1			
Hauling															
CAT 785D - 140 t	1005 kW	**3**	1		1			1			1	4	2	1	
Articulated Volvo AF40 - 40 t	530 kW	**4**				3		4			1	4			

Equipment	References	Y-14	Y-15	Y-16	Y-17	Y-18	Y-19	Y-20	Y-21	Y-22	Y-23	Y-24	Y-25	Y-26	Tot.
Drilling															
Atlas Copco Viper 235	596 kW	1		1											**4**
Sandvik DR560	641kW														**1**
Loading															
Komatsu PC3000-6	940 kW			1		1									**5**
Wheel Loader CAT 993K	705 kW			1								1			**4**
CAT 390D (overburden)	390 kW														**4**
Hauling															
CAT 785D - 140 t	1005 kW	1			1				3	1			1		**21**
Articulated Volvo AF40 - 40 t	530 kW														**16**

*Y-0 refers to year -2 to -1 (pre-production period)

Table 16-10: Mine Arnaud Open Pit Service Equipment Purchase Schedule

Equipment	References	Y-0	Y-1	Y-2	Y-3	Y-4	Y-5	Y-6	Y-7	Y-8	Y-9	Y-10	Y-11	Y-12	Y-13
Atlas Copco SmartROC T40	168 kW	1													
Motor Grader CAT 16M	221 kW	1								1					
Track Dozer Komatsu D275	335 kW	2								2					
Track Dozer CAT D7E	221 kW	1								1					
Excavator CAT 390D	390 kW	1								1					
Water Truck Kenworth	380 kW	2													
Service Truck Kenworth	381 kW	2													
Tire Handler Kenworth	382 kW	1													
Fuel/Lube Truck Kenworth	383 kW	2													
Tool Carrier CAT IT62H	158 kW	2								2					
Gooseneck Mover Kenworth	500 kW	1								1					
50 Tonne Crane		1													
Light Towers		6								6					
Mobile Welding Machine		2								2					
Dewatering Pumps & Pipes		3								3					
Pick-up Trucks		15					15					15			
Busses		1													

Equipment	References	Y-14	Y-15	Y-16	Y-17	Y-18	Y-19	Y-20	Y-21	Y-22	Y-23	Y-24	Y-25	Y-26	Tot.
Atlas Copco SmartROC T40	168 kW														1
Motor Grader CAT 16M	221 kW			1								1			4
Track Dozer Komatsu D275	335 kW			2								2			8
Track Dozer CAT D7E	221 kW			1								1			4
Excavator CAT 390D	390 kW			1								1			4
Water Truck Kenworth	380 kW		2												4
Service Truck Kenworth	381 kW		2												4
Tire Handler Kenworth	382 kW		1												2
Fuel/Lube Truck Kenworth	383 kW		2												2
Tool Carrier CAT IT62H	158 kW			2								2			8
Gooseneck Mover Kenworth	500 kW			1								1			4
50 Tonne Crane			1												2
Light Towers				6								6			24
Mobile Welding Machine				2								2			8
Dewatering Pumps & Pipes				3								3			12
Pick-up Trucks			15					15					15		90
Busses			1												2

*Y-0 refers to year -2 to -1 (pre-production period)

16.10.3 Drilling & Blasting

16.10.3.1 Hydro-Quebec Constraints

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

Hydro-Quebec 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 systems 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 explosives 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).

16.10.3.2 **Production Drilling for Ore and Waste**

Production drilling of the ore and waste will be carried out in advance of excavation with a diesel powered Atlas Copco Pit Viper 235 tower drill, as shown in the Figure 16-22 that is capable of single pass drilling the 5 to 10 m benches. A nominal 200 mm (7.875'') drill diameter is specified for the blast design. Over the life-of-mine, two original and two replacement units are needed, resulting in an average of just over 55,000 hours recorded per machine. The drilling output from one production drill is estimated at 446 meters per working day, equivalent to 40,700 tonnes of material, from an availability of 83%.

Figure 16-22: Rotary Tower Drill Atlas Copco Viper 235

A Sandvik DR 560, shown in next Figure 16-23 is recommended for pioneering work, mid range grade control drilling, final wall pre-split drilling and back-up unit for the Viper drills.

Figure 16-23: Drill Sandvik DR 560

This machine is rated for up to 216 mm diameter holes, however, for the pre-split operations the minimum diameter of holes is 76.2 mm. This drill could also be used as a back-up production unit in case of a breakdown of one of the two major production units. Over the life-of-mine only one machine is required.

16.10.3.3 Pre Split and Nelsonite Drilling

Geotechnical recommendations call for the use pre-split drilling and blasting to achieve the final South wall slope attributes. The proposed drilling for the final wall pre-split holes is a minimum diameter of 76.2 mm and 1.5 m spacing. In order to achieve the pre-split requirements upon completion of the detailed pit design, it is estimated that 310,000 metres will be required to complete this operation

In order to manage mining recovery and dilution of the Nelsonite ore, a modified drill pattern 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 rig 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), 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 during operation. 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 metre intervals around the projected contact. Based on sampling results, the projected footwall is refined and blast hole depths are adjusted to maximize mining recovery, while minimizing back break and resultant dilution. The current Opex estimate assumes a regular production blast for the Nelsonite footwall.

16.10.3.4 Ore and Waste Blasting

Blasting is assumed to be conducted on an all inclusive type contract by an explosive supplier. This procedure is well known in the area as all iron producers rely on this option. Budgetary quotes for these services were received from both Dyno Nobel and Orica Canada Inc. in which 100% use of water proof emulsion has been specified. The selected contractor is responsible for all equipment, buildings, and permits required for the contract, including the firing of the blasts. The contractor is also responsible for the hiring and supervision of all their personnel. 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 that all loading and blasting will be done on day shift only. The explosive supply contractor is also responsible for the magazines and maintaining of sufficient blasting supplies. Electric initiation systems are specified for all blasting outside of the Hydro-Quebec right-of-way to improve blast fragmentation, reduce ground vibration, and reduce noise levels. Initiation systems layout and firing are the responsibility of the mine operator. A powder factor of 0.3 kg/t is targeted (for ore and waste) for production benches. The powder factor was estimated using the same density for ore and waste, even if the waste density is slightly lower than the ore. These results in a safety margin for smaller drilling patterns and redrilling or reblasting blasting if needed. Table 16-11 summarizes the typical blast design parameters for the production of both ore and waste with 10 m benches.

Table 16-11: Production Blasting Parameters

Description	Units	Ore & Waste
Material Density	t/m3	3.00 - 3.50
Powder Factor	kg/t	0.3
Bench Height	m	10
Stemming	m	4
Sub-Grade	m	1.7
Hole Depth	m	11.7
Final Hole Diameter	mm	200
Burden	m	5.25
Spacing	m	6.25

16.10.3.5 Loading and Hauling

Open pit mining operations are scheduled to operate 24 hours per day with two 12 hour shifts. 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. The optimized fleet production and overburden removal, as well as costs, were estimated with the collaboration of supplier software, while SGS Geostat provided specifications for distances, ramp grade, salaries, fuel costs, etc,: the main parameters retained in the production estimation are the followings:

- Truck availability: 90%
- Shovels and loader availability: 98%
- Basic fleet availability: 88%

- Site utilization: 95%
- Worked hours per day: 20.4
- Yearly worked hours: 7,344
- Ore and waste truck payload factor: 140 tonnes
- Shovel passes per ore and waste truck load: 5
- Overburden trucks payload: 40 tonnes
- Shovel passes per overburden truck load: 4

16.10.3.6 Excavators & Loaders

Overburden

All overburden will be mined with a dedicated equipment fleet made up of Caterpillar 390D backhoes (presented in Figure 16-24) and 40t articulated Volvo haul trucks. The specific number of each machine required at different times of the operation was presented earlier in section 16.10.2 Mining Equipment Purchase Schedule. A 4m³ bucket capacity is necessary to limit four passes per truck load. Each backhoe averages just less than 30,000 hours of service. An additional Caterpillar 390D backhoe, or equivalent, is listed under ancillary equipment for general purposes but could also be used for production if required.

Figure 16-24: Excavator CAT 390D

Ore and Waste

Production loading of ore and waste rock will be performed by up to two diesel powered Komatsu PC3000-6 excavators (presented in Figure 16-25), or equivalent units. Three equipment replacements are required over the life-of-mine. This results in approximately 60,000 hrs being recorded per machine. A 15 m³ bucket is specified for five passes per truck load. Two excavators 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.

Figure 16-25: Front Shovel Komatsu PC3000-6

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.

One wheel loader CAT 993K, shown in Figure 16-26, is recommended for stockpile reclaim operations and for back-up units. This machine is capable of side loading the 140 truck fleet in six passes and feeding ore to the crusher. Based on the stockpile reclaim schedule, one loader is required with a further three replacement units throughout the life of mine. This would also allow the front end loader to be utilised in the open pit operation when required to be a replacement for an excavator.

Figure 16-26: Wheel Loader CAT 993K

16.10.3.7 Haul Trucks

Overburden

A dedicated truck fleet is matched to the overburden loading equipment. According to Mine Arnaud recommendations, the selected articulated trucks will be the Volvo A40F, primarily for their low environmental impact. Four 40 tonne capacity articulated trucks, shown in Figure 16-27, are specified for continuous operation from the start of the operation. A maximum of seven trucks will be required during years 8 to 10. An average of approximately 30,000 hours per truck will be undertaken during this stripping period. At least four trucks are paired with each loading unit at a time, resulting in a total of 16 trucks being required. Working with a larger truck size fleet might reduce the costs of the overburden stripping operation. Articulated trucks of 50 and 60 tonnes payload, manufactured by Bell Equipment UK, are now available on the market. A 50 tonne truck fleet would reduce the maximum fleet to 6 units, instead of 7, and lower the manpower cost accordingly, and the 60 tonnes truck would lower the maximum units to five instead of seven. The subcontracting alternative should also be considered in future detailed studies.

Figure 16-27: Volvo 40 t articulated trucks

Ore and Waste

A fleet of Caterpillar 785D haul trucks, (see Figure 16-28), with 140 tonne payloads are specified for ore and waste rock haulage. The truck fleet starts at three (3) units in Year -1 and hits its peak in the period Years 13 to 18 with ten units in operation. The total quantity of trucks to purchase will be twenty-one units. The loader/truck match of 140 tonnes payload trucks and 10 to 15 m^3 excavators is a selection which is in accordance with the recommendations from the heavy equipment suppliers of Caterpillar, Komatsu and Hitachi for the Sept-Îles project.

Figure 16-28: Hauling Truck CAT 785D (140 tonnes payload)

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.

A pit haul road width of 25.4 m is specified for Caterpillar 785D two way traffic; this includes a single shoulder screen berm and additional width for drainage of surface water. This is illustrated in the above Section 16.7: Open pit design parameters.

The Figure 16-29, 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.

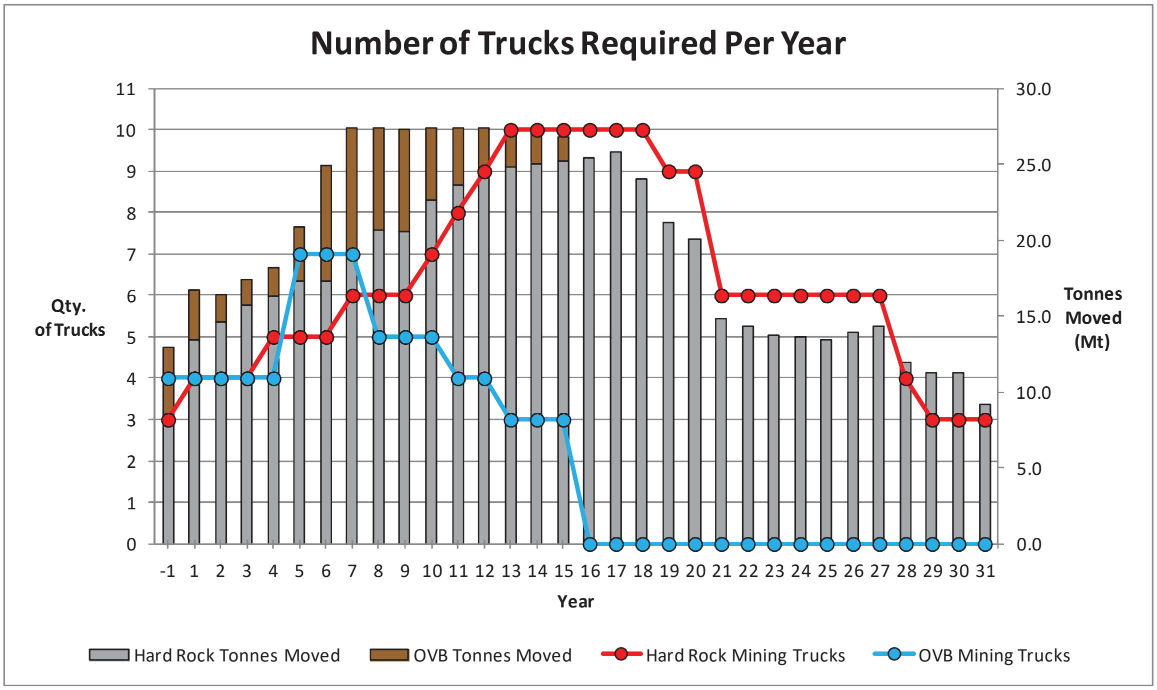


Figure 16-29: Trucks Requirement for OVB and Hard Rock Mining

16.10.3.8 Service Equipments

A fleet of support and service equipment is provided to assist the mine operation as shown in the Table 16-12.

Table 16-12: Ancillary and Support Equipment

Ancillary and Support Equipment
Atlas Copco SmartROC T40
Motor Grader CAT 16M
Track Dozer Komatsu D275
Track Dozer CAT D7E
Excavator CAT 390D
Water Truck Kenworth
Service Truck Kenworth
Tire Handler Kenworth
Fuel/Lube Truck Kenworth
Tool Carrier CAT IT62H
Gooseneck Mover Kenworth
50 Tonne Crane
Light Towers
Mobile Welding Machine
Dewatering Pumps & Pipes
Pick-up Trucks
Busses

- To comply with the environmental regulations of noise control and low exhaust emissions the project includes an Atlas Copco Smart Drill T40 top hammer machine, shown in the Figure 16-30. It is equipped with a silencer package, and that will be used for all secondary workings, like ramp start-up, sumps excavation, water drainage, etc.

Figure 16-30: Atlas Copco SmartDrill T40 with Silencer Package

- The project also includes a new CAT D7E track dozer, shown below in Figure 16-31, is described as an electric unit powered by a diesel generator which feed electric motors for drive. According to Caterpillar, it is a very economic and low emissions unit, which represents the future trend in dozer design. This unit will be mainly used for the construction of the screen berm, during day shift only, in accordance with an environmental recommendation. The dozer will be available as a support to mining operations during night shift and after completion of the screen berm.

Figure 16-31: Electric Track Dozer CAT D7E

- Ramps and haulage roads will be maintained using a CAT 16M grader.
- Two Komatsu D275 (similar to the Cat D9) track dozers will be available for cleanup around truck loading sites and for the proper stockpile arrangement, as mentioned above the Cat D7E will be available as a back-up unit.
- A CAT 390D excavator will be used for wall scaling, road and sump construction and mine cleaning purposes.
- Two customized T880 Kenworth fuel and lube trucks will service mining equipments; primarily for the production drills, the excavators, the dozers to avoid delays in operations.
- Two customized Kenworth trucks will also be available to the mechanical crews for troubleshooting and on-site preventive maintenance.
- A truck mounted tire handler will also be supplied by Kenworth to assist the mechanical crews.
- Two water trucks, also supplied by Kenworth, will provide the dust control for the roads during summer. These trucks are customized to have the water tank replaced by a sand spreader for the winter conditions.
- Two tool-carriers will be used primarily for assisting the mechanical crews but also for the blast hole stemming.
- A minimum of six diesel-run portable lights are available to assist the night mining crews.
- The table above denotes the remaining service equipment is: buses, 50 tonne crane, flatbed moving truck, welding machines and dewatering pumps.

16.11 Hydrological Study and Pit Dewatering

16.11.1 Hydrological Study

Hydrogeology characterization of the Arnaud Mine site was based on studies conducted for the feasibility study and for the environmental impact assessment between 2011 and 2013.

Work Summary

A total of 26 drill holes, including 25 observation wells, were completed to assess the hydrogeological units and the baseline groundwater quality of the aquifer located on the site. Two seismic refraction surveys (GPR, 2013) were performed to calculate the mean thickness of clay (6.6 m) at the junction of the bay's coastline.

Groundwater level monitoring data was taken from a total of 21 wells across the property: six in overburden, 12 in bedrock and three in the fault zone. In 2011, constant head (Lugeon) packer injection tests were performed on nine core holes (ninety-eight tests total) during drilling to determine the permeability of the rock in the area of the planned open-pit. Additional bedrock hydraulic testing was completed in 2012 and 2013, including slug testing (six in overburden, seven in bedrock) and constant head (Lugeon) packer injection tests to evaluate the permeability of fault zones (16 zones in 5 holes).

Two constant rate pumping tests were also performed in 2011, one in the overburden and the other in the bedrock to evaluate groundwater elevations and hydraulic characteristics of both units.

Hydrostratigraphy

Four hydrostratigraphic units were identified in the study zone, from the surface:

1. Coastal sand horizon, only in pit area (class II);
2. Clayey silt horizon with varying thickness and proportions of sand (class III);
3. Till horizon (silty sand and gravel) (class II); and
4. Bedrock (class II).

Hydrogeological formations Class I being of high importance and Class III of lower importance.

Water Levels

The groundwater flows towards the Sept-Îles Bay, in agreement with the local topography. No major fault zone has been intercepted in the pit area. Two fracture systems of NE-SW and NW-SE directions have been identified at the site. Minor fractures have been identified during drilling. Locally, preferential flows occur in the direction of the main fracture networks and ultimately, towards the Bay. The hydraulic gradients calculated in this sector range from 1.3% to 4.25% to 1.9% average gradient.

Water levels measured in the tailings area are close to the topsoil (between 0.07 and 2.81m deep) or above the topsoil (0.04m to 0.45m). Water levels ranged from 0.2 to 4.61m below the soil surface in the pit area. More shallow wells (4 to 5m) presented levels near the surface.

Groundwater Modeling

The data collected has been incorporated into numerical models to simulate groundwater flow.

A 2D cross-section model and analytical calculations were used to assess the groundwater inflow rate under tailing area and the future screen berm. The results are less than the maximum inflow daily stipulated by Directive 019 (3.3L/m²)

A 3D model was used to evaluate the dewatering flow rate and the drawdown cone extension. The model allows to verify the effects of dewatering on the surface watershed (mine Northwest lakes) and, secondly, to identify the origin of the water, which will be pumped outside the pit during dewatering. Steady numerical simulations have been completed for four different scenarios: in full-scale mining excavation activities (28 years), or after 7 years, 15 years and 23 years of operations. The dewatering flow rate of the pit has been estimated to be, 4,572 m^3/d after 7 years, 8,621 m^3/d after 15 years, 9,370 m^3/d after 23 years and 10,811 m^3/d at the end after 28 years.

At the end of the mine expansion (28 years), the proposed pit will reduce the groundwater elevations in an area bounded by 1m drawdown cone, which will extend to the Sept-Îles Bay and to 1.2km to the

Northwest. In the NE-SW axis, 1 m drawdown will extend to approximately 1.6km from sections and sides of the pit.

The simulation shows the low contribution of groundwater to surface water flow. The expected flow reduction of the water courses is mainly related to the reductions in the watersheds, the base flows representing only a small amount of the total flows (between 1 and 15%). Considering the proximity of the Sept-Îles Bay (salt water) from the pit, the risk of saline intrusion during the mining operations was evaluated. At the end of mine expansion, approximately 7% of the total groundwater volume flowing to the pit could be salt water. No saltwater intrusion is expected during the first seven years of operations.

16.11.2 Pit dewatering

Pit dewatering will be accomplished by pumping water from two main sumps inside the open pit. Horizontal drain holes in the pit walls are likely to be required as per the Hydrogeological Investigation Report. The Sandvik Drill DR560 type 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 drill 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 planned. There will also be the option to pump water between sumps if required in the event of an extended shutdown or moving of one sump. The selected pumps will be electrical ones of 75 kW each, capable of pumping 150 l/sec at 35 m static head. The number of pumps is varying by the pit depth like shown in the Table 16-13, the maximum daily dewatering requirement is estimated to be 11,000 m^3/day at the termination of mining for an electrical power installation of 425 kW. For safety purpose SGS Geostat has estimated the open pit electrical requirement to be twice the pumping need given by WSP, this safety margin will allow for unexpected flood rain and for small electrical services installations: staff offices, lunch rooms, etc., in other words the total power required at the end of the operation is 850 kW.

Table 16-13: Number of running dewatering pumps by periods

Items	Yrs. 1-7	Yrs. 8-15	Yrs. 16-23	Yr.s 24-30
Dewatering Pumps (qty.)	2	4	6	8

17. Recovery Method

The information contained in this item is an update of Item 17 of the SGS PFS report dated July 2013, as well as Item 6.4 of the Roche-Ausenco 2012 Feasibility Study and considers additional metallurgical testwork performed after those referred to in the SGS report.

17.1 Introduction

The process plant design criteria are based on Roche-Ausenco's 2012 feasibility study and on various sources of information. These sources are:

- Information provided by the client;
- Previous studies;
    - Internal Feasibility study conducted by Roche Ausenco in 2012
    - Feasibility study conducted by SGS Geostat in 2013
- Testwork conducted at SGS Lakefield, COREM, or by equipment suppliers before September 2012;
- Testwork conducted at COREM in September and October 2012;
- Testwork conducted at COREM in May and June 2013;
- Test work conducted at COREM on samples representing the ore at depth that would be processed between year 20 and 28 of the mine life;
- Roche-Ausenco's calculations, layouts, and/or recommendations of phosphate consultants;
- Industry standard practices or literature.

The plant capacity is established at 11,282,880 tpy Run-of-Mine (ROM), based on an ore processing rate of 1,400 tph, a plant availability of 92%, and 365 days of operation per year. The plant has been sized to meet the general design criteria and parameters as indicated in Table 17-1. The detailed design criteria are presented in APPENDIX E – Design Criteria.

The apatite grains are, for all practical purposes, liberated at 125 microns with power requirements being 10.1 kWh/t for ore crushing and 12.5 kWh/t for ore grinding (Bond ball mill work index). The concentrate will be dried between 0.5% and 1.5% moisture for ship transportation and subsequent processing by Yara.

Table 17-1: Mine Arnaud General Design Criteria

Parameter		Value	Unit	Source
Days per year		365	Days	A – D
Plant Capacity		1,400	tph	C
Process plant availability		92.0	%	D
Process plant operation		22.08	tpd	D
Ore processing per year		11,282,880	tpy	A - D
Average ore processing		30,912	tpd	D
Overall concentrate weight recovery		10.50	%	D
Concentrate produced total		1,184,702	tpy	D
Concentrate grade		≥39	%P_2O_5	D
Recovery		≥88	%P_2O_5	D
Tailings produced total		10,098,178	tpy	D
Reserves average grade		4.57	%P_2O_5	D
Reserves average grade		0.03	%Cl	D
Ore type proportion (%)	Nelsonite	20.0	%	A
	Combined	80.0	%	A
Apatite liberation size		125	µm	B
Concentrate specifications	P_2O_5 (%)	>39.0	%	A
	Fe+Al (%)	<1.0	%	A
	Ca/P	<2.2	Ratio	A
	Cl	<0.14	%	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% P2O5) material from stock piles processed after year 28.		Sources :		A. Mine Arnaud B. Corem C. Roche D. SGS

17.2 Flowsheet Process Description

The process description was written in association with the schematic drawings listed in Table 17-2. To help understand the flowsheets, the corresponding area descriptions can be found in the subsections below, and a Simplified Flow Diagram can be found in Figure 17-1 below. Detailed flowsheets supporting this report could be found in APPENDIX F – Flowsheets.

Table 17-2: Process Flowsheet Drawing List

Drawing No.	Process Area
15685-05-DR-FS-001	Crushing and Ore Handling
15685-03-DR-FS-001	Grinding
15685-03-DR-FS-002	Low Intensity Magnetic Separation
15685-03-DR-FS-003	Flotation
15685-03-DR-FS-004	High Intensity Magnetic Separation (optional)
15685-03-DR-FS-005	Concentrate Thickening, Filtering and Drying
15685-03-DR-FS-006	Tailings Thickening, Disposal and Water supply
15685-03-DR-FS-007	Reagents
15685-05-DR-FS-002	Silo Load-out

17.2.1 Crushing and Ore Handling

The ore will be hauled by 140-tonne mine trucks (Caterpillar 785D). The trucks will discharge into a 1,372 mm x 1,905 mm (54” x 75”) primary gyratory crusher driven by a 450 kW (600 HP) motor, with a nominal feed rate of 2,800 tph. The gyratory crusher circuit is designed for the throughput required as per client request, based on their 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. A hydraulic hammer will break down any oversize rocks in the crusher cavity. The crusher will discharge onto a 1,800 mm x 15,900 mm (6 ft x 52 ft) variable speed apron feeder, which will discharge onto a 1,220 mm (48”) wide 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 crushed ore is retrieved by five (5) 2,438 mm x 7,925 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. Each feeder has a design capacity of 1,680 t/h which is the same capacity of the mill feed conveyor. A weight scale is used to monitor and control the ore addition into the SAG mill.

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 55 tonne capacity hydraulic jib crane with a 7 tonne capacity auxiliary hook as well as 3 monorail hoists with 7.5 tonne capacity each

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.

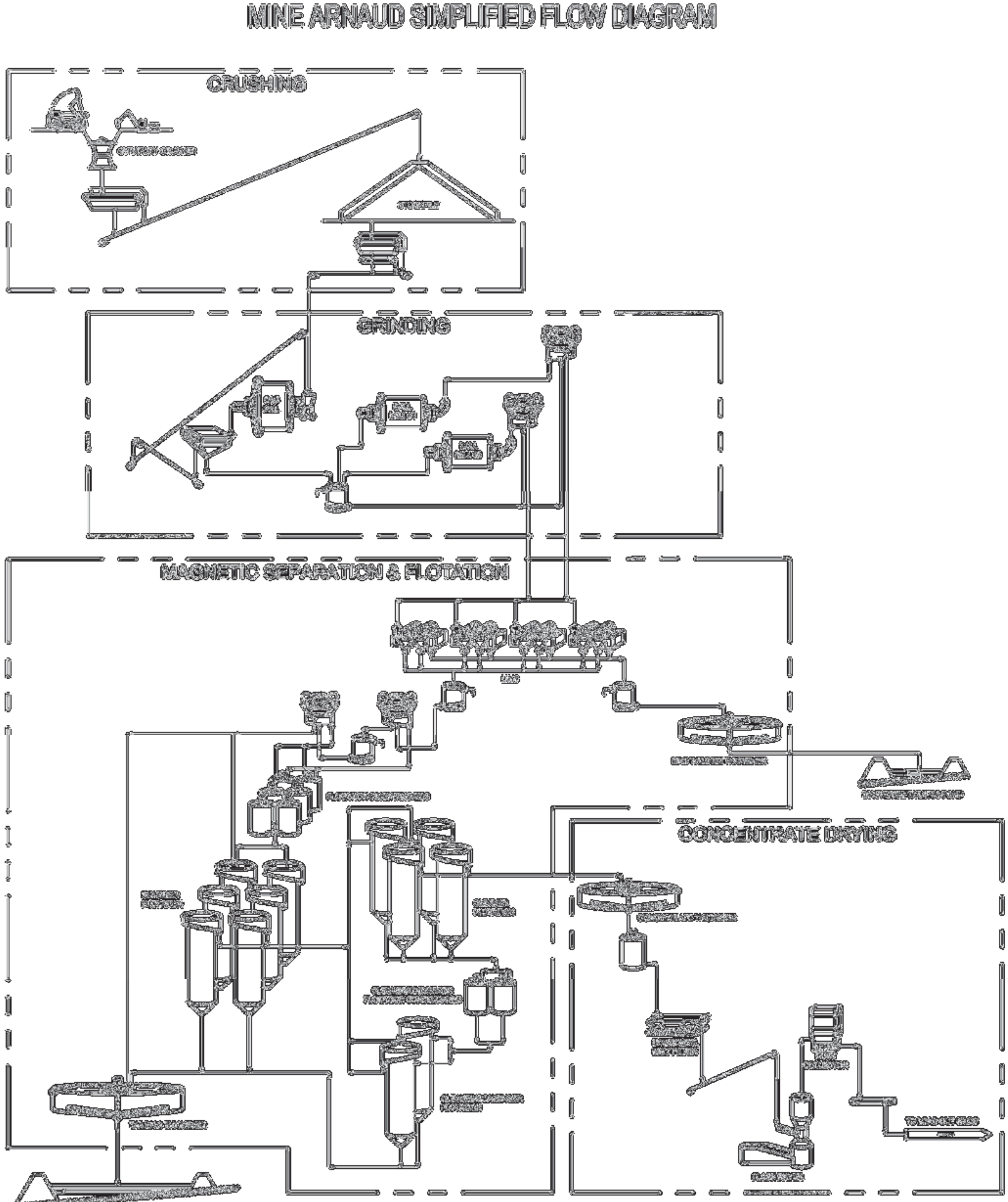


Figure 17-1: Mine Arnaud Simplified Flow Diagram

17.2.2 Grinding

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 be 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, with one (1) on stand-by. The cyclone underflow will flow by gravity to their respective mill feed spout. The cyclone overflow will have a P_{80} of approximately 125 µm.

The auxiliaries include 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.

17.2.3 Low Intensity Magnetic Separation and Dewatering

The overflow from the two classifying cyclone clusters is directed to two distributors and feed four (4) double drum Low Intensity Magnetic Separators (LIMS) (1,200 mm diameter x 3,200 mm long) to remove the titano-magnetite. The LIMS feed 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 pond cell.

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. A part of the overflow of the second stage is re-used for dilution after flotation reagent conditioning to adjust the feed solids prior to flotation. The remaining slurry is directed to the tailings thickeners. 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

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 dewatering cyclone overflow and then distributed to rougher stage #1 flotation column cells.

Each line of flotation will have two stages of rougher flotation column cells. The first stage consists of two (2) 4,880 mm x 14,000 mm rougher flotation column cells in parallel. The underflow from the first stage, or tailings, is pumped to a single second stage rougher flotation column cell (4,880 mm x 14,000 mm). Thus the froth or overflow from the two rougher stages (rougher concentrate) is combined in a froth launder and will 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. Froth from the cleaner/scavenger stage is recycled to the cleaner stage feed. Tailings from the cleaner/scavenger stage is combined with the stage 2 rougher tailings and is discarded as final tailings, and pumped to the tailings thickener.

The cleaner concentrate (final concentrate) should have a grade of 39% P_2O_5 containing less than 1% iron and aluminum combined (Fe+Al), with less than 0.3% magnesium (Mg), a calcium-to-phosphorous (Ca/P) of < 2.2 and less than 0.14% chlorine (Cl). The final apatite concentrate is pumped to the apatite concentrate thickener.

Six (6) low-pressure rotary screw compressors rated at 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.

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. Removing the WHIMS area from the plant layout would result in major space savings. The WHIMS area was initially planned to give robustness to the process. Since that time, multiple testwork campaigns have been performed and the need for WHIMS has never been confirmed.

17.2.6 Thickening, Filtering and Drying

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 transfers 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 14.5 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.

17.2.7 Apatite Concentrate Silo Load out

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 39 covered hopper fereigh cars (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. A detailed description of the load out system is available in Item 18.7.3.2.

17.2.8 Flotation Tails Thickening and Tailings Handling

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

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

17.2.9 Tailings Management

Plant tailings, with the exception of the magnetic product from the LIMS, will feed the tailings thickener. The tailings thickener overflow is directed to the process water reservoir. The tailings thickener feed is 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 the 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 water is pumped from a barge located on the flotation tailings pond. The barge house contains two (2) reclaim water pumps, each with capacities of 270 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. A portion of the storage pond water will be treated and used as gland seal water and reagent preparation water. 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 and gland seals for the slurry pumps. Seal water will be stored in a 5,790 mm diameter x 6,100 mm height tank for distribution to the slurry pumps. The design pressure for the seal water system is 120 psig.

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)	30,912
Mine Life (years)	28
Apatite concentrate grade	39% P_2O_5
Apatite Concentrate (m^3/t)	0.312 m^3/t
% Water in Ore	5%
% Solids in Ore	95%
Non Magnetic Tailings dry density ($tonnes/m^3$)	3.19
Magnetic Tailings dry density ($tonnes/m^3$)	4.58
Water to non-magnetic tailings pond ($m^3/hour$)	666
Water to magnetic tailings pond ($m^3/hour$)	189
Overall mill process water requirement (m^3/h)	4,142
Water recirculate within the mill (m^3/h)	3,614
Process mill water from tailings ponds (m^3/hr)1	≈529 m^3/h (See note 1)
Non-magnetic tailings (% by weight of total tailings)	77.7%
Non-magnetic tailings production rate	973t/hr
Non-magnetic tailings solids content (% by weight)	59.5%
Non-magnetic tailings water content	666 m^3/hr
Non-magnetic tailings pulp content	981 m^3/hr
Non-magnetic tailings specific gravity ($tonne/m^3$)	3.09
Magnetic tailings (% by weight of total tailings)	22.30%
Magnetic tailings production rate	280 t/hr
Magnetic tailings solids content (% by weight)	62.0%
Magnetic tailings water content	189 m^3/hr
Magnetic tailings pulp content	233 m^3/hr
Magnetic tailings specific gravity (tonne/m3)	4.58 t/m^3

Notes:

1. It is assumed that all of process water will be supplied by both tailings ponds
2. The gland seal water and reagent preparation water will be supplied from a water treatment plant treating a portion of the storage pond water
3. Domestic water will be connected on the main distribution line of the city of Sept-Îles. When used the domestic water will report to the septic tank and leaching bed

17.2.10 Reagent Preparation

17.2.10.1 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 equipped with an agitator and 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.

17.2.10.2 **Soybean Fatty Acid (LIACID 1800)**

This reagent is an apatite flotation collector. 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 indoors. The tank is equipped with a 200 kW solution heater as well as an outdoor vent to release vapour to the atmosphere.

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.

17.2.10.3 **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.

17.2.10.4 **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.

17.2.10.5 Hydrated Lime

Hydrated lime helps the flocculation process for the flotation tailings in the tailings thickener. The lime is received pneumatically by trucks and is discharged into a 60 tonne vertical silo. As needed, lime is reclaimed from the silo by conveyor to a 5,110 mm dia. x 5,490 mm mixing tank equipped 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.

17.3 Mass and Water Balances

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 4.57% 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 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 mill / Ball mill grinding circuit varies by ± 15% from average. Therefore, these variations have been taken into account by the sizing of the equipment downstream of the grinding circuit. 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 G – Mass Balance. The mass balance based partly on pilot plant results held at COREM and on COREM's reports from projects T1405 and T1518. Table 17-4 and Table 17-5 are taken from the mass balance, and the results shown 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, and/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 and Losses

Stream	Weight percentage (%) (from fresh feed)	Weight percentage (%) (from flotation feed)
Feed	100.00	
LIMS Magnetic product	20.00	
2nd Stage dewatering cyclone overflow	2.24	
Flotation feed	78.88	100.00
Rougher stage 1 & 2 concentrate to cleaner	13.44	17.04
Cleaner concentrate (flotation concentrate)	10.50	13.31
Cleaner/scavenger concentrate to cleaner	0.18	0.23
Flotation tailings	68.39	86.70

The magnetite or titano-magnetite content in the ore can vary considerably and can reach up to 30 or 35%. In 2012, it was demonstrated that there is a fair correspondence between the P_2O_5 content in the ore and the magnetic mass pull giving approximately 20.0% weight recovery for feed grade varying between 4.5 to 5.5 % P_2O_5.

For flotation, the weight recovery by stream is based on the result of the second pilot plant campaign held at COREM. The chemical assays have been balanced and the weight and recovery for each stream calculated.

Based on adjusted pilot plant results, the P_2O_5 grade increases from 4.57% (in the feed) to 5.57% at the LIMS separation. There is no significant grade change at the dewatering stage. Table 17-5 shows an example of the P_2O_5 grade variations during pilot plant testing for each flotation stage.

Table 17-5: P_2O_5 Grade by Flotation Stream

Stream	P_2O_5 grade (%)
Feed	4.57
Flotation feed	5.57
Rougher stages 1 & 2 concentrate to cleaner	31.22
Rougher stage 2 tails	0.30
Cleaner concentrate (flotation concentrate)	39.01
Cleaner tails	3.98
Cleaner/scavenger concentrate to cleaner	12.85
Cleaner/scavenger tails	3.44
Flotation tailings	0.43

The process plant water balance is presented in APPENDIX H – Water Balance, in cubic meter per day. This water balance covers only the processing plant, and is calculated with an average availability of 92%, or 22.08 hours per day. The global site water balance, which includes the tailings pond precipitation and evaporation, mine dewatering, site drainage, and others, is included in the Item 20 (Environmental studies, permitting and social or community impacts)

17.4 Plant Design Layout

17.4.1 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. The equipment list is provided in APPENDIX I – Equipment List.

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

17.4.2 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 a 54-75 crusher requested by the client based on experience with similar operation.

17.4.3 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 such 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.

17.4.4 Grinding Area

The SAG mill circuit will grind the material from F_{80} of 170 mm to a P_{80} of 2 mm. The installed power for the SAG mill processing 1,400 tph is 12 MW (16,000 HP). The power requirements are calculated using experience with similar equipment, simulation work, and the work index of the ore. The SAG mill 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. For added grinding flexibility, the SAG Mill will be variable speed. The secondary grinding mills require 12 MW (16,000 HP) to produce a 125 micron product. The power requirements are calculated using experience with similar equipment, simulation work, and the ball mill work index of the ore. The secondary grinding will be done by two parallel ball mills, each with 6 MW (8,000 HP) motor. 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 performed by two (2) double deck vibrating screens of size 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; each containing our (4) 800 mm (31.5”) cyclones.

17.4.5 Low Intensity Magnetic Separation Area

The low intensity magnetic separation circuit has a designed throughput of 1,400 tph at 30% solids w/w. Four double drum magnetic separators have been selected and sized based on this throughput (one per flotation line). Each magnetic separator is independent from the others except for the communal magnetic and non-magnetic product pump boxes, 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 secondary stage. The primary cyclones have a diameter of 250 mm while the secondary stage cyclones 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 metres.

17.4.6 Flotation Area

The number of the conditioning tanks and their sizes are selected in order to have one tank per reagent addition. The retention time is based on the pilot plant testwork and on the advice of KEMWorks as per Table 17-6.

Using the calculated flow rate from the mass balance, the effective volume for each conditioning tank was calculated.

Table 17-6: 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

The flotation column sizes have been calculated based on 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/m².

Since 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 unacceptably 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 are very high. Residence time will not be an issue, and limiting factor will be carrying capacity. To ensure a consistent high quality product, a slightly lower carrying capacity range has been chosen and the columns are expected to operate between 2.5 - 3 tph/m². 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.

17.4.7 Thickening Area

The concentrate and tailings thickeners sizing was performed 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 and tailings thickeners diameters were estimated at 20 meters and 40 meters, respectively.

17.4.8 Filtering Area

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 m² horizontal vacuum belt filter. The belt filter has been sized according to the testwork done by Delkor with a filtration rate of 6 tph/m².

17.4.9 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 total 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.

17.4.10 Reagent Preparation Area

Reagents tank sizing was based on the consumptions obtained during the COREM 2011 pilot plant testwork and shown in Table 17-7 gives an indication of the probable reagent consumption. Recent LCT's performed at COREM have shown that reagent consumption will vary and will have to be closely monitored with the mineralogy of the ore.

Table 17-7: Reagents Addition and Consumption

Reagent	Addition Point	Consumption (g/t feed)
Depressant – Wheat starch (WW82)	Starch addition conditioner tank	300
	Cleaner/scavenger conditioner tank	25
Collector – Soy bean oil	Collector addition conditioner tank	165
pH regulator – NaOH	pH regulator conditioner tank	710
Flocculant – Flomin 905MC	Apatite concentrate thickener	15
	Tailings thickener	15
Settling agent - Lime	Tailings thickener	30.8

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

17.4.11 Silo and Load Out

Silo and load-out facilities at the mine site were sized 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, consisting of adjustable loading spouts.

17.4.12 Building Layout (Dimensions)

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 high.

The crushed ore stockpile is contained within a storage dome, with a 70-meter diameter footprint and a height of 27 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.

17.5 Instrumentation and P&ID

Preliminary P&ID's have been developed based on the updated flowsheets and are presented in APPENDIX J – P&ID's, drawings 15685-03-DR-IC-001 to 007. The P&ID's were necessary to establish the instrument list, 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.

17.5.1 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.

17.5.2 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.

17.5.3 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.

17.5.4 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 required 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.

17.5.5 Tailings Thickening, Disposal and Water Supply

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 storage pond discharged to the environment and other tailings that are routed in different ponds.

Process water is composed of the recovered thickener overflow, supernatant water from the tailings ponds and fresh water make-up. 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.

17.5.6 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.

18. Project Infrastructure

Unlike previous studies, where the plant and related infrastructure were located on the south side of the Hydro Quebec power line corridor, most of the site infrastructure has been moved to the north side of this corridor to minimize the noise and visual impact, to the neighbouring community. Similarly, the main access road which will connect the mine site to Highway 138 will follow the north part of the Hydro Quebec Power line for approximately four kilometres.

Mine trucks will access the crusher and the ore stockpile through a haul road which will run underneath the power line corridor and will be able to access the plant site through an overpass running over the railroad tracks.

A block diagram showing all the related infrastructure and connections is presented in APPENDIX K – Block Diagram, drawing 15685-01-DR-GE-007.

18.1 Major Site Infrastructure

18.1.1 Bulk Earthworks, Landscaping, Fencing

Following the completion of the geotechnical work and topographic analysis, the site pad was placed at a location showing sound properties for the foundations of the concentrator and the administration buildings (Figure 18-2). 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 material will be excavated, of which 90,750 m^3 of rock 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-metre wide gates and one pedestrian entrance turnstile.

18.1.2 Plant Building

The concentrator plant is divided in two separate buildings, the concentrator and the flash dryer building APPENDIX L – Concentrator, drawings 15685-03-DR-GE-002 to 004. 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. It was developed later on during the Original Feasibility Study and is at a less advanced level than the concentrator building.

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;
- 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;
- 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 M – Mine Shops and Warehouse, drawings 15685-04-DR-GA-001 & 002) will contain offices on the second floor, while the laboratory, infirmary, 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. The building's dimensions and general arrangement are based on accommodating 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 by 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 required 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 by 16 m covered cold storage area and a 20 m by 23 m mechanical inventory pad adjacent to the administration building have also been accounted for in the estimate. The gatehouse dimensions and general arrangement are based on a recent similar mining project in the Sept-Îles area, whose price was updated with inflation. The gatehouse building is rectangular and measures 7.3 m by 19.5 m. Once again it has been considered that the soil bearing capacity is sufficient to allow for standard foundations (no pilings). A plan view of the gatehouse is shown in Figure 18-1. The gatehouse serves several purposes, including a class room to welcome visitors to the site. There, visitors can be shown a presentation concerning the safety guidelines that are applicable to all personnel on site. Offices and a rest area are also included in the structure, as well as a cafeteria, showers, locker room, and restrooms. This ensures the gatehouse personnel do not need to leave their work area for their daily activities. There is also allotted space outside for emergency vehicles and storage spaces for safety equipment.

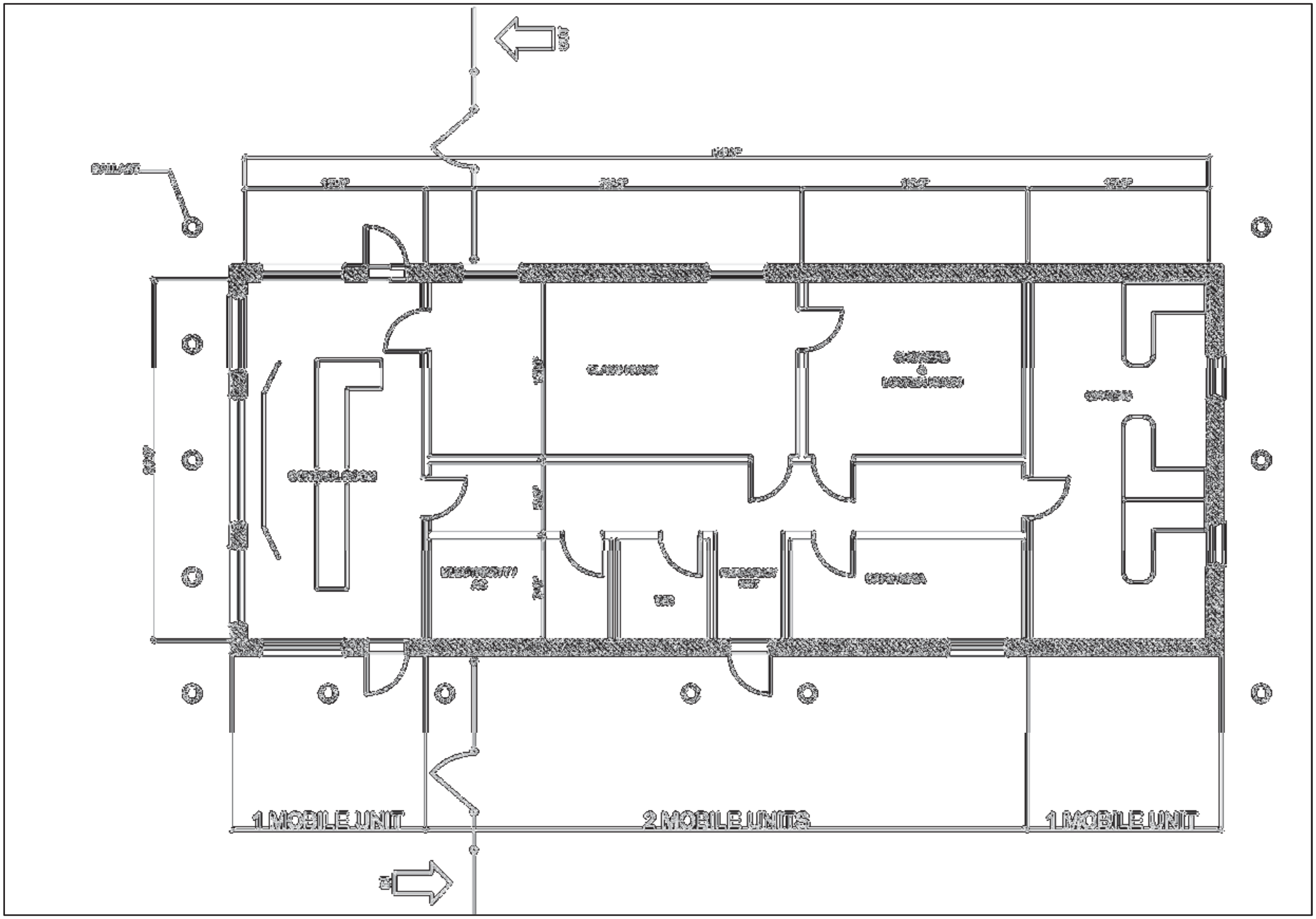


Figure 18-1: Gatehouse Layout

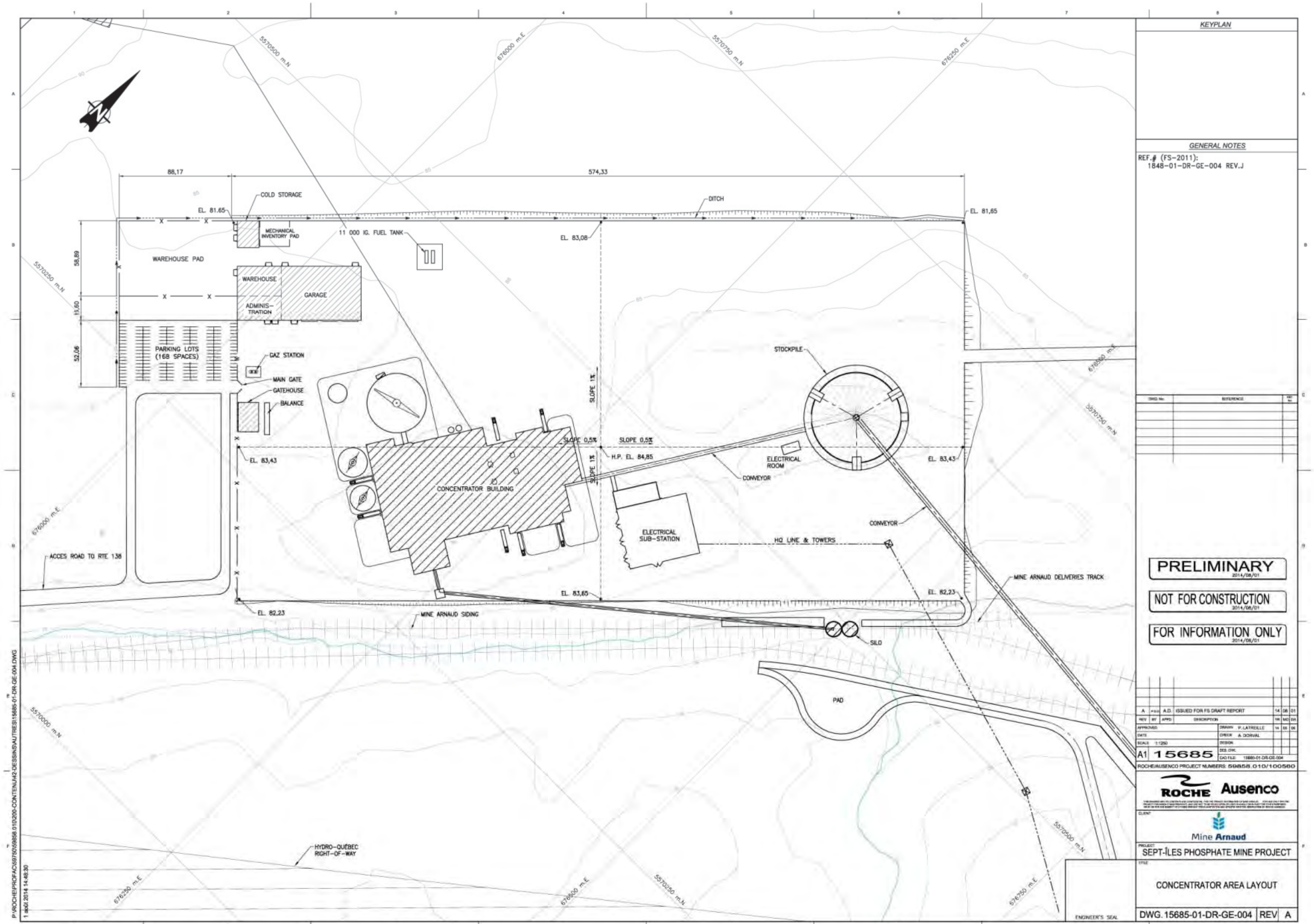


Figure 18-2: Concentrator Area Layout

18.1.4 Access Roads

Access road to the property will be through a new two lane 4.2 km length road from provincial highway 138 with proper signage. It will be used by personnel during construction as well as for transportation and operations. The road will be built with overburden and waste rock materials from mine pre-production. Crushed aggregate will be used for the upper grade topped by sand and a 50 mm asphalt surface. The width of the asphalt will be 10 m with 1 m sidings on each side. A total of ten (10) culvert type water crossings will be required. Culverts will be used at all water crossings (10 culverts, 900 mm dia.) as well as side ditches wherever required.

At the intersection with provincial highway 138, new entry, exit, turning and bypassing lanes will be constructed as required by the Ministry of Transports of Quebec.

A typical section of the proposed access road is shown on Figure 18-3.

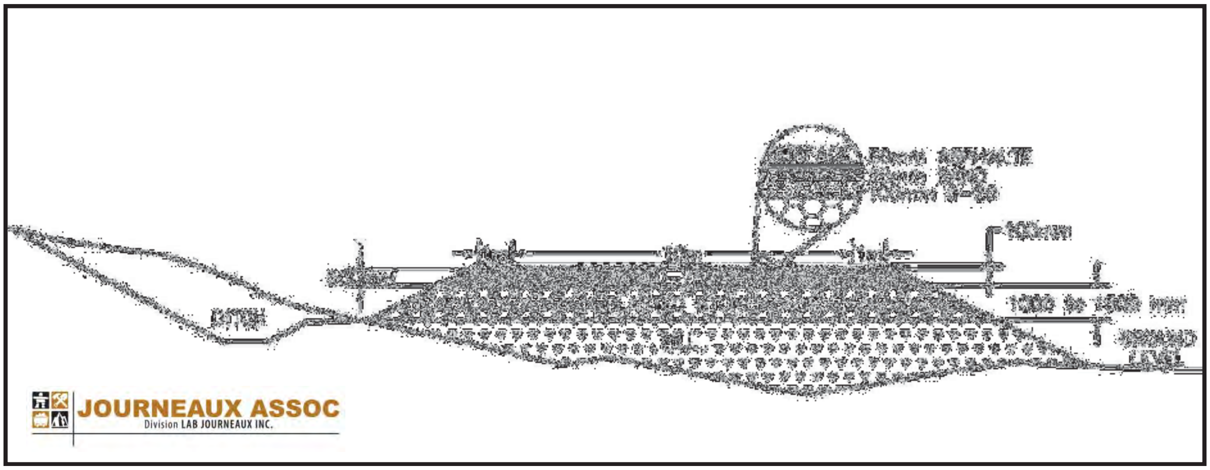


Figure 18-3: Typical section of the access road to property

Table 18-1 lists the materials required for construction of the access road base and pavement. Other accessories, not detailed here, are lighting, guard-rails, etc.

Table 18-1: Access road construction materials

ACCESS ROAD TO PROPERTY CONSTRUCTION MATERIALS
ROAD STRUCTURE
General fill
Unsorted waste rock fill (blast stone)
Screened waste rock fill 0-75 mm
Screened waste rock fill 0-25 mm
Crushed stone 0-112 mm (MG112)
Asphalt
CULVERTS
Galvanized corrugated culvert steel pipe, 900 mm dia., 2.8 mm thickness
Crushed stone 0-20 mm (MG20)

18.1.5 Overpass & Parking Lot

An overpass will allow the haul trucks to pass over the railway to reach the waste rock dump and concentrator area. The overpass has two train tracks that allow for a main track and a Mine Arnaud siding underneath it. The design for this overpass respects the "Standard Respecting Railway Clearance (TC E-05)". A section view of the overpass is shown in Figure 18-4.

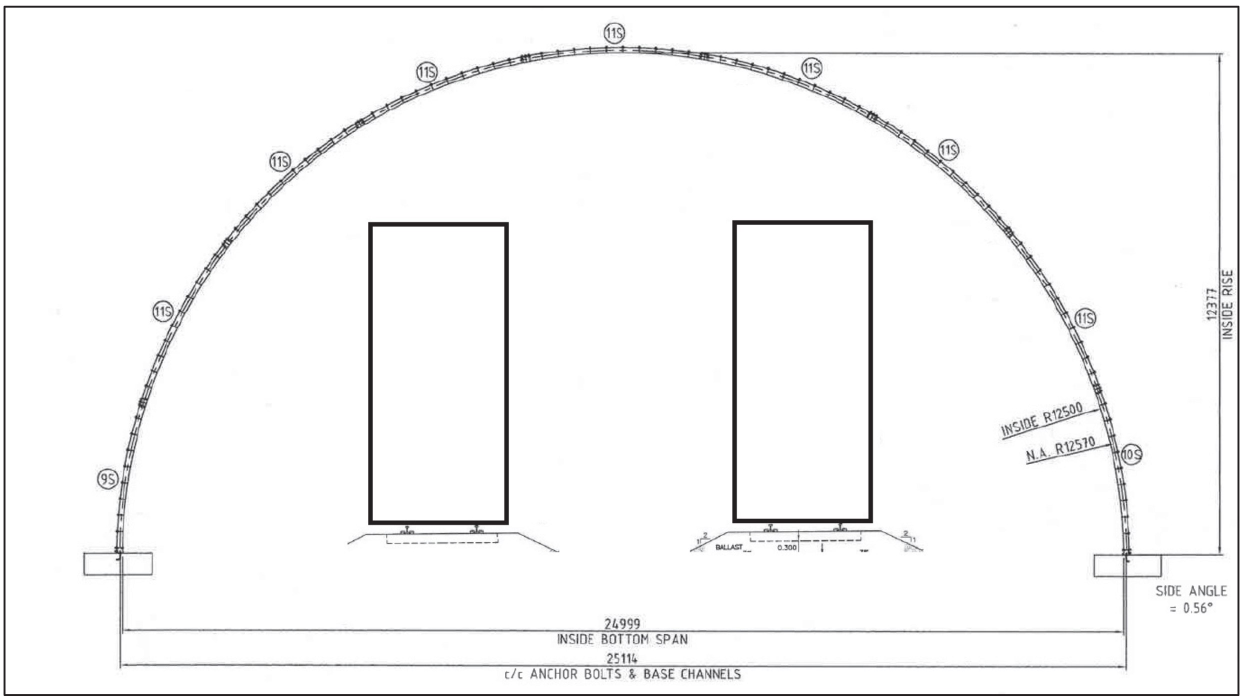


Figure 18-4: Overpass Section View

A parking lot connected to the main administration building is located east of the concentrator and provisions were made for lighting poles. Figure 18-5 shows the location of the parking lot.

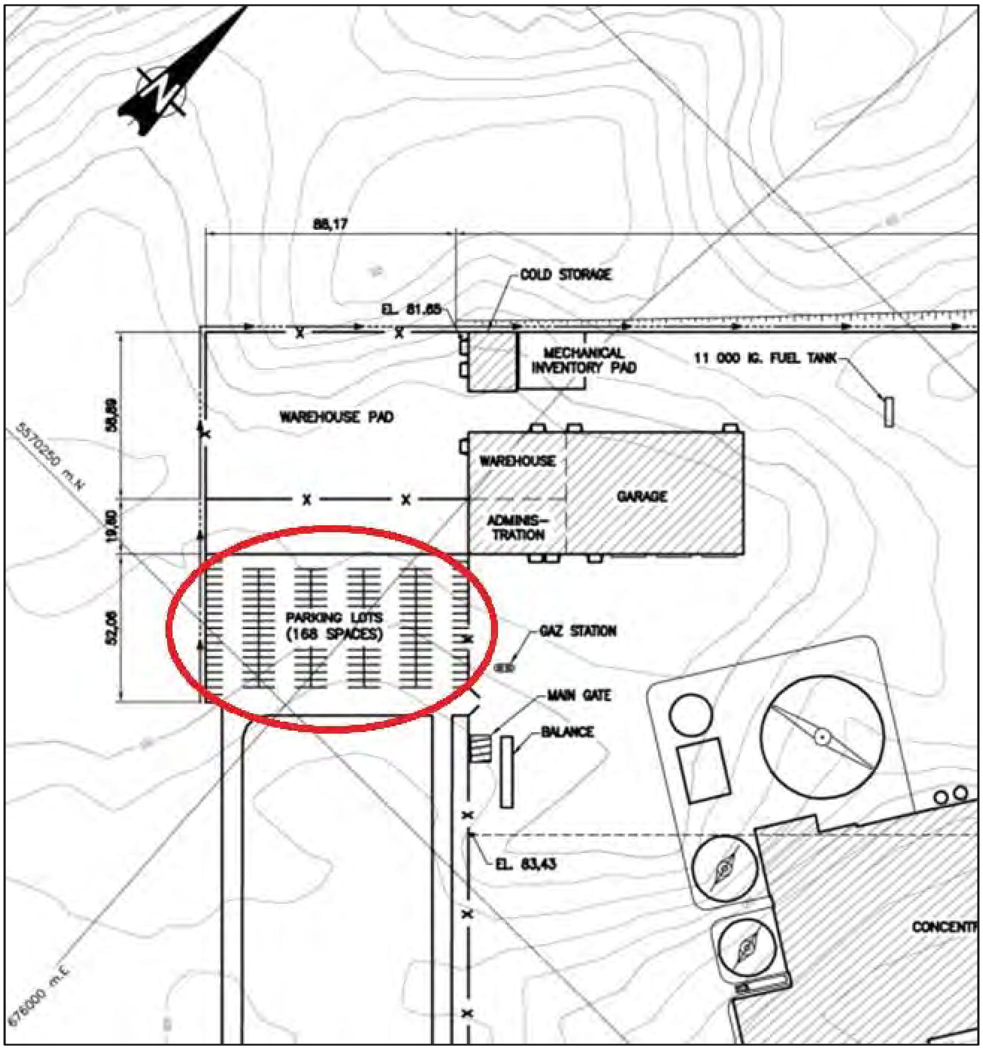


Figure 18-5: Parking Lot Location

18.1.6 On-site roads

Other roads will be required to be built on-site, such as:

- A haul road connecting the pit and crusher area to the concentrator and waste rock dump area;
- A road to access the explosives storage area from the concentrator;

The haul roads will be composed of run-of-mine material overlay by a layer of aggregate and will be built using the mining equipments such as tractor, grader, trucks, etc. The width will be at least 20 meters, which represents three times the width of the widest equipment, i.e., the CAT 785 mining truck.

The road to access the explosives storage area will be built to accommodate pick-up trucks, delivery trucks as well as road maintenance equipment (grader, etc). The maximum width will be 10 meters.

18.1.7 Fuel Storage

The fuel storage consists of a diesel/gasoline 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.

18.1.8 Process Water

The process water is pumped from a barge located on the flotation tailings pond. The barge house contains two (2) reclaim water pumps, each with 270 m^3/h capacity and one (1) de-icing pump. For more details related to the process water please refer to Item 17.2.8 Tailings Management .

18.1.9 Potable Water

A 100 mm PVC DR-18 pipe is used for potable water requirements. This pipe will connect to the existing water line that services the Sept-Îles municipality. The length of the pipe is approximately 3,820 m with a water head loss of 280 kPa. The maximum output flow is assumed to be around 10 m^3/h with a maximum service pressure of 1 MPa. Figure 18-6 shows the proposed trajectory of the water pipe. Starting with an intake at Highway 138 and will pass underground beneath the Hydro-Quebec Power Line for 1,220 m. To compensate for the loss of pressure, a booster pump-house will be constructed at the end of the 1,470 m first segment. The pump house increases the water pressure to allow for the water to be pumped up to the plant building. Figure 18-7 shows a proposed set-up for the boosting pump-house.

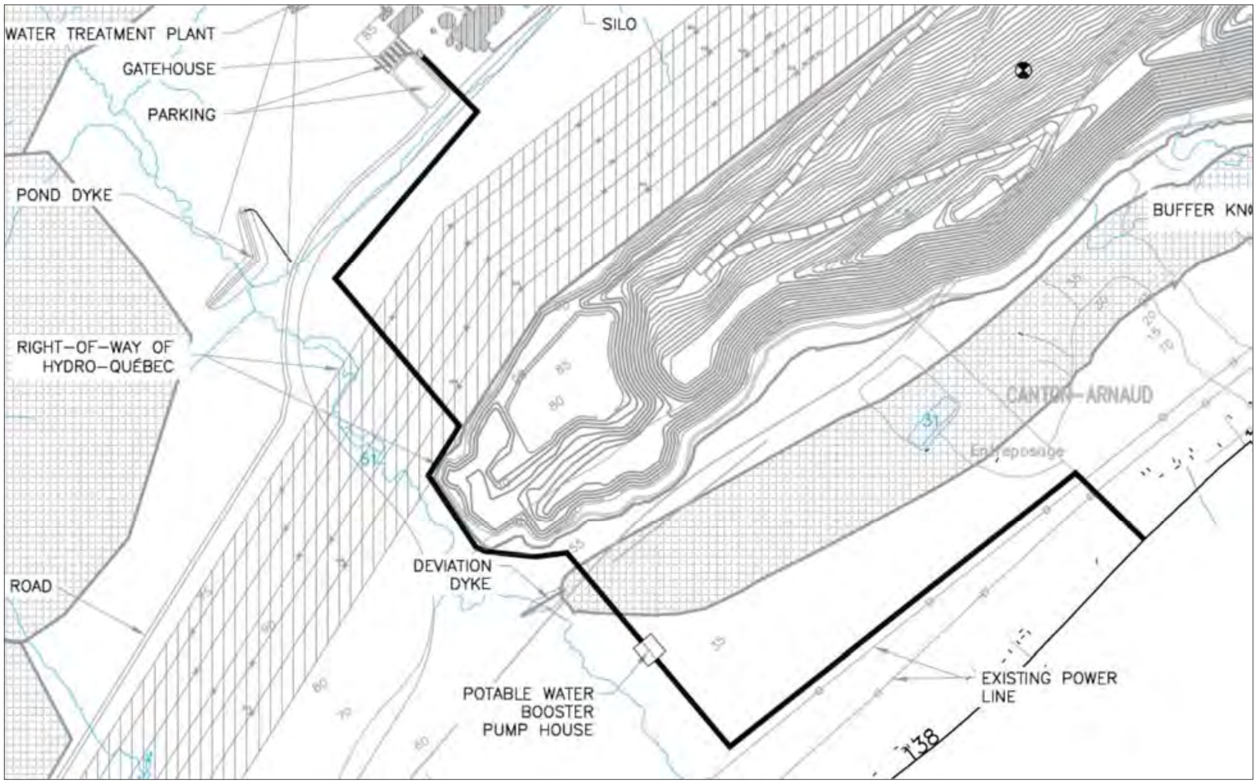


Figure 18-6: Proposed Water Pipe Layout

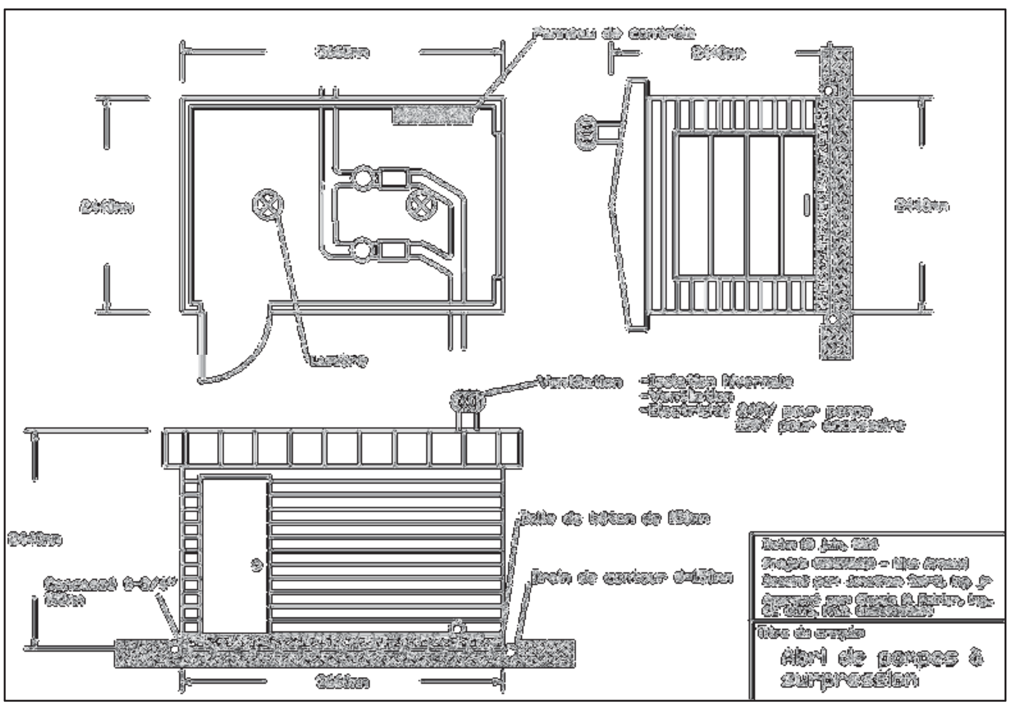


Figure 18-7: Proposed Boosting Pump-House Layout

18.1.10 Safety and Fire Protection

The safety fire protection equipment includes:

- A 4x4 fire truck, with a mini pump and foam system;
- A 4x4 ambulance;
- A first-aid room as well as provisions for safety showers, eye wash stations, and fire extinguishers;
- A 2.45 m (8 ft) high fence to secure the open pit area at the South, East, and West ends.
- An allocation for eight (8) Safety showers and eye wash stations for the garage, assay lab, wet lab, mechanical shop, and process plant was considered;
- An allocation for a few fire extinguishers was also added to the safety supplies.

18.1.10.1 General

Fire protection infrastructure and systems will comply with NFPA-13-2007 standards, Québec's National Building Code, 2005.
Standpipes systems will comply with NFPA-14.
Water loop system will comply with pertaining AWWA standards, NFPA-24 and BNQ 1809-300/2004.
Fire pumps and generator sets will comply with NFPA-20.
The complete system will be supervised by fire alarm system.
It is assumed that plant operation will provide a local firefighting team and equipment.

18.1.10.2 Water Supply

Fire protection water will be supplied by a tank with a minimum reserve capacity of 1,250,000 L [330,000 USG], to be able to supply the highest hazard risk conditions for 2 hours. This tank will be kept filled by the fresh water line. The fire protection water will distributed by a 10'' (254 mm) diameter water loop plus 6'' (152.4 mm) and 8'' (203.2 mm) diameter branches to different buildings. This loop will be pumped to a pressure of 160 psi to feed all sprinkler systems and stand pipes without additional pump.

The main fire protection loop will also include 14 hydrants strategically located for firefighters use. The estimate includes installation costs including piping, hydrants, excavation, and backfilling.

18.1.10.3 Sprinkler Systems

Sprinkler 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 conveyors: ordinary risks, group 2, 0.50 USGPM (range from 0.2 to 0.8 depending on detail design to be confirmed) density and 1,500 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, 0.45 USGPM density and 5,000 sq ft area, plus 500 USGPM for standpipes, area per sprinkler head 100 sq ft. The maximum flow will be 2,750 USGPM for 2 hours. The density and area required for the garage is set according to FM Global Standards considering most stringent case.

18.1.10.4 **Standpipe Systems**

Standpipe systems will be provided for tall buildings more than 46 ft in height (concentrator and crusher) and will supply 2 ½" fire hose with 500 USGPM at 100 PSI. Standpipes will also be installed at stockpile and garage.

18.1.10.5 **Equipment Included**

- Sprinklers (pendant, recessed or upright depending on use);
- Siamese connectors for fire department connections on every buildings;
- Alarm valves;
- Fire alarm system;
- Fire extinguishers;
- Seismic restraints as required;
- Accessories;
- Tests and start-up of all fire protection related equipment and systems;
- Design and drawings as required by NFPA.

18.1.10.6 **Fire Protection Systems Description**

- Site fire protection loop supply:
    - Fire loop and distribution piping;
    - 14 hydrants;
    - One electrical fire pump (400 hp) on electrical emergency supply and one diesel 400 hp fire pump in a heated room and properly ventilated.
- Crusher:
    - Dry sprinkler system with 6’’ water supply in a heated room;
    - Dry standpipe system with single interlock.
- Crusher conveyer to stockpile:
    - Pre-action system with 6’’ water supply in a heated room;
    - Detection system (protecto-wire).
- Garage:
    - Wet sprinkler system with 10’’ water supply;
    - Standpipe system.
- Stockpile:
    - Dry sprinkler system with 6’’ water supply;

    - Standpipe system.
- Conveyor (from stockpile to concentrator building):
    - Pre-action sprinkler system and 6'' water supply;
    - Detection system (protecto-wire).
- Electrical room:
    - Pre-action system with double interlock.
- Concentrator:
    - Wet and dry sprinkler systems depending on area served with 6'' water supply;
    - Standpipe system.
- Conveyor (from concentrator to silo):
    - Pre-action sprinkler system with 6'' water supply.
- Silo:
    - No protection provided.
- Transformers (3 pieces):
    - Vortex sprinkler system.

18.1.11 Domestic Waste Water Treatment

Considering 100 workers per shift, a daily flow rate of 60 m³/day has been calculated using 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 effluent of this water treatment could be released into either the ground or Ruisseau Clet. The system includes:

- One 100 m³ septic tank with filter;
- One 225 m³ equalization basin;
- Six Rotofix units;
- One 15 m² settling tank;
- One 35 m³ sludge storage basin;
- One UV disinfection unit.

18.2 Dykes

18.2.1 Deviation dyke Construction; Pre-Operation

The purpose of a deviation dyke is to deviate the water and tailings away from the Clet creek and the downstream fields in the rare event of breach of the tailings park dyke upstream of the Clet creek and the polishing pond. Water and tailings will deviate to the ditch between the Open pit and the screen berm, north-east of the Clet creek.

The 6 m width crest of the deviation dyke will be at elevation 55 m, with the side slopes constructed at 2H:1V on both the upstream and downstream sides. It will be constructed with waste rock graded from finer in its core to coarser on the slopes, the coarser rock as erosion protection.

Four (4) small culverts, screened at their entrance, installed along the Clet creek will permit flow of the creek while in the event of a tailings dyke breach will allow some water through but will be blocked by the sudden high flow of water and tailings resulting in re-direction of water and tailings to the ditch between the Open pit and the screen berm.

The construction material types are presented in Table 18-2. A plan and a typical section of the dyke are shown in Figure 18-8 and Figure 18-9.

Table 18-2: Materials for construction of the deviation dyke

DEVIATION DYKE - CONSTRUCTION MATERIALS
Coarse Rockfill (waste rock >600 mm)
Rockfill (waste rock 300-600 mm)
Fine Rockfill (waste rock 0-300 mm)
Crushed stone 0-20 mm
Galvanized corrugated culvert steel pipe, 900 mm dia., 2.8 mm thickness

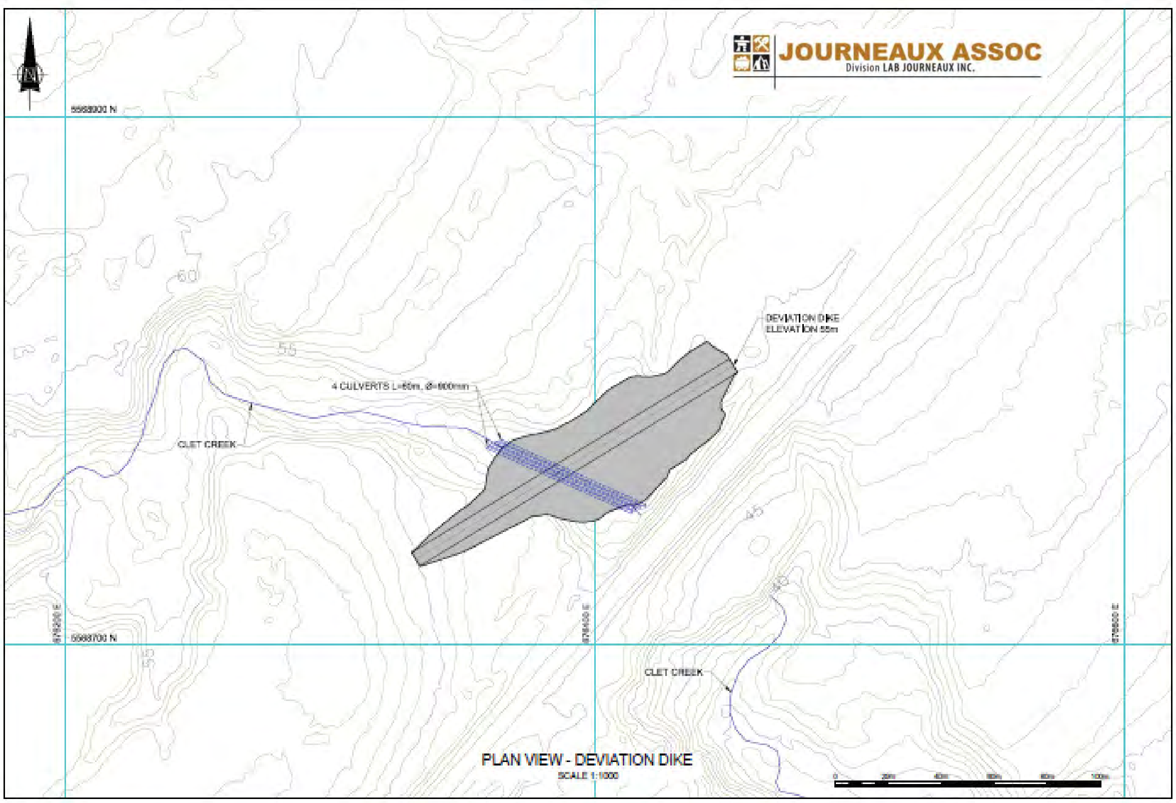


Figure 18-8: Deviation dyke layout plan

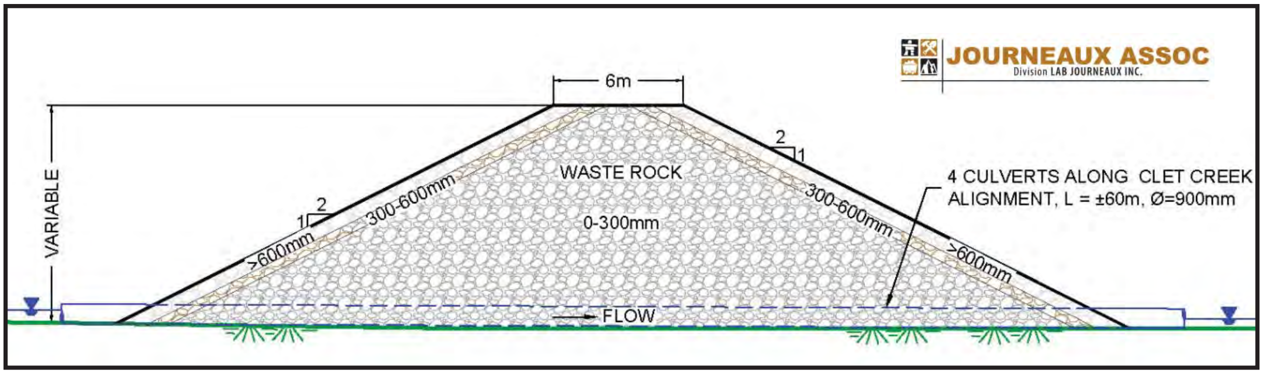


Figure 18-9: Deviation dyke typical section

18.2.2 Storage Pond Dyke Construction; Pre-Operation

The storage pond will be located on the south-east side of the tailings park. It will be constructed in the pre-operation period such as to collect water for starting processing.

The dyke of the storage pond will be constructed to elevation 75 m with side slopes of 2H:1V and 3H:1V on the upstream and downstream sides respectively and a crest width of 15 meters.

Selected waste rock materials (for erosion protection and global stability) and an impermeable till or clay core, are necessary to retain the precipitation and the free water transferred from the tailings park. This water will still contain very fine particles to settle in the storage pond.

As part of the dyke is constructed, on the clayey bed of the Clet creek, wick drains will be installed at its foundation footprint to accelerate dewatering (consolidation) of the clay strata and thus increase its strength and bearing capacity. A spillway will be constructed on the north end of the storage pond dyke and will overflow to a channel lined with geotextile and rip-rap that will direct overflow water (in a case of an extreme event) towards the Clet creek downstream.

The construction material types are presented in Table 18-3. The layout and a typical section of the dyke are shown in Figure 18-10 and Figure 18-11.

Table 18-3: Storage pond dyke contruction material types

STORAGE POND DYKE CONSTRUCTION MATERIALS
Coarse Rockfill (waste rock >600 mm)
Rockfill (waste rock 300-600 mm)
Fine Rockfill (waste rock 0-300 mm)
Sand or sand/gravel
Till or clay impermeable core
Geotextile

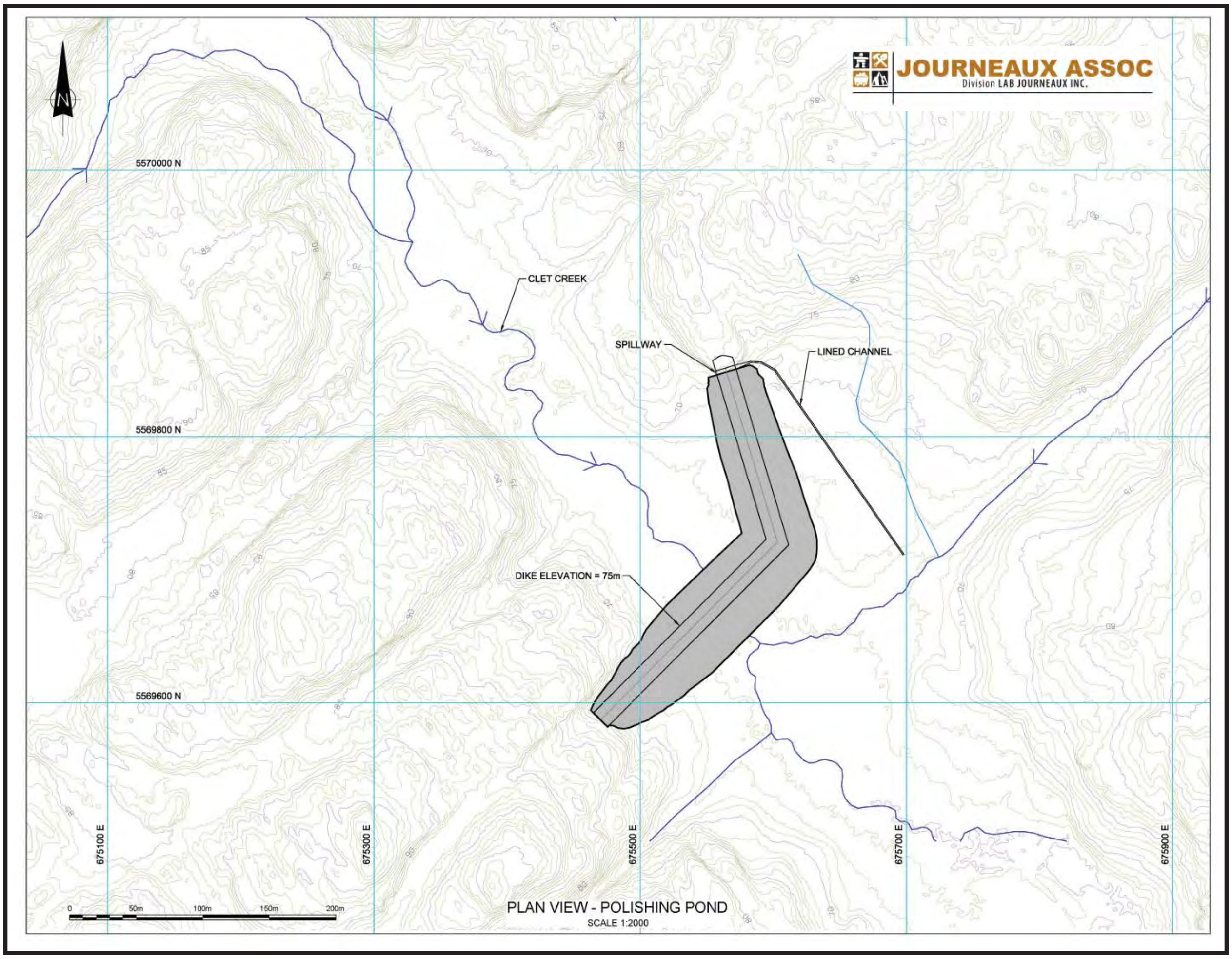


Figure 18-10: Storage pond dyke layout plan

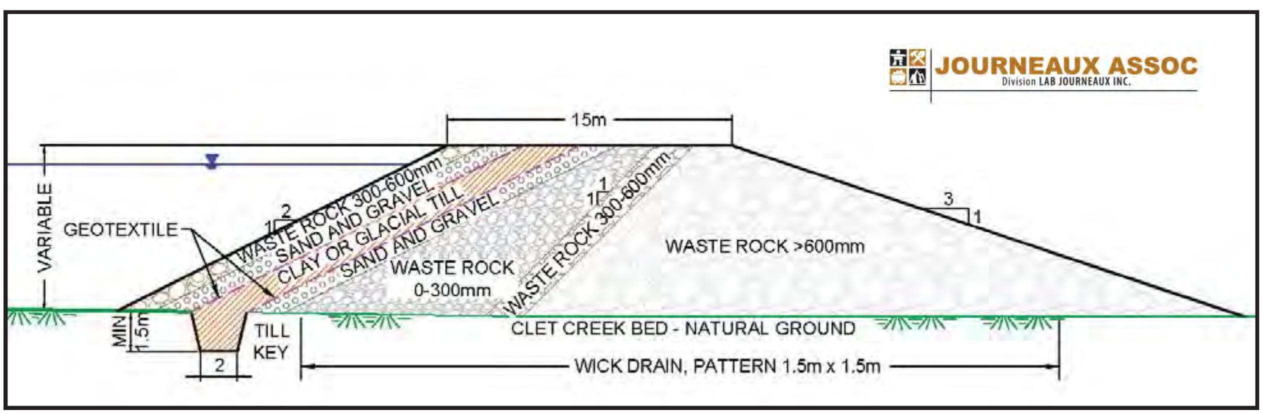


Figure 18-11: Storage pond typical impermeable dyke section

18.2.3 Tailings storage dykes; Pre-Operation and Operation

The tailing storage facility (TSF) is located approximately one kilometre west of the process plant. It uses a natural water body frequented by fish for mine waste disposal. Under the Fisheries Act, an assessment of the alternatives measures for mine waste disposal in order to comply with the Guidelines for Assessment of Alternatives for Mine Waste Disposal (hereafter, Guidelines). In December 2012, Mine Arnaud submitted a first version of its assessment, and then a revised version in September, 2013, which included comments from the Canadian Environmental Assessment Agency and recent changes made to the Guidelines. In these reports, the projected TSF consisted of five floatation tailings cells, and two magnetic tailings cells. Comments on the revised report were received in January, 2014 and many exchanges with CEAA took place during spring 2014.These discussions led to the optimization of the TSF in order to limit the use of natural water bodies frequented by fish for mine waste disposal. Consequently, the West Cell of TSF was removed and the other cells were modified to contain tailings from the removed cell.

The TSF presented in this FS consists of six cells (see Table 18-4). Five of them will contain floatation tailings (#1 to #5) and the last one will store magnetic tailings (North Cell). Cell 1, 2 and north will be constructed first.

The TSF is designed to store approximately 320Mt of tailings; 242Mt of floatation tailings and 78Mt of magnetic tailings, distributed as indicated in the Table 18-4.

Table 18-4: Capacity of TSF

Cell	Capacity *Mt*	Years of operation
1	37,3	1 – 5
2	15,9	6 – 7
3	39,5	8 – 12.5
4	101,9	12.5 – 26.5
5	47,5	26.5 – 31
North	78,4	1 – 31
TOTAL	**320.5**	

Each of the TSF cells will be built in stages while the open pit is being mined. Once the open-pit mining has ceased, low-grade ore will be processed for approximately three years and those tailings will report to cell #5 and north.

The starter dam at TSF Cell #1 and north will provide storage for tailings produced during the first year of mineral processing, approximately 5.7Mt of floatation tailings and 1.8Mt of magnetic tailings. Dam raises will be completed annually, based on the centreline construction method. Some specific area, where thick clayey soils are encountered, will be constructed with the downstream method. TSF Cell #2, 3, 4 and 5 will be built using the same general methods as Cell 1.

18.2.3.1 Pre-operation

As mentioned, the tailings park consists of six cells to be constructed and filled during the mine life. One cell will receive the magnetite tailings while the remaining five cells will receive floatation tailings. The dykes of all cells will be raised sequentially as the cells are filled up. Construction of the starter dykes for the magnetite tailings and cells 1 and 2 for the floatation tailings is required. Unsorted waste rock is used for construction of the starter dykes (except over water streams in one location during pre-operation) where starter dykes are constructed with till or clay (lower permeability materials). Starter dykes will have side slopes of 2H:1V on the upstream and downstream sides and a crest of 10 metres wide.

18.2.3.2 Operation

During operation, starter dykes of existing cells will be raised while for new cells starter dykes will be constructed as described previously.

Generally, dykes will be raised using the center axis (or centerline) construction method and have a 10 m crest and 2H:1V side slopes upstream and downstream. The dyke core will consist of 0-150 mm waste rock on the downstream side and compacted coarse tailings borrowed from the tailings pond beach on the upstream side (tailings will be deposited using spigotting around the ponds). Waste rock of sizes 150-600 mm will be placed on the downstream side of the dykes, the larger size on the exterior of the slope for erosion protection.

Over four water streams, the dykes will be raised using a downstream construction approach, have a 10 m crest and side slopes 2H:1V and 4H:1V upstream and downstream, respectively. The types of construction materials, methods and layering sequence are similar to the center axis construction dykes.

At any stage of construction, the crest of a dyke will be 1.5 meters above operating levels (tailings/water) to account for freeboard and emergency spillway.

Figure 18-13 and Figure 18-14 display typical dyke sections for the proposed tailings park.

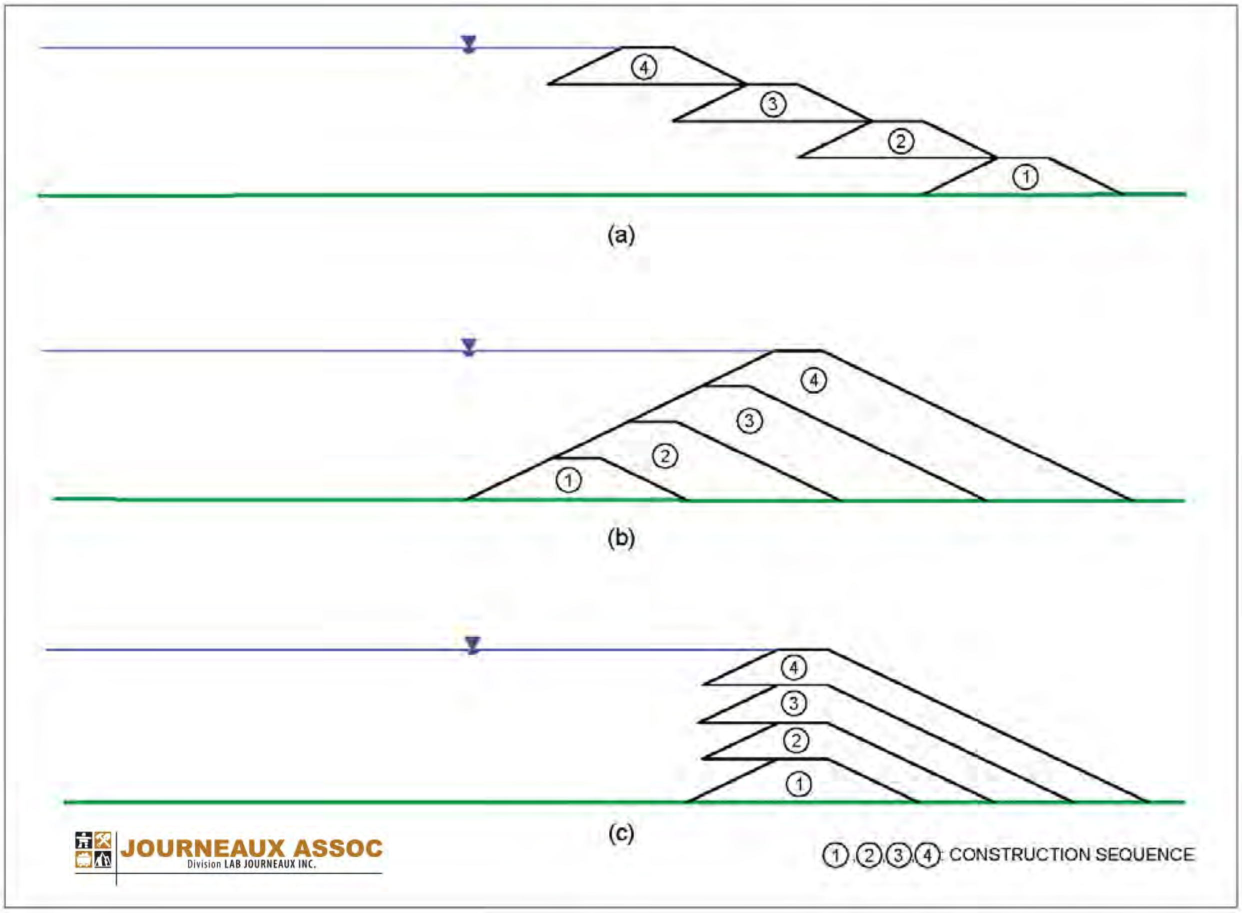


Figure 18-12: Typical construction methods for tailings dykes; (a) upstream construction, (b) center axis (or centerline) construction, (c) downstream construction

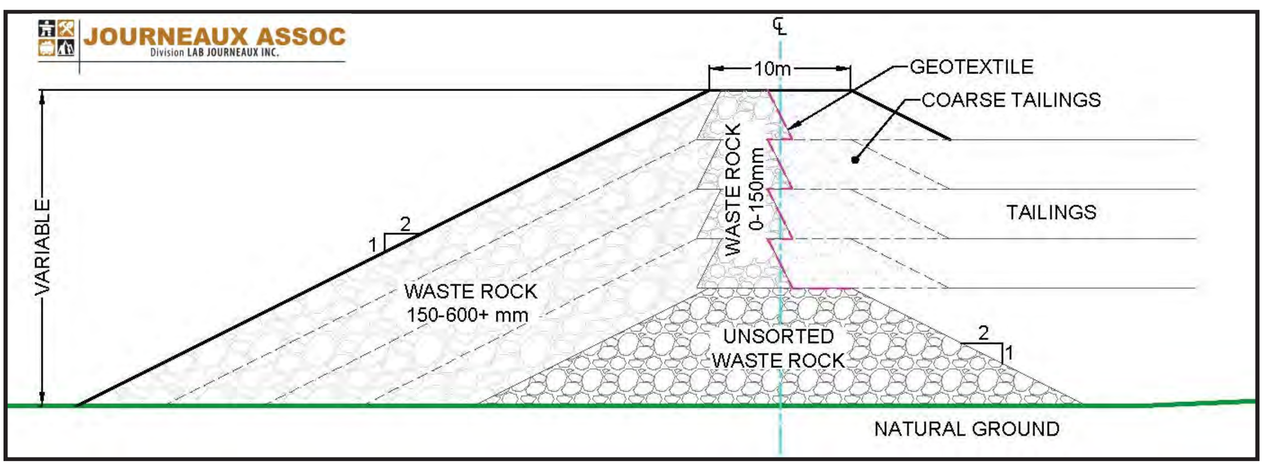


Figure 18-13: Typical section of a tailings dyke (center axis or centerline construction method)

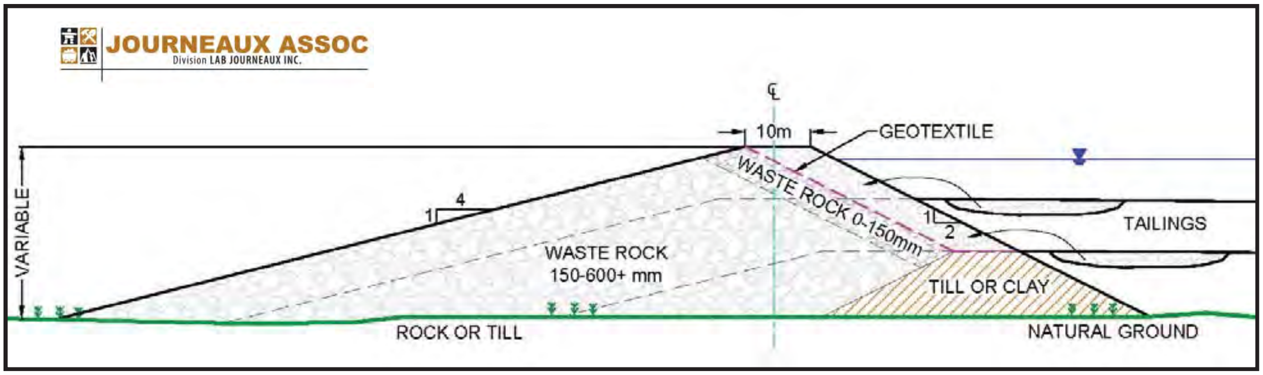


Figure 18-14: Typical section of a tailings dyke constructed over a water stream (downstream construction method)

The sequence and crest elevation of the various dykes/cells are displayed in Table 18-5 and the construction material types in Table 18-6

Table 18-5: Tailings Park estimated construction sequence

TAILINGS PARK CONSTRUCTION STAGES											
	CELL NO.	PRE-OPERATION	OPERATION								
		EL, m	EL, m	EL, m	EL, m	EL, m	EL, m	EL, m	EL, m	EL, m
CAPACITY TO YR		*1*	*5*	*7*	*8*	*12.5*	*12.5*	*26.5*		*30*
CONSTRUCTION YEAR		-2 AND -1	1,2,3,4	5,6	7	8,9,10,11	11	12,13,14, 15,16,17, 18,19,20, 21,22,23, 24,25	25	26,27,28, 29
NO. YRS CONSTRUCTION			*4*	*2*	*1*	*4*	*1*	*14*	*1*	*4*
F=FLOATATION M=MAGNETITITE	1M	106	112	115	116.5	123		143		149
	1F	115	149							
	2F	87.3		112						
	2F	87.3		112						
	2F-4F	87.3		112				132		
	3F				113	139				
	4F-5F						95	132		
	4F						95	132		
	4F						95	132		
	5F								100	126
	5F								100	126

Table 18-6: Tailings Park construction material types

TAILINGS POND DYKES - CONSTRUCTION MATERIALS
Till or clay
Coarse tailings
Fine Rockfill (waste rock 0-150 mm)
Medium-large size Rockfill (waste rock 150-600+ mm)
Unsorted Rockfill (waste rock)
Geotextile

18.3 Geotechnical Studies

18.3.1 Geotechnical Summary

At an average elevation of 100 to 110 m, the study area is part of a high plateau dissected by deep faults and valleys that control the drainage of many small water ponds in the region.

During glaciation, the softer rocks in the region were eroded and their surfaces, shaped by glaciers, were covered with a thin layer of sand outwash and gravel or till deposited on the surface of the parent rock. During the glacial retreat period, the area was flooded by the sea and saline marine clays were deposited on the ground, in the inland bays and other depressions. In addition, in highlands, the clay layers are relatively thin in the depressions while they are much thicker at elevations lower than 80 m. Numerous water filled depressions appeared as ponds and lakes following the rebound of the land mass. Peat deposits have therefore accumulated in some of these depressions.

Considerable thicknesses of sand or sand and gravel were then deposited on the marine clays following the onset of outwash deltas and sandy beaches accompanying the continuous rise of the land mass and the lowering the groundwater table. Layers of sand, with high iron content, oxidized, and even cemented are clearly visible in the region.

Within the scope of work for the Mine Arnaud project to exploit an open pit apatite (phosphate) mine, two (2) geotechnical investigations were carried out by Journeaux Assoc., a division of LAB JOURNEAUX INC.

The first investigation (Journeaux Assoc. 2011), in which field work was carried from March 17 to April 21, 2011, identified a stratigraphy composed of topsoil (peat) resting on sand followed by a layer of stiff clay becoming soft at depth. The latter covers a till deposit overlying the bedrock. It should be noted that in some soundings, the absence of the sand was noted and the topsoil rested directly on the clay.

Within the scope of work for the first investigation, a combination of seventy-nine (79) boreholes and test pits established the soil stratigraphy at the proposed location of the following structures: plant, ore storage shed, crusher, silos, waste dumps, tailings impoundment area, polishing pond, railway and the bridge over Des Rapides River (to the waste dump area).

The second investigation (Journeaux Assoc. 2014), in which thirty six (36) soundings were put down from April 22 to May 2, 2014 and from May 19 to May 24, 2014, confirmed the stratigraphy identified in 2011 and noted the presence of a silt layer occasionally inserted between the sand and the clay. It should be noted that some boreholes identified the presence of two (2) distinct layers of clay.

The structures associated with this second geotechnical investigation are: access road to property, storage pond dyke, waste dump, diversion dyke, screen berm along Highway 138, water treatment plant, overburden pile, and the tailings impoundment dykes.

18.3.2 Tailings Impoundment Area

In addition to the soundings carried out in 2011, in 2014 eight (8) geotechnical boreholes were put down at the location of the Clet creek in order to characterise the foundation soils for the dyke over this water stream which will be sequentially raised during mine life.

The general soil stratigraphy at the location of the dykes for the various cells (6 cells) of the tailings impoundment area is a sandy/silty till with gravel, cobbles and boulders with the exception of the beds of a few water streams where the soil is siltier. Along the Clet creek, the stratigraphy indicated a thin layer of topsoil resting on the 1 to 4-metre thick stiff to soft clay layer. The bedrock was encountered on the surface and at 4 metres in depth.

18.3.3 Waste Rock Dump

The waste rock dump site, located north of the mining property, was investigated for the first time in 2011 with five (5) test pits and then three (3) boreholes in 2014.

The soundings identified a clay layer a maximum of 2 metres thick underlying a thin layer of topsoil and resting on bedrock.

18.3.4 Overburden Pile

Located directly northeast of the open pit, the site for the overburden pile was investigated during the 2014 geotechnical drilling program.

The geotechnical boreholes noted a thin layer of topsoil overlying sand resting on bedrock. The depth of the latter varied from 0.5 to 1.2 m which is considered a good foundation for the overburden pile.

18.3.5 Access Road to property

Connecting the Arnaud Mine site to Highway 138, the access road extends over 4.2 km. It follows the Hydro-Quebec electrical transmission line and crosses the Clet Creek. The topography of the access road indicates a fairly rugged profile with many small depressions including that of the Clet Creek and two (2) small insignificant water streams.

The soil stratigraphy along the axis of the access road has been identified in nine (9) boreholes put down in 2014 combined with three (3) others put down in 2011. Below the topsoil layer, ranging from 0.6 to 1.8 m thick, a layer of sand and silt (0.4 to 1.2 m thick) was encountered resting on bedrock. However near the Clet Creek, a clay layer of variable thickness (5.7 to 10.5 m) was identified resting on bedrock and partly on till. In fact, the bedrock outcrops along the majority of the road (about 3.2 km).

18.3.6 Screen Berm

Located south of the proposed mine open pit and parallel to Highway 138, the screen berm, as its name suggests, will isolate and reduce noise coming from mining operations. Measuring about 4.0 km, the screen berm will be most probably constructed with waste rock and overburden material borrowed from the work site.

Five (5) geotechnical boreholes were put down in 2014 within the proposed footprint of the screen berm. A layer of topsoil was encountered covering a layer of sand of variable thickness (2.0 to 4.5 m) resting on clay followed by bedrock. The thickness of the clay varies from 2.0 to 9.5 m. In general in this area, the bedrock is located at a depth of 4.0 m (East Sector) to 14.0 m (West Sector and near the Clet Creek).

18.3.7 Polishing Pond

Occupying a section of the Clet Creek valley, the storage pond is delimited by the tailings impoundment area and a water holding dam on the opposite side.

The soil stratigraphy at the location of the storage pond dyke was identified in 2014 by putting down four (4) boreholes. Below a thin layer of topsoil, 12.0 to 19.0 m of relatively soft clay was encountered resting on a thin layer of till above the bedrock. During the 2014 program, the groundwater table was observed to be on the surface.

18.3.8 Diversion Dyke

Located west of the open pit and positioned across the Clet Creek bed, the diversion dyke will be used to divert flow in case of a rupture of a tailings impoundment or storage pond dyke, to the drainage ditch located between the open pit and the screen berm.

Two (2) geotechnical boreholes were drilled in the right-of-way of the dyke. The soil stratigraphy indicated a thin layer of topsoil overlying sand to a depth ranging from 1.5 to 6.0 m. A fairly thick layer of clay (7.0 to 10.0 m) followed to the bedrock.

18.3.9 Water Treatment Plant

Located north of the polishing pond, the water treatment plant is proposed in an area where the bedrock is near the surface.

The three (3) geotechnical boreholes drilled in 2014 identified a layer of topsoil less than a metre thick overlying a 1.0 to 2.0 m thick clay layer resting directly on bedrock.

18.3.10 Railway

A railway, property of Cliffs currently passes through the work site and the proposed open pit location. Relocation of a section of the railway is necessary. Geotechnical boreholes were drilled in 2011 along the proposed axis of the section to be relocated.

The soil stratigraphy was investigated by putting down a combination of 23 boreholes and test pits. Topsoil varying in thickness from 0.3 to 3.0 m is underlain by either a 1.0 to 5.3 m thick layer of sand and silt or by a fairly thick layer of clay (4.5 to over 12.0 m) extending to the bedrock encountered at variable depths.

18.3.11 Mining Complex - Plant

The sector associated with the mining complex has also been investigated by putting down numerous soundings in 2011. Twenty seven (27) soundings (geotechnical boreholes and test pits) were put down to determine the bearing capacity of the soils at proposed locations of various structures (plant, concentrator, conveyors, various warehouses, garages, etc.). As a result the investigation, heavier structures were relocated such as to bear on bedrock. Recommendations on construction methods were also issued..

18.4 Stockpiles and Water Management

18.4.1 Waste Rock Piles

The open pit mining operations will generate 159.9 Mt of waste rock. The material will be utilized in the construction of various site facilities including, for example, the tailings dams, mine roads and channels. The remaining will be stored in a waste rock stockpile located approximately 1.1km from the mine entrance. This stockpile will be composed of rock material that does not contain enough mineralized material to be economically processed. A volume of approximately 33 Mm^3 is expected and it 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. From ground elevation (natural topographic elevation in the area varies between 73 and 120 m) to elevation 138 m, it will have an overall 26° slope angle. From an elevation of 138 m, the overall slope will be reduced to 10° to insure stability of the pile. The planned final elevation is approximately 158 m. Refer to APPENDIX N – General Site Layout (15685-01-DR-GE-003) for location of the waste rock pile and to the Figure 18-15 for a typical cross-section.

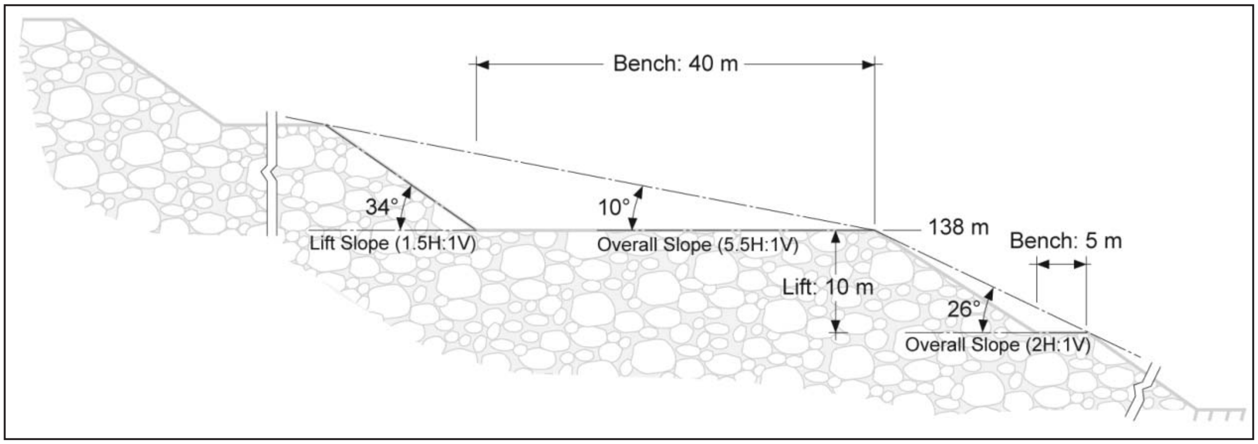


Figure 18-15: Typical Cross-Section - Waste Rock Dump (by WSP)

In addition to this waste rock dump, West section of the open-pit will be used to dump waste rock. It will be possible to store approximately 26.9 Mt of material in this area, starting around year 17.

18.4.2 Overburden Piles

An overburden material stockpile will be erected at approximately 800 m from the mine entrance/exit. This stockpile will be primarily made up of top soil material that will be removed in order to reach hard rock containing mineralization. It will have a volume of approximately 1.0 Mm^3. The pile will be accessible through a 10% access ramp. It will have an overall slope of 18 degrees and a final elevation of approximately 110 m considering that the natural topography in the area varies between elevations 56 to 98 m. Refer to APPENDIX N – General Site Layout (15685-01-DR-GE-003) for the location of the overburden piles and to the Figure 18-16 for a typical cross-section.

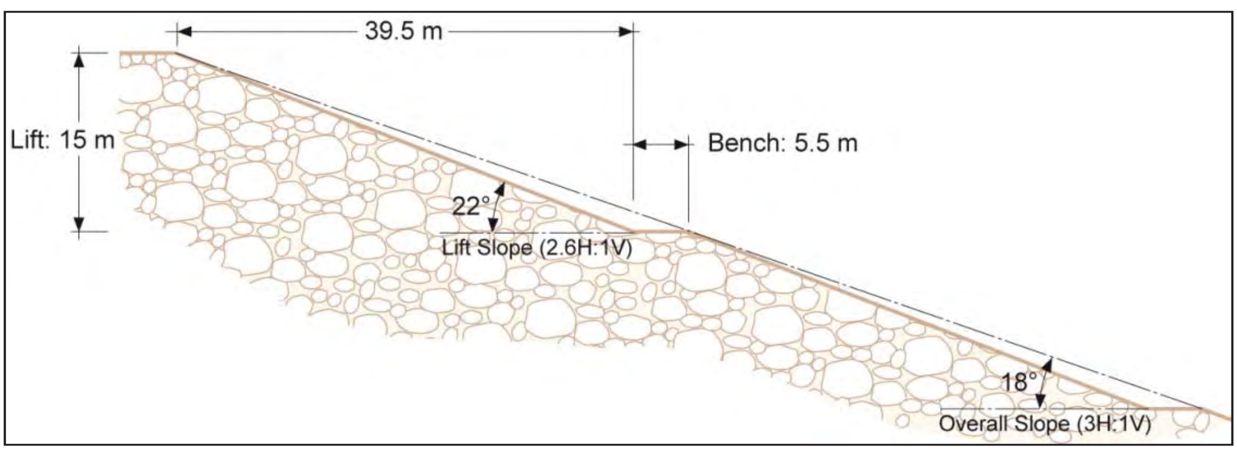


Figure 18-16: Typical Cross-Section of the Overburden Piles (by WSP)

The stockpile footprint is limited on each side by the final pit crest, the Hydro-Quebec power lines and railway. The resulting stockpile capacity is therefore not sufficient to dispose of all of the overburden material. A second overburden stockpile will be available (refer to section 18.4.3 Screen Berm) and the remaining material will be used to cover a portion of the tailing storage facility.

18.4.3 Screen Berm

Mining at the site will consist of advancement of an open pit to extract the ore for processing. The mining activities will require blasting as well as operation of large mining equipment (haul rock trucks, loaders, excavators, etc) to extract the ore and mine waste rock. A screen berm is proposed to mitigate and deflect noise and dust from the local population that is located to the south of the planned open pit. The proposed screen berm will be aligned along the South wall of the open pit and will be constructed with local borrow materials (overburden stripping) and mine waste rock.

A small scale site investigation was completed to support preliminary assessments of the proposed screen berm by collecting in situ sub-surface soil conditions with mechanical directional drilling. Four (4) boreholes were advanced in the proposed footprint area of the screen berm. The sub-surface soil conditions from the four (4) boreholes advanced indicates large varying thicknesses of overburden (to bedrock) ranging from approximately 4 m to 14 m in depth. The overburden material generally consists of a sand layer overlying a clay/silt layer. The sand layer was identified as being very loose to loose/compact in density. The underlying clay/silt layer was noted as being very soft/soft too stiff with undrained shear strengths from in situ vane testing ranging from approximately 18 kPa to 80 kPa.

Considering the number of boreholes in the proposed footprint and large variability of soil properties and thickness, conservative values were used for stability analysis. Preliminary stability assessments of the proposed screen berm have indicated that a maximum fill height of overburden of approximately 10 m to 15 m can be adopted while maintaining acceptable safety factors. This resultant height does not achieve the required embankment height to meet the design objectives and therefore a buttress berm has been included with the proposed screen berm cross-section to increase the potential height of the berm. Stability analyses completed on the screen berm, with the inclusion of stability buttress berms, indicate that an embankment height of approximately 30 m can be achieved on the eastern and center sections of the screen berm. The buttress berms would be included on the upstream and downstream sides to achieve the required minimum stability safety factor. Buttress berms would be required on the downstream side of the screen berm for the western section. In order to limit footprint of mining infrastructure and/or to reach a higher elevation, soil improvement or establishing a noise impeding fence can be considered as the project is advanced. Additional site investigations should be used to delineate the bedrock profile and overburden thickness along the berm alignment to optimize the design of the screen berm (refer to recommendations, section 26.4).

The screen berm will be accessible through a 10 % access ramp; it will have an overall slope of 18°. Refer to APPENDIX N – General Site Layout (15685-01-DR-GE-003) for the location of the screen berm.

Approximately 15.8 Mm^3 of overburden will be required for this construction. Approximately 6.0 Mm^3 of waste rock could be used for construction of the buttress berms.

18.4.4 Water Treatment Plant

All contact water is directed to the Storage Pond. This includes excess water from the TSF, TSF dam seepage collected in ditches, pit dewatering, the waste rock dump, the overburden pile and the ore pile. Water from the Storage Pond is either pumped to the Concentrator for process purposes, to a treatment system to feed the Gland Seal Water Tank or to a water treatment plant (WTP) to then be discharged to the Clet creek.

A treatment system with membrane will be implemented to treat water from the Storage Pond to the Concentrator pumps gland seal, which was included in the capital costs. The system would take 350 m^3/h at the intake to produce about 260 m^3/h of permeate to feed the Gland Seal Water Tank in the Concentrator. The concentrate from the backwash would be sent back to the TSF.

From the water balance completed in October 2013 by WSP, excess water will have to be discharged to the environment at Year 1 of the project at a rate of about 1040 m^3/h. A water treatment plant will be installed due to possible high levels of iron, total suspended solids and phosphorus. The WTP would include a reactor mixer, a rapid mixer, a clarifier mechanism and a filter press for sludge management. The sludge would be trucked to the TSF.

18.4.5 Surface Water Management System

The water management plan must facilitate the mining operations through a wide range of climatic conditions, while at the same time protecting the environment. The prime objectives of the water management plan will be to:

- Provide a reliable water supply to the concentrator.
- Facilitate mining of the ore deposit by limiting inflows to the open pit and by the timely removal of groundwater discharges and precipitation falling on the incremental catchment of the open pit.
- Provide sediment control.
- Collect and treat contact water that could otherwise impair water quality of receiving streams.
- Protect mine infrastructures during extreme flood events.

The water management plan revolves around changes throughout the mine life cycle. Each phase incorporates diversion structures, sumps and pumping stations that manage contact and clean water separately as the overall surface area, or footprint of the mine, expands. Each sump and pumping station will be accessible through a secondary maintenance road.

Surface water runoff that comes into contact with tailings is identified as “contact water.” This water, which includes runoff and seepage from the tailings dams, will be collected by a network of channels and sumps located around the TSF cells, and pumped back into the active TSF. The concentrator will reclaim as much of the TSF water as possible to minimize the treatment required for the contact water. Any contact water surplus will be treated at the water treatment plant.

Surface water runoff that comes into contact with waste rock or overburden is also considered as “contact water”. This water will be collected by a network of channels and sumps located around waste rock dump, overburden stockpile and screen berm. This contact water will be pumped into the Storage Pond where it can be used by the concentrator for reclaim or as a freshwater source. Water resulting from the dewatering activities of the open pit will also be pumped to the Storage Pond.

A total of 33 sumps and 33 pumping stations will be located in low elevation areas throughout the TSF, screen berm and various stockpiles, where water conveyance by gravity flow is not feasible. These components will be implemented at various times throughout the development of the mine site. Three of the sumps will collect clean water. This water will be pumped to the adjacent natural watercourse.

The collection ditches will be at least 1 m deep, with a bottom of at least 1.0 m wide and side slopes of 2H:1V (in soil) and about 0.1H:1V (in bedrock). The layout of the ditches network is shown on Figure 18-17, while typical cross sections of the collection ditches and of the secondary access road are shown on Figure 18-18.

Typically the sumps are rectangle, with a capacity ranging from 700 m^3 to 70 000 m^3 (average of 8600 m^3) and a depth of 2.6 m. Water from the sumps will be pumped from spring snowmelt to autumn. It is not planned to heat pumping stations and operate them during winter.

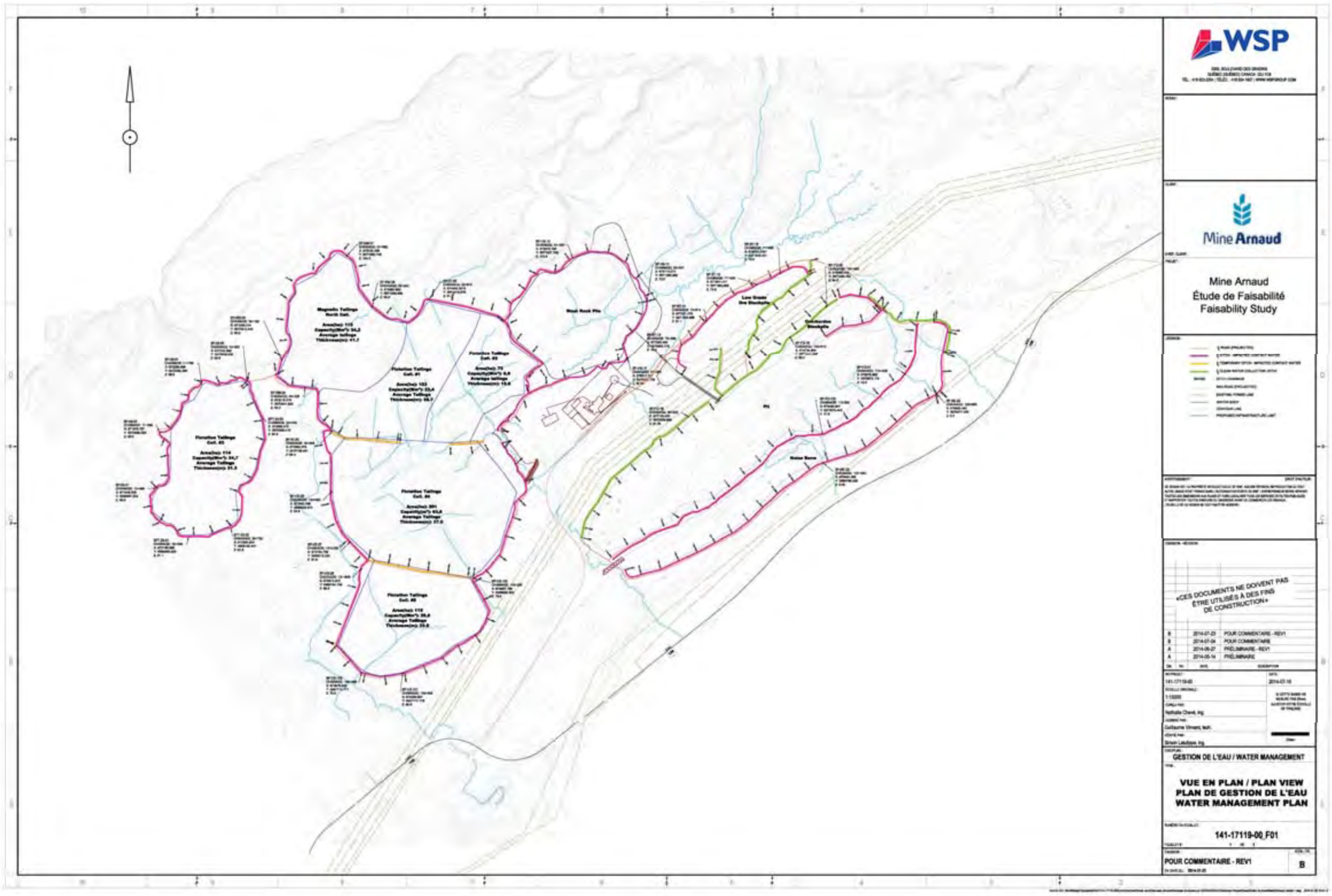


Figure 18-17: Layout of the ditch network

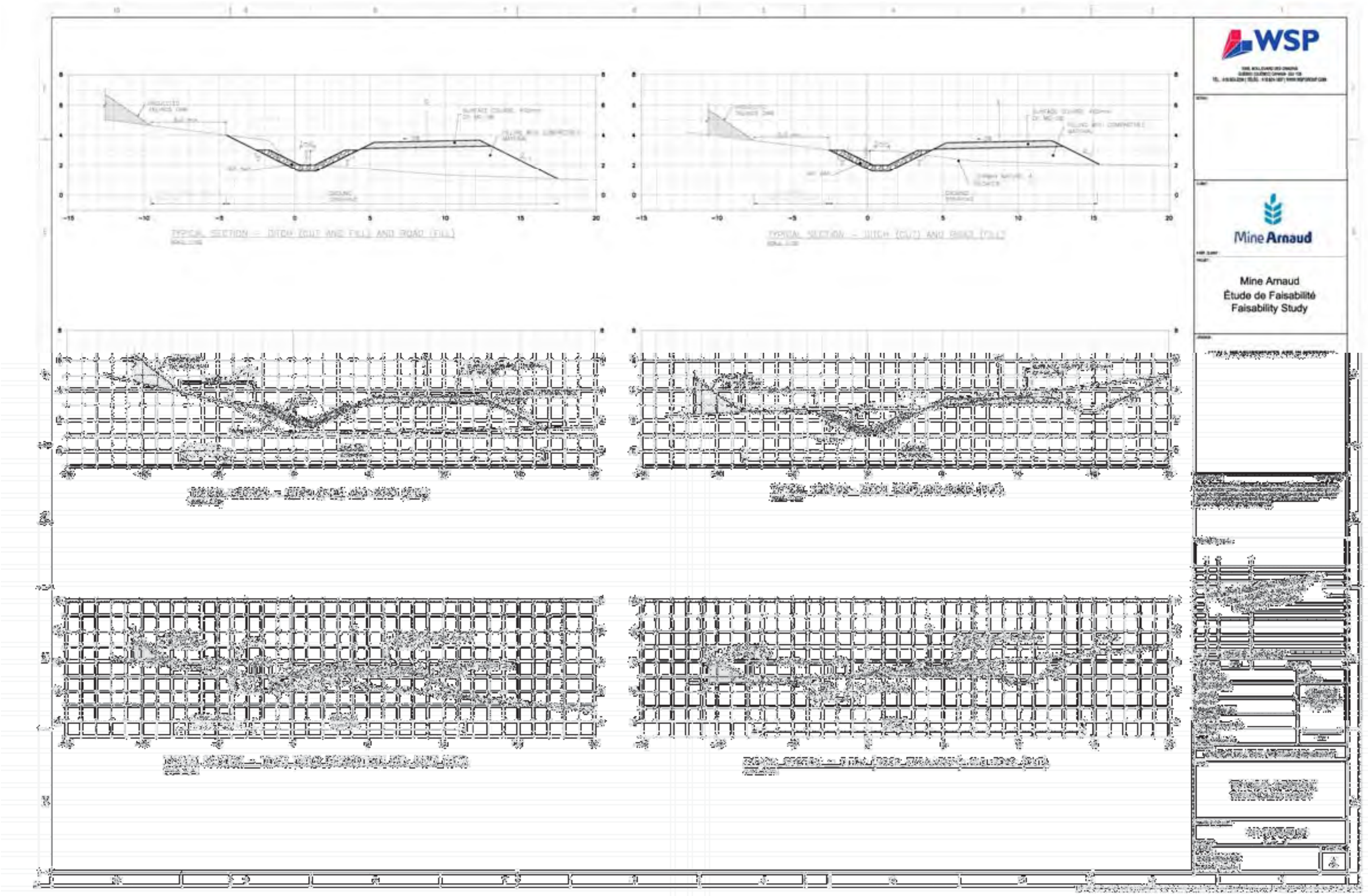


Figure 18-18: Typical Cross Section of collection ditches and secondary access road

18.4.6 Storage Pond

The Storage Pond is located south of the TSF cell #2 and it has a capacity of approximately 0.25 Mm^3. It will receive runoff water from TSF, various stockpiles and dewatering water from the pit. Water in the storage pond is assumed to be contact water and act as the feeding pond for the water treatment plant. After treatment, water will be directed to the concentrator or to Clet creek.

Since the footprint of the TSF, various stockpiles and the pit will be small during the first years of operation, the water coming from these components will be limited and quantity may be inadequate for the ore treatment. It is therefore planned to build the storage pond early in the construction phase. An additional 0.5 to 1 Mm^3 of water will be required during the first years of operation,This will be done by constructing a portion of the TSF Cell #2 dike, which crosses the Clet Creek, at the construction phase (year -1). This dam will allow the accumulation of enough start-up water for year 1 of mining operations.

18.5 Electrical Infrastructure

18.5.1 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-Quebec line L1619. 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-Quebec.

Substation general arrangements drawings 15685-04-DR-EL-011 – 161kV Substation Plan and 15685-04-DR-EL-012 – 161kV Substation Sections are presented in APPENDIX O – Electrical Drawings.

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 Concentrator Electrical Room 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 electrical rooms are planned:

- Concentrator 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.

18.5.2 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.

18.5.3 Load Flow and Short-Circuit Study

To establish equipment ratings, preliminary load flow, short-circuit, motor starting and harmonic studies were prepared for the project in 2011 (see APPENDIX P – Load Flow and Short Circuit Studies).

18.5.4 Electrical Load

The electrical installed load and running loads for the concentrator-mine site and for the port facilities are indicated in the Table 18-7 and Table 18-8.

Table 18-7: Concentrator - Mine Site

Area	Electrical Installed Load kW	Electrical Running Load kW
Concentrator	73,335	55,554
Rail Load-Out	604	521
Garage, warehouse & administration	2,395	1,955
Crusher	1,849	1,576
Stockpile area	935	792
Tailings & Water Treatment	4,622	3,487
Mine Pit	324	195
TOTAL	84,064	64,080

Table 18-8: Port Facilities

Area	Electrical Installed Load kW	Electrical Running Load kW
Port-Shiploader	1,122	949
Port-Receiving	708	606
Port-Storage	528	409
TOTAL	2,358	1,964

The detailed list of loads is shown in the Electrical Load List presented in APPENDIX Q – Electrical Load List.

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 is calculated using the electrical running load, but does not include the 10% contingency.

18.5.5 Switchgear and MCC's

The 13.8 kV switchgears at the Concentrator Electrical Room will supply power to the 13.8 kV mills 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 the Concentrator 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 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.

18.5.6 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, could be considered to determine if there is a technical or cost advantage to the project.

18.5.7 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.

18.5.8 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, three 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 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 sets 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 could 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.

18.5.9 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 minutes 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.

18.5.10 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 Concentrator Electrical Room at the Mine Site, and at the 600V busbars of both Low Voltage Switchgears at the Port electrical buildings.

18.5.11 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.

18.5.12 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.

18.5.13 Electrical Reference Documents

Details of the proposed electrical equipment are contained in document number 15685-01-DC-EL-001 Electrical Design Criteria (APPENDIX R – Electrical Design Criteria) and in the single line diagrams (APPENDIX S – Single Line Diagrams) drawings 15685-04-DR-EL-001 to 007 and drawing 15685-08-DR-EL-008).

18.6 Controls and Communication

Details of the proposed control & communication systems and equipments are contained in document number 15685-01-DC-IC-001 Design Criteria for Instrumentation and Controls (APPENDIX T – Design Criteria for Instrumentation and Controls) and in the system architecture drawings (APPENDIX U – System Architecture Drawings, 15685-04-DR-IC-401 to 403).

System architecture drawings 15685-04-DR-IC-401 and 15685-04-DR-IC-403 cover the different systems located within the mine area, while drawing 15685-04-DR-IC-402 cover the different systems located within the port area.

18.6.1 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 fiber 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.

18.6.2 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.

18.6.3 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.

18.6.4 Access System

The control access system will permit acknowledgment of entrance and exit of individuals and vehicles 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 personel.

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 personel movement, archiving of video and movement at specific control point, etc.

18.6.5 Radio Communication System

45 Watts mobile radios installed in vehicles and 5 Watts hand held radios will supplement communication between individuals 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 Cliffs train operators.

18.6.6 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 fiber, or Cat6 according to distance. Digital video recorder will allow recording and archival of process and restricted area activities when required.

18.6.7 Process Control System

Process control system will consist of redundant operation stations 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 MCCs and switchgears will be non redundant.

18.6.8 Instrumentation

All instruments supplied by a package supplier or the Owner shall be as much 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.

18.6.9 Controls and Communication Reference Documents

System architecture drawings 15685-04-DR-IC-401 and 15685-04-DR-IC-403 cover the different system located within the mine area, while drawing 15685-04-DR-IC-402 cover the different system located within the port area (APPENDIX N – General Site Layout).

18.7 Transportation and Relocated Railroad

18.7.1 Background

In the early phase of the Original Feasibility Study in 2010, Roche-Ausenco performed a study 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.

A Preliminary Alternative Transportation Study Report was issued to Mine Arnaud in October 2010. The report provided a description of each of the alternatives followed by a direct comparison of the transportation alternatives from the Capex, Opex, 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 Cliffs Natural Resources (Cliffs), the owner of the Chemin de Fer Arnaud (CFA) railway. Although the capital cost for rail is higher than the road alternative, it requires less labour and can benefit from experienced Cliffs personnel as well as 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 Cliffs and the Port of Sept-Îles, the rail alternative appeared to be the most interesting option and the one to pursue further for the Original Feasibility Study of the Mine Arnaud project. The rail alternative is therefore the preferred mode of transportation retained for the project.

Earlier in 2014, Mine Arnaud engaged the engineering firm Strudes to compare transportation modes and further develop transportation by trucks, which would be the fall back transportation mode if Mine Arnaud would not be able to reach an agreement with Cliffs. A summary of an alternative transportation mode by trucks was provided by Mine Arnaud in the Item 18.8. The complete study performed by Strudes can be found in APPENDIX V – STRUDES.

18.7.2 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 Cliffs Natural Resources. The railway is used to transport iron ore to the Cliffs Wabush Mines and Cliffs Quebec Iron Mines (Bloom Lake) terminals in the Pointe Noire area of Sept-Îles. It is also used for access to a ferry-rail service at the La Relance Terminal of the Port of Sept-Îles. The ferry-rail service allows transfer of railcars between the Québec North Shore and the southern shore of the St-Lawrence and is operated by Cogema, a subsidiary of 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 off the main track. Although there are many different property owners near the mine site to the south, only Cliffs has right-of-way for the railway corridor.

18.7.3 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 CFA (Cliffs) 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.

18.7.3.1 Apatite Concentrate Characteristics

Apatite concentrate (apatite) is to be transported. 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.

18.7.3.2 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, 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 enclosed belt conveyor with a capacity of 250 tonnes per hour. The conveyor will be elevated over the silos at its head end. 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. 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 three (3) telescoping spouts, each feeding its corresponding railcar hatch one after the other. The spacing between the silos will allow feeding two (2) adjacent railcars at a time.

For each silo, an electric pivoting gangway will provide access and guarding to the railcar roof. A hydraulically operated indexer located at ground level along the rail track will move and position the whole train so the loading hatches of the next two (2) empty cars will be aligned with the silo spouts.

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 loading station will be equipped with rapid rolling doors at both ends and with an annex including a control room, a kitchen, toilets and a mechanical (hydraulic) room. An enclosed stairway adjacent to the annex will provide access to the conveyor gallery on the top of the silos.

18.7.3.3 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 CFA rail crews for transport and delivery to the port, approximately 17 kilometers away;
- A new 2.28 km rail spur will be built to by-pass the existing Cliffs Wabush Yard loop track and avoid any potential conflicts or delays;
- Railcars are delivered in blocks of up to 20 railcars to the unloading facility by contracted CFA rail crews;
- Staff at the unloading facility to unload railcars and prepare them for return to the mine site.

18.7.3.4 Equipment

Locomotives

The locomotives to perform this movement will be supplied under contract by the rail operator. These locomotives should be capable of handling the 39 loaded cars on the grade present on the CFA subdivision, each having a capacity to carry 105 tonnes of apatite concentrate. 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. This operation requires 39 rail cars in service. With a bad order factor of 5%, the total number of railcars to be purchased is 41.

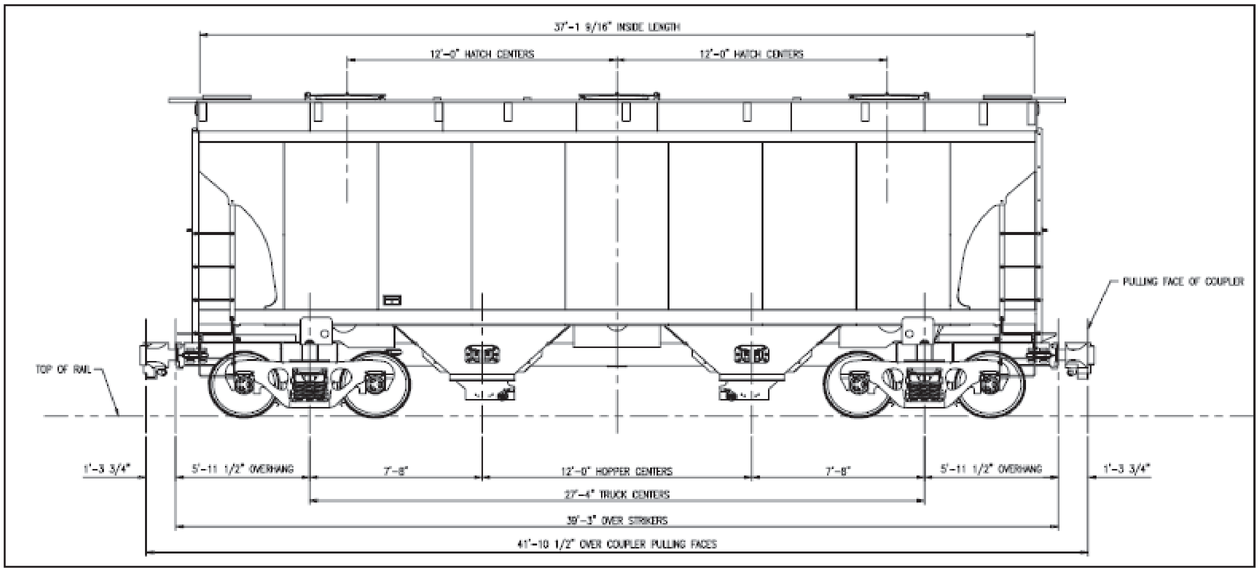


Figure 18-19: Railcar Drawing

18.7.3.5 Operating Plan

It was estimated that that there would be 336 working days per year (Table 18-9).

Table 18-9: Rail service Data outlines the service requirements and assumptions

Parameter	Unit	Value
Operating days	days/y	336
Tonnes per Year – Design	t/y	1,300,000
Tonnes per Operating Day	t/day	3,869
Tonnes per railcar	t/car	100
Railcars per Day	cars/day	39

Switching service would be provided by CFA under contract. CFA would provide the crews and the locomotives and be in charge of the dispatch. CFA would also control switching priorities.

The 39-car train is expected to be able to handle 1.30 Mt/y of annual throughput. The train will spend its idle time at the mine, without locomotives, waiting for product.

A 39-car train has been selected as an optimal choice because it makes decent use of the locomotives' power minimizing the sidings length required and allowing efficient train operations based on a constant schedule of one train per day. Making decent use of the locomotives' power capacity reduces the number of trips and distance-associated maintenance required.

The unloading process is being designed to unload all 39 rail cars within 5 hours. 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 eight hours, at the mine. The Table 18-10 presents the proposed train cycle time.

Table 18-10: Proposed Cycle

Event	Duration (h/cycle)
Loading Window	9.25
Allowance	4.24
Loaded Transit	0.5
Spotting at Unloader	0.35
Train Segmenting & Unloading Window	4.07
Pick-up from Unloader & Train Regrouping	0.35
Allowance	4.24
Empty Transit	0.5
Spotting at Loader	0.5
Gross Cycle Time	24.0

18.7.4 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.

18.7.5 Diversion Track

The diversion track consists of a new main line and a loadout siding to be constructed to relocate the existing main line, for the purpose of accommodating the proposed mine site. The siding will allow the loading of the railcars without blocking the main line.

18.7.5.1 Earthworks

The cut and fill quantities were calculated per the assumed cross-section shown in drawing 15685-04-DR-RA-007 (APPENDIX W – Rail Drawings). 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 0.30 m.

18.7.5.2 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-Quebec power lines. The diversion then turns back to the northeast and plateaus at elevation 79.10 meters. The load out siding track branches off at this point. The diversion track alignment then runs roughly north-easterly, 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-Quebec right-of-way again and follows a 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. 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).

18.7.5.3 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.

18.7.5.4 Overhead Powerline

Hydro-Quebec has four 735 kV powerlines and one 161 kV line within a right-of-way that runs southwest to northeast. This Hydro-Quebec 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.

In a letter dated 4 July 2014, Hydro-Quebec TransÉnergie advised that the rail passages considered by Mine Arnaud under the corridor for the four 735 kV transmission lines (7031, 7032, 7033 & the new Romaine II–Arnaud) are acceptable based on the train not transporting dangerous goods. However, the 161 KV line L1619 needs to be stabilized.

18.7.6 Track Construction Methodology

The existing track is to remain usable during the construction of the diversion track with minimal interruption to Cliffs operations.

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.

The civil portion of the work will take place in the first year of construction, with rail added the following year using one winter season for settlement.

As the existing track will be abandoned afterwards, rail, fasteners, and ties can be re-used elsewhere.

The diversion track is to be constructed to final top of rail elevation prior to tying in to the existing track.

18.7.7 Mine Arnaud Load-out Siding

The Mine Arnaud Load-out 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 northeast to southwest. This is to ensure that the southwest portion of the siding bears the heaviest load.

It is preferable that the Mine Arnaud Siding be completed before the diversion track is opened for traffic for safety risks and construction delay reasons.

18.7.8 General Delivery Track

The General Delivery track is a short extension intended to be a storage track for up to six (6) wagons in total carrying any goods to be delivered by rail to the Mine Site as well as storage for two spare 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.

18.7.9 Reference Documents for Rail

As part of this Feasibility Study Update, the Rail Design Criteria 15685-01-DC-RA-001 and the rail drawings 15685-04-DR-RA-001 to -007,-009 and -010 were updated.

The Silos and Rail Load-out are shown on drawings 15685-05-DR-FS-002, 15685-05-DR-IC-002 and 15685-05-DR-ME-010.

A new drawing 15685-08-DR-RA-001 was created for the new 2.28 km rail spur to bypass the Cliffs Wabush Yard (see APPENDIX W – Rail Drawings).

18.8 Alternative Transportation Modes

Mine Arnaud mandated the engineering firm Strudes, outside of this Feasibility Study, to evaluate alternative transportation modes for the concentrate to the Port of Sept-Îles. The few lines below present what has been evaluated and what was retained as a Plan B for the transportation of concentrate, in case of a major problem with the plan to carry the material by train.

Three options (other than the train) were considered:

- Slurry Pipeline;
- Conveyor;
- Trucks.

For all these options, evaluations were based on technical feasibility, Capex, Opex and consideration for the environment.

18.8.1 Slurry Pipeline

Strudes has investigated transportation of the phosphate rock in wet form, i.e. pumping of the phosphate slurry to a dewatering station located near the Sept-Îles port. This requires that the filter and the flash dryer, currently located at the concentrator site, be relocated near the marine terminal. The pumps required to pump the slurry needed to operate at high pressure and therefore became very costly. This option was then rejected due mainly to the capital cost.

18.8.2 Conveyor

A few types of conveyor were considered:

- Overland conveyor – long conventional trough belt;
- Rope (cable) conveyor with full enclosure;
- Pipe (tubular) belt conveyor.

Overland conveyor and rope belt conveyor systems were rejected due to potential environmental issues related to dust. To fully contain the dust and eliminate the potential hazards made the cost of the systems excessive.

The tubular belt conveyor with a proposed cover is acceptable for containment of potential dust spills; however the corresponding costs are prohibitive.

18.8.3 Trucking

Trucking was the selected alternative for the transportation of apatite concentrate. This retained option was evaluated in more details and shows that the capital cost is in the same range as the one with the train. It would require some design changes at the loading and unloading station, but would keep all the other infrastructures for the train as is. The operating costs were evaluated at a slightly higher price per tonne than the train but remained acceptable. The transportation price used in this feasibility by train is CA /t of concentrate and the alternative way with truck is estimated at CA$ /t of concentrate. For more details about this scoping study, please refer to (APPENDIX V – Strudes).

18.9 Port Facilities

Apatite concentrate is transported by train up to the Mine Arnaud Port Facilities (Port Facilities) located around Anse à Brochu in the Pointe Noire area of the Port of Sept-Îles, about 17 km away from the mine site. The Figure 18-20 shows the location of the port facilities vis-à-vis the mine site.

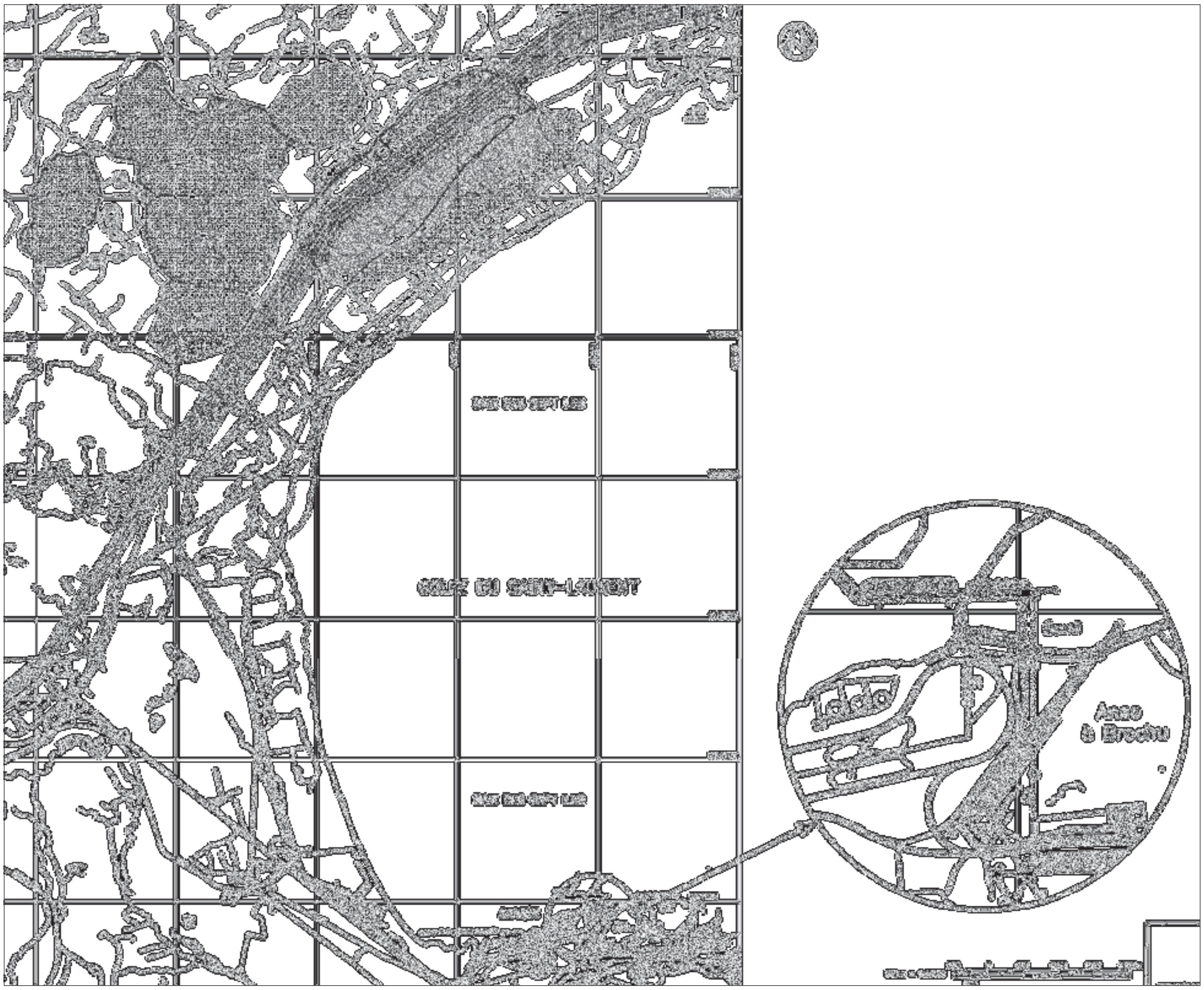


Figure 18-20: Location of Port Facilities

During the course of the Original Feasibility Study in 2011, different concepts for the Port Facilities were considered at the La Relance Terminal of the Port of Sept-Îles. The existing Berth 41 and Berth 42 at the Port cannot be used by Mine Arnaud. Therefore, a new dock (Berth 39) located at the end of the existing Berth 42 was required. Mine Arnaud port facilities included some high vertical structures, such as silos, that required bedrock located near the surface in the ground to minimize construction costs. The Terminal La Relance is mainly reclaimed land with thick filling material, except near the entrance of the Terminal on the south side. At that location, a road overpass for the terminal was required to allow unrestricted rail access to the area. The concept developed in 2011 at the La Relance Terminal had therefore two major drawbacks: the need to build a costly new dock and a road overpass.

In May 2014, Mine Arnaud requested Roche-Ausenco to develop a concept for port facilities located near the existing Berth 31 in a different location of the Pointe Noire area of the Port of Sept-Îles. Berth 31 is at the east end of a dock owned by the Port of Sept-Îles and currently used by Cliffs. The following are key design criteria considered:

- Since 2011, there have been discussions between Cliffs and Mine Arnaud regarding the use of Chemin de Fer Arnaud to transport apatite to the port facilities. It appears that a rail line that would bypass Clifs Wabush Yard would be more acceptable to them. Such a rail link would isolate Mine Arnaud port operations and make splitting up trains less of an issue. Splitting up trains in two means that the port facilities require less land, the port terminal is more compact and can be located closer to Berth 31;
- The existing Berth 31, currently used by Cliffs to ship its iron ore from Bloom Lake, is believed to become available for use by Mine Arnaud in the near future. Cliffs is notably planning modifications at Berth 30 for capesize vessels that would free up Berth 31. The Port of Sept-Îles is also building a new multi-user dock for iron ore shipment. For this Feasibility Study Update, the Port of Sept-Îles confirmed in a letter the anticipated availability of Berth 31 with the beginning of operations at the new Multi-User Dock (expected to be at the end of 2014);

As a result of the above, Roche Ausenco developed a new concept for the Mine Arnaud Terminal based on the following:

- Cliffs will discontinue the use of Berth 31 for loading of iron ore to ships;
- If Cliffs extends the travel of the tripper conveyor at Berth 30 towards Berth 31 or replace it entirely to accommodate capesize vessels, it will limit the usable space at Berth 31. A maximum travel distance for the tripper conveyor feeding Berth 30 was estimated by Roche Ausenco as no dimension was available;
- The existing fixed shiploader and associated feed conveyor used by Cliffs at Berth 31 will be removed by Cliffs or the Port of Sept-Îles;
- Mine Arnaud will ship apatite concentrate using only Handysize ships (37,000 tonne DWT) on the space left at Berth 31;
- The Port of Sept-Îles will make the necessary investments at Berth 31 to reinforce the dock to suit the new travelling shiploader and tripper loads, extend it to the east, add a platform for a transfer tower and realign the access bridge;
- Cliffs will allow Mine Arnaud to build a new 2.28 km rail spur with siding on its property to bypass its Wabush Yard;
- An area for the terminal within the Port of Sept-Îles property at Anse à Brochu will be available to Mine Arnaud. At that location, the unloading station and the storage silos can be built directly on the bedrock.

The new concept is shown in the following Figure 18-21. assuming that Cliffs will proceed with upgrades at the west end of the dock (Berth 30) to accommodate receiving and loading capsize vessels.

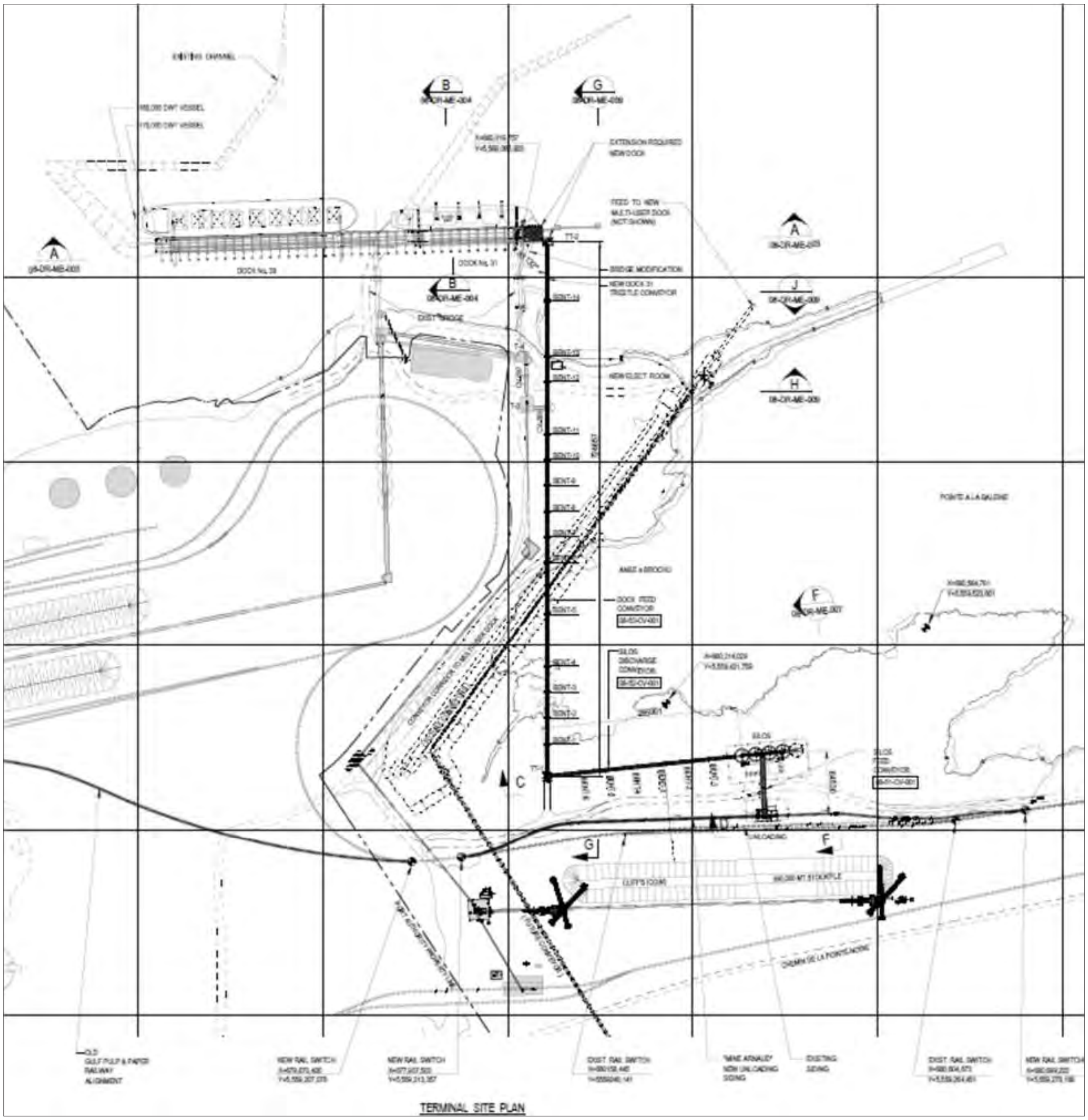


Figure 18-21: Port Facilities

Mine Arnaud rail unloading and storage areas are located in the southern part near the existing rail tracks within the Port of Sept-Îles property and where bedrock is anticipated near the surface. The four (4) storage silos, with provision for a future fifth silo, are located near rail and an access road. From the silos, a series of two (2) conveyors enclosed in tubular galleries will transfer the apatite concentrate to the Berth 31. The first conveyor runs at grade in the westward direction to the first transfer tower. From the transfer tower, a second elevated conveyor oriented from south to north will cross the Bay of Sept-Îles at two (2) locations as well as over future iron ore installations for the new multi-user dock currently being built by the Port of Sept-Îles. The second elevated conveyor will terminate at a second transfer tower located at the east side of Berth 31.

18.9.1 Port Track Infrastructure

Roche Ausenco developed a new concept for the Mine Arnaud Terminal based on the addition of a new 2.28 km rail track within Cliff's property to bypass Cliffs Wabush Yard in order to reach the existing main track within the Port of Sept-Îles property.

A new 2.28km by-pass of the existing Wabush loop track will be built to avoid potential conflicts or delays with operations on the Wabush loop track. The proposed spur provides a more direct route to the Mine Arnaud terminal and follows the alignment of the old (abandoned) Gulf Pulp & Paper Railroad on the south side of the Wabush loop track.

Based on the information available, the proposed rail alignment appears to be feasible. Details of the alignment, soil conditions and available clearances must be reviewed and confirmed in the next stage of project development.

A 20-car long storage track (siding) has also been included on the spur. This will allow CFA to split the Mine Arnaud train into two blocks (19 and 20 cars) for delivery to the unloading station.

The existing main track within the Port of Sept-Îles property gives access to the La Relance Terminal and to a rail-ferry operated by Cogema (CN), from which railcars can reach the North American rail network via Matane in Gaspesia. This rail section is very infrequently used and mostly for inbound supplies to the iron ore mining companies. Near Pointe-à-la-Baleine, a 771 m long unloading and storage siding will be added for the Mine Arnaud unloading operations. This new Mine Arnaud siding is set around an unloading station, which includes a dumping pit, a railcar indexer, a bottom gate opener and all required services. The siding is long enough to take half of a complete Mine Arnaud train on each side of the dumping pit. The existing access road for the new Multi-user Dock will need to be modified at two (2) locations to allow for that siding.

The Mine Arnaud operations are envisioned to be as follows:

The switchers (yard locomotives) will drop half of the consist (20 loaded wagons) on the siding of the 2.28 km track and proceed with a first group of 20 loaded wagons to the Mine Arnaud unloading siding;

At the unloading siding:

- Loaded wagons will be dropped at the indexer and the switchers will return via the main track for their next assignment;
- The 20 loaded wagons will be processed over a period of 5 hours and empties will remain on the Mine Arnaud siding;

- After this first unloading period of two hours, the switchers will pick up the first 20 empty wagons and will position them on the siding of the 2.28 km track, then they will push the last 19 loaded wagons on Mine Arnaud unloading siding at the indexer;
- Twor hours later, the switchers will return to the Mine Arnaud unloading siding, pick up the empty wagons, consolidate them with the empty ones on the siding at the 2.2 km track and proceed to drop them on the siding at the Mine Arnaud mine.

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 movement of wagons as they go through the unloading station. The unloading siding finished grade profile is flat to minimize the requirements, size and power of the indexer.

18.9.2 Apatite Concentrate Characteristics

For materials handling purposes, apatite is assumed to be a dry easily-flowing powder that should not be spilled, 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 systems, similar to bulk cement.

18.9.3 Concentrate Unloading

Roche-Ausenco analyzed 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 and Boot-Lift type connector is believed to ensure a clean transfer of apatite concentrate, while minimizing dust and moisture pickup.

The railcar unloading station will consist of a shed housing the indexer, the bottom gate opener, the railcar unloading equipment, the unloading pit and its ancillary systems. The station is located as close as possible to the west and to Berth 31, while allowing for 20 wagons (half a train) to be parked upstream and downstream of it on the Mine Arnaud siding. The station is also located adjacent to the access road for the new Multi-user Dock. A fence will control access to the station that will house the control room for the complete Mine Arnaud port facilities.

The railcar unloading station will have a capacity of 1,325 tonnes per hour. It will include a receiving bin with sufficient volume for 125% of one railcar volume and an extracting screw conveyor. A flexible wall belt type high angle conveyor will collect the apatite and elevate it, via an inclined gallery, to the top of the silos. The unloading station will shelter all the required ancillary systems such as a dust collector, hydraulic unit for the indexer and sump pump. The dust collector is designed to keep a negative pressure in the receiving hopper to eliminate fugitive dust emissions on the ground floor via the unloading openings at the screw conveyor discharge point. The dust is reintroduced downstream on the flexible wall conveyor.

18.9.4 Concentrate Storage

The capacity of the apatite transfer equipment between the unloading station and the storage silos has been established to be 125% of the nominal railcar unloading rate, which corresponds to ten (10) railcars per hour loaded at the maximum permissible load of 106 tonnes (1060 t/h), given the time constrains for unloading by gravity, opening/closing the bottom gates and indexing the railcars.

The 1524 mm (60”) wide, 116 meter long, flexible wall belt type, Silos Feed Conveyor with a capacity of 1325 tonnes per hour will transfer and lift concentrate at an angle of about 42 degrees to the top of the silos located about 84.5 meters away for the unloading station. This angle will also allow the conveyor to be sufficiently high over the access road to the multi-user dock to clear road traffic.

Apatite concentrate storage at the port will consist of four (4) silos, 18 meters in diameter by 60 meters nominal height (44.2 m active height) for a total of 60,000 tonnes of concentrate storage capacity. This is equivalent approximately to 1.9 times the capacity of the largest vessels expected to handle Mine Arnaud apatite for Yara (37,000 DWT vessels with about 32,000 tonnes of product). 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 1067 mm (42”) wide by 86 meter long, shuttle belt type Silos Distributing Conveyor with a capacity of 1325 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 as well as two (2) stair cases (one adjacent to the freight elevator and the other at the other end) to provide access to the silo roof and to the gallery.

The silos at the port will be similar in design to 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 fluidized. The technology considered for this study is the CFI Silo System from FLSmidth. The silo bottom ring will be fluidized and equipped with multiple discharge spouts, where the apatite will be extracted, at controlled rate, and conveyed to a central tank via air slides. The central tank, equipped with a de-dusting filter, will feed a downstream chute. The silos will be equipped with bin vent and fan systems located on the silo roof and accessed by the top gallery. The silo floors will be elevated to provide room and clearance for the Silos Discharge Conveyor underneath.

The silos are located adjacent to the access road for the new Multi-user Dock. A fence will control access to the silos area which will also include and administration and maintenance building and an electrical sub-station.

18.9.5 Concentrate Ship loading

When a vessel is ready for loading, a 1372 mm (54”) wide by 324 meter long Silo Discharge Conveyor located underneath the silos will transfer, at controlled rate, the apatite concentrate from the bottom of the silo in operation to a Transfer Tower No. 1 located to the west, onshore south from Berth 31. The shiploading portion of the Port Facilities will have a design capacity of 2,500 tonnes per hour, capable of loading a 37,000 DWT ship in less than 24 hours. The Silo Discharge Conveyor is a conventional belt conveyor, enclosed in an unheated, uninsulated tubular gallery, and equipped with a belt scale at its head end. The transfer tower will allow feeding the Dock Feed Conveyor of the port facilities. The transfer tower will be equipped with a dust collector and with a 7.5-tonne hoist for the maintenance of the conveyor head/drive pulley and drive components.

The Dock Feed Conveyor, which will cross the area from the onshore south to the Berth 31 on the offshore true north side, will be an elevated 1372 mm (54”) belt conveyor enclosed in an unheated, uninsulated tubular gallery identical to the one for the Silos Discharge Conveyor. The 738 meters long conveyor will transfer apatite concentrate from the Transfer Tower No.1 near the silos to a Transfer Tower No.2 located at the east end of Berth 31.

The conveyor will be delivered for installation in sections of 35 meters, which will correspond to the spacing of bents on land. The conveyor will cross the Bay of Sept-Îles at two (2) locations, where additional support steel will allow spans of about 70 meters. This will minimize requirements for supports in the water. The selected locations for those supports correspond to areas of rocky islands or shallow water. A 70 meters section will also be included around Cliffs conveyor CV-008 and its emergency pile.

The elevated conveyor system will allow for truck traffic underneath. It will also allow for the future addition of belt conveyors underneath in the corridor designated by the Port of Sept-Îles for iron ore transport to the new Multi-user Dock.

The Transfer Tower No. 2 at Berth 31 will require a new structure of 14 metres by 11 meters. To minimize its size, a new electrical room for the new equipment will be located on land near Bent-13 of the Dock Feed Conveyor. The Transfer Tower No. 2 will be equipped with a dust collector.

In the Transfer Tower No. 2, the apatite concentrate will be transferred to a 1524 mm (60”) wide by 182 meters long Shiploader Tripper Conveyor, a conventional enclosed belt conveyor. The Shiploader Tripper Conveyor, installed in an elevated gallery open on one side to provide the passage of the travelling Shiploader Shuttle Conveyor and equipped with a travelling tripper, will feed a travelling shiploader along the Berth 31. The travelling tripper will be towed by the shiploader.

As Berth 31 should ideally be designed for multiple purpose usage, the shiploader will be elevated and travel on rails flush with the deck floor, with the possibility for parking away at the east end of the Berth 31. A modification to the berth access bridge will be required to relocate the access east of the shiploader’s end of travel position in order not to block access to Berth 31 at any position of the shiploader.

The shiploader boom will have luffing capability. The 1400 mm (55.1 inch) wide by 40 meters long Shiploader Shuttle Conveyor will be equipped with a 15 meter long telescoping portion allowing the loading of vessels up to 37,000 tonne DWT. 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 vessel holds to trim the apatite concentrate.

18.9.6 Marine Structures for Berth 31

The investments required to reinforce the dock for the new travelling shiploader and the Shiploader Tripper Conveyor, to support the new Transfer Tower No. 2 and to modify the access bridge at Berth 31 are by the Port of Sept-Îles.

A preliminary evaluation of the space available at Berth 31 for Handysize type ships, if Cliffs adopts Capesize type ships at Berth 30, shows that an extension of at least 25.333 meters will also be required on the east side of the dock at Berth 31.

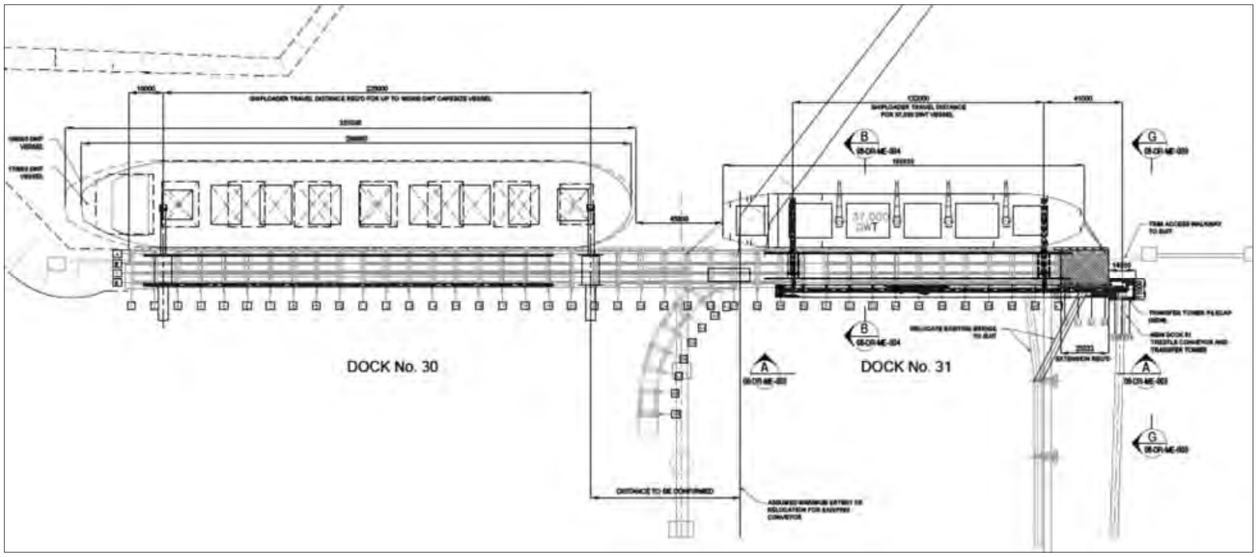


Figure 18-22: Berths №`s 30 and 31

The design of marine structures is by the Port of Sept-Îles. Therefore, marine structures have not been defined in details and are not included within the Mine Arnaud project Capex estimate. However, it will be indirectly charged to Mine Arnaud as an additional yearly fee from the Port of Sept-Îles in the Opex estimate.

Roche-Ausenco retained Portha as a sub-contractor for a very high level opinion on the capacity of Berth 31 to support a travelling shiploader. Portha has worked on Berth 31 in the past and has a good knowledge of its design and condition. Their conclusion is as follows:

'It appears that the deck beams and slab at Berth 31 were not built to support a travelling shiploader unless some reinforcements are added. However, according to some previous analyses of that structure, the lower tier, which includes the piles and lower cast-in-place reinforced beam system were designed uniformly for both Berths 30 and 31 and should be structurally adequate for such an application to load Handysize ships with a travelling shiploader of similar size and weight to the original ones at Berth 30.

> 'It appears that the deck beams and slab at Berth 31 were not built to support a travelling shiploader unless some reinforcements are added. However, according to

some previous analyses of that structure, the lower tier, which includes the piles and lower cast-in-place reinforced beam system were designed uniformly for both Berths 30 and 31 and should be structurally adequate for such an application to load Handysize ships with a travelling shiploader of similar size and weight to the original ones at Berth 30" (Portha, 2014).

Further study work is required to verify the dock structure and establish the cost to reinforce it for a travelling shiploader. Portha has identified two (2) alternatives if this scenario is found to be too expensive:

- Two fixed radial shiploaders could very likely cover the design vessel. Further check is necessary, but it seems it would be possible. The locations of the support towers of the fixed shiploaders would have to match the existing major pilecap baylines - so the hatch coverage would need to be evaluated for these discrete positions;
- The other option is to retain the existing fixed shiploader and the incoming conveyor and use it in combination with warping, and perhaps with some adaptation of the shiploader and spout - this could be the least expensive option. This would need verification as to the berth occupancy, the extent of warping to avoid clash with the capesize vessels on Dock 30 and the need for additional dolphins at the end of warped position.

18.9.7 Services for the Port Facilities

The Mine Arnaud Facilities will be located near the new Multi-user Dock and Berth No. 31 in an area already equipped with many services. 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-Quebec 25 kV transmission line nearby via a sub-station located near the silos. Please refer to Section 18.5.2 for the electrical and Section 18.6 for the automation portion of the Port Facilities.

18.9.8 Reference Documents for the Port Facilities

As part of this Feasibility Study Update, the following drawings prepared for the Port Facilities (APPENDIX X – Ausenco Feasability Study):

Flowsheet and P&ID: 15685-08-DR-FS-001, 15685-08-DR-IC-001;

Site and Mechanical: 15685-08-DR-ME-001 to -009.

19. Marketing Studies and Contracts

19.1 Overview

19.2 Phosphate rock types and key characteristics

20. Environmental studies, permitting and social or community impacts

All information contained in this section was drawn from the Environmental Impact Statement (EIS) dated March 2012 and its supplements, as well as all commitments made by the proponent during public hearings, including those held by the BAPE [Board of Public Environmental Hearings].

The EIS and its supplements can be viewed on Mine Arnaud's Website:

(http://www.minearnaud.com/fr/étude-impact).

20.1 Legal environmental context

The legal framework for construction and operation of the planned facilities is a combination of provincial, federal and municipal policies, regulations and guidelines. The project must be designed and environmentally managed in accordance with this legal framework.

20.1.1 Provincial procedure

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 tonnes or more per day, or any other ore[4], where the processing capacity of the plant is 500 metric tonnes or more per day.

[4] Other than a metalliferous, asbestos or uranium ore.

(p) The opening and operation of a metals mine[5] or an asbestos mine that has a production capacity of 7,000 metric tonnes or more per day, or any other mine[6] that has a production capacity of 500 metric tonnes or more per day.

Thus, this 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 notification was filed with the MDDELLC on December 14, 2010. A week later, having reviewed the project notification, the MDDELLC[7] provided Mine Arnaud with guidelines defining the required scope and contents of the environmental impact assessment of the project.

Under the *Canada-Québec Agreement on Environmental Assessment Cooperation* signed in May 2004, the MDDELLC sent a copy of the project notification to the Canadian Environmental Assessment Agency (CEAA) so that the latter could determine whether the project is also subject to the *Canadian Environmental Assessment Act*. As the project is subject to federal procedure (see below), the project will undergo a cooperative environmental assessment and the project notification will be entered in the public registry under the *Canadian Environmental Assessment Act*.

20.1.1.1 Analysis of the Environmental Impact Statement (EIS)

It is essential that the project be well defined in order to produce an EIS that takes into account all project features and to ensure that the project is considered acceptable by the authorities early in the procedure. Following receipt of the EIS, the MDDELLC will conduct an analysis of its admissibility. The analysis will include consultation with a number of departments and agencies. Generally, the proponent can expect to receive questions and comments, which it must address before the EIS can be considered admissible. Once the proponent has answered this first set of questions, it is possible that a second series of questions will be submitted for response. To avoid delays associated with this procedure, it is particularly important that the EIS follow the guidelines issued by the MDDELLC as closely as possible from the outset.

[5]. The term "mine" denotes all surface and underground infrastructure used for the extraction of ore.

[6]. Other than a metalliferous, asbestos or uranium ore.

[7] Formerly known as either MDDEFP or MDDEP

20.1.1.2 Public consultations

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 consultations provided for by regulation of the Government."

Thus, once the impact study is declared admissible, the Minister will ask the BAPE to prepare the case for public consultation. The consultation process will last 45 days (section 11 of the *Regulation respecting environmental impact assessment and review*).

20.1.1.3 Public hearings

Section 31.3 of the EQA also states that: "Any person, group or 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 BAPE 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* [internal management and rules of procedure relating to the conduct of public hearings] (Q-2, r. 45).

Following the public hearings, the BAPE will file its report with the Minister. The commission will be given four months to execute its mandate and file its report. The Minister will then have 60 days to publish the BAPE report.

20.1.1.4 Governmental decision

Based on the BAPE report and the environmental analysis report from the MDDELLC, the Minister will analyze the case and make a recommendation to the Government. As specified in section 31.15 of the EQA[8], the Government will render its decision by Decree, i.e., it will authorize the project, with or without any changes and conditions it establishes, or will refuse it. The maximum period allotted between the time the BAPE report is published and rendering of the Government's decision is not specified in the EQA or its regulations.

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[8] 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.

20.1.1.5 Application for certificates of authorization (CofA)

Following issuance of the Government Decree and until the construction of the project begins, the proponent shall submit to the Regional Directorate of the MDDELCC all the plans and specifications for construction that are necessary to obtain the required permits (CofA).

20.1.2 Federal procedure

As prescribed by section 5 of the *Canadian Environmental Assessment Act*, the project will have to undergo a federal environmental assessment, since the federal authorities will have to issue permits or licences prescribed by regulations. The federal assessment is also necessary 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*, only one impact assessment is to be conducted for both authorities, in accordance the Canada-Québec agreements in this regard.

The Canadian Environmental Assessment Agency, which will be responsible for coordinating the federal environmental assessment, has indicated that a comprehensive study-level assessment would be required for this project, given that the mine would involve 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 to be a “metal mine” and is thus subject to the *Metal Mining Effluent Regulations* (MMER). As such, it is currently possible to dispose of tailings in fish habitat areas, provided that the water bodies are listed in Schedule 2 of the Regulations. This approach requires conducting an assessment of alternative solutions for mine waste disposal. To this end, a first report was submitted in December 2012. A revised version was subsequently issued in September 2013 and included the comments received from the Agency in May 2013, as well as changes made to the Guide sur l’évaluation des solutions de rechange pour l’entreposage des déchets miniers, published as a revised version by Environment Canada in June 2013. Comments and requests for additional information on the September 2013 report were sent to Mine Arnaud in January 2014. The scope of the requests required that additional characterization fieldwork be carried out, which resulted in delays, as the work could only be done in the spring. Mine Arnaud is currently preparing a third version of the alternatives assessment report, which should be filed in September 2014.

20.2 Physical environment

The information in this section is a summary of the EIS and its supplements. For more details, or to consult the figures and tables, please refer to these documents (accessible on the Mine Arnaud website).

20.2.1 Air quality

Air quality in Sept-Îles is a source of concern for some residents. Sept-Îles is characterized by industrial activity linked to the presence of metallurgical facilities (i.e., pelletizing and aluminum plants) and a deep-water port for transhipment of large quantities of bulk goods, which could affect 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 heating of residences[9], are also contributing to the degradation of air quality.

MDDELLC has implemented an air-quality monitoring network in Québec. However, there were no monitoring stations on the North Shore, east of the Saguenay River prior to 2013. At the federal level, a similar network is managed by Environment Canada. The National Air Pollution Surveillance Program (NAPS)[10] sampling stations closest to the project site on the North Shore (Forestville[11] and Mingan[12] stations) are located in rural areas, about 250 km and 165 km respectively from the study area. These stations are therefore not representative of conditions in the study area.

MDDELLC conducted a study in 2013 to assess the current air quality situation. According to the 2013 data, the air quality in Sept-Îles is considered excellent. The study compared conditions in 30 cities in Québec, and the results indicated that the air quality is better in Sept-Îles than in all other cities subjected to such an assessment (presentation made by MDDELCC at the BAPE hearings in August 2013).

20.2.1.1 Modeling

Several atmospheric dispersion models have been produced for the Mine Arnaud project. The latest model, dated June 2013, serves as a reference for the project. Produced by Genivar (as of January 1, 2014, it became Groupe WSP Global Inc) using CALPUFF (Genivar, 2013), this model was deemed acceptable by the MDDELCC, who is currently analyzing it.

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[9]. Wood heating appears to be responsible for 46% of particulate emissions and 25% of emissions of volatile organic compounds (VOCs) in Québec.

[10]. The objective of the National Air Pollution Surveillance network is to provide accurate, long-term data on air quality obtained using standard Canadian methods. The network was established in 1969 to monitor and assess the quality of ambient air in populated areas of Canada.

[11]. Station 51901 is located in a rural forest area, 15 km from Highway 385.

[12]. Station 55601 is located in a rural undeveloped area.

All major sources of air emissions were included in the model, and two reference scenarios were used as a baseline: years 6 and 10 of the planned mining project. These two scenarios are considered conservative, but realistic. Year 6 was selected because of the relatively high tonnage—approximately 25.3 Mt—to be excavated during the year, and the fact that most of the work will be conducted aboveground during this period. Year 10 was chosen as the second scenario since it represents the maximum tonnage to be excavated in one year, according to the mine plan.

The modeling results show that in order to avoid exceedances, Arnaud Mine will have to implement mitigation measures such as frequently watering roads during dry periods. Since road dust is responsible for approximately 85% of emissions, Mine Arnaud decided to purchase two watering trucks to ensure that it can maintain dust control if one of the trucks breaks down.

The models indicate that even with the proposed mitigation measures, exceedances could occur, for less than 1% of the time. A *plan de gestion des opérations pour le projet Arnaud afin d'éviter les dépassements de particules totales dans l'air ambiant* [operations management plan for the Arnaud project to prevent exceedance of total particles in ambient air] was therefore prepared and filed in October 2013. The plan proposes operations management that avoids exceedances of the regulation at all times. Monitoring will be conducted during operation to validate the models.

20.2.2 Noise

Ambient noise levels were recorded on a continuous basis from 6 p.m. on July 19 to 6 p.m. on July 20, 2011 (Genivar, 2011), as well as from 10 p.m. on October 25 2012 to 7 a.m. on October 27, 2012 (Genivar, 2012). Readings were taken at four sites and MDDELCC used the results to establish the day- and nighttime noise levels to be complied with, in accordance with Directive 019.

The noise level criteria in residential areas were established at 42 dBA for nighttime and 49 dBA for daytime. Daytime noise limits were established at 49 dBA during operation and 55 dBA throughout construction. At night, the noise level limits were set at 42 dBA during operation and 45 dBA during construction.

The municipality of Sept-Îles has no regulations to govern noise level limits. However, the city does have regulations for nuisances, i.e., the *Règlement concernant la paix, le bon ordre et la sécurité publique* (No 2005-63) [regulation concerning peace, order and public safety], which include certain clauses that define ambient noise levels.

Day- and nighttime noise models were produced for key years of the project, namely, years -1, 1, 3 and 10. For each year: the number, type and locations of equipment were selected to realistically reflect the operating scenarios. Simulations have revealed the need for mitigation measures in order to comply with Directive 019 regarding noise emissions from the project.

To comply with the various regulations, the following main mitigation measures have been incorporated into the project:

- Construction of a screen berm
- No screen berm construction at night
- Installation of efficient mufflers on CAT785trucks
- Use of rubber liners in the CAT785 trucks
- Soundproofing of power *R-120* drills
- Use of smaller bulldozer (CATD7) for the construction of the screen berm
- Use of Volvo articulated trucks as they are quieter than other makes
- Installation of white noise backup alarms on mobile equipment

Noise simulations that took the mitigation measures into account have shown that noise generated from the mining site activities should meet the maximum level for residual noise (Genivar, 2012).

20.2.3 Vibrations

Mining the ore deposit will require blasting. Due to the proximity of residences and Hydro-Québec power lines north of the pit, special precautions will be required to avoid damage caused by blasting to the power lines and nearest residences caused by blasting. Simulations conducted done by SNC-Lavalin Environnement (SNC-Lavalin, 2011) have established specific drilling and blasting parameters to keep the vibration levels within acceptable limits.

According to the simulation calculations, blasting in the south sector of the pit should not be subject to constraints with regard to the control of vibrations and rock projections because the nearest residences are located more than 800 meters from the pit limits.

However, blast-induced air overpressure at the closest residences might exceed the limit of 128 dB if the blasting clearing axis is oriented towards the residences along Highway 138. Blasting will therefore have to be planned so that the clearance axis is perpendicular to Highway 138 (i.e., the free face is oriented towards the west or North to minimize the air overpressure at the closest residences). It would be advisable to mine the deposit in an east-west or north-south direction along the dominant ore lens, and directing the shots northward, except in the area of the Hydro-Québec power lines, where the height of the benches will have to be limited to 5 metres and the of blasting clearance axis oriented southward to avoid any damage to Hydro-Québec's infrastructure. Weather conditions will require that particular attention be paid to minimizing overpressure at the nearest residences while, to the extent possible, making sure that blasting is carried out when the following conditions are in place:

- Blasting mat used to limit rock projections
- Use of electronic detonators
- Use of emulsion explosive
- Monitoring of vibrations (installation of seismographs near Hydro-Québec towers and nearest residences)
- No blasting at night

- Limit blasting events to conditions of either clear sky or scattered clouds and where the cloud ceiling is more than 300 metres high
- Speed of wind blowing towards residences does not exceed 25 km/h.

According to the simulations, vibrations and the distance of projected rocks will comply with acceptable limits and will not cause inconveniences to the nearest residences or Hydro-Québec's infrastructure.

20.2.4 Hydrogeology and groundwater quality

As part of the environmental impact assessment, Ausenco-Vector completed a preliminary hydrogeological study of the site, to determine the hydrogeological conditions and characteristics of some hydrostratigraphic units at the location of the mining project (Ausenco-Vector, 2011). Thus, fieldwork provided data on water depth, the hydraulic properties of some hydrostratigraphic units in the area of the pit and information on the main hydrogeological systems. Hydraulic conductivity tests (packer tests) were carried out on the rock formation in the area of the pit. These tests provided the hydraulic conductivity of the formation at a given location. In addition, numerical modeling of the flow in the area of the pit was performed to verify the effects of dewatering during the period of operation. The modelling completed by Ausenco-Vector was based on the old configuration of the pit; the configuration has been slightly modified since.

In spring 2011, a geotechnical and hydrogeological study was also conducted by Journeaux and Associates (Journeaux, 2011) in the area of the future infrastructure and north of the Hydro-Québec power lines. Over 75 trenches and boreholes were completed and six monitoring wells were installed.

The information collected includes stratigraphic descriptions of the environment, geotechnical characteristics of certain materials, as well as preliminary information on groundwater quality.

WSP completed further work in 2012 and 2013 (Genivar, 2012 and 2013). This work was designed to obtain additional data on the stratigraphy of the site, the hydrogeological context and the hydraulic properties of the in-situ soil and rock. Additional boreholes and monitoring wells were completed. The geochemical characteristics of the groundwater were evaluated as well as the influence of tides on the flow regime in the area of the pit. Finally, two geophysical surveys, supervised by WSP, were conducted in 2013 to assess clay thickness and the depth of the rock on the shore of the Bay of Sept Îles.

The data collected in these studies, as well as those compiled during the the study of the fault systems have been incorporated into a numerical model. The objective of numerical modeling was to determine whether future dewatering of the pit would have an impact on the water table and the surrounding hydrological environment.

20.2.4.1 Groundwater flow regime

Groundwater in the area flows towards the Bay of Sept-Îles, following the local topography. Two fault systems (NE-SW and NW-SE) were identified at the site based on available geological information.

Locally, preferential flow in rock occurs in the direction of major fracture networks and groundwater ultimately empties into the bay. Calculated hydraulic gradients in the area range between 1.3% and 4.25%, with an average gradient of 1.9 %.

20.2.4.2 Geochemical characterization of groundwater

Considering that resurgence of groundwater in the study area could occur in surface water, the results of chemical analyses were compared to the criteria for *Résurgences dans les eaux de surface et infiltration dans les égouts* (RESIE) [resurgence into surface water and seepage into sewers] established by the MDDELCC in its policy on soil protection and restoration of contaminated soil. Potential receptors are lakes, streams and the Bay of Sept-Îles. The RESIE criteria are taken from the *Critères de qualité de l'eau de surface au Québec* (MDDEP, 2009) [criteria for surface water quality in Québec] established by the *Direction du suivi de l'état de l'environnement* [environmental monitoring division] of the MDDELLC. The value used for each parameter corresponds to the lowest of the following four values:

- 1 X CVAA (*CVAA: Critère de vie aquatique, aiguë* [criterion for aquatic life, acute]);
- 100 X CVAC (*CVAC : Critère de vie aquatique, chronique* [criterion for aquatic life, chronic]);
- 100 X CPCO (CPCO: *Critère de prévention de la contamination des organismes aquatiques* [criterion for the prevention of contamination of aquatic organisms]);
- 100 X CFP (*CFP: Critère de faune terrestre piscivore* [criterion for land wildlife and fish]).

Specifically, the comparison criteria are as follows:

Table 20-1: Comparison Criteria

Parameter	RESIE(1)	Directive 019(2)	Parameter	RESIE(1)	Directive 019(2)
Aluminum	0.75	-	Manganese	0.6	-
Silver	0.00004	-	Mercury	0.00013	0.1
Arsenic	0.34	5	Nickel	0.067	-
Barium	0.11	100	Lead	0.0044	5
Cadmium	0.00021	0.5	Selenium	0.062	1
Chromium	-	5	Uranium	-	2
Cobalt	0.37	-	Zinc	0.017	-
Copper	0.0016	-			

Notes :

1. Criteria for "Résurgence dans les eaux de surface ou infiltration dans les égouts"establishd in the MDDELCC policy.
2. Concentration (in mg/l) at of which tailings are considered high-risk according to Directive 019 sur l'industrie minière [Directive 019 on the mining industry]

The RESIE criteria for metals were adjusted according to a hardness of 10 mg/l, which is representative of water hardness in the surrounding streams. Given the low hardness measured in the host environment, the RESIE criteria for some metals are very low and even fall below the detection limits used by the laboratory. This is the case for cadmium.

Analysis results from the groundwater samples indicate exceedances in 12 of the 27 water samples collected in 23 wells for one or more of the following metals: Al, Ag, Ba, B, Cu, Mn, Ni, Zn. Results for all other metals are below the RESIE criteria.

20.2.4.3 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 MDDELCC groundwater classification system is a tool to ensure that use of the territory is in line with groundwater resources, in accordance with the classes listed below (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 community water supply)
- 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 m^2/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)

Based on the above definitions, the rock aquifers at the mine site constitute Class II aquifers that can be used to supply small wells. However, the water for the residents of the municipality of Sept-Îles is supplied 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 be considered Class I or II aquifers, 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.

Only 25 wells or boreholes have been reported and recorded in the MDDELCC's hydrogeologic information system (S.I.H.) for the Sept-Îles area. Their use (for drinking water or irrigation) is not known. However, all of these wells are more than one kilometre from the limits of the mine site, and none are located between the mine site and the bay.

20.2.4.4 Hydraulic properties

The permeability tests (packer tests) show mean hydraulic conductivities of 4.92×10^{-8} m/s for the clay layer, 2.34×10^{-7} m/s for the till layer and 1.97×10^{-8} m/s for the rock formation. Since these hydraulic conductivities measured in the rock formation are low, the water will tend to run off on the surface and infiltration will mostly take place in the most fracture zones. Few fractures were identified during exploration drilling, and the RQDs[13] are generally above 90%. Results from the Lugeon tests conducted in boreholes located in the fault zones have estimated an average hydraulic conductivity for the fractures of 1.23×10^{-7} m/s in the targeted horizons.

Monitoring of the wells in the area of the pit using Levelogger pressure sensors (made by Solinst) was performed to assess the influence of tides on changes in groundwater levels. Piezometric elevations were compared with tidal fluctuations. Water levels fluctuate in phase with the tidal cycle in only one well located within 200 m of the bay. None of the other wells in the main stratigraphic units of the sector (units of clayey silt and rock) appear to be influenced by the tide.

Based on the analysis of hydraulic properties and on other hydrogeological observations and geochemical characteristics of groundwater, it can be concluded that the hydraulic connections between groundwater and surface water are not significant. Indeed, overburden, when present, consists of low permeability material that restricts exchange between groundwater and surface water.

The geochemical signatures of groundwater and surface water are fairly different, which indicates that direct connections are not present. Poor drainage observed in the study area shows a slow flow and low infiltration rates in rock. Finally, water levels within the monitoring wells installed in the rock formation near the bay are not affected by the tides; only one well partially installed in overburden seems to react to tide changes.

20.2.4.5 Use of the water at the site

Water required for ore processing and other activities at the industrial site will be supplied by the storage pond collecting water extracted from the pit (i.e., from dewatering operations) as well as all contact runoff water. Reuse of this water makes it unnecessary to use a fresh water make-up, and greatly reduces the volume of effluent to be discharged into Clet Creek.

The water required for domestic needs will be provided through a connection to the city of Sept-Îles water supply system.

20.2.5 Hydrography and hydrology

20.2.5.1 Hydrographic network

The southern part of the future mine site (namely, the pit area) drains into the Bay of Sept-Îles via a series of brooks that flow more or less parallel to one another. The western portion of the mine site flows into

[13]. RQD: Rock Quality Designation

Clet Creek, which empties into the bay. The northern part of the site (including the waste rock dump[14] and a portion of the industrial area) is in the catchment area of an unnamed brook which flows into Rivière des Rapides [Rapides River], approximately 1 km upstream of the Bay of Sept-Îles.

20.2.5.2 Hydrological regime of watercourses

According to section 3.3.2.2 of Directive 019 on the mining industry, the environmental assessment of a mining project requires establishing the annual and summer minimum flows in the streams at the possible discharge sites for mining effluents in the host environment. Two locations, Clet Creek and an unnamed creek flowing into the Rivière Des Rapides, were considered for this study and Clet Creek was ultimately selected.

Most of the streams in the study area drain areas of less than 1 km^2 and are intermittent. Four streams drain larger areas: Clet Creek, R10 creek, R11 creek and Gamache Creek.

The study area's lakes are small and drain small watersheds (less than 1 km^2), with the exception of the Petit lac du Portage, which drains a watershed of 2.2 km^2.

The mining project will change the hydrological regime of rivers and lakes by reducing the size of the watershed. At the maximum extent of the project, the waterhseds of Clet, R10, R11 and Gamache creeks will be reduced by 64%, 46%, 68% and 27%, respectively. This will alter the concentration time of the watersheds, as well as runoff volumes and flow rates. These effects will be more significant during rainfall events that produce more than 5 mm of precipitation, since rainfall of less than 5 mm produces no significant runoff. River flows will change in proportion to changes in the watershed area.

In the case of lakes, the duration of these impacts will be limited to a few days. Indeed, the data show that following precipitation, lakes found their original level and output rate 4 to 7 days after the start of precipitation.

20.2.5.3 Downstream uses

Because water is supplied to the residences along Highway 138 through the municipal aqueduct, the surface water and groundwater downstream of the mine site are not used for any specific purpose, except by a small business (Le Végétarien), which uses surface water to water its gardens (flowers, fruit and vegetables).

[14] Surface drainage from the waste rock dump would naturally flow into the unnamed creek (at a point located slightly upstream of the confluence with the outlet of Lake Gamache) but will pool and form a small settling pond at the toe of a small dam before being pumped back into the tailing accumulation area.

20.2.6 Sediment and surface water quality

A characterization study of the surface water and sediment quality at the mining site was carried out in October 2010 and July 2011. About 20 sampling stations were set up to cover most of the watercourses and water bodies in the study area, including reference stations located upstream of the proposed infrastructure sites.

Additional fieldwork was carried out in lakes and streams at the site in 2012 and 2013 to support the fish and fish habitat characterization study, and to determine water quality in the Clet and R8 creeks. These activities included measuring the physico-chemical parameters of the surface water.

20.2.6.1 Surface water

The parameters used to monitor water and sediment quality were selected based on the recommendations made in Directive 019 governing the mining industry.

Surface water at the mining site is generally acidic and its pH value is almost always below the criterion for chronic effect (pH< 6.0) for the protection of aquatic life. Some values are even below 5. Dissolved oxygen concentrations vary greatly, but meet the MDDELCC criteria for the protection of coldwater biota. However, measurements taken under the ice in winter 2012 showed that the dissolved oxygen concentrations in four of the five lakes of were below the MDDELLC criteria. The electrical conductivity of the water is low and indicates the low electrolytes content in the water. These waters contain virtually no suspended solids. In addition, the water has low alkalinity values and hardness and is also low in nutrients such as nitrogen (Kjeldahl, nitrites/nitrates and ammonia). Although relatively low, phosphorus concentrations are quite variable between water bodies and in some cases, exceed the criteria for the protection of aquatic life.

In general, surface water contains few metals or metalloids, given that concentrations generally remain below the detection limits of the laboratory. This is the case for the following elements: Sb, Ag, As, Ba, Cd, Cr, Co, Cu, Hg, Mo, Ni, Pb, Se and Zn. However, the high aluminum content in all samples taken exceeds the chronic toxicity criteria for the protection of aquatic life; and in some samples it also exceeds the acute toxicity criteria for aquatic life. Iron also exceeds the chronic toxicity criteria (provisional criteria) for the protection of aquatic life in some streams. Finally, samples collected from Clet Creekin 2012 showed beryllium and lead concentrations above the chronic toxicity criteria for the protection of aquatic life.

20.2.6.2 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 below than the analytical 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 concentrations 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 cause rare effects on organisms. In addition, high calcium and iron content was noted in sediments collected at stream R8.

Sediments are generally characterized by a high proportion of sand, except in some rare instances. Streams R8 and R10 are characterized 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 in Clet Creek are characterized by a very low content of fine particles.

20.2.7 Biological environment

The information in this section constitutes a summary of the EIS and its supplements. For more details or to consult the figures and tables, please refer to these documents.

20.2.7.1 Flora and wetlands

The bay of the Sept-Îles is recognized on a provincial scale for the presence of wetlands of ecological importance. The ZICO[15] of Sept-Îles covers an area of approximately 242 km^2. With its marsh, the bay's salt meadows and islands and Checkley Plain, the ZICO constitutes an important area for water birds and the conservation of wetlands.

The presence of a wetland of major ecological significance within the study area (i.e., Checkley Plain, a flat peat bog officially recognized as being a natural reserve), as well as small peat bogs and riparian marshes near the mining site, justified conducting an in situ characterization and special inventory in October 2010 and September 2012, in order to evaluate the ecological value of the wetlands present where mining infrastructure is planned.

The study of wetlands in the study area and adjacent coastline of the Bay of Sept-Îles has revealed the presence of the following types of inland wetlands: beaver pond, shallow water (aquatic plant), marsh, marshy swamp, low shrub swamp, high shrub swamp, riparian open bog, pond open bog, uniform shrub bog, tree bog, uniform shrub fen and uniform open fen. Along the coastline, the following wetlands were identified: shallow water (marine eelgrass), low marsh, high marsh, low shrub swamp, high shrub swamp and rocky shore. In total, the 113 wetlands mapped (excluding the marine eelgrass) cover more than 425 ha (or7.3% of the study area).

[15] *(Zone Importante de Conservation des Oiseaux [Important Bird Area] – a science-based initiative to identify, preserve, and monitor a network of sites that provide essential habitats for bird populations)*

The broadest wetland areas mapped were wooded bogs (118 ha). Shrub swamps are second at 84 ha. These environments are rarely very large (Gamache Creek), but are spread out over the study area. The low marshes of the littoral zone are next in importance at 76 ha.

Wooded and open bogs dominated by sphagnum were also observed within the study zone. Fewer than 20 vascular species were recorded in the wooded bogs of the study area, which indicates an ecosystem of very low specific richness. Open bogs offer a greater specific richness, particularly when ponds are present within the bog. No special-status species were observed in the study area.

Marshes are dominated by herbaceous species, including carex and rushes accompanied by some low shrubs such as leatherleaf and sweet gale. A total of 32 vascular species were observed in the marshes but only 18 in the riparian bogs. The pioneering nature of these species reflects the relative youth of these environments. In the absence of further flooding by beaver, swamp leatherleaf and sweet gale will gradually invade these marshes. The same dominant shrubs are found in the riparian bogs, but the herbaceous plants and large sphagnum cover are typical of the bogs. The study area's riparian marshes and high shrub swamps are dominated by speckled alder and are primarily found on the banks of the main creeks such as Clet Creek.

The low shrub swamps are dominated by low shrub species, mainly leatherleaf and sweet gale. High shrubs such as speckled alder, larch and black spruce are present. A total of 54 vascular species are found in the low shrub swamps.

In the Bay of Sept-Îles, the structure of the littoral plant community corresponds to a succession of salt meadows, salt marshes, mud flats without vegetation, eelgrass beds and, finally, algae beds.

A large population of twin-scaped bladderwort *(Utricularia geminiscapa)*, a plant species likely to be designated threatened or vulnerable in Québec (CDPNQ, 2008), was identified in the marshy swamps in lake PE-7.

20.2.7.2 **Fish**

Fieldwork was carried out between 2010 and 2014. Ten lakes and more than thirty streams were surveyed during this period. In general, most of the rivers and watercourses at the mine site and in the region shelter populations of brook trout and/or stickleback (three- or ninespine). Brook trout is fished for sport, particularly in the Matimek Zec.

In regard to special-status species, the American eel is found in the Rivière Hall, Lac des Rapides [Rapides Lake] and Rivière des Rapides [Rapides River]. This species is not found in the water bodies or creeks located on the project site and will not be impacted.

20.2.7.3 **Birds**

Bird inventories were conducted in 2011 and 2012. Existing regional data were also used to complete the studies. According to available data and fieldwork, the Arnaud Township sector is likely to be frequented by nearly 200 species of birds on an annual basis. Inventories during nesting and migration helped identify 92 species. Among them, nesting was confirmed for 14 species and considered likely for 11 species and possible for 51, while 16 species were observed in unsuitable habitat or nesting outside their breeding season.

In regard to special-status species, the olive-sided flycatcher and the common nighthawk were observed during inventories.

Grass beds in the Bay of Sept-Îles are recognized as reproduction and staging areas for waterfowl. Most of these grass beds, composed of Spartina alterniflora, are located to either side of the Rivière des Rapides, i.e., between Clet Creek and the Rivière au Foin [Hay River].

In terms of the inland portion 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 creek and the peat bog upstream of Lac à l'Anguille) represent favourable habitat for waterfowl breeding. However, these habitat areas are smaller than those in the bay.

20.2.7.4 **Bats**

Survey results suggest that, although the study area itself does not offer a large amount of suitable habitat for bats, they nevertheless use it to travel between resting, breeding and feeding sites. In fact, the cliffs located on the north shore of Lac Hall (north of the study area) are bordered with mature forest stands that provide prime habitat for breeding bats, in addition to the extensive mature forest stands associated with chains of lakes located to the east. Furthermore, the mature stands and wetlands bordering the study area to the west and the large adjacent bog areas constitute prime feeding sites for this insectivorous species.

20.2.7.5 **Amphibians**

No species of anuran or urodela of special or endangered status were identified in the study area. Only some common and widely distributed species in Québec were detected or observed on site. The *Centre de données sur le patrimoine naturel du Québec* (CDPNQ) [centre for data on the natural heritage of Québec] does not report any species designated as special, threatened, vulnerable, or likely to be designated as such in the project area (Roche, 2012).

20.2.7.6 **Mammals and micromammals**

Land wildlife species that frequent the future mine area or its vicinity include moose, black bear, weasel, marten, squirrel, otter, hare, red fox, muskrat, beaver and mink. The majority of these species have been hunted or trapped over the last five years, weasel and marten being the species captured most often.

Because of the presence of human activities (Highway 138, power lines, railroad, ATV/snowmobiles), the mine area does not provide a good habitat for moose. In addition, the map of environmental elements sensitive to the establishment of electric infrastructure produced by Hydro-Québec (Hydro-Québec, 1987) does not report any wildlife habitat of particular potential for moose in the study area. However, according to the representatives of the Matimek Zec, the Lake Hall sector, located to the northwest of the study area, may offer some potential for moose hunting.

In the case of micromammals, the results obtained from the inventory studies suggest that the areas surveyed (which are representative of the study area) do not offer particularly good habitat in terms of population density or diversity of micromammal species. The most interesting areas are represented by mature forest stands near rivers, which is not particularly surprising, given the habitat preferences of the species that may be present.

Regarding special status species, the rock vole, which appears on the list of species likely to be designated threatened or vulnerable in Québec, was the only species identified during the inventories.

20.2.8 Human environment

The Mine Arnaud project is located approximately 15 km from the city of Sept-Îles and the Innu communities of Uashat mak Mani-Utenam by road, (approximately 9 and 7 straightline kilometers respectively), in the Côte-Nord [North Shore] administrative region of the Province of Québec. The region is divided into six regional municipalities (RCMs, or Regional Municipal Counties), including the Sept-Rivières RCM, the main city of which is Sept-Îles.[16]

Covering an area of more than 32,000 km^2, the Sept-Rivières RCM is characterized by a sparsely populated coastal strip and a vast hinterland mainly devoted to the exploitation of natural resources (forestry, hydropower, wildlife harvesting, etc.). In addition to Sept-Îles, the RCM also includes the town of Port-Cartier to the west and the undeveloped area of Lac Walker and Rivière Nipissis to the north.

[16] As a result of the municipal mergers that occurred in 2003, the city of Sept-Îles now includes the former municipalities of Gallix and Moisie.

Moreover, the study area overlaps two Innu [17] communities known as Uashat mak Mani-Utenam (Uashat and Mani-Utenam). Although separated by some 15 km, the two communities are grouped under one council forming a single band.

20.2.8.1 **Demographics**

Sept-Îles

At the time of the 2006 census, Sept-Îles had a population of 25,725, which is slightly lower than that recorded in the census done by Statistics Canada in 2001. Since then, according to estimates published by the Institut de la statistique du Québec [statistical institute of Québec], the population of Sept-Îles has increased steadily every year and reached 26,196 in 2010. This contrasts with the situation in the rest of the North Shore region, where the regional population declined over the same period.

The city’s population has a profile somewhat similar to those of the Province of Québec and Canada in terms of its breakdown by age groups. Individuals under the age of 25 make up slightly less than one-third of the total population. However, as in the rest of Québec the population is aging and this could eventually be reflected in the number of available workers.

Although 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 is much lower (9.6% vs. 16.5% in Québec as a whole). One-third of the population aged 15 years and over does not have a diploma.

Current economic turmoil in Sept-Îles and the surrounding area has had a direct effect on the main economic indicators of the population. In 2009, the local labour force comprised about 17,000 individuals, but 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 in North Shore region, and some observers even comment of a situation which tends more and more towards full employment. Such a context 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

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[17] Previously known as Montagnais, the Innu (which means “human being” in their language, Innu-Aimun) are the Aboriginal inhabitants of an area they refer to as Nitassinan, which includes most of the northeastern portion of the Province of Québec and some of the western part of Labrador.

population continues to increase, but at a slower pace than that recorded between 1980 and 2006, when it more than doubled.

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 increased at all academic levels.

Data from the 2006 census point to a relatively high level of economic activity but high 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 and creates mostly seasonal jobs. Uashat mak Mani-Utenam has crab and lobster fishing, shrimping and demersal fish trawling fleets. The community also operates a seafood processing 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.

20.2.8.2 Economic conditions

The economic structure of the Sept-Rivières regional municipality is mainly based on the exploitation and, to a lesser extent, transformation of its natural resources. Mining, forestry, hydroelectric power, fishing and hunting and 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 production, 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 brought the city 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 also closely linked to mining activities taking place at the iron ore deposits of Fermont and Schefferville in Northern Québec, as well as Labrador City and Wabush, in the Province of Newfoundland and Labrador. The Iron Ore Company of Canada (IOC), Cliffs Natural Resources (Wabush Mine and Bloom Lake Mine – recently bought from Consolidated Thompson) transport their respective production to the port terminals, located on both sides of the Bay of Sept-Îles, for shipment to the world steel industry.

Over the years, the business relationships developed with mining companies operating in the hinterland and Northern Québec have resulted in the development of several specialized small and medium enterprises (SMEs) that offer 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 and supply companies that not only serve local and regional customers, but now export their expertise to Mexico, the United States, Brazil, Scandinavia, Russia, India, and elsewhere throughout the world in the areas of technology, products and services, especially for companies involved in iron ore mining.

Finally, the tertiary sector of the Sept-Îles economy is also well organized with the Center for Health and Social Services Sept-Îles, various educational institutions and government offices (federal, provincial, regional and local), which are major employers in the services domain. In addition, the city boasts several shopping centers, retail stores, professional offices and financial institutions. Overall, the trade and services industry employs about 10,000 people locally.

20.2.8.3 Land use and occupancy

Most of the project site being developed by Mine Arnaud is located within the boundaries of the city of Sept-Îles, it partially overlaps the undeveloped territory of Lac Walker, but is well away from the urban centre located about 15 kilometres to the east.

A sparsely populated residential area (about fifty residences) stretches along Highway 138 between Rivière Hall to the west and Rivières 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) located just east of Clet Creek.

The northern part of the future mine site intersects with the Matimek ZEC (zone d'exploitation contrôlée, or controlled harvesting zone), an area dedicated to hunting, fishing and recreation activities. The southern part of the ZEC is accessed from Highway 138 via a logging road leading to Lake Hall, or via a gravel road (known as Chemin Allard), which crosses the Arnaud property and leads to the lakes in Sept-Îles' hinterland. Some of the lakes and rivers within the limits of the study area, including those in the Gamache sector, are also used for sport fishing. Their adjacent lands are popular with big and small game hunters.

The project site also overlaps with three traplines (Nos. 09-11-320, 09-11-321 and 09-11-0326), where furbearing animals are trapped by non-Aboriginal residents of the Sept-Îles area. Inventories have also identified some hunting and fishing camps and lodges within the study area.

Study on Aboriginal Land Use and Wildlife Harvesting

Part of the following section was taken from Supplement No. 11 of the Environmental Impact Statement (EIS) (Chapter 9 – questions 109, 110 and 111).

In its Environmental Impact Statement (EIS), the proponent characterized the Uashat mak Mani-Utenam Innus' use of the territory based on a review of the literature, the most recent data of which dates from 2004–2006 for linear development projects[18]. According to these studies and given that the project is located outside of the Sept-Îles division of the Saguenay beaver reserve; the mining facilities will have no impact on any traplines or family lands belonging to the Uashat mak Mani-Utenam Innu.

Furthermore, meetings with certain land users (such as Jean-Guy Vigneault, a trapper who owns a furbearer management unit) confirmed that the Innu make little or no use of the project site and have no permanent facilities there. It should be noted that Mr. Vigneault, who has been trapping furbearers in this region for over 50 years, has never seen any Innu or found evidence of Innu activity of any kind in this area. Since goose hunting usually takes place along the Bay of Sept-Îles rather than on the Mine Arnaud project site (pers. comm.), this activity will not be affected by future mining operations.

For the purpose of validating the data, Mine Arnaud sought to obtain official information from Innu Takuaikan Uashat mak Mani-Utenam (ITUM) about the traditional activities practised at the proposed project site. In order to accomplish this and in accordance with the guidelines set out in the EIS, Mine Arnaud mandated ITUM to carry out a study of the Innu community. Among other things, the study had to include a section on "Land Occupation." During hearings held by the BAPE (board of public environmental hearings) in 2013, Mine Arnaud learned that no report would be filed or made public by the mandated consultant (Serge Ashini Goupil).

Trans-Québec snowmobile trail 3 runs parallel to Highway 138 within the corridor of a 161-kV power transmission line. From there, the trail runs inland to the Lac des Rapides sector. Outside of the winter season, the trail is also used by ATV enthusiasts.

Lastly, the study area crosses six power transmission and distribution lines in two different corridors.

20.2.8.4 **Transportation**

The road system in the Côte-Nord [North Shore] administrative region is essentially made up of Highway 138, a major national highway that spans more than 800 km between Tadoussac and Natashquan. Highway 138 is the only major road along the North Shore and crosses the southern portion of the study area. Average traffic along this road is estimated at a little less than 5,000 vehicles per day.

The secondary road system starts at Highway 138. The Route de la Pointe-Noire [Pointe-Noire road] provides access to the Cliffs Natural Resources pelletizing plant in the Pointe-Noire sector of the Port of Sept-Îles, as well as to the Alouette aluminum smelter.

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[18] These are the environmental assessment of the Romaine complex and Minganie power transmission system expansion project, by Hydro-Québec

A railway line operated by the Arnaud Railway (owned by Cliffs Natural Resources) crosses the study area. The line is about 38 km long and links the industrial infrastructure at the port of Pointe-Noire with the regional railway system (QNS&L, or Quebec North Shore and Labrador Railway). Every day, the train makes four return trips between Pointe-Noire and Sept-Îles Junction (QNS&L terminals). Information obtained from the company indicates that about six million tonnes of concentrate were transported on the rail line every year from 2006 to 2010.

Sept-Îles' strategic location at the entrance to Gulf of St. Lawrence and the St. Lawrence Seaway results in Sept-Îles special status as a major hub for the international shipment of bulk goods. The Port of Sept-Îles, which is accessible year-round because of its deep, semi-circular bay 10 km in diameter, handles approximately 24 million tonnes of merchandise per year. With potential new iron ore production from new deposits, the port could receive 30 million metric tonnes from the Fermont-Labrador City and Schefferville areas over the next few years. Furthermore, if various major projects currently under development in the Schefferville region (Taconie, Kémag and LabMag) get underway, they could generate an additional 22 million metric tonnes by 2016, which would bring total shipments to nearly 80 million metric tonnes per year. At this rate, Sept-Îles would soon become the largest mining port in North America.

Lastly, Sept-Îles airport is the hub for air transportion services to and from the North Shore region. With daily connections to Québec City and Montréal, the airport also serves Northern Québec and Labrador. Sept-Îles is now one of the busiest airports in Eastern Canada and is still growing.

20.2.9 Stakeholders and consultation with the communities

In order to provide a thorough understanding of its planned open-pit mine and the project's potential impacts, Mine Arnaud began to hold discussion meetings with the different communities in the region in 2008. The meetings were held in the early stages of development in order to establish a forum for discussion and to keep all local stakeholders with direct or indirect ties to the project as thoroughly informed as possible.

The following stakeholders have been identified and consulted over the past few years:

- Members of the community of Uashat mak Mani-Utenam
- Members of the community of Sept-Îles (vacation camp owners, Matimek ZEC (controlled harvesting zone), trappers, residents of Arnaud Township, owners of the Quad/ATV and snowmobiles club, etc.)
- Local elected representatives
- Local entrepreneurs and regional economic stakeholders
- Representatives of various interest groups
- Representatives of environmental agencies (CRÉ [Regional Conference of Elected Representatives], OBV [watershed protection group], CPESI [Sept-Îles Environmental Protection Corporation], CPQAE [Air and Water Quality Protection Committee] and RSGBSI [Coalition for the Protection of Sept-Îles Bay])

- Representatives of certain segments of the media
- Provincial and federal government representatives

The one-on-one or group meetings were held before, during and after the impact assessment process. Discussions are still ongoing and are now overseen by a monitoring committee. The exchange of information has been supported by a number of e-mails, letters and documents.

20.2.9.1 Consultation with the community of Sept-Îles

In 2008, since the economic climate was favourable for phosphorus and fertilizer production, the shareholders re-evaluated the Mine Arnaud project and made the decision to go ahead with development. A number of meetings were subsequently held with the different communities concerned. A detailed list of stakeholders with a potential interest in the project was drawn up in 2010. The list has been continually updated and expanded as the project evolved.

Meetings in 2009

As of 2008, information on various topics, including the possibility of mining development began to be disseminated to the population through the media. In 2008 and 2009, representatives of the Société générale de financement du Québec (SGF) [General Investment Corporation of Québec] held several meetings with the mayor of Sept-Îles, representatives of the Sept-Îles Port Authority and Hydro-Québec, as well as certain stakeholders from the industrial sector of Sept-Îles. The objective of the meetings was to present the Arnaud Mine project and examine the economic and logistical impacts of developing such a project in the region.

Meetings in 2010

In June 2010, a series of meetings was organized to inform and consult certain local stakeholders about the project. The meetings were held with representatives of the Sept-Rivières RMC (Regional County Municipality), the Sept-Îles Chamber of Commerce, the Sept-Îles Port Authority, Sept-Îles Economic Development, the city of Sept-Îles, the town of Port-Cartier and the North Shore regional conference of elected representatives. In addition, a meeting was held with Mark Fafard, spokesman for Sept-Îles sans URANIUM (SISUR) [Sept-Îles without uranium]. Mr. Fafard also visited the site and the Mine Arnaud warehouse that year.

Three public meetings were also held at the Arnaud Township Chapel on August 4, September 14 and November 16, 2010. They provided opportunities to meet and hold direct discussions with some of the township's residents.

Meetings in 2011 and 2012

Between 2011 and 2012, Mine Arnaud held dozens of meetings with individuals or groups to gather their comments concerning the mining project. The following table provides a summary of what these meetings entailed:

Table 20-2: Summary of Meetings held in 2011

Date	Individuals or groups consulted	Comments
January 19, 2011	*Sept-Îles Environmental Protection Corporation (CPSES*)	The objective was to respond to a series of previously submitted questions.
June 16–17, 2011	DESI, Maniketish school at Uashat, Uashat Band Council, Gervais Gagné (City Councillor for Arnaud Township), Jean-Marc Cassista (Chairman of the Arnaud Township Citizens' Committee), Jean-Guy Vigneault (trapper), Colette Arsenault (resident), Lorraine Richard (Deputy for Duplessis)	Discussion on some of the project's potential impacts (economy, workforce, housing, land use, relocation of some residents, etc.)
July 13–14, 2011	DESI, mayor of Sept-Îles	Working and information meeting
November 11–16, 2011	Residents' interest groups	
December 12–14, 2011	ITUM, MRN, MDDELLC, RMC, Residents, Media, Land Users, ZEC	
December 19–21, 2011	Nicole Harvey (resident) and Jean-Guy Vigneault (trapper)	Discussion on drilling, logging and the project's impacts on the territory (buyback, relocation, claims for compensation, etc.)
January 11–13, 2012	Sept-Îles Port Authority, Sept-Îles Environmental Protection Corporation (CPSESI), residents of Arnaud Township, Québec Department of Natural Resources and Wildlife, vacation camp owners, ATV and snowmobile clubs	

Several participants in these meetings highlighted the positive impacts of the project such as job creation and economic diversification in the Sept-Îles region. However, a number of issues were submitted to Mine Arnaud representatives. They are summarized as follows:

- **Water quality** (Lac des Rapides, where the city's water intake is located, Rivière des Rapides (Rapides River), the Bay of Sept-Îles, compensation for loss of fish habitat)
- **Air quality** (noise and dust affecting vacation camps in the Lake Gamache area)
- **Noise and vibrations** (Lake Hall campground vs. work on the dikes, noise affecting vacation camps in the Lake Gamache area)
- **Environment** (cumulative effects, the need for mitigation and compensation measures for wildlife habitats)
- **Employment** (jobs, workforce availability)
- **Housing** (lack of housing, higher housing costs)

- **Land use** (loss of land use, snowmobile trail relocation that is acceptable to the two clubs and ZEC, compensation to Mr. Vigneault for loss of land use, access for owners of vacation camps, relocation/compensation for loss of hunting camps)
- **Compensation to Arnaud Township residents** (adequate compensation to homeowners in the township)
- **Economic impacts** (negative impact on the economy, particularly in terms of tourism, cruises and sightseeing tours)
- **Health and safety** (physical and psychological well-being)
- **Innu involvement**

Pre-consultation Meetings (Preparation for BAPE Hearings)

After filing its Environmental Impact Statement (EIS) in March 2012, Mine Arnaud initiated a pre-consultation process to prepare for the BAPE hearings. Mine Arnaud opted to work with the firm Transfert Environnement to establish the pre-consultation process, which enabled concerned individuals and groups to become familiar with the EIS and participate in the project's environmental assessment by expressing their comments and concerns before the evaluation process was completed.

At that time, Mine Arnaud also decided to establish an environmental policy to provide a framework for its environmental initiatives.

The pre-consultation meetings took place in three phases:

- Sector-based meetings: discussion meetings between Mine Arnaud and representatives of the main sectors (municipal affairs, environment, economic environment, community groups, public health, abutting land owners, unions, education community, Innu community, recreation and tourism), which made it possible to group the issues expressed since 2010 into key topics
- Thematic workshops: open to all, the workshops were held in the presence of Mine Arnaud representatives to allow for discussion of previously identified main topics (infrastructure, landscaping and development, air and water quality, noise, vibrations and traffic, impacts and socioeconomic benefits)
- Feedback and validation session: after completion of the first two phases, a pre-consultation report on the results of the exercise was produced and subsequently presented to the population at an evening information session

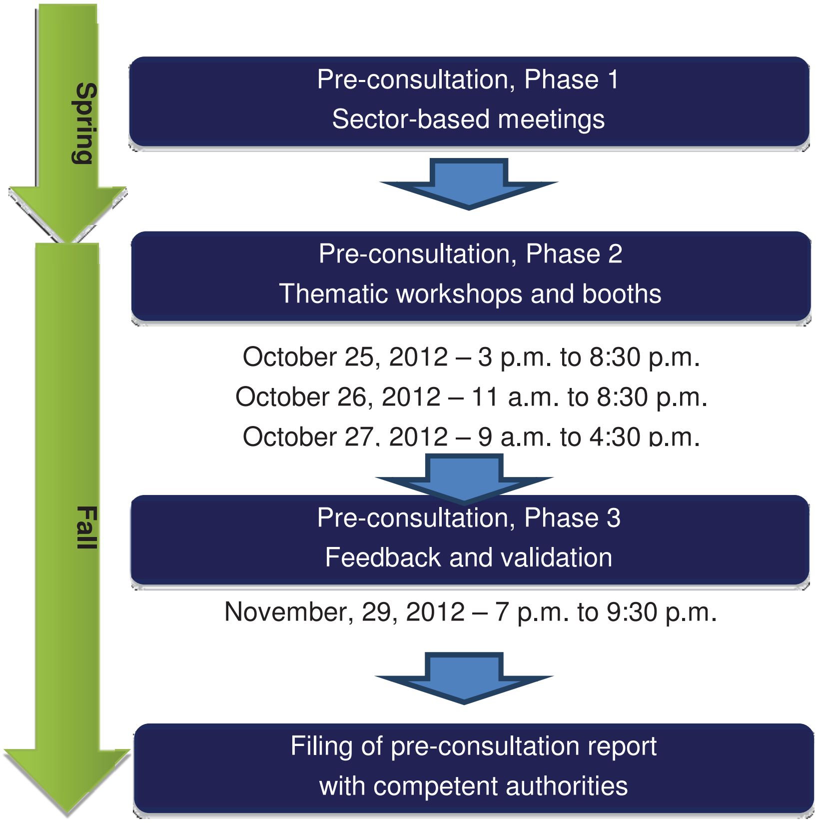

Figure 20-1: Pre-consultation Schedule

The pre-consultation meetings focused on the following main issues or project components:

- Infrastructure and activities
    - Access routes
- Socioeconomic benefits and impacts
- Air quality
- Water quality
- Quality of life
    - Noise
    - Vibrations
- Information and participation
- Agreements and acquisitions
- Corporate matters
- Landscape
- Wildlife and vegetation

The pre-consultation report served as a basis for incorporating as many of the participants' comments as possible into the pre-consultation activities as part of the environmental impact assessment process. During the feedback and validation session, the participants were also consulted about the means through which they wanted to be informed on and participate in the project's development.

Framework for Agreements and Acquisitions

On June 20, 2012, a framework for agreements and acquisitions was presented to the residents of Arnaud Township. The meeting was attended by 60 of the 220 people invited.

20.2.9.2 Relations and communication with the communities of Uashat mak Mani-Utenam

The communities of Uashat and Mani-Utenam are respectively located about 7 and 20 km from the project, as the crow flies. Mine Arnaud considers the Innu to be major stakeholders in the equation and vital to the success of the project. This is why the proponent began holding discussions and consultations about the project with the Innu community in 2010 and continues to maintain a positive dialogue with them on a regular basis.

Discussion and Communication

The first official contact with Uashat was made in June 2010, with a letter sent to Chief Georges-Ernest Grégoire informing him of Mine Arnaud's plan to develop an open-pit mine on the community's ancestral Nistassinan lands. Since 2010, the proponent has held periodic meetings with the members of the Band Council and residents of Uashat mak Mani-Utenam to discuss a variety of topics, including traditional activities. More than a hundred exchanges have taken place between the community and the proponent (through e-mail messages, meetings, telephone calls, site visits, letters, etc.). No agreement has been signed as yet.

Three information sessions were organized for the Uashat mak Mani-Utenam communities. The first two took place in Mani-Utenam and Uashat on the evenings of February 27 and 28, 2012, respectively. In total, some 15 people attended the sessions, during which the following main concerns were discussed:

- Land claims and access to the territory
- Impacts on the environment and human health
- Employment and workforce training
- Housing

Mine Arnaud returned to Uashat on August 19, 2013, to continue the consultation process. About a dozen people attended the meeting.

Table 20-3: Summary of Relations with ITUM

Date	Activity	Details
August 2010 to September 2012	Periodic meetings with ITUM	Presentation of the project notice and feasibility study Project status report Presentation of the Environmental Impact Statement (EIS) Financial assistance from Mine Arnaud to enable ITUM to review the EIS and prepare consultation activities Presentation of the pre-consultation process Preparation of information sessions Discussion on negotiation of an agreement between ITUM and Mine Arnaud concerning impacts and benefits
September 2012 to December 2012	Suspension of discussions between ITUM and mining companies	ITUM wishes to re-examine its vision of land development with the community ITUM and Mine Arnaud continue to communicate by e-mail during this period, but all planned meetings are cancelled
December 2012 to April 2013	ITUM now ready to continue its evaluation of the Mine Arnaud project	Consultation with the community resumes: ITUM invites Mine Arnaud to present its project to the population Discussion on negotiation of an agreement between ITUM and Mine Arnaud concerning impacts and benefits
February 27 and 28, 2013	Two information sessions	Presentation of the project to two communities: Total of 15 people in attendance
April 2013	Mike McKenzie elected as the new Chief A first meeting is held with Chief McKenzie and certain members of the Band Council on June 20, 2013	
April 2013 to present	Periodic meetings with ITUM	Project status report Preparation and organization of the first information session Discussion on negotiation of an agreement between ITUM and Mine Arnaud concerning impacts and benefits Meeting to establish a means of disseminating project information to the Band Council and Uashat mak Mani-Utenam communities
August 19, 2013	Information session	Presentation of the project to Uashat community with a total of 12 people in attendance

20.2.10 Archaeological resources

The information summarized in this section was drawn from studies conducted by Jean-Yves Pintal (1997 and 2011).

Separate but complementary research methods were used to conduct the study on prehistoric and historic components in the target area.

The database of the Inventaire des sites archéologiques du ministère de la Culture, des Communications et de la Condition féminine [survey of archaeological sites by the Québec Department of Culture, Communications and the Status of Women] identifies 44 archaeological sites in the vicinity of Sept-Îles.

One is located within the study area; it is the remains of an old whaling post established southeast of Pointe Noire.

With respect to the potential of Amerindian occupation, 41 areas have been identified within or in the immediate vicinity of the study area.

The potential for Euro-Canadian occupation was assessed by analyzing cadastral and topographic maps dating from before 1950. Eleven potential areas were located within or just outside the study area. This potential is much more important around the periphery of the Bay of Sept-Îles.

Of all the potential sites identified above, one Amerindian archaeological site is the only one located within the mining infrastructure. The potential site is located along a creek south of the screen mound. In the case where archaeological sites would be discovered, work would be interrupted and the Ministère de la Culture, des Communications et de la Condition féminine [Québec Department of Culture, Communications and the Status of Women] would be notified.

20.3 Characterization of ore, overburden, waste rock and tailings

This section summarizes the results of the analyses conducted in 2011 and 2013 of the material to be excavated (overburden, waste rock and ore) and of tailings[19] (solid and liquid phases) that will be generated. These analyses were performed to determine their storage method. The specific objectives of this geochemical characterization program were to:

- Classify waste rock according to the Québec Directive 019 on the mining industry and the Guide de valorisation des matières résiduelles inorganiques non dangereuses de source industrielle comme matériau de construction [guidelines for reclaiming non-hazardous inorganic waste such as construction material] for waste management planning (including possible avenues for reclamation and reuse)
- Determine probable mining effluent quality and identify chemicals of environmental interest related to water and mine waste management

The methods used to characterize material are consistent with the above-mentioned guidelines (Ministère de l’Environnement du Québec [Québec Department of the Environment], 2002) and with the most recent methods advocated by the Centre d’expertise en analyse environnementale du Québec (CEAEQ) [Québec Centre for Expertise in Environmental Analysis].

[19] Ore samples were treated in pilot plants to determine the best ore processing method. During the pilot phase, tailing pulp samples were collected.

The elemental composition of the material was compared to the MDDELLC's Politique de protection des sols et de réhabilitation des terrains contaminés [policy on soil protection and restoration of contaminated sites] (MDDEP Policy, 1998, revised in 2001). Leachates generated during TCLP leaching tests (EPA 1311) were compared to Table 1 in Appendix 2 of Directive 019. Leachates were also compared with the criteria for the protection of groundwater from the RESIE policy [resurgence into surface water and seepage into sewers] as presented in section 20.2.4.2 Geochemical characterization of groundwater. The results obtained in 2011 and 2013 and presented in this section were thus compared to the same criteria. However, it is noted that the detection limits of the analytical methods used in 2011 differ from those used in 2013, sometimes making the comparison with the existing criteria not relevant.

Leaching tests to simulate natural conditions were also performed on the samples tested for TCLP. Thus, leaching SPLP tests (EPA1312) simulating acid rain as well as CTEU-9 tests simulating water leaching have been conducted. These analyses allowed for classification of residual material according to the guidelines prepared for this purpose.

20.3.1 Characterization of ore

20.3.1.1 Elemental composition

Most of the parameters analyzed on the ore samples showed results below the Policy's A quality criteria offset, with the exception of cobalt, copper, manganese and nickel, which presented concentrations in excess of their respective A criteria and sometimes, B criteria (copper, cobalt and manganese).

It should be noted that the MDDELLC is reviewing the manganese background levels.[20] The value corresponding to criteria A should be revised from 1,000 ppm to 1,445 ppm. According to this new value, there would be only one ore sample with a concentration exceeding criterion A.

20.3.1.2 Leaching tests

The concentrations of leachates produced from TCLP, SPLP and CTEU-9 tests were below the limits set out in Directive 019 for defining high-risk materials. Therefore, the ore was not classified as a high-risk material.

All leachate parameters tested met the applicable criteria with the exception of copper, manganese and barium, which exceed the RESIE criteria, especially for the TCLP test. Aluminum, cobalt and nickel also exceed these same criteria for some CTEU-9 and/or SPLP tests.

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[20] MDDEP, Cadre de gestion des teneurs naturelles en manganèse dans le sol, 5^{e} conférence environnementale MAXXAM, Québec, September 20, 2012.

As for mercury, the only two results exceeding the criterion were not representative of the majority of the results and were too close to the analytical detection limit to be meaningful.

20.3.1.3 Acid generation potential

A material (such as ore, waste rock or tailings) can be defined as potentially acid generating when:

- It contains sulphides in concentrations above 0.3%, and
- It shows a net acid neutralization potential[21] below 20 kg $CaCO_3$/t or an ANP:AGP ratio below 3.

These results showed that the ore has no acid generation potential, and its total mean sulphide content (0.19%) remained below the 0.3% criterion.

20.3.2 Characterization of overburden

20.3.2.1 Elemental composition

The results show that all of the parameters analyzed for the 2011 and 2013 samples were below the applicable criteria except for barium, which sometimes exceeded the provincial soil protection A criterion. However, all barium concentrations were below the B criterion.

20.3.2.2 Leaching tests

The concentrations of all leachates produced from TCLP, SPLP and CTEU-9 tests were below the limits set out in Directive 019 for defining high-risk materials. Therefore, the overburden was not classified as a high-risk material.

The analytical methods used in 2011 did not allow for comparison of all the parameters with their respective RESIE criteria, since some detection limits were not sufficiently low. However, when comparison was possible, the parameters measured in the leachates were below applicable environmental criteria with the exception of copper, aluminum, barium, manganese, mercury and nickel, which sometimes had concentrations above the provincial soil protection criteria for resurgence in surface

[21] 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 $CaCO_3$/t. A static acid generation potential test, such as the modified Sobek test used in this project, allows determining both the ANP and AGP of a rock sample.

water. It should be noted that the mercury results were too close to the analytical detection limit to be meaningful.

Only SPLP tests were conducted in 2013, this method being considered more representative of the surrounding environment (water in lakes and streams is slightly acidic). Much lower analytical detection limits allowed for comparison of all parameters with their respective RESIE criteria. As for the tests performed in 2011, concentrations of aluminum and, in some cases, of copper, exceed their respective criteria. All other parameters were below the applicable RESIE criteria.

20.3.2.3 Acid generation potential

Acid generation potential testing was not performed on overburden as this fine, oxidized material is not likely to generate acid, especially considering the inert properties of the bedrock.

20.3.2.4 Reuse of the overburden

Barium was the only parameter above the A criteria set out in the Policy. However, all concentrations were below the B criteria. The TCLP tests showed concentrations above the RESIE criteria for barium but no exceedance was observed for the SPLP test, which was considered more representative of site conditions.

The overburden can therefore be used onsite for infrastructure construction and site restoration.

20.3.3 Characterization of waste rock

20.3.3.1 Elemental composition

Some of the metals analyzed in the waste rock samples (manganese, cobalt, selenium and/or copper) exceeded the provincial A criteria and sometimes exceeded the B criteria for soil protection. All other parameters were below the A criteria.

20.3.3.2 Leaching tests

The concentrations of all leachates produced from TCLP, SPLP and CTEU-9 tests in 2011 and 2013 were below the limits set out in Directive 019 for defining high-risk materials. Therefore, the waste rock was not classified as a high-risk material.

The analytical methods used in 2011 did not allow for comparison of all the parameters with their respective RESIE criteria, since some detection limits were not sufficiently low. However, when comparison was possible, the parameters measured in the leachates were below applicable environmental criteria with the exception of copper, aluminum, barium, manganese and mercury, which sometimes had concentrations above the provincial RESIE criteria.

It should be noted that the mercury results were too close to the analytical detection limit to be meaningful.

Only TCLP tests were conducted in 2013. Much lower analytical detection limits made it possible to compare all parameters with their respective RESIE criteria. The tests performed in 2011 revealed aluminum concentrations that exceeded the criteria for this metal. All other parameters were below the applicable RESIE criteria.

20.3.3.3 Acid generation potential

With the exception of one of the 12 samples, the results of the tests performed on the waste rock in 2011 showed that this material had no acid generation potential. Based on the average values obtained in the results, waste rock showed no potential for acid generation, with a net acid neutralization potential of 21.6 kg $CaCO_3$/t and an average total sulphide content of less than 0.2 %.

20.3.3.4 Reuse of waste rock

Four parameters (manganese, cobalt, selenium and copper) showed concentrations above the A criteria set out in the Policy. Among them, only copper and manganese exceeded the RESIE criteria in TCLP tests performed in 2011. The SPLP tests, which were more representative of site conditions, showed manganese exceedance in only one sample. The TCLP tests conducted in 2013 only showed exceedances for aluminum.

The waste rock was not considered to be high-risk or acidogenic and its grain-size was greater than 2.5 mm. Based on these tests, the waste rock could be used for civil construction on the mine property with no restriction.

20.3.4 Characterization of tailings

20.3.4.1 Elemental composition

Two types of tailings will be generated at the site: magnetic tailings and flotation tailings.

The flotation tailings analysed in 2011 and 2013 showed concentrations of manganese, cobalt and/or copper above the B criteria for some samples, while concentrations of nickel, chromium and/or zinc occasionally exceeded their respective A-criteria.

The magnetic tailings analysed in 2011 and 2013 showed concentrations of manganese, cobalt, nickel and/or chromium above the B criteria for some samples, while copper and zinc exceed their respective A criteria.

20.3.4.2 Leaching tests

The results of TCLP, SPLP and CTEU-9 tests performed on magnetic tailings in 2011 and 2013 showed concentrations of aluminum, barium, copper, manganese, nickel and/or zinc above their respective RESIE criteria.

With respect to flotation tailings, the TCLP, SPLP and CTEU-9 tests performed in 2011 revealed exceedances of the RESIE criteria for aluminum, barium, copper, manganese, nickel and/or zinc. The CTEU tests conducted in 2011 showed exceedances for phosphorus. The 2013 results showed the same exceedances with the exception of zinc, nickel and phosphorus, which remained below their respective RESIE criteria.

It should be noted that the SPLP tests (1312), which were more representative of site conditions, showed very few exceedances of the RESIE criteria. In fact, for the magnetic tailings, the SPLP test showed exceedances for copper and manganese in only one sample (test performed in 2011). The results of the SPLP tests conducted in 2013 were all below the RESIE criteria, with the exception of silver and nitrites in one sample. However, the proximity of the results with respect to the detection limit of the analytical method used and the fact that the results for these parameters have always been undetected for all other samples makes these exceedances questionable.

With respect to the results of the SPLP test performed on flotation tailings in 2011, only copper exceeded its RESIE criteria in some samples. All SPLP test results obtained in 2013 were below the RESIE criteria.

20.3.4.3 Acid generation potential

Acid generation potential tests were conducted in 2011 and 2013. However, the 2011 tests combined both types of tailings.

Results on the combined samples indicate that they had no potential for acid generation.

The 2013 test results on magnetic tailings showed potential for acid generation in the majority of samples analyzed. However, none of the tests done on flotation tailings showed any potential for acid generation. It should be noted that in order to eventually develop their commercial potential, the magnetic tailings will be stored in separate cells in the tailing ponds to allow for an eventual reclaimation.

20.3.4.4 Reuse of tailings

Both types of tailings are leachable. According to Directive 019, the tailings storage area must be designed to meet a maximum daily percolation rate of 3.3 l/m^2 at the bottom of the accumulation area. The tests to date demonstrate compliance with this criterion for the proposed tailing ponds ground materials.

As previously discussed, the magnetic tailings will be stored separately from the flotation tailings in order to eventually reclaim this material. The only proposed use for this by-product is that it be sold to another company for recovery and use, for example as source of iron for the iron and steel industry.

20.3.5 Environmental characteristics of wastewater from tailings

Water samples[22] corresponding to the supernatant from the tailing pulp were analyzed. Tailings water had a pH value of 10.4 to 10.7, which corresponds to the pH at which the apatite floats. These pH values are above the criteria for the protection of aquatic life as well as those set out in Directive 019 for an effluent.

Total suspended solids (TSS) in the tailings water remained above 15 mg/l (Directive 019 criteria), probably because the sedimentation time allotted in the laboratory during preparation of the supernatant samples was too short for the solids contained in the tailing pulp samples to settle sufficiently.

With the exception of aluminum, all of the metals, metalloids, nutrients and ions analyzed in the supernatant were below the existing criteria for the protection of aquatic life as well as those in Directive 019. Process water will be treated, including pH control, prior to being discharged into the environment in order to ensure that it meets all applicable environmental criteria.

It should be noted that natural background concentrations of aluminum, cadmium and iron, among other metals, already exceed the criteria for protection of aquatic life at the project site (Genivar, 2013).

20.4 Environmental management

The environmental management plan (EMP) to be developed by Mine Arnaud will present all measures the proponent has committed to implement, in addition to what the regulation requires, to ensure that environmental impacts are verified, that regulations are complied with, that the company's environmental management activities are monitored and that its environmental objectives and targets are achieved.

The environmental monitoring program for the mine will be an integral part of the operation, and will be implemented to generate information for environmental management and reporting. The monitoring program will involve the monitoring of physical, biological and socioeconomic changes in the environment and of the company's project activities, including the application and efficiency of the mitigation measures.

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[22] These samples were taken during pilot tests conducted by COREM to optimize ore processing. The analyses of wastewater samples from tailings produced during pilot tests conducted by SGS Lakefield were not used, since the ore processing method in this case was completely different from the method later selected for the project.

The monitoring program will play a key role in ensuring that the trends for specific parameters are detected and tracked, and will provide information concerning compliance with legislative standards, set guidelines or desirable operational limits. It will also 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 the results provided by monitoring data.

20.5 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 purpose of this Act is to promote mineral prospecting, exploration and development in keeping with the principle of sustainable development, while ensuring that Quebecers get a fair share of the wealth generated by mineral resources and taking into account other possible uses of the territory” (s.17).

Section 232.1 of the Act states that:

> “Every operator who engages in mining operations determined by regulation in respect of mineral substances listed in the regulations must submit a rehabilitation and restoration plan to the Minister for approval and carry out the work provided for in the plan. The obligation shall subsist until the work is completed or until a certificate is issued by the Minister under Section 232.10.”

Hence, as part of the project, a rehabilitation plan will have to be prepared (and approved by the MERN). 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.

20.5.1 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 and waste rocks areas as well as 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.

20.5.2 Conceptual mining site rehabilitation plan

20.5.2.1 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 successive 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.

20.5.2.2 Final rehabilitation and restoration

The conceptual final rehabilitation and restoration plan can be summarized as follows:

Tailings Accumulation Cells and Waste Rock Piles

The waste rocks piles will be covered with a layer of top soil/overburden mixture and revegetated. The flotation tailing cells and dikes will be partially covered with a layer of top soil/overburden and vegetated in order to create vegetation islets. As for the magnetic cell, due to its chemical characteristics the surface of tailings will be covered with a protective sand layer, a Bentofix geomembrane, a mixture of top soil/overburden and revegetated.

The threshold of all tailing dikes' spillways will be lowered as much as possible to minimise the possibility of accumulation of water in the various cells of the tailings pond. To protect the dikes against erosion, drainage channels composed of rip rap will be developed every 200 m around the cells perimeters.

Water management infrastructure

After 5 years of post-monitoring, or once it is confirmed that water complies with regulations, the water treatment plant will be dismantled. Water in the sedimentation and polishing ponds will be pumped out and the dam will be breached to allow free-flowing of surface runoff. The area of the ponds will be covered with topsoil/overburden, and seeded to help vegetation growth, which will also consolidate the material that has accumulated into the pond.

Pumping stations and piping networks will be removed.

Access and Haul Roads

The access road to the site will be left intact and retroceded 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.

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 fill 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.

Port Infrastructures

Specific Arnaud project port infrastructures (Rails and port conveyor) the will be dismantled. The site will then be returned to a responsible entity (e.g., the ministry).

20.5.2.3 Environmental aspects

- Drainage
  - Whenever possible, the surface water drainage pattern will be restored to conditions resembling the original hydrological system.
- Topsoil management
  - During site construction and ore body stripping, the overburden and topsoil will be salvaged separately and used for revegetation purposes. The 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 at appropriate site
  - All non-contaminated waste will be sent to a 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 removed from the site

Final restoration of the mine site and port facilities will be completed within three years following the end of commercial production.

20.5.3 Monitoring program and post-closure monitoring

At the end of the operation, Mine Arnaud will submit a request to move into the post-operational monitoring phase. The monitoring program to be developed must be approved by the MDDELLC before implementation. The duration of the monitoring will depend on the time required to complete the restoration process, in order to then proceed with post-restoration monitoring. With respect to post-operational monitoring, a program will be developed by Mine Arnaud for the post-restoration phase. This program must also be approved by the MDDELLC before implementation.

20.5.3.1 Physical stability

The physical stability of the tailings accumulation areas and the waste rock piles will be assessed, and signs of erosion will be noted. These components will be monitored on an annual basis for three years following mine closure.

20.5.3.2 Environmental monitoring

Monitoring of water quality (surface and groundwater) at specific locations such as tailing accumulation areas will continue for five years after the site is restored.

A program to monitor surface and groundwater quality at target locations such as the tailing accumulation area will be carried out for at least five years after the site is restored. Discontinuation of the post-restoration monitoring program must be authorized by the MDDELLC.

20.5.3.3 Agricultural monitoring

The purpose of the agricultural monitoring program is to assess the effectiveness of revegetation done as part of the mining site rehabilitation efforts.

To document the success of revegetation of the accumulation areas, agricultural monitoring will be undertaken following the establishment of a plant cover in the areas subject to the progressive restoration program. Monitoring will be conducted annually for three years following revegetation.

Once the mine site is 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 required, reseeding will be carried out at spots where revegetation is not deemed satisfying

20.5.3.4 Water Treatment Plant Operations

The water treatment plant will be in operation for five years after the mine closure. After five years of operation during post-closure monitoring, it is assumed the water will complies with regulations.

21. Capital and Operating Costs

21.1 Capital Expenditures (CAPEX)

This portion of the Feasibility Study (FS) report is partly prepared from the assumptions and parameters selected by Roche-Ausenco for their report, then revised by SGS Geostat in 2013 under the title Pre-Feasibility Study, effective date July 24, 2013. The actual report is relying on budget prices obtained from recognized suppliers, generally a minimum of two of them. The retained quotations are included in the appendices of this report. Following the recent slowdown of the mining industry during the past two or three years, one would believe that the mining equipments prices would be the same or below the ones of 2012-2013, but unfortunately this is not the case as most of the mining fleet equipment prices went up significantly. It is important to recall that this study is prepared using a currency rate lower than the previous study, when the CA and US dollars were at par.

The Capex Estimate for the Sept-Îles Phosphate Mine project can be found in APPENDIX Y – CAPEX Estimate. A summary is presented in Table 21-1 below. The Capex estimate is split in two (2) main parts as follows:

The first portion covers all the Direct Costs broken down per area as detailed in Section 21.1.3 below. The second portion covers all the Indirect Costs plus other items such as Contingency, Escalation, etc. as detailed in Section 21.1.4 below.

In order to assess the Capex of the project, the following assumptions and qualifications were made:

- There will be no major delays in the project such as those associated to environmental permitting;
- There is sufficient accommodation available in the Sept-Îles area for the manual and non-manual workers as the cost of a camp is NOT included in the estimate;
- It will be possible to ensure safety without incurring significant loss of efficiency;
- Metric units are used through the estimate.

21.1.1 Summary

The Capex planned to be spent during the construction and pre-production period (years -2 and -1) is amounting to CA$ 854 M as shown in Table 21-1. This amount is the reference for the economic analysis shown in Section 22 and does not include an amount of CA$ 70.06 M attributable to the pre-mining period.

Table 21-1: Capital expenditure (Capex)

DIRECT COSTS		Amount (CA$)
200	MINING	
210	Mining Equipment	
220	Open Pit & Auxiliary Services	
280	Tailings Management Facilities including Water Treatment Plant	
290	Rehabilitation & Mine Closure Costs	
300	PROCESS FACILITIES	
320	Crusher, Storage & Conveying	
340	Concentrator	
370	Silos & Load-Out	
400	SITE INFRASTRUCTURE	
421	Bulk Earthworks, Landscaping & Fencing	
423	Water Intakes & Distribution System	
424	Sanitary System	
425	Roads, Overpasses & Parking	
426	Safety	
427	Fuel Distribution and Prevention	
428	Fire Detection and Prevention	
440	Auxiliary Buildings - Non-Process	
460	Rail Diversion	
480	High Voltage Substation	
490	Automation, Instrumentation and Communications	
500	APATITE TRANSPORTATION TO PORT	
540	Rail Transportation	
800	PORT FACILITIES	
820	Site & Material Handling	
840	Marine Structures	
TOTAL DIRECT COSTS		
INDIRECTS COSTS		
	EPCM COSTS (per Mine Arnaud)	
	FIELD INDIRECTS COSTS:	
	TEMPORARY SITE INSTALLATIONS AND SERVICES	
	COMMISSIONING	
	COMMON CONSTRUCTION EQUIPMENT	
	FIRST FILLS	
	VENDORS COSTS	
	THIRD PARTY SERVICES, TESTING & INSPECTION (0.5% of directs costs)	
	OWNER'S COSTS:	
	Mine Arnaud Project Construction Team	
	Training (0.5 % of direct costs)	
	Two years spare parts (per Mine Arnaud based on consignment)	
	Capital spares	
	Construction Insurance (0.5% of directs costs)	
	TAXES & DUTIES (Not included)	
TOTAL INDIRECTS COSTS		
CONTINGENCY		
	CONTINGENCY (15 % on all Total Direct and Indirect Costs per Mine Arnaud)	
TOTAL CONTINGENCY		
TOTAL DIRECT AND INDIRECT COSTS WITH CONTINGENCY		
	OTHER COSTS:	
	HYDRO-QUEBEC TIE-IN TO NETWORK (excluding back-up & costs at Arnaud Sub-Station)	
	HYDRO-QUEBEC - STABILIZATION OF 161 kV TRANSMISSION LINE L1619	
	LAND / PROPERTY ACQUISITION (per Mine Arnaud)	
	ESCALATION (Not included)	
TOTAL COSTS (CA$):		

21.1.2 Accuracy of the Estimate

The purpose of this feasibility study phase is to update a capital cost estimate performed in 2011 to an accuracy of to costs as of June 2014.

In order to obtain this level of accuracy, the consultants have done the following:

- Get major equipment procurement cost based on manufacturers' quotations;
- Perform a detailed evaluation of All-In Construction Labour Rates to reflect the most recent practices in Sept-Îles;
- Confirm unit prices based on collecting information from various sources;
- Verify calculations of quantities to reflect the latest engineering documents issued.

The base date of the cost estimate is June 2014.
The estimate is expressed in Canadian Dollars (CA$).

For reference, the currency conversions rates used during the estimate preparation are as per instructions from Mine Arnaud:

- 1 CA$ = US$;
- 1 CA$ = Euro.

21.1.3 Direct Costs

Direct costs, as presented in the Capex summary, cover for the cost that can be directly related to a specific project area, including equipment supply and delivery, materials costs, installation labour, etc.

21.1.3.1 Construction Labour Rates

The All-In Labour Rates used reflect Yara's description of typical inclusions. The rates to use are summarized in Table 21-2.

Table 21-2: All–In labour rates

	Tradesmen Rate ACQ (note 1)	Tradesmen Contractor Base cost (note 1)	All-In Rates at 40 hrs/wk	All-In Rates at 58 hrs/wk
Civil and Earthworks	55.89	62.90		
Concrete	55.56	65.64		
Structural Steel	56.96	71.96		
Architectural	55.56	65.64		
Mechanical	56.24	63.88		
Piping	56.57	65.44		
Electrical	57.69	64.39		
Automation	57.69	64.39		

Note 1: Per Quebec Construction Collective Agreement as of May 1st, 2014

Work Week

In the Province of Quebec, 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 in 2011 that most of the major projects in the Sept-Îles area were offering fifty-eight (58) working hours per week schedule in order to attract workers. The fifty-eight (58) hours 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, the following detailed information is provided in the previous table:

- Construction base rates as of May 1st, 2014 as per the Quebec 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, an 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 ll

At the time of the Original FS preparation in 2011, the demand for workers was higher than it is now because of a slowdown in the overall mining industry. By selecting to retain the hours per week option for the construction period, Mine Arnaud is on the conservative 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.

In 2014, the following changes were made to the 2011 All-In Labour rates estimate:

- Base rates have been adjusted to the 2014 Quebec Construction Collective Agreement rates;
- Percentage of non-local workers have been reduced from 86% to 60% as per Mine Arnaud evaluation;
- Large capacity cranes (100 tonne and higher) have been removed from the all-in rates and transferred to Common Construction Equipment.
- Rates have been developed from the total hours budgeted in the concentrator area (Area 340);
- of productivity loss factor was added to the hours per week schedule.

Man-Hours and Productivity

A % productivity loss factor was added to the hour per week schedule (base on a NRC study) to take into account the loss of productivity on working hours at regular rates.

No other productivity factor has been applied separately to take into account the local conditions for construction work at Sept-Îles.

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

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:

- Mobilization and demobilization 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.

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;
- Rental of all cranes under 100 tonnes

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 or Temporary Site Installations.

- Mobilization and demobilization of crane equipment to site;
- Rental for all cranes 100 tonnes and higher;
- Construction installations such as office complex, first aid room, sanitary facilities, etc.

21.1.3.2 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. Some major unit prices used for this Study are presented in the following Table 21-3 and Table 21-4. These prices have been validated during the Feasibility Study.

Table 21-3: Unit price (steel)

Item	
Structural Steel (including installation)	
Miscellaneous Steel	
Embedded Metals	

Table 21-4: Unit price (civil and site preparation)

Item	
Tree cutting	
Backfill material (from excavation)	
Excavation (overburden)	
Rock Excavation <100,000 m^3	
Rock Excavation >100,000 m^3	
Concrete (including form work & rebar)	

21.1.3.3 Mining

The amount related to mining direct cost is equal to CA$ M and it includes the following items:

- Mining Equipment;
  - The mining equipment list contains the initial purchase of the mine fleet and the ancillary equipments. The value of the mining equipments is CA$ and the content is presented in Table 21-5
- Open Pit and Auxiliary Services;
- Tailings Management Facilities including Water Treatment Plant.

Table 21-5: The mining equipments Capital Expenditure Cost

21.1.3.4 Process Facilities

The amount related to process facilities direct cost is equal to CA$ M and it includes the following items:

- Crusher, storage and conveying;
- Concentrator;
- Silos and load-out.

21.1.3.5 Site infrastructure

The amount related to site infrastructures direct cost is equal to CA$ M 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.

21.1.3.6 Apatite transportation to port

The amount related to apatite transportation to port direct cost is equal to CA$ M and it includes the following items:

- Rolling equipment;
- A 2.28 km new rail to bypass the Cliffs Wabush Yard.

21.1.3.7 Port facilities

The amount related to port facilities direct cost is equal to CA$ M and it includes the following items:

- Site and material handling;
- Marine structures (not included).

21.1.4 Indirect Costs

The indirect costs are composed of the items described in this section.

21.1.4.1 Engineering, Procurement and Construction Management (EPCM)

As agreed with Mine Arnaud, the Capex estimate for the 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, an amount was provided by Mine Arnaud to cover the cost of the EPCM services, which Mine Arnaud based in part on costs provided by consultants. The EPCM value is equivalent to about % of total direct costs, which include Detailed Engineering, Procurement, Project Management and Home Office Services as well as Construction Management as described in the table below:

Table 21-6: Definition of EPCM services

Detailed Engineering	Includes the drawings and documents for the complete engineering package necessary to procure and 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, preparation of as-built documentation and field support from the office.
Procurement	Includes both local and foreign purchasing. Procurement encompasses request 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, receving and storage, site inspection, but not all foreign source inspection. Sub-contracts for installation and other services are also included.
Project Management and Home Office Service	Includes specialist personnel necessary to support the engineering and construction plans. These services involve project management, cost control, scheduling, engineering, estimating, project accounting, construction administration, industrial relations, personnel, administration, support to permitting, quality and OH&S.
Construction Management	Includes construction management services at site such the required field expenses for construction management personnel including travel and relocation expenses to the site.

21.1.4.2 Temporary Site Installations and Services

For the purpose of this Feasibility Study, a detailed evaluation was made in 2011 and updated in 2014 to cover the cost of the Temporary Site Facilities. It includes the project management team's site installation requirements for the project execution.

21.1.4.3 **Commissioning**

For the purpose of this Feasibility Study, an evaluation was made in 2011 and updated in 2014 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 and 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 and commissioning.

Vendors' representative costs are estimated separately.

21.1.4.4 **Common Construction Equipment**

For the purpose of this Feasibility Study, an evaluation was made in 2011 and updated in 2014 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 mobilization and demobilization of crane equipment to site. In this item the minimum requirement of lifting equipment for the lay down area has been included and an allowance for the rental cost of the major cranes (100 tonnes and higher).

21.1.4.5 **First Fills**

For the purpose of this Feasibility Study, an evaluation was made in 2011 and updated in 2014 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.

21.1.4.6 **Vendors Costs**

This item covers the cost for the vendors' representative for:

- Erection assistance;
- Start-up and Commissioning assistance;
- Training.

The estimate included costs for vendor assistance on major pieces of equipment that are specialized. The costs were based partly on vendors' information collected during the Request for Quotation process. An allowance to capture vendor assistance on other additional equipment was included. The amount estimated has a value equivalent to % of direct purchase costs less mine equipment.

21.1.4.7 **Third Party Services, Testing & Inspection**

This item is to include for testing and inspection by third parties that are not included elsewhere in the estimate, such as but not limiting to: soils testing, concrete testing, non-destructive testing on piping, testing on specific field welds, etc.

A percentage of % of Direct Costs was used to cover for this item.

21.1.4.8 Owner's 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.

Mine Arnaud Project Construction Team

An amount was provided by Mine Arnaud.

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 salary mass higher than CA$1 M, 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.

One Year Spare Parts

An evaluation on one year spare parts was done for consideration by Mine Arnaud. No allowance is included for one year spares based on direction by Mine Arnaud. Their one year spares strategy will be based on consignment with vendors.

Capital Spares

An estimate on capital spare parts was calculated for major pieces of equipment. The source of pricing was partly provided by vendors' information collected during the Request for Quotation process. For other smaller equipment, an allowance has been made in areas where there is substantial amount of equipment.

A cost savings opportunity was identified on the crusher package in consideration of the client's consignment spares strategy and the amount allowed for spares was reduced from the original vendor recommended value.

For mine equipment, no capital spares are included.

Without considering mine equipment, the capital spares included is equivalent to % of direct purchase.

Construction Insurance

A percentage of % of Direct Costs was used to cover for the cost of construction insurance.

21.1.4.9 Taxes and Duties

There is no amount included for taxes and duties as confirmed by Mine Arnaud.

21.1.5 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.

Mine Arnaud has provided a value of % of Direct Costs and Indirect Costs (excluding Land/Property Acquisition and Hydro-Quebec) to include for this estimate.

21.1.6 Tie-In to Hydro-Québec Network

This item is to cover for the cost provided by Hydro-Québec to tie-in the mine site to the Hydro-Québec network. In 2011, Hydro-Québec performed a planning study. At that time, the least expensive option for Mine Arnaud was to tie-in to the 161 kV transmission line L1617 near the Bay of Sept-Îles, even though the 161 kV transmission line L1619 is literally crossing the mine site between the open pit and the plant.

In 2014, a request for a new planning study was submitted to Hydro-Quebec. A meeting was held with Hydro-Quebec on 9 July 2014 in which Hydro-Quebec presented the preliminary results of that study on which the costs used in this estimate are based.

There have been changes to the network since 2011 and, as a result, Mine Arnaud can now be tied to the 161 kV line L1619. Hydro-Quebec provided verbally a cost of CA$ for the tie-in, including CA$ for a pre-project study. A total of CA$ M has been included in the estimate and includes any associated contingency.

Hydro-Quebec also provided the additional optional cost for firm tie-in to the network consisting of back-up tie-in to line L1617 of CA$ M, including CA$ for a pre-project study. This is not included in the Capex estimate as Mine Arnaud is planning not to go for this option.

Before issuing their planning study at the end of August 2014, Hydro-Quebec still have to evaluate the impact of adding Mine Arnaud to their Arnaud sub-station feeding both lines L1617 and L1619. Hydro-Quebec mentioned that the costs to upgrade the sub-station would be their responsibility, unless the Mine

Arnaud electrical load would trigger something major. As this is unlikely, Mine Arnaud suggested that no contingency be included in this estimate.

Note: The Tie-In to the Hydro-Quebec Network should not be confused with the 161 kV line securisation, which concerns the changes required to allow the addition of (1) haul road and (2) rail passages under line L1619. This is covered separately in the next section. Both items may ultimately be combined by Hydro-Quebec as they affect the same transmission line.

21.1.7 Hydro-Quebec – Stabilization of 161 kV transmission line L1619

In a letter dated 4 July 2014, Hydro-Quebec Trans Énergie advised that the rail passages considered by Mine Arnaud under the corridor for four (4) 735 kV transmission lines (7031, 7032, 7033 & the new Romaine II–Arnaud) are acceptable based on the train not transporting dangerous goods. However, the 161 KV line L1619 needs to be stabilized.

Hydro-Quebec provided verbally a cost of CA$ to CA$ M to stabilize the line, including CA$ for a pre-project study. This includes both rail passages as well as the haul road and includes provision for dead-end tower. The amount will be confirmed when Hydro-Quebec issues its planning study at the end of August 2014. A total of CA$ M has been included in the estimate and includes any associated contingency.

21.1.8 Land Property Acquisition

An amount corresponding to a fixed budget was provided by Mine Arnaud and included in the Capex estimate to cover for land property acquisition. This amount includes any contingency associated with this cost.

21.1.9 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 -2 and -1 of the project. As inflation will be taken into account in the Economic Analysis, Mine Arnaud requested that no amount for escalation be included in the Capex Estimate.

21.1.10 Exclusions

The following items are not included in the initial capital cost estimate:

<u>Labour Relations</u>

- Loss of efficiency due to negotiations in the construction industry;

- Allowances for industrial dispute or lost time arising from industrial actions;
-).

Community and Native Relations

- Costs for Community and native relations and services. are not presented in the Capex Section but they are included in the Opex section of this report.

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 economic analysis);
- Any requirements related to project financing;
- Financing Fees;
- Resettlement / Relocation costs;
- Other Owner's costs;
- Sunk cost;
- Sustaining capital or deferred capital costs are provided separately (Section 21.2 on p.327);
- Working capital is provided separately.

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;
- Rehabilitation and Mine Closure Costs are provided separately (Section 21.2.5).

Permits, Fees, Taxes and Legal Costs

- Cost of Permits;
- License and Royalty fees;
- All Owner payable taxes;
- Legal costs;
- Force majeure issues.

Operating and Maintenance Costs

- Any operational insurance such as business interruption insurance and machinery breakdown;

- Operating and Maintenance Costs (Opex) are provided separately in the Operating Costs Estimate (Section 21.3).

21.2 Sustaining Capital Expenditures

21.2.1 Summary of Sustaining Capital Expenditures

Sustaining capital contains costs that will be capitalized, i.e. added to the carrying amount of the asset; money spent to acquire or upgrade physical assets such as: tailings facilities, Tailings Cells, replacements and new purchases of the mining equipments and the mine closure costs and rehabilitation. Based on the project WBS we have four main areas which bring important sustaining capital expenditures: the mine, the tailings cells, tailings facilities and the closure and rehabilitation activities. A summary of these costs in thousands of CA$ with afferent contingencies is presented in Table 21-7.

Table 21-7: Summary of Sustaining Capital Expenditures

21.2.2 Sustaining Capital for Mine Equipments

The mining sustaining capital does not contain the pre production period, years -2 and -1, and includes both the cost to purchase the initial mining fleet required to achieve production level based on the mine plan, as well as, mining capital equipment purchased as replacements for ongoing equipments.

The sustaining capital is based on estimations of replacement life for major and ancillary equipments. The replacement life of the selected production fleet is 70,000 hours for the trucks, 60,000 hours for the excavators and production drills, the overburden equipments have a replacement life of 30,000 working hours. For ancillary equipments the replacement lives are based on suppliers' recommendations and on internal data base. The next two tables are showing the sustaining capital cost before and after the % contingency, and the estimated replacement life of all mining equipment

s.

Table 21-8: Mining Sustaining Capital Cost

Table 21-9: Replacement Life of Mining Equipment

21.2.3 Sustaining Capital for Tailings Cells

These costs represents costs for tailings park constructions, it contains six basins named: 1M, 1F, 2F, 3F, 4F and 5F. A time frame of these costs is presented in the next table, which contain % contingency. For basins dams erection the waste from the mine is utilized as well as sand and gravel, till or clay and a geo textile membrane.

Another item of this sustaining category is the cost for 2nd barge for magnetic tailings pond and extra length of pipeline (Section 21.4.1 of Capex) for 1.5 km to complete the pipeline for the magnetic tails cell.

Table 21-10: Tailings Park, Cells Sustaining Capital Cost

21.2.4 Sustaining Capital for Tailings Facilities

Tailings facilities sustaining capital costs contains costs that occurs at the fourth year up to the 11th year of the operations for the tailings facilities constructions which will be completed at that time. It contains items like: Tailings management piping network, water treatment units, tree cutting and stripping, site preparation for Screen Berm, Stocking Area and Waste Dump.

Table 21-11: Tailings Facilities Sustaining Capital Cost

21.2.5 Closure and Rehabilitation Plan

A cost estimate for the closure and rehabilitation of the mine site has been produced in compliance with the requirements of Québec's Ministry of Energy and Natural Resources (MERN) presented in the document entitled "Guidelines for Preparing a Mining Site Rehabilitation Plan and General Mining Site Requirements", and indicated in the Mining Act. The costs have been estimated based on prices given by contractors as well as costs monitored in other rehabilitation works or presented in various rehabilitation plans.

The total cost of closure and rehabilitation is estimated at CA M. This amount includes the rehabilitation of the accumulation areas and the cost of other closure and restoration activities. Table 21-12 (WSP 1) shows the costs detail.

Table 21-12: Closure and Rehabilitation Cost Summary

An amendment to Article 111 of the Regulation respecting on mineral substances other than petroleum, natural gas and brine (chapter M-01.13, r. 2) was adopted on July 23, 2013 (Decree 838-2013). Thus, the proponent of a mining project must now provide a financial guarantee which amount corresponds to the total projected costs to complete all works included in its closure and restoration plan. As stated in Article 113, the financial guarantee will be provided in three installments. The first installment is 50% of the total amount of the guarantee and will be provided within 90 days of receipt of the closure and restoration plan approval. Each subsequent payment (25 % each) will be made on the anniversary date of the closure and restoration plan approval.

Table 21-13: Estimate for yearly payment of financial guarantee

21.2.5.1 Environmental Monitoring

Environmental monitoring will be conducted in compliance with Directive 019 and Metal Mining Effluent Regulations (MMER).

21.3 Operating Costs (Opex)

21.3.1 Summary

The Opex estimates for specific areas were estimated by the responsible consultants and compiled by Roche. The Opex estimate 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 project Opex estimate is based on the following parameters:

- Tonnes of ore & waste mined per year: 21,463,174;
- Tonnes of ore milled per year: 11,282,880;
- Average stripping ratio in the mine, over the project life year -1 to year 31: (tonnes of waste & overburden per tonnes of ore) 0.71
- Tonnes of apatite concentrates produced per year (dry): 1,184,702;
- Total manpower resources required for operation: 276 employees (payroll).

Excavation of overburden and transportation of concentrates to the port will both be conducted using owner operating personnel.

The overall Opex for the Mine Arnaud project is estimates at CA$ M per year or CA$ per tonne of apatite concentrate. A summary of the operating costs for the project is presented in Table 21-14. All costs presented in this section are in Canadian Dollars (CA$) per year and CA$ per tonne of apatite concentrate. Please note that the Summary Process Opex is considering an annual tonnage milled of 11,282,880 tpy which correspond to the design plant throughput. Any variations with the annual through put as per the Mine planning will have an effect on the average cost per tonne milled or tonne of concentrate produced because some of the operating costs, such as the salaries, are not proportional to the processing rate. Complete details are further described in the Opex Document shown in APPENDIX Z – OPEX (Document 15685-01-ES-GE-002-0C).

Table 21-14: Operating Cost Summary

21.3.2 General and Administration Costs

The General and Administration (G&A) Opex is estimated at CA$ 13.81/t of concentrate. This includes all costs which are not related to the mining and processing parts of the project's operation. This includes: administration man power, general costs, contracts, marketing, taxes and transport & ship loading activities. The total cost for each item is shown inTable 21-15. Additional details are provided in the relevant sections below.

Table 21-15: Summary of General and Administration Costs

21.3.2.1 Manpower

The Administration group consists mostly of yearly salaried employees, which includes the Environment and Administration Departments. The environmental group (5½ employees) is composed of a coordinator, technicians, and laborers. The administration group (25 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. Details of the manpower cost are indicated in Table 21-16.

Table 21-16: Administration Manpower Cost

21.3.2.2 General

Additional cost allocations have been made 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 21-17.

Table 21-17: General Cost Details

21.3.2.3 Contracts

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. The estimates for the costs of contracts are based on information provided by locally established companies as well as information provided by Mine Arnaud. Details are presented in Table 21-18.

Table 21-18: Contractor Cost Details

21.3.2.4 Marketing

A placeholder 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 placeholder in the OPEX was left in case the marketing strategy changes at some point in the future.

21.3.2.5 Municipal Taxes

Municipal taxes have been supplied to Mine Arnaud, by AEC SYMMAF a specialized consultant and are composed of Sept-Îles municipal and school taxes. Mining rights yearly cost has also been estimated and is integrated into the financial model. The details are shown in Table 21-19.

Table 21-19: Municipal Taxes and Mining Rights Cost Details

21.3.2.6 Cost for community (IBA, Impact Benefits Agreement)

The operational cost estimate contains an allowance for supporting the local community, with of CA $

. These cost are expressed in the Table 21-20 below:

Table 21-20 Impact Benefits Agreement allowance

21.3.2.7 Transport and Ship Loading

An evaluation of the operating costs for the rail transport and ship loading activities are summarized in Table 21-21.

Table 21-21: Transport and Ship Loading Cost Details

Cost Items	Annual Cost (CA$)	Cost per Tonne of Apatite Concentrate* (CA$/t)

*Based on 1,196,669 tonnes of apatite concentrate (which contains 1% moisture)

Rail transportation of concentrate to the port

The transport of concentrate from the mine site to the port will be provided by Chemin de Fer Arnaud or Arnaud Railway, a subsidiary of Cliffs Natural Resources. Negotiations between Mine Arnaud and Cliffs need to be carried out. Mine Arnaud has suggested a rate of CA$ per tonne of apatite concentrate, which includes maintenance of the railcars. As an alternative to transportation by rail, Mine Arnaud has a quote of CA$ per tonne transported by truck.

Unloading railcars and ship loading – Operation and Maintenance

The railcar unloading is to be handled by additional employees (2) included in the Manpower estimate. Mine Arnaud will sub-contract operation and maintenance for ship loading. A local terminal operator was contacted to provide a rough budget of what operations and maintenance of the ship loading facilities would cost. The cost used is based on their evaluation.

The annual ship loading operational and maintenance is estimated at CA$ based on the experience of the local operator. Additional stand-by fees apply for time spent waiting between loads. This cost was estimated at CA$ or the equivalent of CA$/hr on stand-by, of waiting per ship, each with a capacity of tonnes of concentrate.

Port of Sept-Îles fees

There are many costs related to the shipment of apatite concentrate overseas. As the concentrate price used for the economic analysis is at the Port of Sept-Îles, most of the shipments costs will be absorbed by the clients. Per Mine Arnaud, the only applicable annual fee to be paid to the Port of Sept-Îles is the wharfage fees. The rate for similar type of bulk material is CA$ 1.48/t. This is equivalent to about CA$ 1.77 M per year. This needs to be confirmed by the Port of Sept-Îles.

Capex recovery for the Port of Sept-Îles investment at Berth 31

As mentioned in the description of the port facilities, the Port of Sept-Îles will have to make investments at Berth 31 to adapt it for Mine Arnaud. The Port will have to reinforce the dock to suit the new travelling ship loader and tripper loads, extend it to the east, add a platform for a transfer tower and realign the access bridge. This will be indirectly charged to Mine Arnaud as an additional yearly fee.

Further study work is required to verify the dock structure and establish the cost to reinforce it for a travelling ship loader. At this stage, Mine Arnaud is considering that beyond CA$ million, an alternative to Dock 31 would be considered. Therefore, a yearly payment corresponding to a % interest over years for CA$ million is included.

21.4 Mining Costs

21.4.1 Basis of Estimate of the mining cost

The mining Opex estimate was developed for the project base case, which reflects the following key assumptions:

- A mine plan based on a waste to ore ratio of 0.71 over the life of the project, between year -1 and year 31th.
- An assumed ramp up tonnages is described in the Table 21-22 and Figure 21-1, below.

Table 21-22: Ramp Up of the quantities of tonnes mined and tonnes of ore

	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5
Tonnes Milled ('000)	8,461	9,591	10,719	11,283	11,283
Tonnes Mined ('000)	13,653	16,441	17,366	18,156	20,862

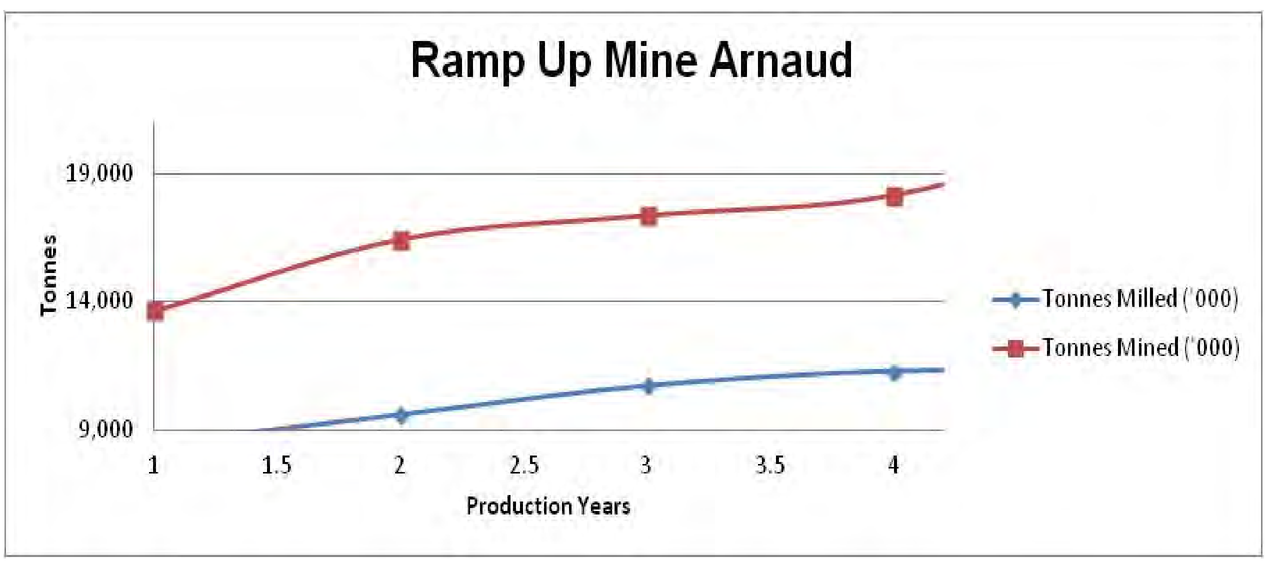

Figure 21-1: Mine Production and Concentrator Feed Ramp-Up

- Mining tonnages are assumed to match the above production requirements and provide the requisite ROM feed to the beneficiation plant;
- Life-of-mine based on 30 years of operation, utilising best practice for an owner-operated mining and processing operation;
- Electrical Power will be provided by the local power authority from the national grid at a pre-determined rate per kWh of CA$ per kWh and also CA$ monthly fee for subscribed kW, given by the L tariff of Hydro-Quebec. Credits as applicable to the rates for Mine Arnaud have been applied to the published rates;
- A salary basis was established according to salaries average in the same geographic area for the similar projects. The fringe benefits are estimated at % of the gross salary for each employee. A summary of salaries per pay category can be found elsewhere in the Section 21.5.1.1 of the FS report. The salary costs represent the full cost of employment to the project and include numerous local taxes and all benefits;
- Organizational chart is developed according to the production requirements and is approved by the client;
- The annual working hours are not including vacations and sick leaves and they are estimated at 2,016 for each employee in working schedule A and 1,800 in working schedule B. A complete description of every working schedule is presented in the Section 21.5.1.1;
- Diesel fuel cost of CA$ per litre, which was provided by the client;
- Safety consumables and general consumables refer to fringe benefits for site based personnel to cover clothing, hard hats, etc;
- No allowance was included for the closing activities in the present chapter of mining cost estimate.
- Maintenance and fuel consumption for majors and ancillary equipments are given by an Estimator's Guide, Mine and Mill Equipment Costs, 2012-2013. An inflation factor of 2% was applied;
- Infrastructure and site road maintenance are based on the mobile equipments hourly costs and any other additional expenses like signals, traffic regularizations, materials, dewatering, etc.
- Explosives and accessories costs are based on annual tonnages given by the mining plan and the blasting design. The unit prices for explosives products and accessories were obtained from Orica.

21.4.2 Escalation, Currency and Contingency for the mining cost

- The base date of the estimate is 2nd quarter of 2014. All escalation beyond that date is excluded;
- The currency for this estimate is CA$;
- No escalation has been applied to the Opex.
- No contingency has been applied to the Opex it is applied on the financial model;

21.4.3 Methodology of the mining cost estimate

In accordance with the project's WBS, the mining cost estimate was broken down into cost types: fixed and variable.

The mining cost is broken down by areas:

- Major Equipment Cost, or Load and Haul Cost
- Ancillary Equipment Cost
- Pumping Cost
- Drilling and Blasting Cost
- Technical Services and G&A
- Production Supervision Manpower Cost

The battery limit of the mining cost is presented in the next Figure 21-2.

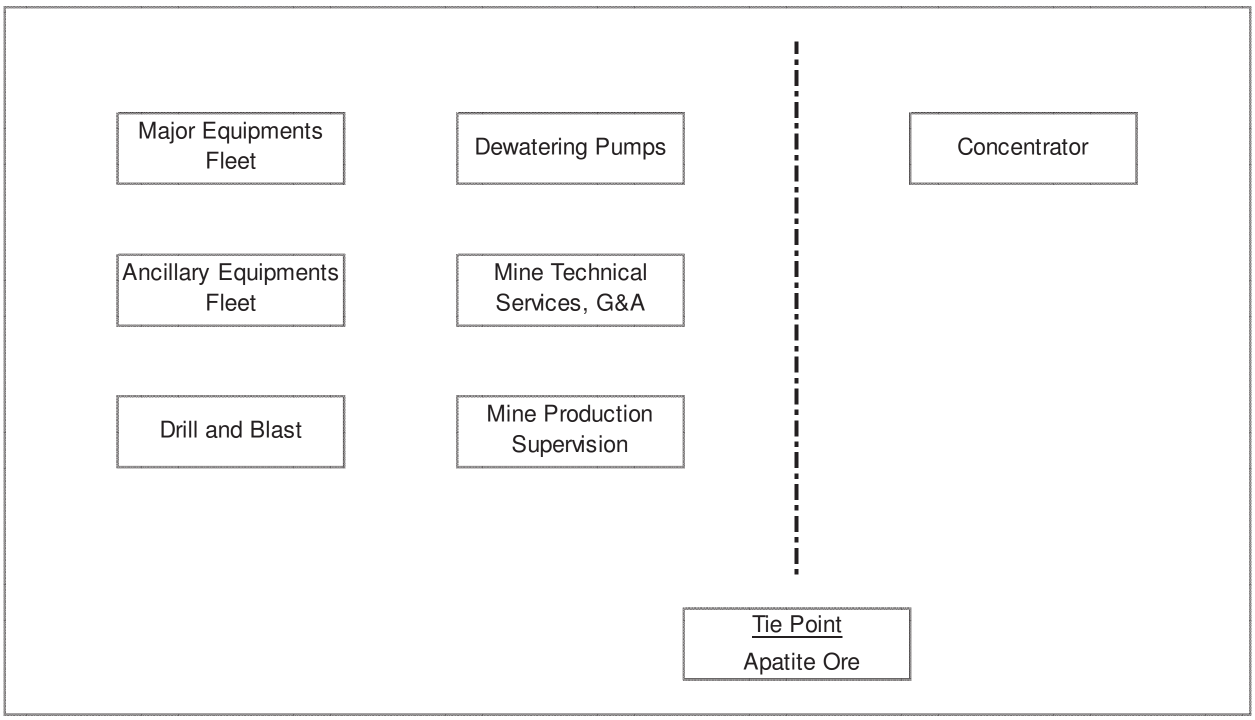

Figure 21-2: Battery Limit of the operating cost estimate

The mine plan and equipment recommendations were developed by SGS and are included in Section 16 of this report.

Operating costs for the Mining Operation can be categorised either as fixed or as variable.

Fixed costs are not dependent upon variations in production levels or operating efficiency, they include:

- All labour costs associated with the technical services and production supervision activities;
- All Operating costs for dewatering pumps;
- Explosives Equipments (the explosives plant) and the contractor labour for blasting.

Variable costs are dependent upon variations in production levels and operating efficiency, they include:

- Mining fleet and ancillary mobile equipment operating and maintenance costs;
- Consumption of fuel and lubricants, where not already included in hourly operating costs for equipment;
- Accessories for drilling, like rods, bits, etc.
- Explosives and accessories for blasting.

The Opex estimate covers the fixed and variable costs described above, for all personnel directly involved in the mining operations, and for the management, planning, supervision and other operations within the mine area.

21.4.3.1 Manpower

The manpower for the mining operations is presented in the following table which first gives the annual average manpower and cost for Year-2 to Year-27 and also the maximum manpower and cost that occurred at Year-7. The annually average manpower and cost period of years 2 to 27 was selected because these years are typical of the project duration following the ramp up and the ramp down periods of the apatite concentrate production.

Table 21-23: Average and Maximum Mine Manpower and Costs

The mining activity is scheduled for 365 days per year. The mine labour force is based on the production requirements and on the maintaining in year-round operation of all ancillary equipments. The schedule, named schedule A, assume 2016 working hours for every person on the mine payroll, there will be four crews working 12 hours shifts, two shifts per day. The personnel for blasting activity: blasters, blasters helpers are not included as they are part of the sub contract proposed by a contractor and included in the drilling and blasting cost described previously. The next three tables are presenting the average manpower and cost for every mine manpower area.

Table 21-24: Manpower cost, Mine Technical Services, G&A

Table 21-25: Manpower Cost, Mine Production

Table 21-26: Manpower cost, Mine Maintenance

21.4.3.2 Mining G&A

The mine general administration expenses include all consumables and supplies needed for the mining activity, computers, software and survey equipments and also allowances for consultants and business travelling. These expenses were estimated using the best practices in the industry and the internal data base. They were validated with the client and they are presented on Table 21-27.

Table 21-27: Annual allowances for Mining G&A

21.4.3.3 Mine Operational Costs

For reporting purposes the mining cost is presented on an average annual basis, from the second year to the 27th year inclusively and shown in Table 21-28. These years were chosen because we trimmed the ramp up and the ramp down periods of the apatite concentrate production, which are not considered like typically production years.

Table 21-28: Summary of the mining cost

21.4.4 Mine Drilling and Blasting

All rock in the open pit will be drilled and blasted before being loaded and hauled to the crusher, stockpiles or the waste dumps. The drilling equipment operating and explosive cost estimation, including the operators and maintenance labour are outlined in Table 21-29.

Table 21-29: Drill and Blast Cost, Years 2 to 27

The drills Opex parameters are presented in Table 21-30 and the unit cost of explosives and blasting accessories are shown in Table 21-31. These unit costs were obtained from an explosives manufacturer.

Table 21-30: Production and Pre–Split Drilling Operating Parameters

Table 21-31: Explosive Products Unit Costs in CA$

21.4.5 Mine Loading and Hauling

The loading and hauling equipment unit costs, including the cost of equipment operators and maintenance labour, are outlined in Table 21-32.

Table 21-32: Major equipments cost, Load and Haul

The equipments chosen for this project are the mining trucks CAT 785 D working with the shovel Komatsu PC 3000-6 for ore and waste loading and hauling and the match given by the articulated truck Volvo A40F working together with the excavator CAT 390D for overburden. More details about these equipments are in Section 16 of the present feasibility study (FS).

The loading and hauling annual costs are calculated for every year of the mining life. The average cost/t of material is outlined in Figure 21-3, the trend indicates the annual increase in cost due to lengthening of haul roads and deepening of the pit.

Figure 21-3: Haul and Lift Cost Variation, Year 2 to Year 27

The average cost for loading and hauling ore and waste will vary due to increase in hauling distances as the pit is deepened.

21.4.6 Mine Ancillary Equipments

In this category we have all mining support mobile equipments and also all auxiliary and service mining equipments, light trucks inclusively. A detailed list of mine ancillary equipments is shown in Table 21-33, which also gives the hourly consumption rates.

Table 21-33: Ancillary Equipments Operating Costs

For every Caterpillar equipment we utilized the manufacturer hourly rates and for the rest of equipment a guide estimator was used, more precisely, Mine and Mill Equipment Costs, 2012-2013. An inflation factor of 2% was applied. The annual costs, expressed on an average basis, between years two and 27 are presented in the Table 21-34, shown below.

Table 21-34: Annual operating cost of Ancillary Equipments

21.4.7 Pit Dewatering Cost

In order to insure the dewatering of the pit, electrical dewatering pumps are recommended from the first year of exploitation. Their hourly costs are presented in Table 21-35 and the impact on mining cost is shown in Table 21-36.

Table 21-35: Dewatering Pumps, hourly operating costs

Table 21-36: Dewatering Pumps, Annual Operating Costs

21.5 Processing Costs

The estimated process Opex covers power, labor, consumables, reagents, fresh water and water treatment, tailings pond and some other processing costs such as laboratory supplies, R&D costs, environment services for operating the plant with 11.283 Mt throughput and 1.185 Mt dry concentrate production per year. The plant includes crushing, grinding, LIMS, flotation, concentrate and tailings dewatering circuits. The process operating costs are summarized in Table 21-37.

Table 21-37: Summary of of Processing Costs

21.5.1.1 Manpower

A total of 71 employees are required for the mill operation. This is considering operating 24 hours a day, 7 days a week, 52 weeks per year. The working schedule for most salaried employees will be a standard 40 hours a week, 8 hours a days, 5 days a week, Monday to Friday.

The breakdown of the personnel and costs, including an allowance of 45% for fringe benefit are shown in Table 21-38.

Table 21-38: Process Plant Manpower Cost Details

21.5.1.2 Stockpile Rehandling

It was estimated by SGS that 10% of the ore feeding the processing plant would have to be rehandle at a cost of CA$ t processed.

In order to respect the Cl content restrictions in the apatite concentrate, a blending of the concentrator feed is forecasted. This blend will apply to 10% of the feed, which means around 3,000 tonnes on a daily basis. A CAT 993 loader will be assigned to this task for 12 hours per day. With a time cycle of three to four minutes, this equipment will insure the total concentrator feed in 130 loads daily. Two operators are affected to this operation on the manpower payroll. The hourly Opex of the CAT 993 is presented in the next table.

Table 21-39: Summary of Rehandling Costs

21.5.1.3 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 the eventuality of having equipments 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. Details are shown in Table 21-40.

Table 21-40: Summary of Energy Costs

21.5.1.4 Electricity Consumption

For practical reasons, all electricity consumptions are considered as a processing Opex which represents nearly % 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 Q – Electrical Load List. 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 CA$ and CA$ per subscribed kW for 161 kV and 25 kV lines respectively for the agreed kW with HQ per month and ¢/kWh consumed.

According to HQ's Tarif L, the billing demand rate for the sites supplied at 161 kV line can be calculated as such: the base rate (CA$/kW), less power demand credits (CA$/kW at 161 kV), less transformation loss adjustment (CA$/kW), equals the 161 kV demand rate of CA$/kW. The 25 kV connection for the port has a demand rate of CA$/kW. This rate is calculated in the same way as the site connection, except the power demand credit equals CA$/kW for 25 kV. The Tariff L and the credits applied are described here:
http://www.hydroquebec.com/grandesentreprises/tarification/tarifs-grande-puissance/tarif-l/

To calculate the cost of demand load in relation to the and CA$ 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 ¢/kWh, it is correct to use the subscribed power which is a product of the Running load and utilization rate in percentage. The utilization rate has been partly calculated based on the projected utilization of equipment in different areas and partly estimated based on experience. Table 21-41 shows the details of the estimated annual electrical cost for the Mine Arnaud project.

Table 21-41: Electricity Consumption Cost Details

Note: The Flash Dryer consumption utilization rate has been lowered from 92% to 86% to account for the by-pass of moist apatite concentrate which will not go through the Flash Dryer.The Tailings & Water Treatment consumption utilization rate has been lowered from 92% to 36% to account for the 16% (60 days per year) utilisation of the Run-Off Water Pumps.

21.5.1.5 **Diesel Consumption**

Diesel cost per litre is established at CA$ /l and corresponds to the cost after tax credit/refund from the government. Diesel consumption for 15 diesel powered pick-up trucks was established to be 70 L per truck per day. Diesel for mine mobile equipment is not included in this section. It was calculated as a component of the mining Opex in Section 21.4.3.3. The annual diesel for personnel transportation and maintenance cost is CA$ or CA$ per tonne of apatite concentrate.

21.5.1.6 **Gasoline Consumption**

Gasoline cost per litre is established at CA$ /l 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 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.

21.5.1.7 **Consumables**

Consumables are divided in four sub-groups: liners, grinding media, mechanical maintenance supplies and lubricants. Details are shown in Table 21-42.

Table 21-42: Summary of Process Plant Consumables

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 Table 21-43, the consumption for liners has been evaluated based on the ore's abrasion index laboratory results, suppliers' recommendations and similar projects with similar ores. It represents CA$ M per year or CA$ /t of apatite concentrate.

Table 21-43: Crushing and Grinding Consumables Cost Details

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. Table 21-44 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 21-44: Mill Liner Cost Comparison*

* Prices are from Roche-Ausenco 2012 FS

During the 2011 feasibility study, Roche-Ausenco performed a cost study of using different types of liners such as chrome-moly liners versus normal steel liners as well as high chrome steel balls versus forged steel grinding balls and realized that using chrome-moly steel liners with a combination of forged steel grinding balls at the SAG mill, rubber linners 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 initiating discussion with liner and grinding media suppliers shortly after the purchasing of the grinding mills.

The grinding media 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 consumption in kg/kWh for normal steel. Liner consumption was calculated and given by the mill manufacturers based on supplied data (including ore abrasion index).

The use 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 offsetting the higher unit price.

The relative performance of high chrome steel 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: CA$ /t for 4" and CA$ /t for 1.5";
- 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. Table 21-45 presents the estimated cost difference related to the consumption of the grinding media.

Table 21-45: High Chrome vs Forged Steel Grinding Media Cost

Table 21-45 shows clearly that high chrome grinding media has a great potential for cost saving for the ball mill operation. High chrome-grinding media become rapidly profitable due to lower consumptions compared to the predicted consumption of conventional forged steel balls.

The mechanical maintenance supplies have been estimated using f the direct cost of the concentrator equipment, which is CA$ M per year or CA$ /t of concentrate.

The instrumentation supplies have been estimated using of the direct cost of the instrumentation for the concentrator, which is CA$ M per year or CA /t of concentrate.

Table 21-46 shows lubricant consumption and costs which were estimated with an in-house mining estimation guide based on hourly cost and estimated hourly operational hours for each major equipment.

Table 21-46: Lubricants Cost Details

21.5.1.8 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. Table 21-47 details the cost of reagents. A lot of CA M has been included for the reagents required for the water treatment plant.

Table 21-47: Reagents Cost Details

21.5.1.9 Other Processing

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. These costs are shown in Table 21-48.

Table 21-48: Other Processing Cost Details

21.5.2 Fresh Water and Water Treatment

The Fresh water cost and Water treatment cost are included in the various process operating costs.

21.6 Tailings Pond Costs

Tailings costs have been estimated based on Roche's experience and database or factorized from similar mining operations shown in Table 21-49. The maintenance and reagent consumption costs for annual maintenance of tailings ditches and dikes as well as the water treatment unit are included in Sections 21.5.1.7 (Consumables) and 21.5.1.8 (Reagents), respectively.

Table 21-49: Tailings Pond Cost Details

22. Economic Analysis

The economic evaluation of the Mine Arnaud project was performed using the discounted cash flow model. The capital and Opex estimates input into the financial analysis model were based on the mine plan developed in this study to process 11,282,880 tonnes of ore per year. The IRR on total investment was calculated based on 100% equity financing. The NPV, based on a discounting rate of resulting from the net cash flow generated by the project was also calculated, as the payback period as an additional financial measure. Finally, a sensitivity analysis on key variables and parameters was performed.

The following assumptions were made for the financial analysis:

- Phosphate market price of CA$ per dry tonne of P_2O_5 concentrate "Freight On Board";
- Concentrate transportation costs (from port) are included in the P_2O_5 market price;
- The construction period was estimated at 30 months divided in three years (-3, -2, -1) which represent duration periods of 6, 12 and 12 months respectively.
- For simplification purposes, the costs related to the first 6 months, representing year -3, were transferred to year -2.
- Constant exchange rate of $ (US$:CA$);
- Salvage value of CA$ was considered.
- The restoration costs that will occur once the project will be terminated, i.e., after year 31, were discounted at year 31 (at %) and then treated as a normal charge.

The economic analysis of the Mine Arnaud project was done with and without tax consideration. An accountant consulting firm, PWC based out of Montreal, reviewed the entire economic model. PWC did a detailed review of all the project costs in order to calculate depreciation, applicable credits, etc., to come up with an after taxes result.

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

Table 22-2: Cash Flow Model

22.2 Net present value, internal rate of return and payback period

The financial analysis results of the Mine Arnaud project for the base case scenario are calculated as:

- CA$ M net present value (after tax) at % discount rate;
- % internal rate of return (after tax);
- years payback period (after tax) after start of production.

22.3 Taxes, royalties and interests

22.3.1 Taxes

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	15.0 %	
Provincial income tax rate:	11.9 %	
Quebec mining tax:	16.0 %	(Quebec levies mining taxes under the Mining Tax Act at a flat rate of 16.0 % since 2012 for all material extracted from Quebec soil)

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 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 December 9th 2013, the Quebec Government has voted 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 proposed 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 CA$ 80 M, the royalty will amount to 1% and if the production value is greater than CA$ 80 M, the royalty will be 4% of the value of the ore mined.In addition, the Quebec Government proposed to implement a progressive mining tax rate (instead of actual flat rate of 16 %) for companies that generate profits and that will not be subject to the minimum royalty.The mining tax will be 16% if the profit margin of the company is between 0% and 35%. If profit is between 35% and

50% the tax will be 17.8% and if the profit is between 50% and 100%, the tax will be 22.9%. The part of this study which reflects the new legislation was produced by Price Waterhouse Cooper in accordance with the new legislation, and verified by SGS Canada Inc afterwards.

22.3.2 Royalties

No royalties other than those mentioned above were considered in this study.

22.3.3 Debts and Interests

The opinion of SGS is that a FS should measure the inherent value of a mineral project, not the ability of an owner to finance a project on favourable terms. For this reason, the economical analysis presented in this study is assuming 100% equity cashflows, so no interest attributable to capital financing was considered.

22.4 Sensitivity analysis

The sensitivity of the pre-tax and pre-financed NPV and the 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;

We shown the parameters listed above for a discount rate of which is the base case and .

The results of the sensitivity analysis are presented Table 22-3 and

Figure 22-1 through Figure 22-5:

Table 22-3: Sensitivity analysis (NPV and IRR)

Figure 22-1: Sensitivity Analysis (NPV)

Figure 22-2: Sensitivity Analysis (NPV)

Figure 22-3: Sensitivity Analysis (NPV)

Figure 22-4: Sensitivity Analysis (NPV, IRR)

Figure 22-5: Sensitivity Analysis (NPV, IRR)

The most important impact on the NPV of the project and on the IRR are the price and the weight recovery, and lesser impact are the operating costs followed by the capital cost. An increase in the interest rate increases the impact of the capital cost variation which becomes as important as the operating costs in terms of NPV impact. Any price inferior to US$ of concentrate and weight recovery value less than % will bring the project to the break even point.

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 4 properties are contiguous to the Mine Arnaud Property with one being located inside the Mine Arnaud Property. Two of the contiguous properties are registrerd under Surface Mineral Rights and the two others properties are registered under Philippe Tremblay and Lafarge Canada Inc., on which no information is available.

A search in the SNL Metals & Mining (formely Metals Economics Group) database did not identify any active phosphate projects within 50km of LONG 66°31'38'' W and LAT 50°16'13'' N.

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 30th 2014, three companies had declared previous work on their claims (GESTIM), one of which was Mine Arnaud. The other two companies are Corporation Éléments Critiques, who declared technical work, and Équipements Lalancette Inc. who declared striping and excavating on their Pointe-Noire Property. The Corporation Eléments Critiques claims are no longer active, their claims are expired and some of these claims now belong to various owners (Mario Richard/Armand Boulanger and Glen Griesbach).

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 than Surface Mineral Extraction (peat, sand, gravel and aggregates).

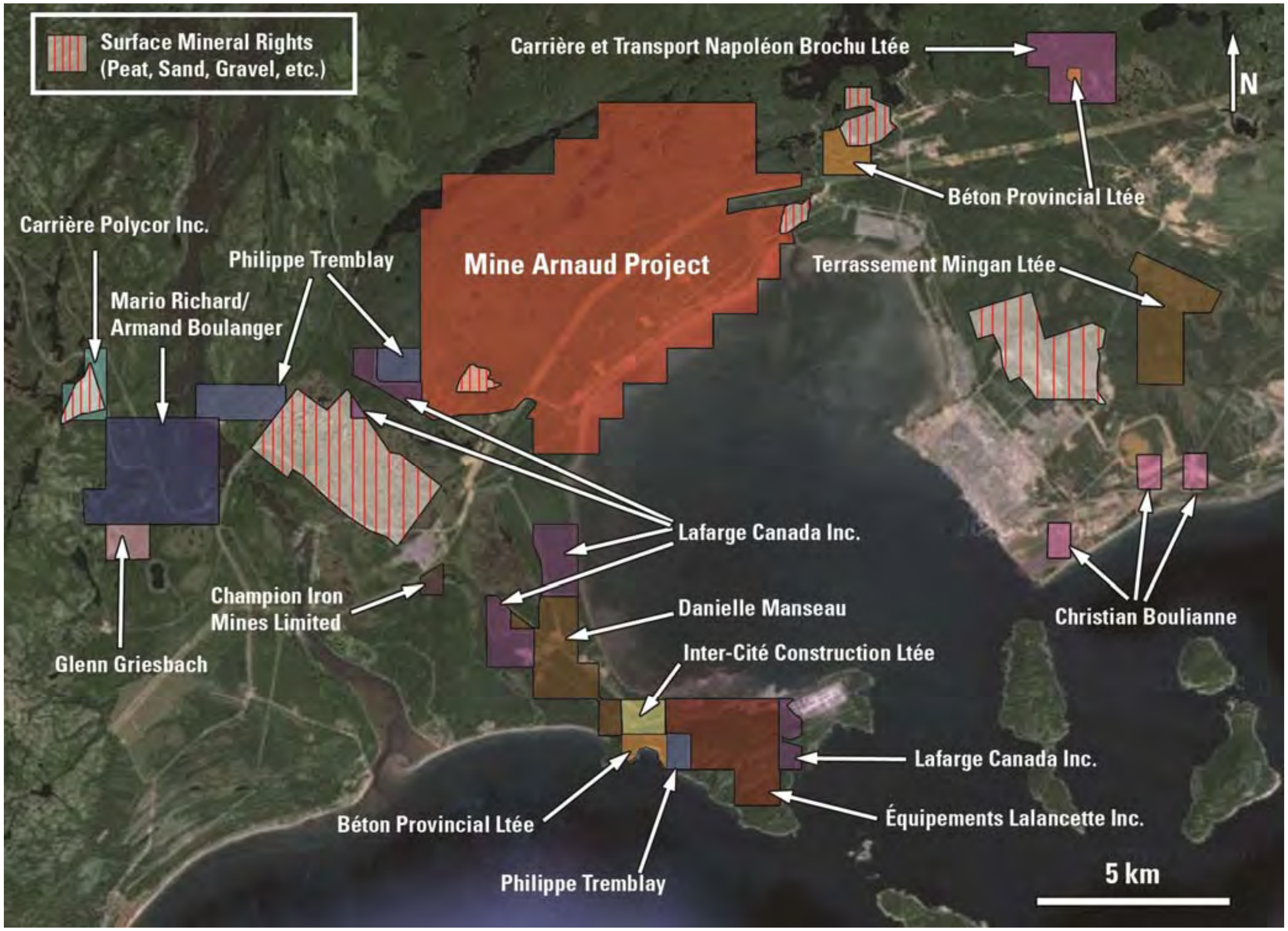

Figure 23-1: Adjacent Properties in the Vicinity of Sept-Îles

24. Other Relevant Data and Information

24.1 Project Schedule

The Project Development schedule has been developed by Roche-Ausenco in close collaboration with Mine Arnaud. This schedule covers the entire time period from the BAPE mandate and hearing to the beginning of commercial operation of the concentrator plant. (Figure 24-1)

The overall project development is divided in three distinct periods:

- Pre-Execution Phase;
- Execution / Construction Phase;
- Ramp-up and Operational Phase.

The schedule considers that the project will be developed as an EPCM type project. If other formulas are considered, some modifications to the schedule might be required. An important milestone is the Environmental decision (decree) from the Government of Québec which will trigger the basic engineering and value engineering phase of the project.

24.1.1 Pre-Execution Phase

Once the decree from the Quebec's Government is received, Mine Arnaud will begin a one year pre-execution phase. The objective during that phase is to minimize the risks associated with the project by:

- Securing agreements with main stakeholders of the project such as Hydro-Quebec and the Port of Sept-Îles;
- Ordering the long lead items that could affect the project's schedule and preparing bidding packages to start the site preparation work shortly after the final decision to proceed with the project (Decision Gate 4, "DG4");
- Value engineering: the optimization of the engineering of the project in order to reduce costs;
- Progressing with the permitting process.

The pre-execution phase deliverable will be a report including a revised Capex, Opex, and project development schedule which will serve as a basis for the final decision to proceed with the project (DG4). The Pre-Execution Phase spans a twelve (12) month period.

24.1.1.1 Securing Agreements with Major Project Stakeholders

Securing commitments with major project stakeholders is essential to the success of the Mine Arnaud project. It is assumed that during the Pre-Execution phase, Hydro-Quebec and the Port of Sept-Îles will proceed with the pre-project studies and that the costs and schedule for the construction work from both parties will be available before DG4. During the same period and prior to DG4, an agreement should be made between Mine Arnaud and the owner of the Chemin Arnaud railway to decide on which terms the railway will be used.

24.1.1.2 Long Lead Items and Other Commitments

During the Pre-Execution phase, some commitments, purchases, and procurement activities will have to be performed at an early stage in order to minimize impacts on the Project Schedule. As such, the SAG mill and the two (2) ball mills will have to be purchased in priority as their delivery times are currently on the order of thirteen (13) months, but may increase with an increase in mining activities. The engineering of the site preparation work will have to be advanced in order to commence construction activities very early during the EPCM stage. Other pieces of equipment including the crusher, ship loader, transformers, and mining haul trucks have long delivery times and in order to be available on-site in time for the erection, purchase will have to be done immediately after DG4, which means that all the procurement activities shall be prepared prior to that date.

24.1.1.3 Value Engineering

Value Engineering will require a complete review of the engineering done so far and will be performed to optimize the equipment sizing, buildings design and site layout. This should lead to savings in the Capex and Opex and the goal is to improve the project's economics.

24.1.1.4 Permitting

The permitting process is ongoing and will have to continue following the established procedures for mining projects in Quebec. With the tabling of the ESIA and the completion of the BAPE public hearings, the government has made its recommendations which Mine Arnaud is addressing. This Feasibility Study will be sent to the government in order to complete the MDDELCC environmental analysis report, which should lead to the issuance of the decree from Quebec's Government. The pre-execution phase will begin after the decree is issued. The process to get specific Certificate of Authorizations will have to begin during the Pre-Execution phase in order to avoid delays in the project's construction.

On the Federal level, the permitting process is also ongoing and will continue through the Pre-Execution phase in order to get a decision from the Minister of the Environment whether or not the project meets the environmental requirements. The permitting process at the Federal level is also crucial for all construction that could potentially affect fish habitats.

24.1.2 Execution/ Construction Phase;

24.1.2.1 Engineering, Procurement, and Construction Management Contract Award

In order the meet the end date, the Engineering, Procurement, and Construction Management (EPCM) Contract will have to be awarded during the Pre-Execution phase. The project execution schedule assumption is that the transition between the Pre-Execution phase and the EPCM phase will be done quickly and smoothly in order to allow for the initiation of site preparation work, procurement, and detailed engineering immediately following DG4. The EPCM phase, from DG4 to the end of construction activities, spans a period of thirty (30) months.

24.1.2.2 Construction

Where feasible, construction activities will be conducted early in the EPCM phase to enable development of certain aspects of the project. Earth moving activities, such as pit pre-stripping and concentrator pad and access road construction will begin shortly after DG4. As described earlier, final purchases of several pieces of equipment will have to be completed immediately following DG4 to ensure their availability on time.

Once the permits are in place, the main construction activities will start with the construction of the new railway line and the removal of the existing one, the erection of the crusher, storage dome, concentrator, and silo, as well as all related infrastructures. In parallel, construction activities at the Port will be conducted. The Port Authority of Sept-Îles is to refurbish an existing wharf at Pointe-Noire. Erection of the railway spur, the unloading system, and the silo will be conducted by Mines Arnaud. The construction period should last twenty-seven (27) months. During that period, the tailing management facilities will be built using rock excavated from the mine. Construction activities include periods varying from 1 to 4 months for the commissioning of the various areas.

24.1.2.3 Commissioning

Once the commissioning of all areas is completed, an additional period of three (3) months has been allocated for the overall commissioning of the operations.

24.1.3 Ramp-up and Operational Phase

Once construction and commissioning are finalized, the six (6)-month ramp-up period will start in order to transition the project to full scale commercial operation. Start of full scale commercial operation is defined by the Canadian Revenue Agency as: *“The CRA’s administrative practice is that a mine has reached production in reasonable commercial quantities when it has operated at 60 percent capacity or more for 90 consecutive days.”*

Figure 24-1: Project Schedule

24.2 Risk Assessment

A risk assessment on the project was performed to measure the variation potential in the different cost elements that make up the capital costs.

The risk was assessed on the major cost areas, as contained in the capital cost estimate. They are summarized in the table below:

Table 24-1: Cost elements of the capital cost estimate

200	Mining Equipment
210	Open Pit & Auxiliary Services
220	Tailings Management & Facilities
230	Crusher, Storage & Conveying
240	Concentrator
250	Silos & Load-Out
260	Bulk Earthworks, Landscaping & Fencing
270	Water Intakes & Distribution Systems
280	Sanitary Systems
290	Roads, Overpasses & Parking
300	Safety
310	Fuel Distribution and Storage
320	Fire Detection and Prevention
330	Auxiliary Buildings - Non Process
340	Rail Diversion
350	High Voltage Substation
360	Automation, Instrumentation and Communications
370	Rail Transportation
380	Site & Material Handling
390	EPCM Costs
400	Indirect Costs
410	Owner Costs
420	Transfer From Operating Costs

Furthermore, other risk factors that can affect costs were evaluated. Those are listed in Table 24-2.

Table 24-2: Other risk factors that can affect project costs

450	Bad EPCM contractor
460	Need for construction camp
470	Flooding of area
480	Mine Arnaud affecting Hydro Quebec lines
490	Strike, lock out
500	Delays in the permitting
510	Decision process for DG4
520	Delivery time of grinding mills
530	Harbour not completed in time
540	Hydro Quebec not completed in time
550	Environmental issues
560	Delayed agreement with rail owner
570	Accident on the mine site, during and after construction
580	Derailment
590	Mine Arnaud cannot find human resources for the project
600	Lack of continuity of personnel on the project
610	Extreme winter weather
620	Managing of thick layer of clay
630	Shortage of manpower
640	Shortage of construction equipment
650	Lack of delivery of consumables
660	Bad productivity of construction labour
670	Change of ship size

The risks to the project were evaluated according to their probability of occurrence, and its potential cost impact in the event of an occurrence. The resulting data was then incorporated in a Monte Carlo simulation. A total of 10,000 iterations were performed by the simulation, resulting in a risk profile that warrants a contingency, based on the P_{50} results.

Details of the A summary of the risk assessment is provided in APPENDIX AA – Risk Analysis.

25. Interpretation and Conclusion

25.1 Deposit

The deposit's geology is now well understood and increasing drilling shall not change significantly the geometry and interpretation of the mineral deposit. However, information related to structures and displacements due to faulting could be better defined and taken into account when conducting further geological modeling.

Work done on chlorine behavior and prediction of the values for the concentrate; enabled 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. The Mineral Resources also have values for Cl in the concentrate under acceptable threshold except for the inferred resources, which show average Cl values for the concentrate above 0.14%.

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 alone accounts for an increase of 15% Cl in 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 and 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 modelling 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 forming a new zone 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 parameters and classification criteria 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 (WRec, Predicted P_2O_5% of the concentrate and Predicted Cl% in the concentrate).

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 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.

Mineral Resources were reported inside the pit shell and under the overburden surface using two different cut off grades. The Nelsonite layer was reported at a 2.05% P_2O_5 cut off whereas all other units were reported using a 1.65% P_2O_5 cut off grade. This variable cut off grade accounts for the different dilution ratio of the thinner Nelsonite layer.

Table 25-1 Mineral Resources for the Mine Arnaud Sept-Îles deposit

Category	Material Type	Cut Off (%P2O5)	Tonnage (Mt)	Grade (%P2O5)	WRec (%Wrec)	Conc. Grade (%P2O5)	Conc. Grade (%Cl)
Measured	California	1.65	27.615	2.94	6.86	38.47	0.1719
	Combine	1.65	319.168	4.01	9.25	38.89	0.1227
	Surrounding	1.65	28.827	2.31	5.46	37.91	0.1524
	Nelsonite	2.05	37.966	5.88	13.41	39.42	0.0823
	TOTAL		413.576	4.00	9.21	38.84	0.1243
Indicated	California	1.65	8.230	3.14	7.30	38.54	0.1556
	Combine	1.65	89.467	4.29	9.87	38.98	0.1213
	Surrounding	1.65	24.951	2.31	5.45	37.92	0.1699
	Nelsonite	2.05	9.264	6.19	14.09	39.48	0.0867
	TOTAL		131.911	3.98	9.17	38.79	0.1302
Inferred	California	1.65	0.001	1.82	4.40	37.23	0.1651
	Combine	1.65	-	-	-	-	-
	Surrounding	1.65	44.634	3.36	5.45	38.67	0.1603
	Nelsonite	2.05	-	-	-	-	-
	TOTAL		44.635	3.36	5.45	38.66	0.1603

Notes:

- The mineral resource estimate has been conducted using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definitions Standards for mineral resources in accordance 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.
- SGS did the supposition that the diluted material will have the same %Cl that the ore blocks where the dilution is applied.
- Resources are constrained by the pit shell and the topography of the overburden layer.
- Effective date 11-08-14.

25.4 Mining

The Mine Arnaud project is a very straight forward mining operation, wide and long open pit, low stripping ratio, etc. The material will be mined by a single open pit, which will have 28 years of production life following a two 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 is expected to be done by the project owners 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. To facilitate the hauling operation two ramps are proposed. The overburden and waste rock material will be hauled to the overburden and waste disposal areas near the pit. Some of the waste rock will be returned to the shallow western part of the open pit to lower the waste left on surface. The run-of-mine mineralization will be delivered by large mining trucks to the primary crusher or stockpiles near the crusher area.

The production mining fleet, for ore and waste, of 140 tonne trucks and 15 m^3 excavators is in line with standards from heavy equipment manufacturers. The overburden will be transported by 40 tonne articulated trucks that were selected because of their low environmental impact. There are possibilities to optimize the developed mine plan using different mining approaches, new high grading scenarios, different mill feed, etc. Those optimizations could impact the project NPV.

25.5 Processing

The process for treating Mine Arnaud Ore was studied intensively over the last two decades. The process as is today should be considered robust. Nevertheless it is recommended to pursue testwork programs until the beginning of the operation in order to help achieving a good understanding of the impact of the ore variability on the efficiency of the beneficiation plant. This should help reducing the ramp up period leading to a production rate at the designed capacity of the plant.

25.5.1 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 is the impact that it will have on the recovery, the consistency of the concentrate grade and eventually the operating cost.

Another risk associated with ore variability concerns the pumping problems that may occur if grade varies considerably. A 10% overdesign factor was 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.

The way to mitigate the effect of the ore variations is to plan for proper blending tools while doing the mine planning. Sufficient equipment and space for material handling of the ore prior crushing will also be required.

Recent testwork program conducted at Corem during winter and spring 2014 with samples taken at depth showed that no major issues are to be expected toward the end of the mine life because the ore at depth responds similarly to the various ore types previously studied. Nevertheless 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 grains 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.

Initially it was identified that 4 ore types with various proportions were present in the Mine Arnaud deposit: Nelsonite (S2) (20%) – Upper (S3) (25%) – Railroad (55%), and California (S4) (0 to 4 %). It is now considered that Upper (S3) and Railroad ore types would be easily combined and are now named Combined. The Combined ore would represent 80% of the feed to the beneficiation plant.

The cleaner flotation at the laboratory scale is still not completely mastered, due to the small scale of the laboratory equipment and due to fast kinetics at the cleaner stage, so it is difficult to extract data from it. The focus at the moment is to improve the methodology in the laboratory and more work is required even if the present laboratory work is going in the right direction.

25.5.2 Dewatering cyclone and Flotation

Dewatering cyclones were added to the circuit to enhance the efficiency of the conditioning. A high pulp density, of 50% solids, should facilitate the contact between the reagents and the pulp and therefore enhance the flotation process. A portion of dewatering cyclone overflow will be used to re-dilute the pulp to 35% solids, prior to flotation. The remaining overflow should represent less than 2% by weight. It will be sent to the tailings pond and will represent a P_2O_5 loss of approximately 2% or less. The only way to reduce this loss would have been to put a thickener instead of dewatering cyclones but the capital cost would have been higher. The risks associated with the dewatering cyclone is that, if very fine grinding of the apatite occurs for some reasons in the grinding mills, there is a potential of additional losses of P_2O_5. The way to mitigate the loss at the cyclone overflow is to have a very good control on the grinding by minimizing the production of slimes or very fine material. In practice, controlling the amount of fines may become a challenge and it is not possible to always foreseen if the ore will create a lot of fines.

The current design of the building allows for some space for the addition of future flotation columns in case of potential expansion. Considerable savings are possible by reducing the size of the flotation area.

25.5.3 Wet High Intensity Magnetic Separation (WHIMS)

All testwork programs indicated that there will be no major problem achieving 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. The current design provides some space in the beneficiation plant if deemed necessary at some point in time during the mine life but major savings would be applicable to the project if the size of the building was reduced by the abandon of this option. If ever needed one day it would always be time to add and an extension to the buildings for a WHIMS operation. 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 by generating a high grade P_2O_5 product that do not meet Yara's specifications for their apatite concentrate.

If for some reason, it is required one day to use WHIMS to achieve the quality specifications, it will be required to find a market for the magnetic by-product or to regrind and reprocess it.

25.5.4 Filtration

A belt filter was selected 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. 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.

25.5.5 Flash Dryer

To mitigate the difficulty of obtaining proper moisture content of 1.0% (+/- 0.5%) in the final dried concentrate, both suppliers, FLSmidth and GEA Barr Rosin, 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.

25.6 Project Infrastructure

25.6.1 Electrical Infrastructure

A request for a new Planning Study was sent to Hydro-Quebec in May 2014. The study is expected to be issued at the end of August 2014. A meeting was held on 9 July 2014, in which preliminary results of that study were presented by Hydro-Quebec. The costs used in this study are based on that meeting. In general, the solutions found by Hydro-Quebec for tie-in to the network and stabilization of the 161 kV transmission line L1619 are adequate and less expensive than in 2011.

25.6.1.1 Electrical Distribution at the Mine-Concentrator Site

The electrical installed load is about 84 MW and the running loads are about 64 MW for the Concentrator-Mine site.

25.6.1.2 Electrical Distribution at the Port Facilities

The power supply for the port site facilities will be taken from an existing 25 kV line. The electrical installed load is about 2.4 MW and the running loads are about 2 MW for the Port Facilities.

25.6.1.3 Risks and Uncertainties

When Hydro-Quebec will issue their study at the end of August 2014, it should be verified for any changes from the meeting held on 9 July 2014. The investments required, if any, at Hydro-Quebec Arnaud sub-station remain to be evaluated.

Hydro-Quebec based its study on transportation of no dangerous goods by rail under its transmission lines under which Mine Arnaud has no control.

25.6.2 Control and Communication Infrastructure

Plant wide communication and integrated control system will facilitate safe and efficient operation at both the concentrator and port areas. Priority will be given to systems where local support is available.

25.6.2.1 Process Control System

A DCS based process control system including operator station located in concentrator building control room and remote I/O and controller cabinet located in different electrical room. PLC based controller and operator station supplied with package will be located in crusher control room, SAG mill, ball mill and in train load-out area. Redundant communication will link the different control system.

25.6.2.2 Communication systems

A fiber optic cable network will provide communication facility between the different buildings and areas. A redundant microwave communication link will provide communication between the concentrator and the port areas. The redundant fiber optic network will serve the different communication and monitoring services such as process control, CCTV, telephone and control access systems. Reliability, safety and non-interference communication between in plant systems and external to the plant will be provided by means or dedicated server, firewall and router.

25.6.2.3 Radio communication

A radio communication system will cover both concentrator and port site. Walkie-talkie and mobile units will serve operation and maintenance vocal and data communication.

25.6.3 Transportation and Relocated Railroad

The Mine Arnaud mine site is crossed by the Chemin de Fer Arnaud, a railway owned by Cliffs Natural Resources. 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 possible the use of that railway for transport of apatite concentrate to the Port of Sept-Îles.

25.6.3.1 Diversion Track

The diversion track consists of a new main line and a load out siding to be constructed to relocate the existing main line, for the purpose of accommodating the proposed mine site. The siding will allow the loading of the railcars without blocking the main line.

The silos and railcar load-out area consists mainly of two (2) concrete storage silos providing the equivalent of about one (1) day of production each.

25.6.3.2 Rail Transportation

Rail haulage must be contracted out to CFA (Cliffs) as they control the rail corridor and the crews that work on it. The locomotives to perform this movement will be supplied under contract by the rail operator. These locomotives should be capable of handling the 39 loaded cars on the grade present on the CFA subdivision, each having a capacity to carry 105 tonnes of apatite concentrate.

This operation requires 39 rail cars in service. With a bad order factor of 5%, the total number of railcars to be purchased is 41. The 39-car train is expected to be able to handle 1.30 Mt/y of planned annual throughput.

25.6.3.3 Risks and Uncertainties

The rail alternative is the preferred mode of transportation retained for the Mine Arnaud project providing that Mine Arnaud can reach a reasonable agreement with Cliffs. During this study, there were no negotiations with Cliffs to firm up an agreement. This will have to take place as the project moves forward.

The transportation cost to be paid to Cliffs used in this feasibility is CA$ 3.00/t of concentrate. It is NOT based on an agreement or recent discussions with Cliffs. Beyond this transportation cost, Mine Arnaud would consider an alternative mode of transportation.

25.6.3.4 Alternative Transportation Modes

Trucking is the selected alternative transportation mode. This option, evaluated by the engineering firm ‘Strudes’ at the concept level, shows a capital cost in the same range as the one with the train. The operating costs were evaluated at a slightly higher price per tonne than the train, but remained acceptable. The transportation price used in this feasibility by train is CA$ 3.00/t of concentrate and the alternative way with truck is estimated at CA$ 3.50/t of concentrate.

25.6.3.5 Risks and Uncertainties

The trucking option has never been studied beyond the concept level. It should be developed further at feasibility level as this may be the most economical alternative.

25.6.4 Port Facilities

Apatite concentrate is transported by train up to the Mine Arnaud Port Facilities located around Anse à Brochu in the Pointe Noire area of the Port of Sept-Îles, about 17 km away from the mine site. For this Feasibility Study, Mine Arnaud requested Roche-Ausenco to develop a concept for port facilities located near the existing Berth 31, in a different location of the Pointe Noire area than in 2011.

25.6.4.1 New Concept Considered for the Port Facilities

Mine Arnaud rail unloading and storage areas are located near the existing rail tracks within the Port of Sept-Îles property and where bedrock is anticipated near the surface. The four (4) storage silos, with provision for a future fifth silo, are located near the rail and an access road. From the silos, a series of two (2) conveyors enclosed in tubular galleries will transfer the apatite concentrate to the Berth 31. The first conveyor runs at grade in the westward direction to the first transfer tower. From the transfer tower, a second elevated conveyor oriented from south to north will cross the Bay of Sept-Îles at two (2) locations as well as over future iron ore installations for the new multi-user dock currently being built by the Port of Sept-Îles. The second elevated conveyor will terminate at a second transfer tower located at the east side of Berth 31.

25.6.4.2 Marine Structures for Berth 31

The Port of Sept-Îles will have to make investments at Berth 31 to adapt it for Mine Arnaud. The Port will have to reinforce the dock to suit the new travelling shiploader and tripper loads, extend it to the east, add a platform for a transfer tower and realign the access bridge. This will be indirectly charged to Mine Arnaud as an additional yearly fee.

Further study work is required to verify the dock structure and establish the cost to reinforce it for a travelling ship loader. At this stage, Mine Arnaud is considering that beyond CA$ 10 million, like building a new dock would be considered. Therefore, a yearly payment corresponding to a 3% interest over 30 years for CA$10 million is included in the Opex.

25.6.5 Screen berm and overburden piles

The weak soil layer identified has effects on stability analysis, maximum height of the screen berm and dimension of buttress berm required to stabilize the pile. Construction of a buttress berm would imply tree cutting and soil stripping on a larger area than supposed, and soil improvement would imply additional soil preparation cost.

25.6.6 Risks and Uncertainties

The new concept for the Mine Arnaud Terminal was developed in a short time and is therefore based on many assumptions that will need to be confirmed. During this study, there was no discussion with Cliffs on the Port facilities and relatively few exchanges of information with the Port of Sept-Iles to firm up the concept. The main uncertainties ar as follows:

- The Port of Sept-Îles will make the necessary investments at Berth 31 to reinforce the dock to suit the new travelling shiploader and tripper loads, extend it to the east, add a platform for a transfer tower and realign the access bridge;

- Cliffs will allow Mine Arnaud to build a new 2.28 km rail spur with siding on its property to bypass its Wabush Yard. The soil conditions will be adequate for the addition of rail;

- An area for the terminal within the Port of Sept-Îles property at Anse à Brochu will be available to Mine Arnaud. At that location, the unloading station and the storage silos can be built directly on the bedrock;

- The Mine Arnaud Facilities will be located near the new Multi-user Dock and Berth No. 31 in an area already equipped with many services. 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.

26. Recommendations

26.1 Geology

1. Use present drilling data in order to establish and position the structural breaks of the deposit and include them in the next resources modeling of the mineralized envelops;

2. 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;

3. Analysis for chlorine in the wedge area was biased by hole 1166-10-83. In order to increase the level of in situ Cl continuity. Hole 1166-10-83 should be re-assay for chlorine. Still, this will have no impact as mill feed ore is control by in situ K_2O to assess Cl in concentrate.

26.2 Mining Methods

1. Assess various mine plans and high-grading scenarios in order to maximize the project profitability;

2. Analyze the pros and cons of using different suppliers and larger size mining fleet in order to lower mining cost, while taking in consideration the impact on the mining dilution;

3. Once the final mine plan will be developed, use specific software (for example TALPAC from Runge Inc.) to precisely estimate the required mining fleet;

4. 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;

5. Precisely recalculate mining costs attributable to the variation of the height of the bench faces (5m and 10m) and include the results into the economic analysis;

6. With future investigations, lower the amount attributable to the contingency by detailing more precisely the project capital expenditure;

7. The sustaining capital should be reviewed by allowing a salvage value for heavy equipment renewals as the present study was done without salvage value, except at the closure of the operations.

26.3 Mineral Processing and Recovery Method

In order to mitigate the risks normally associated with the development of new process plant, Roche recommends the following:

1. To reduce the process operating cost, additional testwork is recommended with similar reagents from different suppliers. With these test results, it will be possible to evaluate the circuit response and reagent consumptions and to optimize the reagent selection.

2. Perform a trade-off study to save energy by using filter press instead of belt filter to reduce the concentrate moisture content prior to feeding the flash dryer. Filter press filtration would involve increased Capex and Opex for the filtration stage, however much less energy is required to reduce the moisture content by filtering versus vaporization; thus there could be savings in the long term. Pressure filtration tests are required to perform the trade-off study between the two technologies.

3. Results received from SGS in July 2011 show that additional grindability testworks should be conducted on the material in order to confirm the proposed SAG-Ball Mill circuit. Additional grindability tests on the newest drill core sample would help assess the variability in terms of grindability and abrasiveness and reconfirm the sizing of the grinding mills.

4. A study to better understand the magnetite distribution will have to be performed with the following objectives:

- To evaluate the variability of the magnetite distribution within the deposit and its impact on the grinding circuit and on P_2O_5 recovery;
- To observe how blending requirements could be achieved using a unique stockpile;
- To fully assess if magnetite can offer an economical value by having a clear evaluation of the magnetite quality and content over the deposit.

5. Building size optimization could result in Capex savings with the following activities:

- Replace the two (2) ball mills by one (1) of a bigger size:
    - Reduction of Capex for the purchase of equipment;
    - Reduction of Capex by reducing the size of the building: eventually the building could have one less bay (7m wide) over 24 bays (7m long each) for a potential reduction of the footprint of the building of 1176 m^2 or approximately 10%;
- Eliminate the space for future columns and Wet High Intensity Magnetic Separator;
    - Reduction of Capex by reducing the size of the building: eliminating the space for additional columns in prevision of future expansion as well the space for

WHIMS can help reducing significantly the foot print of the building. The empty floor space planned for the future equipment corresponds to 588 m² or approximately 5% of the beneficiation plant footprint.

- Optimization of the space requirements for offices, maintenance around the equipment, and equipment installations can potentially help reduce the beneficiation plant footprint by approximately 3 to 5 %.
- Reduction of the height of the Flash Dryer building if possible after discussion with Flash Dryer suppliers.
- Other minor optimization should be considered including reduction of the building height by approximately 0.5m to 1 m for certain portion of the beneficiation plant building.

26.4 Project Infrastructure

1. Railroad Overpass:

- Optimization of the overpass should be studied by changing the location of the railroad.
- A multiple-arch overpass should also be considered at the preliminary engineering phase. The cost could be reduced by decreasing the span and the amount of overburden to stabilise the arches.
- Design one overpass for trains with heights exceeding the height limit. In the event, the safe passage of a railroad train with abnormal height should be insured. One arch should be able to carry out said task safely only if multiple arches are considered.

2. Plant Potable Water Supply:

- Consider purchasing the small strip of land shown in dashed lines, as shown in Figure 18-5. This will impact the need to have a booster pump house;
- Increase the depth of the water pipe so it can survive the impact of a tailing dam failure;
- Add reinforcement to the underground piping near the location of the deflection dam along the Clet river.

3. Gate House:

- Consider the possibility of having a gate house located on the side of the road rather than the middle of the road;
- Consider adding a fire house station near or at the gate house;
- Gate house should not rely on the plant main electrical system but have a separate power system for emergency situations;
- Add an emergency room near or at the gate house for ease of transportation if needed.

4. Parking lot:

- Orientation of the parking lot is parallel to the administration building with two (2) access roads. This should be revised and rotated by 90 degrees to allow greater fluidity when entering and exiting the parking lot;

- Consider relocating the parking lot near the access road, thus having only one (1) road to enter and leave the plant.

5. Electrical Infrastructure:

- Verify Hydro-quebec study when it will get issued at the end of August 2014. It should be verified for any changes from the meeting held on 9 July 2014;

- Hydro-Quebec based its study on transportation of no dangerous goods by rail under its transmission lines on which Mine Arnaud has no control. Mine Arnaud should further clarify this issue with Hydro-Quebec.

6. Controls and Communication Infrastructure:

- There might be savings in optimizing these items, like avoiding redundancy.

7. Rail Infrastructure and Alternative Transportation Mode:

- The rail alternative is the preferred mode of transportation retained for the Mine Arnaud project providing Mine Arnaud can reach a reasonable agreement with Cliffs. During this study, there were no negotiations with Cliffs to firm up an agreement. This will have to take place as the project moves forward.

- The trucking option has never been studied beyond the concept level. It should be further developed, as this may be the most economical alternative.

- Define further the 2.2 km rail extension at the port to bypass Cliffs Wabush Yard by verifying soil conditions and available clearances.

8. Port Facilties:

- For this Feasibility Study, a new concept for port facilities located near the existing Berth 31 was developped A list of assumptions is provided in the report. There was little time to validate assumptions and optimize the design. This will have to take place as the project moves forward.

- An important assumption is that the Port of Sept-Îles will make the necessary investments at Berth 31. Further study work is required to verify the dock structure and establish the cost to reinforce it for a travelling ship loader. Additional study work could be performed to evaluate the existing Cliffs ship loader or an alternative with two (2) fixed ship loaders.

26.5 Water Management

Additional site investigation is recommended to delineate bedrock profile and overburden thickness as the project is advanced to subsequent levels of design. The additional site investigations would be used to delineate the bedrock profile and overburden thickness along the ditch alignment and confirm hypothesis made for ditch design, sumps and pumping stations location.

No drilling or test-pitting has been made in several areas of the TSF, the thickness of overburden and the depth of the rock is unknown. The depth of the rock has been estimated on the basis of available information (boreholes further south, superficial deposit maps, topographic maps, satellite images). Construction costs of infrastructures for water management (ditches, sumps and pump stations) therefore assume that 10% of the volume of material excavated around the TSF will need to be blasted. It should be noted that these costs could increase if more rock is encountered on surface or if additional pumping stations are required.

Water storage at the beginning of the project is considered sufficient with the tailings cell #2 used as a reservoir during the first years. The construction sequence for the tailings cell#2 must fit with the start up period to make sure that both reservoir (tailings cell#2 and storage pond) can provide the process water. A rigorous monitoring of the water balance must be implemented during the start up period.

The discharge of the WTP in the Clet creek considers that no salt water is pumped from the pit dewatering and consequently, it can be discharged in the freshwater of the Clet creek. After the year 7, the hydrological model shows that some salt water could be intruced into the pit and requires management. At that time, if required, the discharge pipe could be extended in the Sept-îles Bay for discharge in salt water.

26.6 Water treatment

The membrane treatment system represents a significant part of the water treatment capital costs. It could be interesting to study other options for the gland seal water and to further evaluate the flow required from this treatment system (ex.: other type of pumps requiring less gland seal water, etc.).

26.7 Stockpiles

Additional site investigations and detailed laboratory testing of the clay material present in the footprint area of the screen berm and overburden stockpile, is recommended as the project is advanced to subsequent levels of design. The additional site investigations would be used to delineate the bedrock profile and overburden thickness along the berm alignment in order to optimize the design of the stabilizing buttress berms and to confirm hypothesis made for stability analysis. Stability models should be updated with the findings and results of the site investigation activities.

Stability models should be completed for interim stockpile layout and slopes as well as the final arrangements to ensure that the stockpiles are stable during the operations. The planned sequencing for developing and filling the rock stockpiles can also be used for scheduling of additional required field investigations.

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