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NI 43-101 TECHNICAL REPORT ON THE FEASIBILITY STUDY ON KEMAG TACONITE PROJECT

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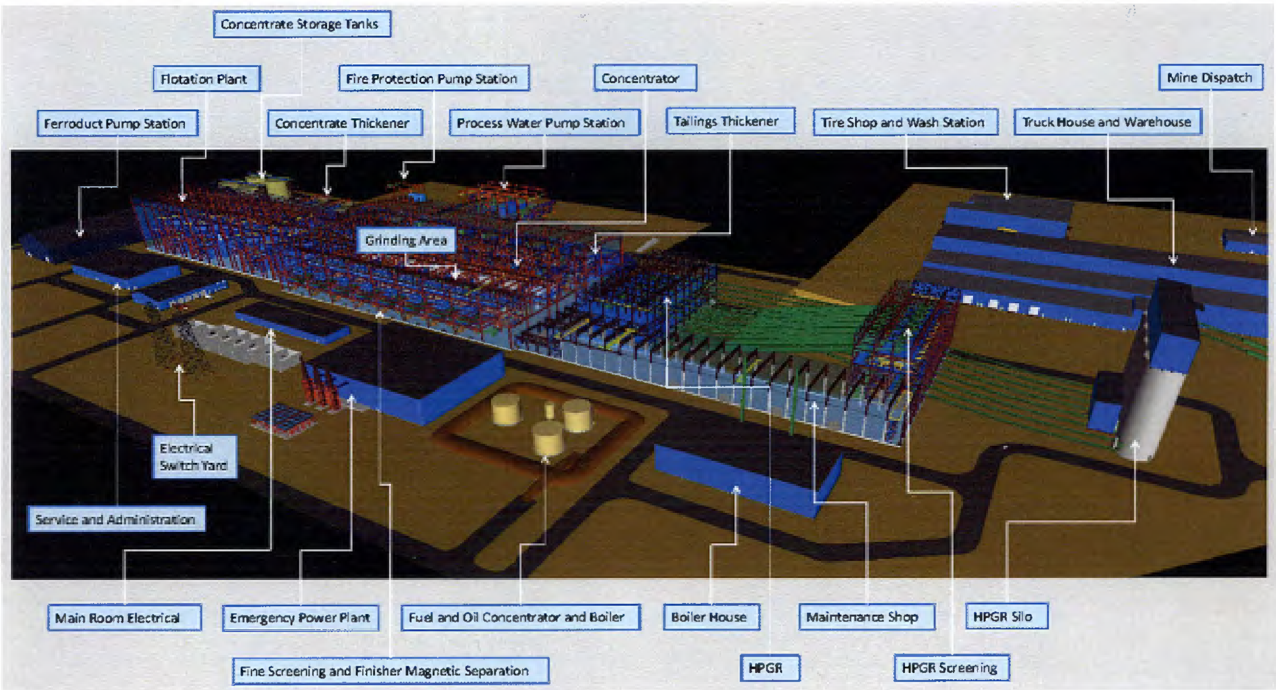
NI 43-101 TECHNICAL REPORT ON THE FEASIBILITY STUDY ON KÉMAG TACONITE PROJECT

Issue Date : May 9th, 2014



FINAL REPORT

Prepared for

New Millennium Iron Corp.

Prepared by

QPs

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Charles Cauchon, Eng.

Effective Date : March 27th, 2014

Project Number : 2013-007

MET-CHEM

IMPORTANT NOTICE

This Report was prepared as a National Instrument 43-101 Technical Report for New Millennium Iron Corp. ("**NML**") by Met-Chem Canada Inc. ("**Met-Chem**"). The quality of information, conclusions and estimates contained herein is consistent with the level of effort involved in Met-Chem's services, based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this Report. This Report can be filed as a Technical Report with Canadian Securities Regulatory Authorities pursuant to National Instrument 43-101, Standards of Disclosure for Mineral Projects. Except for the purposes legislated under Canadian securities laws, any other uses of this Report by any third party are at that party's sole risk.

DATE AND SIGNATURE PAGE – CERTIFICATES

Effective Date: March 27^{h}, 2014

Issue Date: May 9th, 2014

CERTIFICATE OF AUTHOR

To accompany the technical report entitled "*NI 43-101 Technical Report on the Feasibility Study on Taconite KéMag Project*" dated May 9th, 2014 with effective date of December 4th, 2012 (the "**Technical Report**").

I, Yves A. Buro, Eng., do hereby certify that:

1) I am a Senior Geologist presently with Met-Chem Canada Inc. (Met-Chem) with an office situated at Suite 300, 555 René-Lévesque West Blvd, Montréal, Canada;
2) I am a graduate of University of Geneva, Switzerland with the equivalent of a B.Sc. and a M.Sc. in Geology obtained in 1976;
3) I am a member in good standing of the *Ordre des ingénieurs du Québec* (Reg. 42279);
4) I have worked as a geologist continuously since graduation from University in 1976. I have gained direct experience on iron deposits similar to the KéMag Project, as exploration geologist, in Canada, the U.SA., Africa, India, South America;
5) I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined by NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101;
6) I have participated in the preparation of this Technical Report and am responsible for Sections 4 to 12 inclusive, 23, 27 and portions of Sections 1, 3, and 25;
7) I visited the site property that is the subject of this Technical Report between September 18th and 19th, 2012;
8) I have no personal knowledge, as of the date of this certificate, of any material fact or material change which is not reflected in this Technical Report;
9) Neither I, nor any affiliated entity of mine, is at present, under an agreement, arrangement or understanding or expects to become, an insider, associate, affiliated entity or employee of New Millennium Iron Corp., or any associated or affiliated entities;
10) Neither I, nor any affiliated entity of mine, own, directly or indirectly, nor expect to receive, any interest in the properties or securities of New Millennium Corp., or any associated or affiliated companies;
11) Neither I, nor any affiliated entity of mine, have earned the majority of our income during the preceding three (3) years from New Millennium Corp., or any associated or affiliated companies;

12) I am independent of the issuer as defined in section 1.5 of NI 43-101;

13) I have had no prior involvement with the property that is the subject of this Technical Report; and

14) I have read NI 43-101 and Form 43-101F1 and have prepared the Technical Report in compliance with NI 43-101 and Form 43-101F1; and have prepared the report in conformity with generally accepted Canadian mining industry practice, and as of December 4th, 2012, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

This 9th day of May 2014.

Original signed and sealed

(Signed) "Yves A. Buro"

Yves A. Buro, Eng.
Senior Geological Engineer
Met-Chem Canada Inc.

CERTIFICATE OF AUTHOR

To accompany the Report entitled "*NI 43-101 Technical Report on the Feasibility Study on KéMag Taconite Project*" dated May 9th, 2014 with effective date of December 4th, 2012.

I, Schadrac Ibrango, P.Geo., Ph.D., do hereby certify that:

1) I am a Senior Geologist with Met-Chem Canada Inc. ("**Met-Chem**") with an office situated at Suite 300, 555 René-Lévesque West Blvd, Montreal, Canada;

2) I am a graduate of University of Ouagadougou (Burkina-Faso) with a Master Degree in Geology obtained in 1998 and a Ph.D. in Engineering of Darmstadt University of Technology (Germany) obtained in 2005;

3) I am a member in good standing of the "*Ordre des Géologues du Québec*" (1102);

4) I have practiced my profession continuously since 1998. I have gained direct experience on iron projects similar to the Taconite Project, as geologist in Canada;

5) I have read the definition of "qualified person" set out in National Instrument 43-101 ("**NI 43-101**") and certify that by reason of my education, affiliation with a professional association (as defined by NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101;

6) I have participated in the preparation of this Technical Report and am responsible for Section 14;

7) I have visited the site property that is the subject of this Technical Report on September 18th, 2012 and September 19th, 2012 [two (2) days];

8) I have no personal knowledge, as of the date of this certificate, of any material fact or material change which is not reflected in this Technical Report;

9) Neither I, nor any affiliated entity of mine, is at present, under an agreement, arrangement or understanding or expects to become, an insider, associate, affiliated entity or employee of New Millennium Iron Corp., or any associated or affiliated entities;

10) Neither I, nor any affiliated entity of mine, own, directly or indirectly, nor expect to receive, any interest in the properties or securities of New Millennium Iron Corp., or any associated or affiliated companies;

11) Neither I, nor any affiliated entity of mine, have earned the majority of our income during the preceding three (3) years from New Millennium Iron Corp., or any associated or affiliated companies;

12) I am independent of the issuer as defined in section 1.5 of NI 43-101;

13) I have read NI 43-101 and Form 43-101F1 and have prepared the Technical Report in compliance with NI 43-101 and Form 43-101F1; and have prepared the report in conformity with generally accepted Canadian mining industry practice, and as of the date of the certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

This 9th day of May, 2014.

Original signed and sealed

(Signed) "Schadrac Ibrango"

Schadrac Ibrango, P.Geo., Ph.D.
Senior Geologist
Met-Chem Canada Inc.

CERTIFICATE OF AUTHOR

To accompany the technical report entitled "*NI 43-101 Technical Report on the Feasibility Study on KéMag Taconite Project*" dated May 9th, 2014 with effective date of March 27th, 2014 (the "**Technical Report**").

I, Jeffrey Cassoff, Eng., do hereby certify that:

1) I am the Lead Mining Engineer presently with Met-Chem Canada Inc with an office situated at Suite 300, 555 René-Lévesque West Blvd, Montréal, Canada;
2) I am a graduate of McGill University in Montréal with a Bachelor's in Mining Engineering obtained in 1999;
3) I am a member in good standing of the "*Ordre des Ingénieurs du Québec*" (5002252);
4) I have worked as a mining engineer continuously since graduation from university in 1999;
5) I have read the definition of "qualified person" set out in National Instrument 43-101 ("**NI 43-101**") and certify that by reason of my education, affiliation with a professional association (as defined by NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101;
6) I have participated in the preparation of this Technical Report and am responsible for Sections 15 and 16 and portions of Sections 1, 2, 3, 25 and 26;
7) I visited the site property that is the subject of this Technical Report on September 18th, 2012;
8) I have no personal knowledge, as of the date of this certificate, of any material fact or material change which is not reflected in this Technical Report;
9) Neither I, nor any affiliated entity of mine, is at present, under an agreement, arrangement or understanding or expects to become, an insider, associate, affiliated entity or employee of New Millennium Iron Corp., or any associated or affiliated entities;
10) Neither I, nor any affiliated entity of mine, own, directly or indirectly, nor expect to receive, any interest in the properties or securities of New Millennium Corp., or any associated or affiliated companies;
11) Neither I, nor any affiliated entity of mine, have earned the majority of our income during the preceding three (3) years from New Millennium Corp., or any associated or affiliated companies;
12) I am independent of the issuer as defined in section 1.5 of NI 43-101;

13) I have had no prior involvement with the property that is the subject of this Technical Report; and

14) I have read NI 43-101 and Form 43-101F1 and have prepared the Technical Report in compliance with NI 43-101 and Form 43-101F1; and have prepared the report in conformity with generally accepted Canadian mining industry practice, and as of March 27th, 2014, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

This 9th day of May 2014.

Original signed and sealed

(Signed) "Jeffrey Cassoff"

____

Jeffrey Cassoff, Eng.
Lead Mining Engineer
Met-Chem Canada Inc.

CERTIFICATE OF AUTHOR

I, Luc Bélanger, Eng., do hereby certify that:

1) I am Senior Vice-President – Iron, presently employed by SNC-Lavalin Inc. in the Mining & Metallurgy Division, with an office situated at 1140 de Maisonneuve Blvd West, Montréal, Quebec, Canada;

2) I am a graduate of UQAC in Saguenay with a Bachelor's in Geological Engineering obtained in 1973;

3) I am a member in good standing of the "*Ordre des Ingénieurs du Québec*" (26420);

4) I have worked as a Geological, Mining and Process Engineer continuously since graduation from university in 1973;

5) I have read the definition of "qualified person" set out in National Instrument 43-101 ("**NI 43-101**") and certify that by reason of my education, affiliation with a professional association (as defined by NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101;

6) I am responsible for Section 13 of the *NI 43-101 Technical Report on the Feasibility Study on KéMag Taconite Project*" dated May 9th, 2014 with an effective date of March 27th, 2014 ("Technical Report");

7) I have not visited the site property that is the subject of this Technical Report;

8) As at the effective date of the Technical Report, to the best of my knowledge, information and belief, Section 13 of the Technical Report stated above in paragraph 6 contains all scientific and technical information that is required to be disclosed to make Section 13 of the Technical Report not misleading;

9) I have read NI 43-101 and Section 13 of the Technical Report stated above in paragraph 6 has been prepared in accordance with NI 43-101;

10) I have read and understand NI 43-101 and I am considered independent of the issuer as defined in section 1.5 of NI 43-101.

This 9th day of May 2014

Original signed and sealed

Luc Bélanger, Eng.
Senior Vice-President – Iron Ore
Mining & Metallurgy
SNC-Lavalin Inc.

CERTIFICATE OF AUTHOR

I, Éric Giroux, Eng., do hereby certify that:

1) I am Director – Business Development, presently employed by SNC-Lavalin Inc. in the Environment & Water Division with an office situated at 5955, Saint-Laurent Street, Lévis, Quebec, Canada;

2) I am a graduate of Laval University in Quebec City with a Master's in Geology (hydrogeology) obtained in 1994;

3) I am a member in good standing of the "*Ordre des Ingénieurs du Québec*" (108694);

4) I have worked as an hydrogeologist and environmental specialist continuously since graduation from university in 1994;

5) I have read the definition of "qualified person" set out in National Instrument 43-101 ("**NI 43-101**") and certify that by reason of my education, affiliation with a professional association (as defined by NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101;

6) I am responsible for Section 20 of the *NI 43-101 Technical Report on the Feasibility Study on KéMag Taconite Project*" dated May 9th, 2014 with an effective date of March 27th, 2014 ("Technical Report");

7) I have not visited the site property that is the subject of this Technical Report;

8) As at the effective date of the Technical Report, to the best of my knowledge, information and belief, Section 20 of the Technical Report stated above in paragraph 6 contains all scientific and technical information that is required to be disclosed to make Section 20 of the Technical Report not misleading;

9) I have read NI 43-101 and Section 20 of the Technical Report stated above in paragraph 6 has been prepared in accordance with the NI 43-101;

10) I have read and understand NI 43-101 and I am considered independent of the issuer as defined in section 1.5 of NI 43-101.

Original signed and sealed

Éric Giroux, Eng., M.Sc.
Director – Business Development
Environment & Water
SNC-Lavalin Inc.

CERTIFICATE OF AUTHOR

To accompany the technical report entitled *"NI 43-101 Technical Report on the Feasibility Study on KéMag Taconite Project"* dated May 9th, 2014 with effective date of March 27th, 2014 (the "**Technical Report**").

I, Pierre Julien, Eng., do hereby certify that:

1) I am a Project Manager presently employed by SNC-Lavalin Inc. in the Mining & Metallurgy Division with an office situated at 1140 Blvd de Maisonneuve West, Montréal, Quebec, Canada;

2) I am a graduate of Ottawa University with a Bachelor's in Mechanical Engineering obtained in 1984;

3) I am a member in good standing of the "*Ordre des Ingénieurs du Québec*" (40022);

4) I have worked as an engineer continuously since graduation from university in 1984 and as a consulting engineer in mining since 2007;

5) I have read the definition of "qualified person" set out in National Instrument 43-101 ("**NI 43-101**") and certify that by reason of my education, affiliation with a professional association (as defined by NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101;

6) I have participated in the preparation of the *NI 43-101 Technical Report on the Feasibility Study on KéMag Taconite Project*" dated May 9th, 2014 with an effective date of March 27th, 2014 ("Technical Report") and am responsible for portions of Sections 2 (Introduction), and all of Sections 17, 18 and 21;

7) I visited the site property that is the subject of this Technical Report in October 2012 for 3 days and did not make any subsequent visits;

8) As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the Sections of the Technical Report stated above in paragraph 6 contains all scientific and technical information that is required to be disclosed to make such Sections of the Report not misleading.

9) I have read NI 43-101 and the Sections of the Technical Report stated above in paragraph 6 have been prepared in accordance with the NI 43-101.

10) I have read and understand NI 43-101 and I am considered independent of the issuer as defined in section 1.5 of NI 43-101;

This 9th day of May 2014.

Original signed and sealed

INGÉNIEUR
PIERRE JULIEN
40022
QUÉBEC

Pierre Julien, Eng.
Project Manager
SNC-LAVALIN INC.

CERTIFICATE OF AUTHOR

To accompany the technical report entitled "*NI 43-101 Technical Report on the Feasibility Study on KéMag Taconite Project*" dated May 9th, 2014 with effective date of March 27, 2014 (the "**Technical Report**").

I, Joe Poveromo, Ph. D. Eng., do hereby certify that:

1) I am the President of Raw Materials & Ironmaking Global Consulting with an office situated at 1992 Easthill Drive, Bethlehem, PA , USA;
2) I am a graduate of Rensselaer Polytechnic Institute in Troy, New York with a Bachelor's in Chemical Engineering obtained in 1968;
3) I am a graduate of the State University of New York at Buffalo, Center for Process Metallurgy, with a Ph. D. degree obtained in 1974;
4) I am a member in good standing of the Association of Iron & Steel Technology (10361) and the Society of Mining Engineers (4031590);
5) I have worked as a metallurgical engineer continuously since graduation from university in 1974;
6) I have read the definition of "qualified person" set out in National Instrument 43-101 ("**NI 43-101**") and certify that by reason of my education, affiliation with a professional association (as defined by NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101;
7) I have participated in the preparation of this Technical Report and am responsible for Section 19;
8) I have not visited the site property that is the subject of this Technical Report;
9) I have no personal knowledge, as of the date of this certificate, of any material fact or material change which is not reflected in this Technical Report;
10) Neither I, nor any affiliated entity of mine, is at present, under an agreement, arrangement or understanding or expects to become, an insider, associate, affiliated entity or employee of New Millennium Iron Corporation, or any associated or affiliated entities;
11) Neither I, nor any affiliated entity of mine, own, directly or indirectly, nor expect to receive, any interest in the properties or securities of New Millennium Corporation, or any associated or affiliated companies;
12) Neither I, nor any affiliated entity of mine, have earned the majority of our income during the preceding three (3) years from New Millennium Corporation, or any associated or affiliated companies;
13) I am independent of the issuer as defined in section 1.5 of NI 43-101;

14) I have had no prior involvement with the property that is the subject of this Technical Report; and

15) I have read NI 43-101 and Form 43-101F1 and have prepared the Technical Report in compliance with NI 43-101 and Form 43-101F1; and have prepared the report in conformity with generally accepted Canadian mining industry practice, and as of March 27, 2014, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

This 9th day of May 2014.

Original signed and sealed

Joseph J Poveromo

____

Joe Poveromo, Ph. D. Eng.
President of Raw Materials & Ironmaking Global Consulting

CERTIFICATE OF AUTHOR

To accompany the technical report entitled *"NI 43-101 Technical Report on the Feasibility Study on KéMag Taconite Project"* dated May 9th, 2014 with effective date of March 27th, 2014 (the "**Technical Report**").

I, Michel L. Bilodeau, Eng., do hereby certify that:

1) I am a retired (June 2009) Associate Professor from the Department of Mining and Materials Engineering of McGill University, 3450 University St., Montréal, QC, Canada H3A 2A7, and have taught on a contract basis the mineral economics course of the mining engineering program at McGill in the Winter terms of 2010, 2011 and 2012;

2) I am a graduate of École Polytechnique de Montréal with a B.Eng. in Geological Engineering (1970), and of McGill University with a M.Sc. (App.) in mineral exploration (1972) and a Ph.D. in mineral economics (1975);

3) I am a member in good standing of the "Ordre des ingénieurs du Québec" (23799);

4) I have taught continuously in the areas of engineering economy, mineral economics and mining project feasibility studies in the mining engineering program dispensed by McGill University since my graduation from university, and have carried out in the capacity of independent consultant several assignments related to the economic/financial analysis of mining projects;

5) I have read the definition of "qualified person" set out in National Instrument 43-101 ("**NI 43-101**") and certify that by reason of my education, affiliation with a professional association (as defined by NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101;

6) I have participated in the preparation of this Technical Report and am responsible for Section 22;

7) I have not visited the site property that is the subject of this Technical Report:

8) I have no personal knowledge, as of the date of this certificate, of any material fact or material change which is not reflected in this Technical Report;

9) Neither I, nor any affiliated entity of mine, is at present, under an agreement, arrangement or understanding or expects to become, an insider, associate, affiliated entity or employee of New Millennium Iron Corporation, or any associated or affiliated entities;

10) Neither I, nor any affiliated entity of mine, own, directly or indirectly, nor expect to receive, any interest in the properties or securities of New Millennium Corporation, or any associated or affiliated companies;

11) Neither I, nor any affiliated entity of mine, have earned the majority of our income during the preceding three (3) years from New Millennium Corporation, or any associated or affiliated companies;

12) I am independent of the issuer as defined in section 1.5 of NI 43-101;

13) I have had no prior involvement with the property that is the subject of this Technical Report;

14) I have read NI 43-101 and Form 43-101F1 and have prepared the Technical Report in compliance with NI 43-101 and Form 43-101F1; and have prepared the report in conformity with generally accepted Canadian mining industry practice, and as of March 27th, 2014, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

This 9th day of May 2014.

Original signed and sealed

(Signed) Michel L. Bilodeau

Michel L. Bilodeau, Eng., M.Sc. (App.), Ph.D.
Economic/Financial Analyst
Consultant for New Millennium Corp.

CERTIFICATE OF AUTHOR

To accompany the technical report entitled "*NI 43-101 Technical Report on the Feasibility Study on KéMag Taconite Project*" dated May 9th, 2014 with effective date of March 27th, 2014 (the "**Technical Report**").

I, Charles H. Cauchon, Eng. do hereby certify that:

1) I am Senior Process Engineer with Met-Chem Canada Inc. with an office situated at Suite 300, 555 René-Lévesque Blvd. West, Montreal, Canada;

2) I am a graduate of l'École Polytechnique de Montréal with a B.Sc Eng. in Mining Engineering in 1960;

3) I am a member in good standing of the "Ordre des Ingénieurs du Québec" (11811);

4) I have practiced my profession for the mining industry continuously since my graduation from university;

5) I have read the definition of "qualified person" set out in National Instrument 43-101 ("**NI 43-101**") and certify that by reason of my education, affiliation with a professional association (as defined by NI 43-101) and past relevant work experience that includes more than 12 years in concentrators and operating plants and more than 36 years in consulting practice related to mineral processing, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101;

6) I have participated in the preparation of this Technical Report and am responsible for Sections 1, 2, 3, 24 to 26;

7) I have no personal knowledge, as of the date of this certificate, of any material fact or material change which is not reflected in this Technical Report;

8) Neither I, nor any affiliated entity of mine, is at present, under an agreement, arrangement or understanding or expects to become, an insider, associate, affiliated entity or employee of New Millennium Iron Corp., or any associated or affiliated entities;

9) Neither I, nor any affiliated entity of mine, own, directly or indirectly, nor expect to receive, any interest in the properties or securities of New Millennium Iron Corp., or any associated or affiliated companies;

10) Neither I, nor any affiliated entity of mine, have earned the majority of our income during the preceding three (3) years from New Millennium Iron Corp., or any associated or affiliated companies;

11) I am independent of the issuer as defined in section 1.5 of NI 43-101;

12) I have had no prior involvement with the property that is the subject of this Technical Report; and

13) I have read NI 43-101 and Form 43-101F1 and have prepared the Technical Report in compliance with NI 43-101 and Form 43-101F1; and have prepared the report in conformity with generally accepted Canadian mining industry practice, and as of March 27th, 2014, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

This 9th day of May 2014.

Original signed and sealed

(Signed) "Charles H. Cauhon"

____

Charles H. Cauchon, Eng.
General Manager – Mining Group
Met-Chem Canada Inc.

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LIST OF ABBREVIATIONS

Abbreviation	Description
µm	Microns, Micrometre
"	Inch
$	Dollar Sign
%	Percent Sign
°	Degree
°C	Degree Celsius
2D	Two Dimensions
3D	Three Dimensions
≈	Approximately Equal
~	Approximately
AACE	American Association of Cost Engineers
AC	Alternating Current
ACSR	Aluminium Cable, Steel Reinforced
ARD	Acid Rock Drainage
AQ	Drill Core Size (2.7 cm Diameter)
BAPE	*Bureau d'Audiences Publiques sur l'Environnement*
BBA	Breton, Banville et Associés
BF	Blast Furnace
BIF	Banded Iron Formation
BM	Ball Mill
BQ	Drill Core Size (3.65 cm Diameter)
BRASS	BRASS Engineering International
BWI	Bond Ball Mill Work Index

Abbreviation	Description
CA	Collaboration Agreement
CAGR	Compound Annual Growth Rate
Capex	Capital Expenditures
CCR	Central Control Room
CCTV	Close Circuit Television
CDC	*Claim désigné sur carte*
CDP	Closure and Decommissioning Plan
CEAA	Canadian Environmental Assessment Act
CEPA	Canadian Environmental Protection Act
CFLC	Churchill Falls (Labrador) Corporation
cfm	Cubic feet per minute
CFR	Cost and Freight
CIF	Cost Insurance and Freight
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIS	Commonwealth Independent States
cm	Centimetre
Cm^2/g	Centimetre Square per Gram
CMRL	Coleraine Mineral Research Lab
COG	Cut Off Grade
COV or CV	Coefficient of Variation
CPESI	*Corportation de protection de l'environnement de Sept-Îles*
CPM	Critical Path Method
CRM	Certified Reference Materials

Abbreviation	Description
cSt	Viscosity in Centistokes
Cw	Concentration by Weight
d	Day
d/w	Days per Week
d/y	Days per Year
DB	Database
dB	Decibel
dBA	Decibel with an A Filter
DC	Direct Current
DCS	Distributed Control System
DDD	Downdraft Drying
DDH	Diamond drill hole
DDRS	Double Deck Roller Screen Feeder
deg	Angular degree
DEM	Digital Elevation Model
DFO	Department of Fisheries and Oceans
DGPS	Differential Global Positioning System
D-LRS	Dual Liquid Rheostat Starter
DMS	Dense Media Separator
DR	Direct Reduction
DRI	Direct Reduced Iron
DSOP	Direct Shipping Ore Project
DT	Davis Tube
DTC	Davis Tube Concentrate
DTWR	Davis Tube Weight Recovery
DWI	Drop Weight Index
DWT	Drop Weight Test

Abbreviation	Description
DXF	Drawing Interchange Format
E	East
EA	Environmental Assessment
EAB	Environmental Assessment Board
EAF	Electric Arc Furnace
EBS	Environmental Baseline Study
EHS	Environment Health and Safety
EIA	Environmental Impact Assessment
EIS	Environmental Impact Statement
EM	Electro-Magnetic
EMP	Environmental Management Plant
EOH	End of Hole
EP	Environmental Permit
EPA	Environmental Protection Agency
EPCM	Engineering, Procurement and Construction Management
EQA	Environmental Quality Act
ER	Electrical Room
ERT	Endangered, Rare or Threatened
ESBS	Environmental and Social Baseline Study
ESIA	Environmental and Social Impact Assessment
EUR	Euro
FDS	Fused Disconnect Switch
Fe	Iron
FOB	Free on Board
FRCS	Fatal Risk Control Standards
FS	Feasibility Study

Abbreviation	Description
ft	Feet
FVNR	Full Voltage Non Reversible
g	Grams
Ga	Billion Year
GC	Green Cherty
G&A	General and Administration
g/cm^3	Gram per cubic centimetre
g/l	Grams per Litre
g/t	Grams per Tonne
gal	Gallons
GEMS	Global Earth-System Monitoring Using Space
GIS	Gas Isolated Switchgear
g/ml	Grams per milliliter
GoNL	Government of Newfoundland and Labrador
GoC	Government of Canada
GoQ	Government of Quebec
GPS	Global Positioning System
Gr	Granular
GCW	Gross Combined Weight
GOH	Gross Operating Hours
H	Horizontal
h	Hour
h/d	Hours per Day
h/y	Hour per Year
H_2	Hydrogen
ha	Hectare
HBI	Hot Briquetted Iron

Abbreviation	Description
HDPE	High Density PolyEthylene
HFO	Heavy Fuel Oil
HG	High Grade
HGL	Hydraulic Gradient Line
HL	Heavy Liquid
HMC	Hanna Mining Company
HMI	Human Machine Interfaces
HmFe	Hematitic Iron
HoA	Heads of Agreement
hp	Horse Power
HPGR	High Pressure Grinding Rolls
HPS	
HQ	Drill Core Size (6.4 cm Diameter)
H-Q	Hydro-Québec
HR	Human Resources
HSE	Health, Safety and Environment
HSEC	Health, Safety, Environment and Community
HSWE	Health, Safety and Work Environment
HV	High Voltage
HVAC	Heating Ventilation and Air Conditioning
Hz	Hertz
I/O	Input/Output
ICP	Inductively Coupled Plasma
ID	Identification
IDW	Inverse Distance Weighted
IDW1	Inverse Distance to the power of one

Abbreviation	Description
IDW2	Inverse Distance Squared Method
in	Inches
IN	Innu Nation
INREST	*Institut Nordique de Recherche en Environnement et en Santé au Travail*
IOCC	Iron Ore Company of Canada
IRA	Inter-Ramp Angle
IRR	Internal Rate of Return
ISLLP	Innu SNC Lavalin Limited Partnership
ISO	Internations Standard Organization
IT	Information Technology
ITUM	Innu Takuaikan Uashat mak Mani-Utenam
IUCN	International Union for the Conservation of Nature
JBNQA	James Bay and Northern Québec Agreement
JUIF	Jasper Upper Iron Formation
JVE	Joint Venture Enterprise
Kcal/kg	Kilocalories per Kilogram
KE	Kriging Efficiency
kg	Kilogram
kg/l	Kilogram per Litre
Kg/t	Kilogram per Metric Tonne
kl	Kilolitre
km	Kilometre
Km^2	Square kilometre
km/h	Kilometre per Hour

Abbreviation	Description
kPa	Kilopascal
KPI	Key Performance Indicators
KSR	Kriging Slope Regression
kt	Kilotonne ('000 tonnes)
kV	Kilovolt
kVA	Kilovolt Ampere
kW	Kilowatt
kWh	Kilowatt-hour
kWh/t	Kilowatt-hour per Metric Tonne
L	Line
l	Litre
l/h	Litre per hour
lbs	Pounds
LC	Lean Cherty
LCP	Local Control Panels
LCR	Local Control Rooms
LFO	Light Fuel Oil
LG	Low Grade
LG-3D	Lerchs-Grossman – 3D Algorithm
LIF	Lower Iron Formation
LIMS	Low Intensity Magnetic Separator
LM&E	Labrador Mining and Exploration Co
LNG	Liquid Natural Gas
LOI	Loss On Ignition
LOM	Life Of Mine
LRC	Lower Red Cherty
LRGC	Lower Red Green Cherty
LTI	Loss Time Incidents
LV	Low Voltage

Abbreviation	Description
m	Metre
m/h	Metre per Hour
m/s	Metre per Second
m^2	Square Metre
m^3	Cubic Metre
m^3/d	Cubic Metre per Day
m^3/h	Cubic Metre per Hour
m^3/y	Cubic Metre per Year
mA	MilliAmpère
MagFe	Magnetic Iron
Mm^3	Million Cubic Metres
MCC	Motor Control Center
MDDEP	*Ministère du Développement Durable, Environnement, Faune et Parcs*
MENA	Middle East and North Africa
mg/l	Milligram per Litre
MIBK	Methyl Isobutyl Ketone
min	Minimum
min	Minute
min/h	Minute per Hour
min/shift	Minute per Shift
ml	Millilitre
ML	Metal Leaching
mm	Millimetre
mm/d	Millimetre per Day
mm/y	Millimetre per Year
Mm^3	Million Cubic Metres
MMER	Metal Mining Effluent Regulation
MMU	Mobile Manufacturing Units

Abbreviation	Description
MNDM	Ministry of Northern Development and Mines
MNRW	Ministry of Natural Resources and Wildlife
MOE	Ministry of Environment
MOU	Memorandum of Understanding
MPD	Mean Percentage Difference
MPMO	Major Projects Management Office
MRC	Midland Research Center
MRN	*Ministère des Ressources Naturelles*
MS	Menihek Shale
MSDEP	Ministry of Sustainable Development, Environment and Parks
Mt	Million Metric Tonnes
Mt/h	Million Tonnes per hour
Mt/y	Millions of Metric Tonnes per year
MTO	Material Take-offs
MV	Medium Voltage
MVA	Mega Volt-Ampere
MW	Megawatts
MWh	Megawatts per hour
MWh/d	Megawatt Hour per Day
N	North
NAD	North American Datum
Nb	Number
NCC	NunatuKavut Community Council

Abbreviation	Description
NEB	National Energy Board
NEBA	National Energy Board Act
NEQA	Northeastern Québec Agreement
NFPA	National Fire Protection Association
NGR	Neutral Grounding Resistor
NI	National Instrument
NIMLJ	Nation Innu Matimekush-Lac John
Nm^3/h	Normal Cubic Metre per Hour
NNK	Naskapi Nation of Kawawachikamach
NML	New Millennium Iron Corp. (since June 23rd, 2011) New Millennium Capital Corp. (from August 8th, 2003 to June 23rd, 2011)
NN	Nearest Neighbour
NPV	Net Present Value
NQ	Drill Core Size (4.8 cm Diameter)
NRRI	Natural Resources Research Institute
NTP	Normal Temperature and Pressure
NTS	National Topographic System
OB	Overburden
OD	Outside Diameter
O/F	Overflow
OK	Ordinary Kriging
Opex	Operating Expenditures
OT&R	Ore Testing and Research Laboratory

Abbreviation	Description
P&ID	Piping and Instrumentation Diagram
PA	Public Address
PBX	Private Branch Exchange
PCC	Point of Client Connection
PCS	Plant Control System
PEA	Preliminary Economic Assessment
PEP	Project Execution Phase
PF	Power Factor
PFD	Process Flow Diagram
PFS	Pre-Feasibility Study
PGC	Pink Grey Cherty
PGGS	Permit for Geological and Geophysical Survey
p/h	Per Hour
ph	Phase (electrical)
pH	Potential Hydrogen
PLC	Programmable Logic Controllers
POV	Pre Operational Verification
PP	Preproduction
PPE	Personal Protective Equipment
ppb	Part per Billion
ppm	Part per Million
PS	Pumping Station
psi	Pounds per Square Inch
PSTN	Public Switched Telepone Network
QA/QC	Quality Assurance/Quality Control

Abbreviation	Description
QKNA	Quantitative Kriging Neighbourhood Analysis
QMS	Quality Management System
QNS&L	Quebec North Shore and Labrador Railway
QP	Qualified Person
RCM	Regional County Municipality
RCMS	Remote Control and Monitoring System
RMR	Rock Mass Rating
ROE	Return on Equity
ROM	Run of Mine
ROW	Right of Way
rpm	Revolutions per Minute
RQD	Rock Quality Designation
RWI	Bond Rod Mill Work Index
S	South
S	Sulfur
S/R	Stripping Ratio
SAG	Semi-Autogenous Grinding
SAGDesign	SAGDesign Consulting Group
scfm	Standard Cubic Feet per Minute
SCIM	Squirrel Cage Induction Motors
sec	Second
Set/y/unit	Set per Year per Unit
SG	Specific Gravity
SGA	Studiengesellschaft für Eisenerzaufbereitung

Abbreviation	Description
SGS	SGS Canada (since March 9th, 2010) SGS-Geostat Ltd. (from September 8th, 2008 to March 9th, 2010) Geostat Systems International Inc. (from April 7th, 1981 to September 8th, 2008)
SI	Système International d'Unités
SIPA	Port Authority of Sept-Îles
SL	Sanitary Landfill
SLE	SNC-Lavalin inc., Environment & Water Division
SMC	SAG Mill Comminution
SMYS	Specified Minimum Yield Stress
SNRC	*Système National de Référence Cartographique*
SPI	SAG Power Index
SPT	Standard Penetration Tests
SR	Stripping Ratio
SW	Switchgear
t or T	Metric Tonne
t/d	Metric Tonne per Day
$t/d/m^2$	Metric Tonne per Day per Square Metre
t/h	Metric Tonne per Hour
t/h/m	Metric Tonne per Hour per Metre
$t/h/m^2$	Metric Tonne per Hour per Square Metre
t/m	Metric Tonne per Month
t/m^2	Metric Tonne per Square Metre
t/m^3	Metric Tonne per Cubic Metre
t/y	Metric Tonne per Year

Abbreviation	Description
tCO_2 eq/y	Tonnes of CO_2 equivalent/year
TEM	Terrestrial Ecosystem Model
TIA	Tailings Impondment Areas
TIN	Triangulated Irregular Network
TMF	Tailings Management Facilities
TNO	Territoire non organisé
ton	Short Ton
tonne	Metric Tonne
TOR	Terms of Reference
TotFe	Total Iron
TS	Tata Steel Global Minerals Holdings Pte Ltd.
TSE	Tata Steel Minerals Europe
TSF	Tailings Storage Facility
TSH	Tshiuetin Rail Transportation Inc.
TSS	Total Suspended Solids
TSX	Toronto Stock Exchange
U/F	Under Flow
U/G	Under Ground
ULC	Underwriters Laboratories of Canada
UMD	University of Minnesota at Duluth
URC	Upper Red Cherty
USA	United Stated of America
UTM	Universal Transverse Mercator
V	Vertical
V	Volt

Abbreviation	Description
Vd	
VE	Value Engineering
VFD	Variable Frequency Drive
Vl	Limiting Velocity
VLF	Very Low Frequency
VLF-EM	Very Low Frequency - Electro-Magnetic
VoIP	Voice Over Internet Protocol
Vt	
W	Watt
WAN	Wide Area Network
WGM	Watts, Griffis and McOuat Limited
WHIMS	Wet High Intensity Magnetic Separation
WRA	Whole Rock Analysis Method
WSD	World Steel Dynamics
wt	Wet Metric Tonne
X	X Coordinate (East-West)
XRD	X-Ray Diffraction
XRF	X-Ray Fluorescence
y	Year
Y	Y coordinate (North-South)
Z	Z coordinate (depth or elevation)

1.0 SUMMARY

1.1 Introduction

Tata Steel Global Minerals Holdings Pte Ltd. (“**TS**”), New Millennium Iron Corp. (“**NML**”) and LabMag Limited Partnership (“**LLP**”) have entered into a binding Heads of Agreement (“**HoA**”), signed on March 6, 2011, for conducting a Feasibility Study (“**FS**”) to develop the KéMag deposit.

Innu SNC Lavalin Limited Partnership (“**ISLLP**”) was retained in November 2011 to conduct the Feasibility Study. ISLLP in turn appointed sub-consulting services to cover some aspects of the study, namely Met-Chem Canada Inc. to develop the mining plans, capital and operation costs estimates and BRASS Engineering International (“**BRASS**”), a slurry transportation specialist, to conceive the slurry transportation system.

NML/TS appointed Met-Chem to undertake the geological and resource estimation studies, Outotec GmbH to design and estimate the pelletizing facilities including the dewatering of slurry, World Steel Dynamics (“**WSD**”) and Papillon Mineral Services for the market studies and SNC Lavalin Environment for the input regarding the environmental studies, permitting, social and community impact. Based on the above, NML/TS provided ISLLP with chapters on Geology and Mineral Resources, Filtration and Pelletizing, as well as Marketing.

NML holds a 100 % interest in the claims that constitute the KéMag taconite iron ore deposit at Lake Harris in the Province of Québec, Canada. The Property is located approximately 40 km north of Schefferville, Québec,

1.2 Property Description and Location

The KéMag Property, previously known as the Lake Harris Iron Property, is situated in the municipality of Rivière Koksoak in Northern Québec, about 40 km to the northwest of the town of Schefferville, Québec. The Property lies approximately 245 km north of Labrador City, and 550 km due north of Sept-Îles, Quebec.

The Property is comprised of one block of 171 contiguous claims covering an area of approximately 81 km^2 along a NNW-SSE trend and within a large block of claims held by NML. The Property is contiguous on the southeast with NML’s claims and lies approximately 18 km to the northwest of the LabMag iron Property in Labrador. The claims were acquired as Map-Designated Claims by NML. All the claims are currently active and in good standing. The drilled portion of the deposit is located in the southern half of the claims.

In 2011, NML signed a binding agreement with TS to conduct a Feasibility Study for development of LabMag and KéMag deposits and possible formation of a joint venture to develop the deposit (s), subject to the successful completion of the Feasibility Study and TS taking an Investment Decision to develop any or both the deposits.

1.3 Accessibility, Climate, Local Resources, Infrastructure and Physiography

The Property is accessible by well-maintained roads for 25 km northwest of Schefferville and for a further 30 km westward along a trail that reaches *Lac de la Frontière*, seven (7) km away from the Property. No roads connect Schefferville to the populous south, but access is either by airplane or by train.

The Project area is under the influence of sub-arctic continental climate conditions experiencing very severe winters and cool summers. Schefferville survived despite the closing of the iron mines in 1982 and has experienced an influx of workers since 2011 principally triggered by the re-start of iron production in the region. The 2011 census indicated that some 540 members of the Nation Innu Matimekosh-Lake John lived in the nearby Matimekosh community.

The town of Schefferville provides basic services, supplies and equipment, contractors and charter flight operators. However, a significant portion of the labor force required for a mining operation at KéMag would probably come from other parts of Eastern Canada and training programs will certainly be required.

1.4 History

All recorded exploration work prior to staking of the Property by NML had been carried out by Iron Ore Company of Canada ("**IOCC**"). In 1972, IOCC acquired an exploration permit covering the KéMag area but conducted no further work. In August 2003, NML was formed as a capital pool company listed on the TSX Venture Exchange. The KéMag Property was acquired by NML by claim staking between 2004 and 2008.

IOCC conducted reconnaissance mapping, airborne magnetic and electro-magnetic ("**EM**") surveying, and drilled a series of short holes, between 1949 and 1971. Since 2005, NML has conducted different phases of ground and airborne geophysical surveying, drilling and metallurgical test work. NML drilled on the Property and estimated the resources in 2006, 2007 and 2008. Met-Chem considers these resources as historical estimates that are superseded by the current and NI 43-101 compliant estimate that is the subject of this Report.

1.5 Geological Setting and Mineralization

The Property is located in the western margin of the Labrador Trough that extends for more than 1,000 km, from Ungava Bay to Lake Pletipi in Quebec. The belt is about 100 km wide in its central part and narrows considerably to the north and south. The Grenville Front crosses the southern part of the Trough. The rocks in the Labrador Trough are subdivided into an upper volcanic-dominated suite (Doublet Group) and a lower sedimentary sequence (Knob Lake Group) that includes the Sokoman Formation hosting the iron formations found in the region.

The units of the Knob Lake Group underlie the majority of the Property and lie on Archean gneiss. The Sokoman formation is essentially undeformed, strikes northwest and has a shallow dip to the northeast. The iron formation at KéMag has been explored by diamond drilling over a strike length of 9.5 km and it extends beyond the northwest and southeast Property boundaries.

The Sokoman formation has been broken down into individual stratigraphic units on the basis of facies. The taconite at KéMag consists mostly of alternating small-scale beds of chert or jasper and massive or disseminated magnetite, with subordinate amounts of hematite and martite. Gangue minerals are represented by iron silicates, iron and manganese carbonates. Alumina, sulphur and phosphorus generally occur at low levels in the iron formation.

1.6 Deposit Types

The KéMag iron deposit consists of magnetite Banded Iron Formation ("**BIF**") of the Lake Superior type. BIFs are sedimentary rocks composed of alternating mm- to cm-scale beds of quartz (chert or jasper) and iron oxides (predominantly magnetite and hematite). Variable amounts of gangue minerals, mostly silicates, carbonates and sulphides are present. Banded iron formations have greater than 15 % iron content and represent the principal sources of iron throughout the world and are mined in the Great Lakes region of the United States. The Lake Superior-type BIF are associated with typical shelf-type sedimentary rocks with minimal volcanic input and most formed during the Paleoproterozoic era.

The exploration model used by KéMag to design the exploration activities and the drilling is principally based on the interpretation of the drill data indicating the presence of gently dipping, Superior-type iron formation (taconite).

1.7 Exploration

Historical exploration in the KéMag region has consisted principally of field mapping, sampling, drilling, as well as geophysical surveying that included ground magnetic and airborne magnetic and EM surveys.

Resources estimates have been completed following the main phase of drilling. Cumulatively, the exploration and development work has allowed estimating NI 43-101 compliant Mineral Resources in the drilled area.

1.8 Drilling

Historical drilling on the Property consisted of 23 shallow holes testing targets defined by the dip-needle magnetic survey completed by IOCC in 1958. These historical holes were not used in any of the Mineral Resource estimations.

In 2006, NML drilled 3,586 m in 29 holes in order to test airborne anomalies outlined during the 1950s and in 1971. The drilling indicated that the economic stratigraphic horizons are similar to those occurring at the LabMag deposit.

The 2007 drill program, comprised of a total of 4,979 m in 46 holes, was a follow-up of the 2006 program. The deposit was found to be narrower and shallower in the central part than in the rest of the drilled area.

In 2008, 15 holes were drilled for a total of 2,216 m. The drilling on the southern part of the deposit confirmed that the eastern extension of the deposit lies under Lake Harris, *Lac de la Frontière* and the swampy grounds to the south.

1.9 Sample Preparation, Analysis and Security

Each stratigraphic unit was sampled separately, in 2006 to 2008, except for Meninck Schist and Ruth Formation, with sample lengths varying from 1.6 m to 9.1 m, based on the extent of magnetite/hematite mineralization.

The split core samples, and the check samples, were sent to Midland Research Center ("**MRC**"), Nashwauk, Minnesota, USA, for total Fe analysis on heads, Davis Tube Weight Recovery ("**DTWR**") and iron and silica on the -325 mesh concentrates. In addition, trace elements and sulphur were analyzed on 21 crude samples, Davis Tube concentrates and tails.

In 2006, NML monitored the laboratory performance with a total of 13 second half core samples re-assayed at MRC. Seventy-one (71) check samples were submitted in 2007 and a further four (4) were submitted in 2008. NML's QA/QC system did not provide for insertion of standard and blank materials into the sample stream.

MRC applied its own internal QA/QC program, including random selection of samples for re-assay by an external laboratory, Lerch Brothers Inc. ("**LBI**") of Minnesota.

1.10 Data Verification

Geostat visited the site in 2007 and collected 27 samples from the remaining core halves (½). Geostat observed a bias in the head Fe of the check samples, but Met-Chem's coarse rejects check samples submitted to XRF analysis showed a high correlation with the original results. Watts, Griffis and McOuat Limited ("**WGM**") completed a site visit in 2007 and noted that the drill core handling was well done. A BBA's Senior Metallurgist travelled to the site in 2008 and examined some drill core, but did not complete a new validation of the analytical results. Neither WGM nor BBA were required to collect check samples because independent validation had already been completed by Geostat.

The KéMag Property was visited by Yves A. Buro, Eng., and Schadrac Ibrango, P. Geo., Ph.D., both Senior Geologists, at Met-Chem, in September 2012. Met-Chem independently selected 26 samples from the deposit.

The previous Resource Estimates were completed using the density determined by the pycnometer method, without correction for the effects of porosity and permeability. At Met-Chem's recommendation, NML requested the sample preparation laboratory in Chibougamau (*Table Jamésienne de Concertation Minière*) to perform bulk density determination on 167 samples from the KéMag deposit.

The results were used to build a regression function for each seam in the deposit. This matter is discussed in the Section 14.0 of this Report.

1.11 Mineral Processing and Metallurgical Testing

NML owns several multi billion tonnes taconite iron ore deposits in the Labrador Trough. The two (2) main taconite projects are LabMag and KéMag situated about 25 km apart in the same valley, LabMag in Labrador Newfoundland and KéMag in Quebec. The ore characteristics and beneficiation properties in each deposit though not identical are similar enough to allow NML to use identical flow sheet and process equipment for either orebody. Blending will ensure a constant quality of feed to the concentration plant.

The test work carried out by NML since the start of the initial pilot plant tests in 2005 sought to characterize the ore and establish the process flow sheet and design criteria that would allow development of an efficient and economical process. The process has evolved through various stages of development to prepare an optimized feed for the production of quality pellets with a low silica grade for both blast furnace ("**BF**") and direct reduced ("**DR**") iron based steelmakers.

The final flow sheet is composed of primary crushing followed by screening, secondary crushing in closed circuit and High Pressure Grinding Rolls ("**HPGR**") in closed circuit with screens. The screen undersize, at -3 mm, pumped to cobbers where a final tail is rejected to hydroseparators to remove the coarser material while the cobber concentrate is cycloned. The coarse fraction is returned to the ball mill while the fines joined the mill discharge and are pumped to a second stage of cyclones. The cyclone underflow is returned to the ball mill and reground while the cyclone overflow is deslimed in hydroseparators and pumped to the finishers. The finisher concentrate is screened to remove the coarse material which is not ground to liberation yet and is returned to the ball mill. The screen undersize is final concentrate and can go the thickener and to the flotation plant to be upgraded if lower final silica is required.

The main features of the NML Project are the use of secondary crushing with HPGR which reduce substantially the power requirement and grinding cost and, when required for low silica DR pellets, the use of a flotation plant.

A full description of the test work can be found in Section 13.0.

1.12 Mineral Resources Estimates

Although no drilling was performed since the last Resource Estimate by Geostat in 2008, Met-Chem re-estimated the resources after introducing changes in the drill hole database and the geological interpretation. In addition, Met-Chem calculated a new regression function to take into account the secondary porosity in the density model and another regression function to model density with varying iron contents of the mineralization. The estimate was done in accordance with the NI 43-101 Rule (2011) and the guidelines on the resource classification adopted by the Council of the Canadian Institute of Mining, Metallurgy and Petroleum ("**CIM**") (November 2010).

Eighty-nine (89) holes drilled by NML between 2006 and 2008 were used to interpolate 3D blocks constrained within surfaces (top and bottom) related to each geological seam. The same variorum parameters that had been defined for the more densely drilled LabMag deposit were used for grade interpolation. A block model was created using MineSight® software package and Mineral Resources were interpolated on a seam by seam basis. The resource interpolation was performed using the Inverse Distance Weighted to the power of 2 ("**IDW2**") and Ordinary Kriging ("**OK**") was used to validate the results of IDW2. The block size used by Geostat in the previous estimates was adjusted to take into account the drilling density at KéMag. The Mineral Resources summarized in Table 1.1 are reported to a cut-off grade of 18 % DTWR and are not constrained to a pit.

Table 1.1 – Summary of the Mineral Resources – (Cut-Off of 18 % DTWR)

Resource by Category	Tonnage (Mt)	TotFe (%)	DTWR (%)	Concentrate	
				Fe (%)	SiO_2 (%)
Measured	1,507	31.45	26.97	69.69	2.56
Indicated	876	31.95	27.32	69.83	2.51
Measured + Indicated	**2,383**	**31.63**	**27.10**	**69.74**	**2.54**
Inferred	1,007	31.56	26.97	69.31	2.65

The estimate of Mineral Resource may be materially affected by environmental, permitting, legal, title, socio-political, marketing, or other relevant issues. However, Met-Chem is not aware of any known issues that would materially affect the Mineral Resource.

The quantity and grade of reported Inferred Mineral Resource in this estimate are uncertain in nature and there has been insufficient exploration to define the Inferred Mineral Resource as Indicated or Measured Mineral Resource and it is uncertain if further exploration will result in upgrading them to Indicated or Measured categories. Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability.

1.13 Mineral Reserve Estimates

The Mineral Reserves are the portion of the Measured and Indicated Mineral Resources that have been identified as being economically extractable and which incorporate mining losses and the addition of waste dilution. The Mineral Reserves for the KéMag deposit have been developed using best practices in accordance with CIM guidelines and NI 43-101 reporting.

A pit optimization analysis was carried out on the Mineral Resource block model to determine the appropriate cut-off grades and to what extent the deposit can be mined profitably. The pit optimization for the KéMag deposit was done using the MS-Economic Planner module of MineSight® Version 7.05. The optimizer uses the 3D Lerchs-Grossman algorithm to determine the economic pit limits based on input of mining and processing costs and revenue per block. In order to comply with NI 43-101 guidelines regarding the Standards of Disclosure for Mineral Projects, only blocks classified in the Measured and Indicated categories are allowed to drive the pit optimizer. Inferred Resource blocks are treated as waste, bearing no economic value.

The pit optimization analysis determined that the ultimate pit limits for the KéMag deposit should be based on a minimum DTWR cut-off of 19.0 %. In order to ensure a high quality concentrate product with a SiO_2 value below 2.1 %, the optimization analysis also determined that a maximum SiO_2 cut-off of 4.2 % should be applied. The Measured and Indicated Mineral Resources that are contained within the ultimate pit limits and meet the cut-off grade criteria have been converted into Class 1 Proven and Probable Mineral Reserves. The Measured and Indicated Mineral Resources that are contained within the ultimate pit limits but do not meet the cut-off grade criteria have been converted into Class 2 Proven and Probable Mineral Reserves. This material will be stockpiled close the plant site for future processing once the Class 1 Mineral Reserves are depleted.

After the completion of a detailed pit design and the addition of mining losses and waste dilution, the Mineral Reserves for the KéMag deposit have been estimated to include 1,172 Mt of Proven Mineral Reserves and 718 Mt of Probable Mineral Reserves for a total of 1,891 Mt. In order to access these reserves, 231 Mt of overburden, 440 Mt of Menihek Shale and 120 Mt of Inferred Mineral Resources must be mined. This total waste quantity of 791 Mt results in a stripping ratio of 0.33 to 1. The Class 1 Mineral Reserves for the KéMag deposit contain an average DTWR of 27.0 %, an average TotFe of 31.3 %, an average Fe in the concentrate of 69.9 % and an average SiO_2 in the concentrate of 2.2 %. The Class 2 Mineral Reserves for KéMag contain an average DTWR of 19.6 %, an average TotFe of 28.1 %, an average Fe in the concentrate of 64.8 % and an average SiO_2 in the concentrate of 3.6 %. Table 1.2 presents the Mineral Reserves for the KéMag deposit.

A preliminary evaluation determined that approximately 35 % of the Measured and Indicated Mineral Resources are contained below Lake Harris. If Lake Harris were to be left intact, the remaining Mineral Resources would only support a ten (10) year mine life which would most likely not support the Project.

In order to mine the resources that lie underneath Lake Harris, a system of dams and ditches will be constructed. Firstly, a diversion channel will keep the water from filling Lake Harris and Lake Gillespie. It will also drain the overflow from Boundary Lake and keep runoffs from the hills from reaching the mine pit. Cofferdams and dykes will retain water between the diversion channel and the mine pit. As the mine progresses into Lake

Harris in the area that is actually below the water surface, water will be pumped in the main stream and all water from this zone will flow towards the Goodwood River.

Table 1.2 – Mineral Reserves by Category

Category	Tonnage (Mt)	DTWR (%)	TotFe (%)	Concentrate	
				Fe (%)	SiO_2 (%)
Class 1					
Proven	1,172	27.0	31.2	69.8	2.2
Probable	718	26.9	31.4	70.1	2.1
Proven & Probable	**1,891**	**27.0**	**31.3**	**69.9**	**2.2**
Class 2					
Proven	275	20.0	29.5	64.8	3.5
Probable	218	19.1	26.4	64.8	3.7
Proven & Probable	**493**	**19.6**	**28.1**	**64.8**	**3.6**
Total					
Proven	1,448	25.6	30.9	68.9	2.5
Probable	936	25.1	30.3	68.9	2.5
Proven & Probable	**2,384**	**25.4**	**30.6**	**68.9**	**2.5**

The totals may not add up due to rounding errors.

1.14 Mining Methods

The mining method selected for the Project is conventional truck and shovel. Vegetation and topsoil will be cleared using a mining contractor and be carried out with a fleet of dozers, small excavators and articulated haul trucks ahead of the mining operation. Suitable organic material will be stockpiled for future reclamation use. Overburden will then be stripped using a fleet of excavators and hauled to the overburden dump. The ore and waste rock will be mined with 15 m high benches, drilled and blasted then loaded with large mining shovels into a fleet of rigid frame trucks which will haul the material either to the waste dump, the Class 2 ore stockpile or the primary crushers.

The mining operations for the Project will be 365 days per year, operating around the clock on two (2), 12 hour shifts. A total of seven (7) days per year have been accounted for when the mine will be shut down due to severe weather conditions. These conditions may occur during peak rainfall or snowfall periods or at times when the ambient temperature is extremely low.

The 22-year mine plan is based on an annual production of 22 Mt of concentrate. This production is divided in 16 Mt of concentrate which will yield 17 Mt of BF and DR grade pellet and 6 Mt of pellet feed. The mine plan has a ramp up of 60 % in Year-1 (13.2 Mt of concentrate) and 85 % in Year-2 (18.7 Mt of concentrate) before reaching full capacity in Year-3. The mine plan incorporates blending of the different rock types in order to minimize the variations in hardness of the material that is sent to the plant and to supply a constant run of mine feed that is as close to a DTWR of 27 % and a value of 2.2 % SiO_2 in the concentrate as possible.

During peak production, the total number of 363-tonne haul trucks is expected to reach 35, along with eight (8) shovels, ten (10) production drills and a large fleet of support and service equipment. The total mine workforce during peak production is expected to reach 535 employees.

1.15 Recovery Methods

1.15.1 Concentration

The process design for the LabMag concentration plant is based on extensive laboratory and pilot plant test work performed from 2005 to 2012 by NML and described in Section 13.0 of this Report and supplemented by suppliers' test work to support their equipment selection.

The process plant is designed to treat approximately 88 Mt/y of taconite ore (dry basis), at a DTWR of 27 % and Fe grade of approximately 30 %. Any hematite present will not be recovered. Operating 358 days per year 24 hours per day with at an availability of 92 %, (at a nominal rate of 10,026 t/h) the process plant will recover a nominal 22 Mt/y of concentrate (2,491 t/h, dry basis after flotation). The concentrate product quality will be such that DR grade and BF grade pellet feed can be produced as needed. The plant design is based on a 22-year plant life.

The basic process flow sheet consists of primary and secondary crushing and screening circuits feeding HPGRs for dry comminution. After cobbing, the liberation of the iron bearing mineral (magnetite) is completed in a wet grinding circuit. Concentration stages include besides cobbing, roughing and finishing with low intensity magnetic separators. The undersize final concentrate is sent to the thickener and the slurry storage tanks or to the flotation plant for low silica.

HPGRs in combination with secondary crushing have been selected to replace the conventional SAG mill for the primary grinding circuit. The main advantages of HPGR over other mills are their low energy consumption, high reliability, simplicity and ability to dry grind. They also require no grinding media. NML has done extensive pilot scale testing of HPGR for grinding its taconite ore, which is a very hard and abrasive material and the three (3) major HPGR suppliers have successfully tested the ore.

Depending on the product silica level required at the pellet plant, part or all of the concentrate will be floated. Final concentrate will be thickened and pumped to the slurry storage tanks for transport via the concentrate slurry transportation system (645 km).

To produce low silica BF fluxed or DR grade pellets, the concentrate must contain 2.2 % and 1.5 % silica respectively so four (4) flotation lines each with rougher, regrind down to a P_{80} of 18 µm and cleaner have been added to reduce the silica grade. All flotation steps will use tank type flotation cells. Regrinding will be performed in fine ore mills and classified by hydrocyclones.

Concentrator coarse tailings are dewatered for dry stacking disposal. Fine tailings are thickened, pumped and deposited in the tailings storage facility as slurry. The decanted water will be recycled to the process plant.

The KéMag site benefits from a natural depression in the terrain which allows for conventional disposal of fine tailings with few geographical restrictions. The initial dykes would be built with moderate quantities of earthwork prior to the start of production enabling three (3) years of tailings disposal. As tailings are pumped and discharged as slurry in the impoundment area, the coarse fraction settles first on the top of the beach and dewaters naturally. The tailings discharge point is then moved to another location and the coarse fraction is reclaimed to raise the dykes.

A complete description of the beneficiation process and plant facilities can be found in Section 17.0 of this Report.

1.15.2 Pelletizing

Outotec's mandate was to engineer the filtration, additives preparation, balling and pelletizing for two (2) pellet lines each with 8.5 Mt/y of capacity plants complete capital and operating cost estimate and develop design criteria. The plants will be located in Pointe-Noire adjacent to the stockyard and near the shipping facility and all support facilities such as slurry reception, thickening, product handling, stockpiling and reclaim to ship loading are by ISLLP.

Based on SGA and Outotec joint test work in 2012, a second round of tests was undertaken in 2013 at Outotec's facility to fine-tune the design and provide a process guarantee for pelletizing plant throughput and pellet quality. The physical properties of the indurated pellets such as cold compression strength tumble strength and Abrasion Index met or exceeded the desired levels in most tests, especially in the tests which were used as basis for plant design.

After completion of the 2013 tests, Outotec issued a confirmation letter guaranteeing a nominal annual capacity of 8.5 Mt/y for each 816 m^2 pellet machine. It was also stated that the equipment is expected to ramp up to 9.0 Mt/y after a period of stable and routine operation. Each pelletizing line can produce low silica fluxed BF grade pellets or DR grade pellets.

The production is divided as follows:

- Twelve (12) Mt/y of low silica BF pellets with 2.5 % SiO_2;
- Five (5) Mt/y of DR pellets with 1.8 % SiO_2; and

- Six (6) Mt/y of concentrate for direct sales.

1.15.3 Process and Plant Description

The slurry is received at the port facilities at a density of 65 % solids. Through a slurry distribution system, the slurry is fed to thickeners to raise the density to 72 % solids before being pumped to the four (4) slurry surge tanks.

Twenty-two (22) Mt/y of filter cake, with moisture of eight (8) %, will be produced. Of this, 16 Mt/y of concentrate will be used to produce 17 Mt/y pellets and the remaining six (6) Mt/y will be stored and shipped as an iron ore concentrate for sale.

The pellet plant includes wet grinding of dolomite and limestone and dry grinding of bentonite as well as preparation of organic binder when required for low silica pellet.

The screened pellets from each line are transported and stored via dedicated conveying and stacking systems in two (2) pellet piles, each having a capacity of 750,000 Mt.

Pellets for ship loading will utilize a common bucket wheel reclaimer having a capacity of 9,200 t/h for both pellet stockpiles. Both pellet and filter cake concentrate for sale will utilize a common overland conveyor on a time sharing basis for transport to the Sept-Îles Port Authority ship loading facility.

1.16 Project Infrastructure

1.16.1 Slurry Transportation System

BRASS was contracted by ISLLP to prepare a study for the taconite slurry transportation system to transport 22 Mt/y of iron concentrate slurry, from the mine site to the terminal station at Pointe-Noire. The slurry transportation system is designed to operate continuously with an operating range bounded by minimum safe limiting velocity, maximum throughput, and solids concentration by weight ("**Cw**"). The slurry transportation system can only safely transport concentrate within a throughput window equivalent to 18 Mt/y minimum and 24 Mt/y maximum flow rate. The 645 km slurry transportation system will follow a Quebec route. The main pumping station will be located in a building adjacent to the concentrator. There will be one (1) booster pumping station along the line. A summary of the slurry transportation system study can be found in Section 17.0 of this Report.

The design will adopt a "No Freezing, No Plugging philosophy", which means that the slurry transportation system will either be buried below the frost depth (no freezing) or if it is buried above the frost line, it will have the necessary reliability and emergency backup equipment to maintain no-freeze conditions during an emergency stoppage and the slurry transportation system slope will be restricted to avoid plugs even if the line is shut down full of slurry.

The 28 inch (71 cm) slurry transportation system will be in a trench under one (1) metre of cover. Where it is not possible to bury the slurry transportation system, an appropriate thickness of insulation and/or heat tracing will be provided, to guard against freezing of the slurry. A protective three-layer polyethylene external coating will mitigate external corrosion. Cathodic protection systems, either active or passive, will be included in the slurry transportation system design.

There are many stream crossings between the concentrator and Pointe-Noire. Trenching will be the preferred method of construction of the crossings while horizontal directional drilling will be used to cross large streams when trenching is impractical or not permitted.

A fiber-optic "backbone" will carry all slurry transportation system communications, including SCADA system and video surveillance of the booster and valve stations, as well as office data and voice telephone channels between the concentrator and the pellet plant. The multi-fiber optical cable will be installed in a conduit in the slurry transportation system trench.

The current pump station design is robust and has full power backup in the form of diesel emergency generators.

1.16.2 High Voltage Electrical Distribution Network

The utility companies will supply three (3) incoming lines at different voltage to each of the Project installation locations, being at mine and process plant area, slurry intermediate pumping station area and at Pointe-Noire and port area.

The plant will be powered by a new 315 kV aerial power lines by utility companies.

The expected power requirement for LabMag: 383.4 MW at 0.95 power factor giving 403.6 MVA. The sizing of the 315 kV to 34.5 kV main transformers 112/150/186 MVA, was based on the worst case scenario. However, with the load variations during the study, the sizing can be optimized (slightly on low side) at the detailed engineering phase. The sizing of the 315 kV to 13.8 kV main transformers 54/72/90 MVA, can also be optimized (slightly on high side) at the detailed engineering phase.

The slurry transportation system pumping station 2 (PS2) will be supplied from Hydro-Québec by one (1) line of 34.5 kV from their Poste Normand, near route 389 which is part of the grid that supplies nearby Fermont.

Hydro-Québec will feed the Pointe-Noire and port area at Sept-Îles from their Poste Arnaud with one (1) single line of 161 kV.

1.16.3 Emergency Power Plant

The mine, process plant and pumping station 1 (PS1) will be supplied from one (1) set of three (3) diesel generators. The booster station (PS2) will be equipped with four (4) diesel generators to allow operation in case of power failure. The facilities in Pointe-Noire will be equipped with four (4) diesel generators to cover all the emergency loads requirements.

1.16.4 Explosives Preparation and Storage

All required turn-key infrastructure for the manufacturing and the delivery of bulk explosives as well as for a down-the-hole blasting service will be provided and executed by an explosives supplier who will be under contract. The explosives plant will consist as a minimum of an emulsion plant, a storage area for ammonium nitrate, emulsion and explosives and the support installations.

1.16.5 Roads

The main access road from Schefferville to the mine site will be designed to permit heavy traffic to circulate at normal speed as regulated by provincial governments. The existing gravel road from Schefferville to the mine site will be upgraded.

Access for mine haul trucks and other large equipment to maintenance facilities, diesel fuel service and primary crushing areas are by mine roadways. Mine road details are described in Section 16.0.

Plant roads are used for normal operation traffic, which consists of light vehicles, small personnel buses and freight transport trucks. Site access roads are provided for the main entrance to the facility in order to allow delivery of materials, large personnel buses, and light vehicle traffic. Plant roads and access roads have provision for two-way traffic.

After it crosses the St-Marguerite River on the outskirts of Pointe-Noire, the slurry transportation system will parallel the Arcelor Mittal railway at times and then follow route 389 between Fire Lake and Mont-Wright, totaling less than 100 km of shared right of way. It will occasionally benefit from forestry and wood logging routes before climbing to higher grounds beyond Mont-Wright where no infrastructure exists. This last 300 km stretch is entirely undeveloped territory. Access routes to and along the slurry transportation system will be mere bush roads and require slow speed driving in forgiving all-terrain vehicles.

At Pointe-Noire, the main access to the facilities will from provincial highway 138. On-site roads will provide access to all facilities including the water treatment pond and access along the site security fence.

1.16.6 Tailings Management Facilities

The tails are divided in coarse and fine tails. The coarse tails are filtered and hauled by truck to a disposal area as shown and described in Section 17.0. The fine tails are pumped to the TMF near the concentrator and the excess water is returned to the plant.

1.16.7 Process Water Pumping Station

A process water pumping station is installed inside the process plant with eight (8) vertical turbine pumps, fixed or variable frequency driven. Each pump delivers the water necessary for one (1) ore processing line.

The two (2) make-up water pipelines discharge in the pumping station basin. Before discharging into the basin, a branch from the pipeline provides water to the fire and potable water pumping station.

The main contributor to the process water basin is the overflow of tailings thickeners. The overflow from thickeners flows by gravity into the pumping station basin. Some additional water comes from the flotation water pump station.

1.16.8 Temporary Construction Camps

Lodging requirements are estimated based on the total construction hours for the Project and the planned schedule. The construction camp for LabMag is presently sized to 1,700 rooms taking into consideration 400 beds of the permanent accommodation complex at the mine will be utilized during construction. The slurry transportation camp is sized at 1,600 rooms. Camp population will include multidisciplinary construction workers, contractors, management staff, service people, owner's representatives, security and visitors. The design of camps have included all required sub-facilities as catering, recreational rooms and others amenities.

1.16.9 Permanent Residential Complex

The accommodation complex will be designed to meet the sleeping, hygiene, dining and recreational requirements for 400 workers and future employees. Each room will be equipped with shower/water closets and all furniture and related equipment. The central kitchen is used to prepare all meals that are served in the central dining area for the operation personnel.

1.16.10 Pellet Storage and Ship Loading

Product from the two (2) pelletizing lines, each having a capacity of 1,073 t/h, are transported to two (2) dedicated rail mounted stackers and discharged on two (2) parallel stockpiles each having a storage capacity of 750,000 Mt. The pellets will be reclaimed by one (1) single 8,000 t/h bucket wheel reclaimer and transported to ship loaders via one (1) 7.6 km long overland conveyor to the transfer tower and ship loading system that will be operated by the Port Authority of Sept-Îles ("**SIPA**").

The concentrate will be shipped via a slurry transportation system and filtered at Sept-Îles facilities. The concentrate filter cake will feed two (2) pellet plants (see Section 17.0) or be stocked in covered stockpile of 600,000 t. The concentrate will be reclaimed by a bridge reclaimer and will be transferred to ship loaders by the same overland conveyor used for pellets shipment.

1.16.11 Maintenance Facilities

The mine trucks and light vehicles garage and warehouse area is designed for the maintenance of heavy equipment and flexible for light vehicles. It is supplemented by a mine truck washing and tire changing facility.

The port facilities maintenance workshop and main warehouse are sized to support the filtering, balling, pelletizing, stockpiling, reclaim and shiploading equipment.

1.16.12 Boiler Houses and Fuel Storage

For the mine/concentrator site there are two (2) identical boiler house facilities. One (1) boiler house is located adjacent to fire and potable water pumping station. The boilers provide heating for all process buildings and shop areas.

The second boiler house is located adjacent to the main trucks maintenance garage building.

Two (2) diesel storage tanks provide sufficient capacity for mine and process plant operations. Two (2) vehicle refuel stations are provided. One (1) is dedicated to mine trucks and one (1) for light vehicles.

An emergency power plant diesel fuel tank farm provides fuel for diesel generators. Diesel fuel storage is also provided at PS2 and at the port for the emergency generators.

1.17 Market Studies and Contracts

1.17.1 World Steel Demand

The iron ore market is directly related to iron and steel production. NML/TS worked with leading steel information analyst World Steel Dynamic Inc. ("**WSD**") to complete a comprehensive, competitive assessment of the Project. The report concludes that the Project has a strong possibility of succeeding in the coming global steel industry environment. A low steel demand growth path is expected over the period 2012-2016, followed by a transition in the world economy in 2016-2017 to a scenario which would be a period of good growth for steel demand globally.

The mid to long term forecasts for iron ore demand continue to be good, driven by continued robust growth in China, India and the developing world. India's production is projected to more than double from 2011 to 2025, with an average annual growth rate of 6.1 % over the decade 2016 – 2025. Similarly, although the Middle East/North Africa ("**MENA**") region accounted for just 2.1 % of global steel production in 2011 (32.4 Mt), the region is expected to show strong growth in the medium and longer term, driven largely by developments in the Middle East. Over the period 2016-2025, steel production in the region is expected to grow at an average annual growth of 4.5 % per year.

Against this background, WSD sees good market opportunities for the Project. One of the Project's most favorable attributes is the possibility of having TS as an equity investor, as well as off-taker of a substantial portion of production, and with that a core market in the integrated, Blast Furnace/Basic Oxygen Furnace ("**BF/BOF**") steelmaking sector as a supplier to Tata Steel's European operations.

Table 1.3 – World Steel Dynamics' Crude Steel Forecast

Location	2012 (Mt)	2013e (Mt)	2014e (Mt)	2015e (Mt)	2025e (Mt)	CAGR 2014-2025 (%)
Advanced Countries	**464**	**448.7**	**460.8**	**470**	**536**	**1.4%**
Japan	107.2	108.8	110.2	111.0	121.0	0.9%
Western Europe	140.8	132.9	136.9	140.4	158.1	1.3%
United States	88.7	87.0	89.5	92.5	110.0	1.9%
Small Cap. Adv.	126.8	120.0	124.2	126.6	147.0	1.5%
China	717	770	790	802	850	0.7%
Developing World	**368**	**355.3**	**363**	**376**	**486**	**2.7%**
Africa	7.3	6.8	7.2	7.6	10.0	3.0%
Brazil	34.7	33.4	34.0	35.0	42.5	2.0%
CIS	111.5	106.6	108.0	109.0	122.5	1.2%
Eastern Europe	14.2	12.0	12.6	13.5	17.5	3.0%
Developing Asia	21.4	21.5	22.0	24.1	40.0	5.6%
India	76.8	78.5	78.0	82.0	113.0	3.4%
Latin America	31.7	30.0	31.6	34.0	45.0	3.3%
Turkey	35.9	33.5	34.0	34.5	33.9	0.0%
MENA	34.2	33.0	35.1	36.2	62.0	5.3%
World Total	**1,548**	**1,574**	**1,613**	**1,648**	**1,873**	**1.4%**
World Ex-China	831.2	804.0	823.3	846.4	1,022.6	2.0%

urce: WSD estimates

In addition, several factors in WSD's Global Metallics Balance data are especially positive for the Project:

- The expansion of USA steel demand in the next decade, leading to investment in Electric Arc Furnace ("**EAF**") capacity relying on direct reduced ironmaking ("**DRI**") and, in turn, DR grade pellets. Lower natural gas prices in the USA are making the construction of DRI facilities an attractive option for a USA-based steel mill.
- Further expansion of EAF steelmaking in China, the Middle East and Western Europe, the latter two (2) of which, along with the USA, are natural markets for the KéMag Taconite Project.

- Although an overall surplus of steel scrap appears to be developing, the future supply/demand balance for prime scrap is expected to be tight, creating a need for more supply of clean iron units in the form of DR grade pellets, especially as EAF steelmakers move up market into sheet production.

1.17.2 World Pellet Demand

The Project is targeting the production of pellet feed, acid and fluxed pellets for BF/BOF steel making, and DR grade pellets. With its extensive iron ore database, WSD is able to benchmark the quality and cost of the Project's products against competitor parameters, and has concluded as follows:

- BF Pellets – Would be well accepted by any iron making operation as comparable to the best Canadian and Swedish pellets, better than Brazilian pellets, and clearly better than pellets from elsewhere (CIS, India, etc.). Can contribute to improved blast furnace performance at Tata Steel Europe ("**TSE**").
- DR Grade Pellets – Quality will be comparable to the leading producers: Vale, Samarco, LKAB, IOCC and Arcelor Mittal Mines Canada.
- Global Cost Curve Position for Pellets – The KéMag Taconite Project would potentially be the lowest cost pellet producer in North America.
- Pellet Feed – Not a question of quality or cost for the Project, but market analysis suggests a future glut in the supply of sintering ore fines, concentrates and pellet feed. The level of production should be tailored to meet specific market requirements and no more.

1.17.3 Markets for the Project

The Project is expected to require a mix of captive and merchant market sales. Its range of products and a highly competitive cost structure, coupled with access to the new, deep-water ship loading facilities at Sept-Îles, Québec, would give the Project a global market reach for business development.

Tata Steel's European steel works alone could consume 50 to 60% of the Project's output. In January 2007, TSE acquired the Anglo-Dutch steel maker Corus Group, thereby establishing a major presence in the European market to supplement its core production base in India. TSE is pursuing a strategy of greater self-sufficiency in raw materials, and currently the only captive supply is TSMC's DSO product with first deliveries was in mid-2013.

Since TSE's operations are both geographically proximate to and historically familiar with supply from Canada, they are also natural outlets for the Project. In fact, product specifications are the result of collaboration with TSE's supplies team.

TSE's three (3) integrated steel works in Europe (Port Talbot – Wales, Scunthorpe – England and IJmuiden – The Netherlands) are shown in Figure 1.1. As the only European steel maker with on-site pelletizing, the IJmuiden works in the Netherlands provides the Taconite Project with a unique captive market opportunity for pellet feed in what will be a highly competitive open market environment as discussed above.

Figure 1.1 – Natural Fit of the Taconite Project to Tata Steel Europe's Operations

In order to supplement the core off-take requirements of TSE and minimize the Project's market risk, NML and TS are working together to identify reputable and reliable steel or steel tie-in companies with an interest in Project investment and/or a long-term supply contract. The relationships begin with Project introduction and interest is confirmed via a confidentiality agreement. Simply stated, the process continues through technical evaluation and other due diligence which, if successful, then leads to a letter of intent or memorandum of understanding, commercial negotiations and the eventual goal of a long-term and stable relationship with the Project.

To date, this marketing campaign has produced interest from companies in North America, Europe, Middle East and the Asia/Pacific.

1.17.4 Pricing

It is well known that the global pricing mechanism for iron ore has transitioned since the mid 2000's from the historical, annually negotiated benchmark system to shorter term settlements mainly using the 62 % Fe Cost and Freight ("**CFR**") China price as a reference, with upward or downward adjustments for product quality as appropriate.

The year 2012 has seen extreme price volatility, signaling the difficulty of predicting even a one-year price. With regard to the long-term price for iron ore, there are numerous views among industry analysts and forecasting services, resulting in a wide range of predictions.

Steel is a cyclical business, and by extension iron ore prices should reflect this cyclicality over time. The methodology used by WSD for arriving at a long-term price reflects this likelihood by assuming that the steel industry is subjected to boom and bust conditions typical of every cyclical business and, using the 62% Fe CFR China price as a baseline, calculating a weighted average price that reflects the overall impact of the various boom and bust conditions over the cycle being analyzed. Based on the above methodology, the long-term prices, FOB vessel at Sept-Îles, used in the Project Feasibility Study are shown in Table 1.4.

Table 1.4 – Long Term Pricing

Product	Long Term Pricing
Blast Furnace Pellets	US$ 116,61 /dry-t
Direct Reduction Pellets	US$ 126,86 /dry-t
Pellet Feed	US$ 90,00 /dry-t

1.17.5 Market and Strategy Summary

In summary the development of the Project by NML/TS is underpinned by the following key market assumptions and strategic rationale:

- Assuming a positive investment decision in 2013, the Project would come on stream in 2017 at a time of resurgence in the global steel industry;
- The Project's range of high quality pellets and pellet feed and favorable cost curve position gives it a competitive and global market reach;
- With its year-round shipping capability and ability to handle among the largest ore carriers, the Port of Sept-Îles provides a competitive advantage to the Project;
- The Project would have a core market for 50-60 % of its output at TSE's operations;
- Lower natural gas prices and the relatively high price of steel scrap in the USA are making the construction of DRI facilities an attractive option for a USA-based steel mill. This a natural market geographically for the Project;
- Other market opportunities exist in Western Europe, the Middle East and the Asia/Pacific.

Working with WSD, NML/TS has incorporated cyclical steel industry conditions into its analysis in order to develop long-term pricing that is as representative as possible.

1.18 Environmental Studies, Permitting and Social or Community Impact

1.18.1 Applicable Environmental Assessment Regimes and Permitting

The Project is expected to trigger the environmental assessment ("**EA**") regimes of general application established by the Canadian Environmental Assessment ("**CEA**") Act and the Environment Quality Act ("**EQA**") (Chapter I, Division IV.1), in addition to the provincial regimes of section 23 of the James Bay and Northern Québec Agreement ("**JBNQA**") (EQA, Chapter II, Division III) and section 14 of the Northeastern Québec Agreement ("**NEQA**") (*Règlement sur l'évaluation et l'examen des impacts sur l'environnement dans une partie du Nord-Est québécois*).

The federal regimes of section 23 of the JBNQA and section 14 of the NEQA are not expected to be triggered by the Project, because the components of the Project in the areas in question come under provincial jurisdiction. Where impacts on matters of federal jurisdiction are anticipated in the areas in question, the CEA Act, the Fisheries Act, the Metal Mining Effluent Regulations ("**MMER**") or other federal regulatory instruments will apply.

The regulatory regime of the National Energy Board Act ("**NEBA**") is not expected to be triggered for the Project, since the slurry transportation system will not be inter-provincial.

Ideally, the implementation of all the applicable regimes should be harmonized to the extent possible. To that effect, a conference call with the CEA Agency, the Major Projects Management Office ("**MPMO**") and the National Energy Board ("**NEB**") was held on May 3, 2013 to discuss how the respective regimes of the CEA Agency and the NEB might apply, particularly in terms of their harmonization.

Once the regulatory authorities have released the Project from further EA, the applications for the various permits that are required for site-preparation and construction can begin to be filed, followed by applications for permits for the start of operations. In order to expedite the start of construction, preparation of the permit applications can begin before the completion of the EA.

An amendment to schedule 2 of the MMER will be required for draining water bodies or watercourses frequented by fish for purposes of creating tailings disposal areas. An amendment to schedule 2 of the MMER will not be granted unless it can be shown that there is no viable alternative to disposing of them in the water bodies or watercourses in question. A thorough evaluation of the alternatives will be conducted and will be included in the EIS.

Representatives of Environment Canada and Fisheries and Oceans Canada have advised that the process of obtaining an amendment to schedule 2 can take up to 18 months. Until it has been completed, no work affecting the water bodies/watercourses in question can be undertaken. Other construction could proceed, but NML/TS would have to assume the risk that the Project might not be allowed to go ahead if the required amendment is not obtained.

1.18.2 Environmental Impact Statement Status

The drafting of the EIS is underway. At this time, the biophysical data are nearly completed except for the future location of the discharge point for the excess treated water from the slurry transportation system. At a minimum, marine survey should be carried out when the location will be known.

The data collection (desktop study) is nearly completed. Data collection during the public consultations, once the Project description will be tabled, will allow the completion of this part of the EIS.

Other sections of the EIS will be finalized once the EA guidelines will be issued by the governmental authorities.

1.18.3 Timetable

The following timetable for the EA releases assumes that the EIS is filed in August 2015:

- Receipt of questions from regulators (CEA Agency, *Ministère du Développement Durable, Environnement, Faune et Parc*/Kativik Environmental Quality Commission) on the EIS – Q4 2015;
- Submission of responses to questions on EIS – Q1 2016;
- Acceptance of EIS by regulators for purposes of public hearings – Q2 2016;
- Public hearings (BAPE, KEQC) – Q3-Q4 2016;
- Submission of responses to questions filed at the hearings – Q1 2017;
- Issuing of EA releases and conditions of release – Q2-Q3 2017 (taking into account the expected long delay for the KEQC);
- Completion of MMER process – Q3 2019.

1.19 Capital and Operating Costs

1.19.1 Capital Costs Estimate

The capital cost estimate for the Feasibility Study was assembled by ISLLP. Parts of the capital costs were developed by ISLLP and other areas were provided by consultants under the supervision of ISLLP. A matrix of responsibilities and detail of the estimate can be found in Section 21.0 of this Report.

The Project includes the mine, process plant and the slurry transportation system starting at the KéMag mine/concentrator site and continuing always in Quebec to a pellet plant in a common port in Pointe-Noire, Quebec.

The estimate has been developed using approved design criteria, flow sheets, engineering drawings and equipment lists. Enquiries were sent to suppliers for pricing of major equipment.

The capital cost estimate show in Table 1.5 has been developed on the basis of one (1) overall EPCM contractor who will provide the design, procurement and construction activities for the Project. All sub-contracts would be managed by the EPCM contractor. Table 1.6 shown the third party infrastructure excluded from the capital cost and accounted by an annuity in the financial analysis.

Table 1.5 – Capital Costs Estimate Summary

Area	Title	Total Costs ($'000)
1000	Mine Area	106,128
2000	ROM Crushing, Storage and Reclaim	382,938
3000	Concentration, Grinding, Separation and Upgrade	1,006,560
4000	Tailings Disposal and Water Management	23,022
5000	Concentrate Slurry Transport	237,845
5210	Slurry Transport	1,539,965
6000	Port Area (Sept-Îles)	332,376
6200	Pellet Plant (Outotec LSTK)	1,226,084
8100	Infrastructure and Utilities (Schefferville)	531,508
8200	Infrastructure and Utilities (Sept-Îles)	144,359
	Total Direct Cost	**5,740,784**
9000	Indirects Costs	1,421,000
9000	Contingency	867,000
	Total Indirect Cost	**2,288,000**
	Total Direct + Indirect	**8,028,784**
	Owners Cost (by NML)	210,000
	Total Project Capex	**8,239,000**
	EXCLUSIONS FROM Capex	
	Mine Mobile Equipment in Opex	323,000
	Hydro-Quebec Transmission Line in Opex	661,000
	Risk	334,300
	Escalation	Excluded

The above total sums are rounded up to $1,000.

Table 1.6 – Capital Cost vs Leasing

Area	Title	Total Costs ($'000)
	Capex – Mine and Process[1]	5,227,000
	Capex – Infrastructure[2]	3,012,000
	Total Project Capex	**8,239,000**

Costs of major mining equipment and power transmission line are not included in Capex, but servicing costs are in cash cost.
2 Consists of slurry transportation system, product storage and reclaiming system are to be financed on the basis of long term debt.

1.19.2 Basis of Estimate

The estimated capital cost is based on the following key assumptions:

- The estimate is expressed in February 2013 Canadian dollars ("**CAD**") since the majority of the bids were received during that period;
- The following items are excluded; currency exchange fluctuation, allowance for industrial dispute or industrial action, environmental permitting, all taxes and duties, escalation (see construction indirects in Section 21.0), Project financing and interest during construction;
- No allowance is made for acceleration or deceleration of the Project schedule;
- Plant operating costs are excluded from the capital cost;
- Project insurance is included in Owner's cost;
- The estimate base currency is CAD. The capital cost estimate consists of items quoted in various foreign currencies which have been converted into CAD using exchange rates as of February 5, 2013. The vast majority of pricing for equipment and bulk materials are based on CAD. Table 1.7 shows the currency exchange rates and the percentages content in different currencies and the percent content of costs for each of the listed currencies, excluding the pellet plant (by Outotec).

Table 1.7 – Currency Exchange Rates and Percent Content

Currency Code	Currency Name	Canada	Percent Content (%)
CAD	Canadian Dollar	1.00	87.5
USD	US Dollar	1.00	7.0
EUR	Euro	1.30	5.5

The capital cost estimate covers the facilities included in the scope of work described in Sections 17.0 and 18.0.

The proposed construction work week is on the basis of a 60-hour with rotations of 21 days in followed by seven (7) days out [travel during the seven (7) days out].

1.19.3 Cost Basis and Methodology

The labour crew mix wage rates were developed on the basis of the Quebec Labour collective agreement for April 2013. Labour productivity and site considerations for the Project have been evaluated and taken into consideration.

All quantities generated for the estimate are mainly based on engineering material take-offs and deliverables which exclude contingencies of any kind.

1.19.4 Operating Costs

The operating costs were developed for each component of the Project and are presented in detail in Section 21.0 of this Report. The costs at Year-4 of operation are summarized in Table 1.8 below.

Table 1.8 – Summary of Estimated Opex by Area (Average)

Areas	Description	Costs for LOM ($'000)	Unit Costs ($)	Units
1000	Mine	8,380,119	17.76	per tonne of concentrate
2000, 3000, 4000	Concentrator	7,614,038	16.13	per tonne of concentrate
5000	Slurry Transport	421,237	0.89	per tonne of concentrate
6100 - 6200	Pelletizing Plant	4,043,786	11.09	per tonne of pellets
6300 - 6600	Port Operation	1,279,841	2.59	per tonne of total production
8000	G&A and Infrastructure	1,615,287	3.27	per tonne of total production
		Unit Cost for Concentrate	**40.64**	**per tonne of concentrate**
		Unit Cost for Pellets	**49.68**	**per tonne of pellets**
Note: Benefit and other payments are not included in unit costs				

The Opex presented is based on the estimated consumptions of consumables, reagents and power. Maintenance supplies cost factors were used for buildings. Vendor data was used where possible. The accuracy level is in the range of ± 15 %, as required by this estimate class and is deemed appropriate a Feasibility Study level. No escalation or contingency has been applied to the operating costs. The currency exchange rates applied are the same rates used for the Capex presented in this Section.

The base date of the operations cost estimate is February 2013. The capital cost used to factorize operating supplies, maintenance of buildings and of fixed equipment was updated in October 2013. The Bunker C fuel oil is dated August 2013 as well as the fuel price.

The organizational structure for the operations phase is designed to support the successful operation of an iron ore mine and beneficiation plant in a remote location, a slurry transportation system, a pelletizing facility and associated material handling facilities while producing cost effectively quality iron ore concentrate and pellets.

Table 1.9 presents the Project operating labour force at Year-4 of operation.

Table 1.9 – Labour Force Year-4

Area	Senior Executives	Superintendents, Managers, and General Foremen	Supervisors and Engineers	Operators	Maintenance Personnel	Technicians	Others	Total
Mine	1	2	10	344	123	39	4	523
Crushing	0	1	6	33	14	0	0	54
Concentrator	1	7	19	55	55	16	18	171
Slurry Transport	1	4	4	7	27	0	2	45
Pellet Plant	0	6	21	42	73	9	18	169
General Administration and Infrastructure	3	11	0	0	0	72	62	148
Port Facilities	7	7	8	17	3	14	5	61
Total	**13**	**38**	**68**	**498**	**295**	**150**	**109**	**1,171**

1.20 Economic Analysis

The economic/financial assessment of the Project is based on first-quarter 2013 price projections and cost estimates in Canadian currency. An exchange rate of USD 0.90 to CAD 1.00 is assumed to convert the USD price projections into CAD. As well, this rate is used to adjust Opex estimates and 7.0 % of the original Project Capex initially priced in USD at an exchange rate of 1 to 1. A similar adjustment is made to 5.5 % of the original Project Capex initially priced in Euros at an exchange rate of EUR 0.77 to CAD 1.00 to account for a revised exchange rate of EUR 0.71 to CAD 1.00.

No provision is made for the effects of inflation. The evaluation is carried out on a 100 % equity basis. Current Canadian tax regulations are applied to assess the corporate tax liabilities while the recently proposed regulations in Quebec (Bill 55, December 2013) are applied to assess the mining tax liabilities.

The assessment of the Project assumes that some infrastructure elements, namely, the slurry transport infrastructure between the mine site and Pointe-Noire, the mine site accommodations, and the stock yard and materials handling facilities for ship loading are leased. The financial indicators under base case conditions are presented in Table 1.10.

Table 1.10 – Base Case Financial Results (100 % Equity)

Financial Indicators	Results	Units
Before-tax Net Present Value ("**NPV**") @ 8 %	5,261.8	$M
After-tax NPV @8 %	2,241.3	$M
Before-tax Internal Rate of Return ("**IRR**")	17.5	%
After-tax IRR	13.2	%
Before-tax Payback Period	4.9	Years
After-tax Payback Period	5.7	Years

A sensitivity analysis reveals that the Project's viability is not significantly vulnerable to variations in capital and operating costs, within the margins of error associated with feasibility study estimates. However, the Project's viability remains vulnerable to the larger uncertainty in future sale prices.

The complete text of the Economic Analysis can be seen in Section 22.0 of this Report.

1.20.1 Financing

Although the base case assessment assumes 100 % equity financing, the financial model was created to address the case in which debt financing is considered. Here, it is assumed that 70 % of the initial capital cost will be financed through long-term debt sources. To the extent possible, the disbursement of borrowed funds is delayed in order to reduce the amount of interest payable.

As financing arrangements are not yet in place, it is assumed that the annual interest rate on borrowed funds is seven (7) % (current U.S. Prime rate of 3.25 % + 3.75 %).

The assumed terms of repayment are single payments at the end of each year starting at the end of the first full year of production over a period of ten (10) years. It is important to note that no additional work was done to confirm these assumptions with potential lenders. The financing terms are used only to demonstrate the effect of leverage.

1.20.2 Financial Model and Results

Table 1.11 presents the financial analysis results on a before-tax basis.

Table 1.11 – Before-tax Financial Indicators

Before-tax Financial Indicators	Results ($M)
Total LOM Revenue	61,346.6
Total LOM Operating Costs*	24,298.3
Total LOM Lease Payments	6,990.6
Initial Capital Cost	5,291.9
Total LOM Sustaining Capital	1.096.9
Total Cash Flow	23,668.9
NPV @ 8 %	5,261.8
NPV @ 10 %	3,455.0
NPV @ 12 %	2,135.8
IRR	17.5
Payback Period	4.9

* Includes Agreement payments

Table 1.12 presents the financial analysis results on an after-tax basis. The first five (5) lines of Table 1.11 (i.e., total revenue to total sustaining capital) are not reproduced in Table 1.12 because they are identical.

For the purpose of illustrating the effect of debt financing, Table 1.13 compares the financial outcomes with 100 % equity financing versus those with 30 % equity/ 70 % debt financing (i.e. levered). The impact of leverage is greater on an after-tax basis than on a before-tax basis because interest payments are tax deductible. For this reason, the results are shown only for the after-tax assessment.

Table 1.12 – After-tax Financial Indicators

After-tax Financial Indicators	Results ($M)
Total LOM Corporate Taxes	7,272.4
Total LOM Mining Taxes	3,175.9
Total Cash Flow	13,221.6
Net Present Value @ 8 %	2,242.3
Net Present Value @ 10 %	1,153.8
Net Present Value @ 12 %	359.0
Internal Rate of Return	13.2 %
Payback Period	5.7 Years

Table 1.13 – Effect of Leverage

After-tax Financial Indicators	Results ($M)
Net Present Value @ 8 %	2,241.3
Levered Net Present Value @ 8 %	2,697.1
Internal Rate of Return	13.2 %
Return on Equity (“**ROE**”)	21.9 %

1.20.3 Sensitivity Analysis

A sensitivity analysis is carried out on the base case scenario described above to assess the impact of changes in market prices (across-the-board variations in all three (3) product prices), total pre-production capital cost and operating costs on the Project’s NPV @ eight (8) % and IRR. Each variable is examined one-at-a-time. An interval of ± 30 % with increments of ten (10) % was used for all three (3) variables. It is to be noted that the margin of error for cost estimates at the feasibility study level is typically ± 15 %. However, the uncertainty in price forecasts usually remains significantly higher, and is a function of price volatility.

The before-tax results of the sensitivity analysis are shown in Figure 1.2 and Figure 1.3. Figure 1.2 showing variations in NPV indicate that the Project’s before-tax viability is not significantly vulnerable to the under-estimation of capital and operating costs, taken one at-a-time. The vertical dashed lines represent the typical 15 % margin of error associated with the Capex and Opex estimates. The NPV is more sensitive to variations in operating expenses than it is to capital expenditure, as shown by the steeper curve. As expected however, the NPV is most sensitive to variations in prices. Across-the-board reductions of about 25 % in prices result in break-even conditions, i.e., a net present value of zero.

Figure 1.2 – Sensitivity of the Before-tax Net Present Value

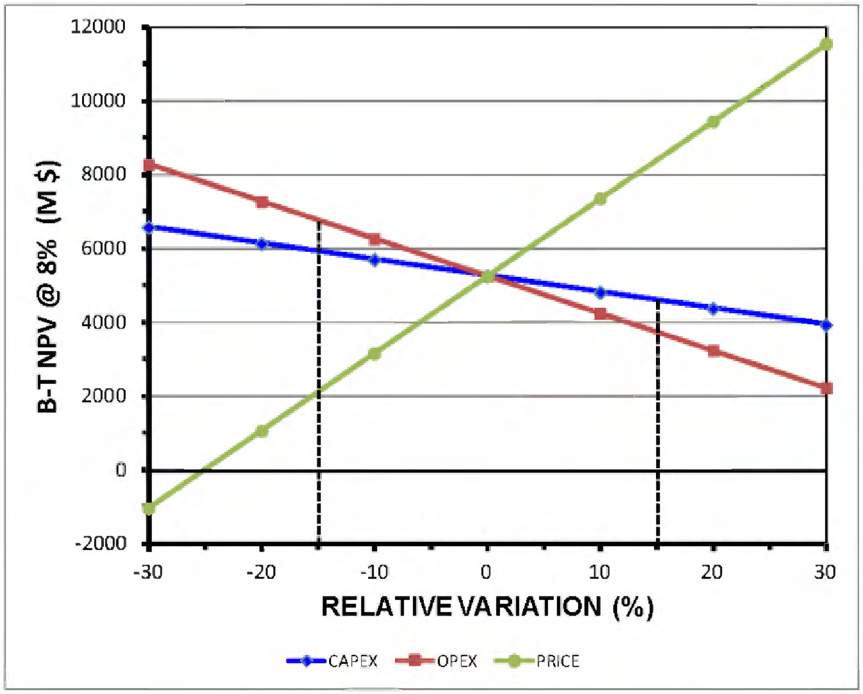


Figure 1.3 showing variations in IRR provides similar conclusions. In contrast with Figure 1.2 which shows linear variations in NPV for the three (3) variables studied, variations associated with IRR are not linear. Due to the different timing of pre-production capital expenditure versus operating expenses over the mine life, the IRR is more sensitive to variations in capital expenditure, especially negative variations. The IRR is reduced to eight (8) %, i.e., break-even conditions (shown by the horizontal dashed line), for the same across-the-board reductions in prices as noted above in the case of the NPV.

The after-tax results of the sensitivity analysis are shown in Figure 1.4 and Figure 1.5. The same conclusions about the sensitivity of the Project's viability to variations in capital expenditure, operating expenses and prices can be drawn here. On an after-tax basis however, break-even conditions are reached at across-the-board reductions in prices of 19 %.

Figure 1.3 – Sensitivity of the Before-tax Internal Rate of Return

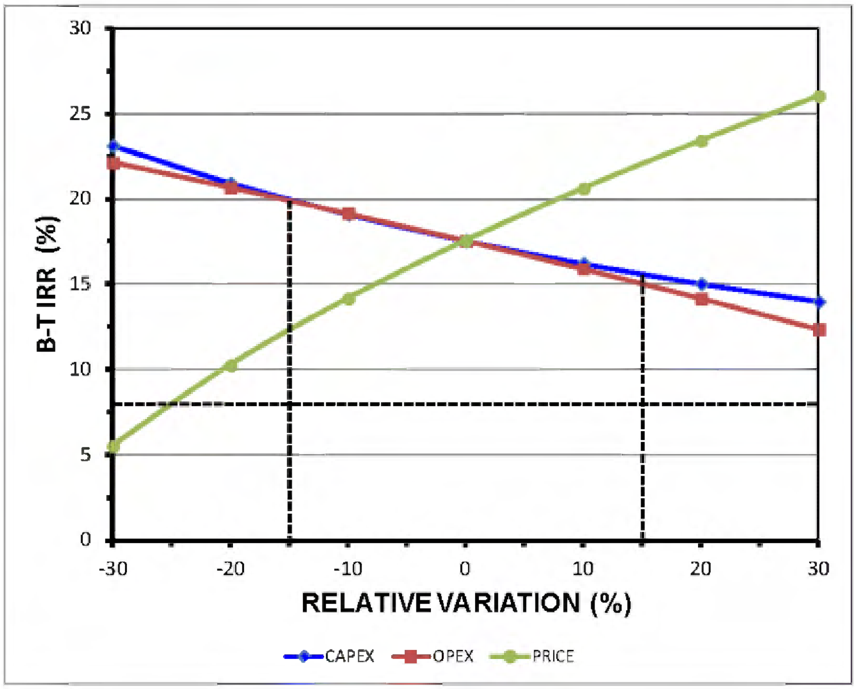


Figure 1.4 – Sensitivity of the After-tax Net Present Value

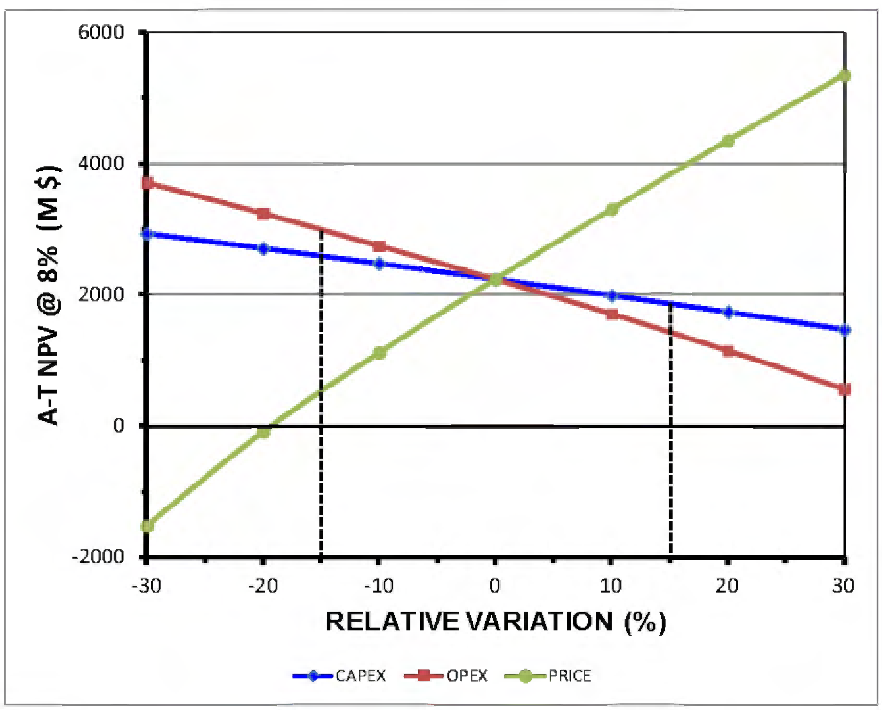
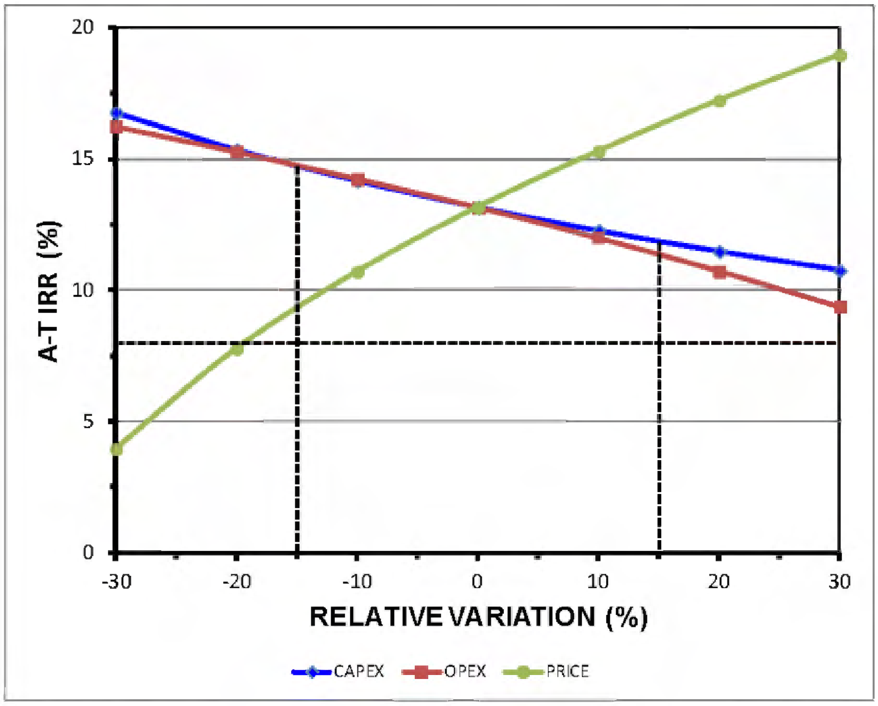


Figure 1.5 – Sensitivity of the After-tax Internal Rate of Return



1.21 Adjacent Properties

The Property is surrounded on all sides by claims held by NML, except for a block of adjacent claims at the southeastern boundary registered under TS. The southeastern limit of the KéMag Property follows the Quebec-Labrador boundary and abuts against the Licenses held by NML, TS and LabMag, the latter hosting the LabMag iron deposit.

1.22 Other Relevant Data and Information

1.22.1 Schedule

A Project master schedule for the Feasibility Study has been developed to cover the major Project milestones. The Project master schedule contains engineering, procurement, construction and pre-operational testing and commissioning activities at a level of detail commensurate with the progress of scope definition.

The major milestones have been tabulated in the following Table 1.14 showing the Project activities through its cycle.

The critical paths are through the procurement of steel, the construction of the slurry transport system and the construction of the pellet plant.

Table 1.14 – Major Milestones

Description	Month
General	
Final FS Study Submitted to NML/TS	0
Stage Gate Approval By NML/TS	4
Geotechnical and Hydraulic Survey Completed	9
Start Basic Engineering	9
EPCM Contract Awarded	11
Start Detailed Engineering	19
Procurement Start	20
Start Construction	21
Design Criteria, Process and Layouts Frozen	22
Establish Control Budget	23
Start Pre-Operational Verification	55
End of Construction	60
End of Pre-Operational Verification	64
Commissioning Completed	68
Plant Operation Start	68
Detailed Engineering	
Start Basic Engineering	9
Start Detailed Engineering	19
Detailed Engineering of Slurry Transport System	22
Detailed Engineering of Structural Steel	35
Procurement of Pellet Plant	35
Construction / Pre-Commissioning / Commissioning	
Start Temporary Camp Installation on Site	18
Start Civil Works Mine Pit	19
Start Slurry Transport System Construction	24
Start Flotation Plant Concrete Construction	35
Start Primary Crushing Concrete Construction	42
Start Structural Steel Installation	42
Start Mechanical Installation	43
Start Electrical Installation	45
Pelletization Plant Commissioning	64
Pre-commissioning Complete	67

1.22.2 Project Execution Plan

The Project Execution Plan ("**PEP**") has been developed based on the Feasibility Study work done to date on the Project. The PEP outlines the plans, resources, mechanisms and procedures that NML/TS expects from the EPCM contractor and covers all aspects of the Project from engineering design through pre-operational verification and assistance during commissioning. This is described in detail in Section 24.2 (and following) of this Report.

1.23 Interpretation and Conclusions

The Project is technically feasible; ore can be mined, treated and delivered into export vessels by incorporating proven processes and technologies.

Based on 22-year cash-flow, the IRR before taxes is 17.5 % and from 13.2 % after taxes assuming 100 % equity and standard mining income taxes. Therefore, the Project has financial viability.

The life of mine can be further enhanced by exploiting the lower grade resources currently being stockpiled.

1.23.1 Project Highlights

The geology and mineral reserve is well known and can supply a consistent quality run of mine to the concentrator for 22 years of production.

The process has been developed after extensive test work and by experienced process engineers, although refinements and confirmation of certain processing steps are possible. There is enough confidence in the flow sheets so that refinements and optimization can be tackled during the basic engineering phase.

Conventional disposal of tailings can be engineered to meet the environmental requirements while remaining economically viable.

The slurry transport system is an economical and reliable mean of transporting iron ore slurry, especially fine taconite concentrate. Frost protection is of paramount importance and can be achieved by a combination of methods according to the geographical location and soil conditions. Such trade-off studies should be assessed early in the basic engineering phase of the Project along with field surveys.

Pelletizing testing has confirmed the high probability of achieving high plant throughput.

A new multi-user dock with a capacity of 50 Mt/y and capability of handling 350,000 DWT vessels is being built by SIPA and is planned to be commissioned by the fall of 2014. The product handling facilities will connect to this deep-sea port terminal. The product handling facilities will connect the pelletizing plant to this deep-sea port terminal. NML has invested $ 38 M in the new dock to obtain the rights to ship 15 Mt/y of product.

The market study indicates a future pellet demand and both the market trends and forecasts indicate higher pellet premium.

The Project is expected to trigger several regimes of EA. Discussions with the applicable governments have been held and it is concluded that a single EIS covering all the components of the Project is proposed to be submitted by August 2015.

Constructability strategy of using pre-engineered structures, skid mounted equipment and modularization to the maximum extent allowed by the transportation channels will lower the risk of cost and schedule overruns.

During the construction phase, the Project will create on average 2,500 construction jobs over 42 months and 300 management and engineering jobs spread over 60 months. The construction labour will peak at 4,600 labourers and 400 EPCM employees in the summer of the last full year of construction at all sites. During operation, 1,100 direct employees will be necessary and numerous indirect jobs will be created through the development of secondary and tertiary industries that will be supporting the operation and maintenance of this Project. This is a major component of the economic development of the Labrador Trough and Sept-Îles.

1.23.2 Risk Evaluation

ISLLP conducted risk reviews workshop with ISLLP experts in their various fields and with NML/TS Project team members. The objectives of the risk sessions were to identify risks, mitigations and action plans in the Feasibility Study phase to pursue up to the execution. The outcome of these workshops was compiled in the risk register and an action log formulated for the Project.

1.23.3 Key Project Concerns

- Mining below Lake Harris however the lakes are shallow and flow can be diverted;
- Lake with fish-habitat is an environmental concern but likely to be permitted. The cost has not been established;
- Long-distance slurry transportation systems in freezing climates, depth of burial review;
- Environmental Assessment, EIS submission delayed to August, 2015;
- Lack of detailed geotechnical assessment could result in surprises and have a significant impact in the construction Capex and hence must be completed before start of basic engineering and the finalization of the Project budget.

The risk level as well as risk qualitative analysis was determined by the participants who selected the level of consequences, mitigation plans including action plans for each identified and evaluated risk. On the positive side, during the next phase, there is good possibility in to improve the flow sheet resulting in reduction of the Capex and Opex.

1.23.4 Conclusion and Next Phase

Early identification of constructability techniques to maximize pre-assembly of concrete, steel, equipment and any other way of reducing the installation time on site, is paramount for a successful project.

Early in the project's lifecycle, an experienced constructability engineering team should be reviewing the design layouts, specifications, objectives, and site conditions to work with the design engineers to establish possible guidelines to standardize buildings, foundations and any other criteria to minimize labour costs and impact on schedules.

The constructability team should also work very closely with the logistics group to determine the ideal transportation and construction philosophy for the engineering team.

In summary, during the next phases of the project development, the following constructability activities should be considered:

- Incorporate an analysis of operations and maintenance into the facility design;
- Review site conditions and access;
- Design development considering construction details and strategy;
- Design quality control;
- Standardize, modularize, coordinate and involve suppliers, consultants and subcontractors;
- Complete the 3D model of the plant and facilities; and
- Refine the scheduling strategy.

1.24 Recommendations

Based on the Feasibility Study schedule, it is recommended to proceed shortly with partial funding for basic engineering for lesser defined areas of the Project. It is also recommended to execute test work and field surveys that are required to support basic and detailed engineering. Topographical surveys would be best executed prior to the basic engineering activities by one (1) year.

At this stage, there are areas of lesser definition that require confirmation from test work carried on representative samples by laboratories and/or suppliers on before or during basic engineering. The main areas of lesser definition are:

- Mine dewatering (hydrogeology);
- Waste rock dumps and run-off water management (acid rock drainage);
- Tailings pumping, dewatering and material handling;
- Slurry transport hydraulic design due to lack of representative rheology testing;
- Slurry transport heat transfer study;

- Frost depth and earthwork for slurry transport;
- Export terminal earthwork and foundation design;
- Water treatment facility at Pointe-Noire due to lack of filtrate sample analysis.

Field work and test work must be done, as required, in each area of lesser definition. This will be followed by trade-offs and complementary studies required for basic engineering for each facilities. During that period of time, the Owner must establish a Project team and continue the environmental approval and permitting processes.

The basic engineering phase will take the Project from the current level of engineering (10-15 %) to approximately 25 %, thus allowing for a better than 15 % estimate accuracy which will provide a control budget for the execution phase.

No long-lead procurement items are deemed to threaten the schedule at this stage. The EPCM approach was selected where engineering is prioritized in the first phase and the EPCM contractor integrates known technologies into a plant as opposed to integrating technology packages.

There are however three (3) construction packages that, by their nature, will need to be tendered early as their duration extends for the full construction period. They are the slurry transport system construction, pelletizing plant and the power transmission lines. Environmental permitting is also key to two (2) of these contracts.

To accelerate the Project an estimated \$ 6 M will be spent for completion of the EIS and miscallenous items mostly related with alternative tailing disposal option and confirmation of the depth of frost penetration along the slurry transport route. The expected cost is shown in Table 1.15 below. A complete list of recommendations per area can be found in Section 26.0 of this Report.

Table 1.15 – Next Phase Estimated Costs

Activity	Estimated Costs (\$)
Completion of the EIS Report	4,000,000
Study of an Alternative option on Dry Stacking of Tailings	1,200,000
Frost Dept Confirmation along the Slurry Transportation Route	600,000
Miscellaneous	200,000
Estimated Total Costs	**6,000,000**

Source: NML

2.0 INTRODUCTION

2.1 Terms of Reference - Scope of Study

Tata Steel Global Minerals Holdings Pte Ltd. ("**TS**"), New Millennium Iron Corp. ("**NML**") and LabMag Limited Partnership ("**LLP**") have jointly commissioned a Feasibility Study ("**FS**") to develop the LabMag and KéMag deposits (referred to as Taconite deposit) which form part of the Millennium Iron Range located in the Province of Newfoundland and Labrador and in the Province of Quebec.

Based on this Feasibility Study, NML has mandated Met-Chem to complete a Canadian National Instrument 43-101 Technical Report ("**Report**") for each of the LabMag and KéMag properties in order to advance the New Millennium Taconite Project.

This Report has been completed through the combined efforts of several companies and individuals, namely:

- Innu SNC Lavalin Partnership ("**ISLLP**") who prepared the NML/TS Taconite Project Feasibility Study which forms the basis for this Report. ISLLP utilized the services of Met-Chem and BRASS Engineering who respectively prepared the mining and the slurry transportation system chapters;
- NML/TS who utilized the services of the services of the following companies to complete the areas not provided by ISLLP, but included as part of the Feasibility Study;
  - Met-Chem who prepared the geological and resources studies;
  - Outotec GmbH who prepared a feasibility study for the design and estimate of the pelletizing and dewatering of slurry;
  - World Steel Dynamics ("**WSD**") and Papillon Mineral Services for the market studies;
  - SNC Lavalin Inc. Environment & Water Division ("**SLE**") who provided the input to the chapter on Environmental Studies, Permitting and Social and Community Impact.

Based on the above, NML/TS provided Met-Chem with chapters on Geology and Mineral Resources, Filtration and Pelletizing, Environment, Permitting, Communities and Social Responsibilities, as well as Marketing and the financial evaluation.

2.1.1 KéMag Deposit

For the KéMag deposit, NML holds a 100 % interest in the claims that constitute the KéMag taconite iron ore deposit at Lake Harris in the Province of Québec, Canada (the "KéMag Deposit"). The property is located approximately 50 km north of Schefferville, Québec, as shown in Figure 4.1.

Early in 2007, given the firm's familiarity with the geology of the area, Geostat Systems International Inc. ("**Geostat**") was again retained by NML, to prepare a mineral resource estimate for the KéMag Deposit. The KéMag Deposit was not as advanced as the LabMag Deposit and did not have as much exploration data but, because of its similarity to the LabMag Deposit, Geostat relied on the knowledge gained from the LIOP and felt confident to apply it in the modeling of the KéMag resources even though at that time the KéMag Deposit only had 29 drill holes drilled in 2006 with 75 % of the cross-sections containing only one drill hole. Subsequent to the 2006 drilling, an additional 45 holes (4,884.1 m) were drilled in 2007 followed by drilling of 15 holes (2,216.1 m) in 2008.

In classifying the KéMag resources, Geostat used the same classification criteria as for the LabMag Deposit and as a result, Geostat was able to delineate Indicated mineral resources on the KéMag Property. A Technical Report compliant with the requirements of NI 43-101 was filed with the Ontario Security Commission in March 2007.

2.2 Source of Information

In addition to public information, reports and information supplied by NML, Met-Chem has used the information contained in relevant reports by Geostat Systems International Inc. and by WGM, as listed under References, in the preparation of the present Report. The Geostat's reports provided data on the previous resource estimates of KéMag and the technical reports by WGM documented the resources of the KéMag deposit, as well as their audits of the Midland Research Center ("**MRC**"), Nashwauk, Minnesota, USA, and of a pre-feasibility study.

2.2.1 Contributing Authors

This Report was completed through the efforts of two (2) companies and individuals: ISLLP, Met-Chem, Joe Poveromo and Michel L. Bilodeau.

2.2.2 Qualified Persons

The main qualified persons ("**QP**") responsible for the development of this Report are Yves A. Buro, Eng., Schadrac Ibrango, P. Geo., Ph. D., Jeffrey Cassoff, Eng, and Charles Cauchon, Eng., all with Met-Chem Canada Inc., Pierre Julien, Eng. and Luc Belanger, Eng. both from ISLLP, Eric Giroulx, Eng. with SLE, Joe Poveromo, Eng. and Michel L. Bilodeau, Eng., M. Sc. (App.), Ph. D.

Table 2.1 provides a list of qualified persons and their respective sections of responsibility. The certificates for people listed as Qualified Persons can be found at the beginning of the Report under Date and Signature – Certificates.

Table 2.1 – Qualified Persons and their Respective Sections of Responsibility

Section	Title of Section	Qualified Persons
1.0	Summary	Charles H. Cauchon, Met-Chem
2.0	Introduction	Charles H. Cauchon and related QPs
3.0	Reliance on Other Experts	Charles H. Cauchon, Met-Chem
4.0	Property Description and Location	Yves A. Buro, Met-Chem
5.0	Accessibility, Climate, Local Resources, Infrastructure And Physiography	Yves A. Buro, Met-Chem
6.0	History	Yves A. Buro, Met-Chem
7.0	Geological Setting and Mineralization	Yves A. Buro, Met-Chem
8.0	Deposit Types	Yves A. Buro, Met-Chem
9.0	Exploration	Yves A. Buro, Met-Chem
10.0	Drilling	Yves A. Buro, Met-Chem
11.0	Sample Preparation, Analysis and Security	Yves A. Buro, Met-Chem
12.0	Data Verification	Yves A. Buro, Met-Chem
13.0	Mineral Processing and Metallurgical Testing	Luc Belanger, ISLLP
14.0	Mineral Resources Estimates	Schadrac Ibrango, Met-Chem
15.0	Mineral Reserve Estimates	Jeffrey Cassoff, Met-Chem
16.0	Mining Methods	Jeffrey Cassoff, Met-Chem
17.0	Recovery Methods	Pierre Julien, ISLLP
18.0	Project Infrastructure	Pierre Julien, ISLLP
19.0	Market Studies and Contracts	Joe Poveromo
20.0	Environmental Studies, Permitting and Social or Community Impact	Eric Giroux, SLE
21.0	Capital and Operating Costs	Pierre Julien, ISLLP
22.0	Economic Analysis	Michel L. Bilodeau
23.0	Adjacent Properties	Yves A. Buro, Met-Chem
24.0	Other Relevant Data and Information	Charles H. Cauchon, Met-Chem
25.0	Interpretation and Conclusions	Charles H. Cauchon, Met-Chem
26.0	Recommendations	Charles H. Cauchon, Met-Chem
27.0	References	

2.3 Effective Date and Declaration

This Report is issued in support of the New Millennium Iron Corp press release, dated March 27th, 2014, entitled "New Millennium Iron Corp, Announces Positive Results for the Taconite Project Feasibility Study". The effective date of this NI 43-101 Report is Mach 27th, 2014 and the issue date is May 9th, 2014.

This Report provides an independent Technical Report for the Feasibility Study of the iron mineralization of the KéMag Deposit, in conformance with the standards required by NI 43-101 and Form 43-101F1. The estimate of mineral resources contained in this Report conforms to the CIM Mineral Resource and Mineral Reserve definitions.

ISLLP and Met-Chem are not insiders, associates, or an affiliate of NML or TS or Tata Steel Minerals Canada and neither ISLLP nor Met-Chem nor any affiliate has acted as an advisor to NML, its subsidiaries or its affiliates, in connection with this Project.

It should be understood that the mineral resources presented in this Report are estimates of the size and grade of the deposits based on a number of drilling holes and samplings and on assumptions and parameters currently available. The level of confidence in the estimates depends upon a number of uncertainties which include, but are not limited to, future changes in metal prices and/or production costs, differences in size and grade and recovery rates from those expected, and changes in Project parameters. In addition, there is no assurance that the Project will proceed to implementation.

The comments in this Report reflect ISLLP and Met-Chem's best judgment in light of the information available at the time of preparation. ISLLP and Met-Chem reserve the right, but will not be obligated, to revise this Report and any conclusions if additional information becomes known to ISLLP and Met-Chem subsequent to the date of this Report.

2.4 Site Visit

A visit to the site was carried out on September 18th and 19th, 2012 by Yves A. Buro, Eng., Schadrac Ibrango, P. Geo., Ph.D., both Senior Geologists, and Jeffrey Cassoff, Eng., Mining Engineer on September 18th, 2012. Pierre Julien, Eng. visited the site October 16th to 18th, 2012. All of the mentioned above are QPs according to the terms of NI 43-101 and have contributed to this Report.

Yves A. Buro and Schadrac Ibrango examined a series of outcrops and hole collars in the field and visited the office and core handling facilities in Schefferville. The core from selected holes was examined and 26 samples from the KéMag deposit were independently selected for check analysis.

Details on site visit and the results from the check samples are discussed under Section 12.4.2 of this Report.

2.5 Units and Currency

In completing this Report, the following terms of reference apply:

- All units of measurement are in the metric system, unless otherwise noted;
- All costs, revenues and financial reviews have been completed in Canadian Dollars (“**CAD**”);
- All metal prices have been quoted in US Dollars (“**USD**” or “**US$**”), unless otherwise noted;
- An exchange rate of USD 1.00/ CAD 1.00 was used to develop the capital and operating cost estimates and USD 0.90/CAD 1.00 was used in the financial evaluation;
- Other standards are specified, as necessary, in individual Sections.

3.0 RELIANCE ON OTHER EXPERTS

This NI 43-101 Report is based on NML Feasibility Study (dated March 27th, 2014) prepared by ISLLP and other specialized firms to development of the NML taconite projects. All the information for this Report was extracted from the Feasibility Study or supplied by NML and the summarized texts were reorganized in the NI 43 101 format.

Met-Chem performed a cursory review of the Feasibility provided to NML by ISLLP and other consultants for development of this NI 43 101 Report. However, each consultant remains fully responsible for his own work and Certificates of Authors that comply with NI 43 101 regulation are included at the beginning of this Report.

The previous Section 2.0 of this Report described ISLLP's organization to produce the Feasibility Study and the responsibility of each group.

All are specialists in their respective fields and Met-Chem has no reason to doubt their conclusions and recommendations.

The responsibility for the various components of the Executive Summary, Other Relevant Information, Interpretation and Conclusions and Recommendations remains with each qualified persons for their specific area of the scope.

Met-Chem has not researched legal ownership information such as property title and mineral rights and has relied on information provided by NML.

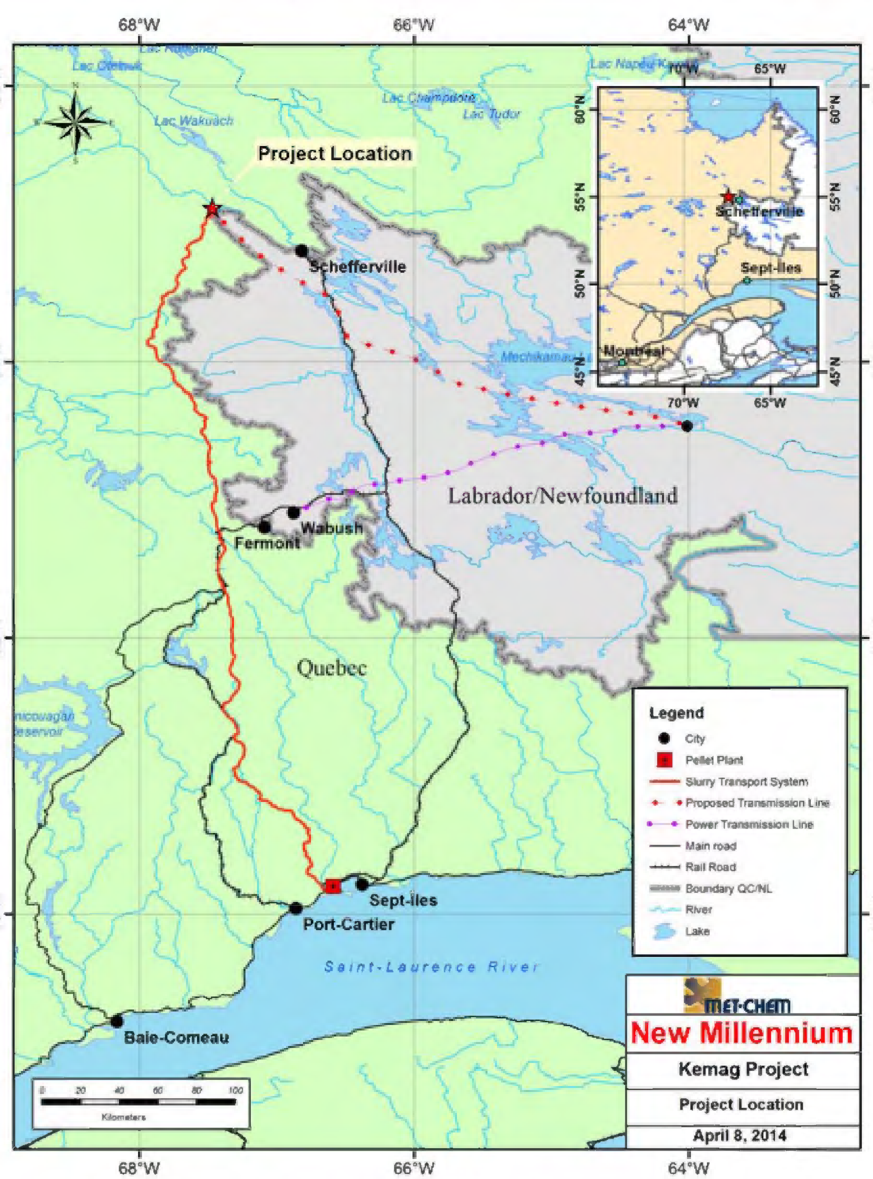
This Report is intended to be used by NML as a Technical Report with Canadian Securities Regulatory Authorities pursuant to provincial securities legislation.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 Location and Access

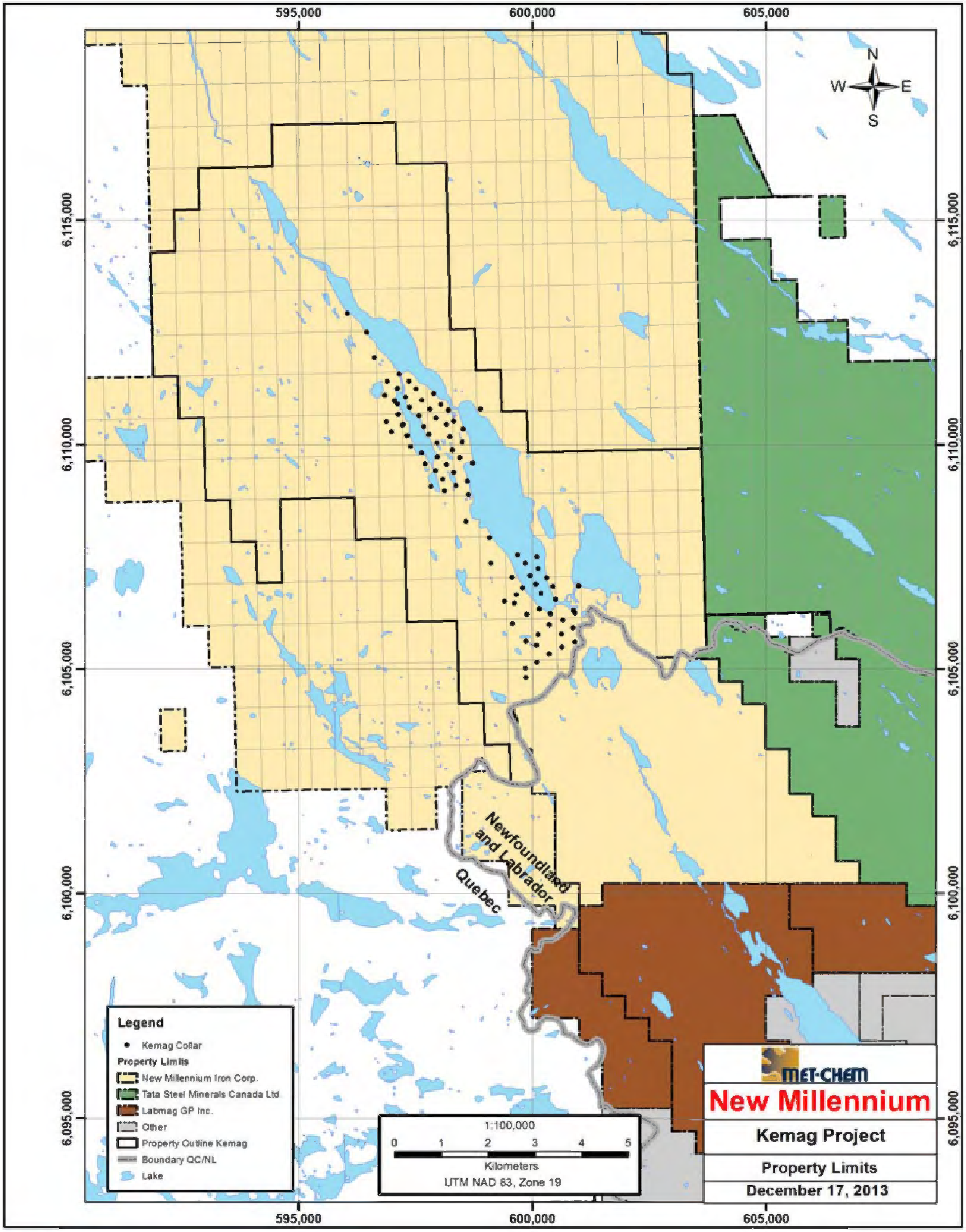
The KéMag Property, previously known as the Lake Harris Iron Property, is situated in the municipality of Rivière Koksoak in Northern Québec, about 40 km to the northwest of the town of Schefferville, Québec. The Property is approximately 245 km north of Labrador City, and 550 km due north of Sept-Îles, Quebec. The location of the Property and other Project elements are shown in Figure 4.1 and Figure 4.2.

Figure 4.1 – Project Location



Source: "© Department of Natural Resources Canada. All rights reserved."

Figure 4.2 – KéMag Property, Claim Map



The area is centered at 55°07'N Latitude and 67°27'W Longitude and is located on 1/50,000 National Topographic Map Reference (NTS or SNRC: Système National de Référence Cartographique) sheets 23O/03 and 23O/04.

4.2 Property Description

The Property is comprised of one (1) block of 171 contiguous claims covering an area of approximately 81 km^2 cut out of a large swath of land extending toward the northwest and northeast covered by claims held by NML (Figure 4.2). The claim group extends for a distance of about 15.5 km along a NNW-SSE trend. The KéMag abuts the boundary of the Province of Newfoundland and Labrador on the southwest and is contiguous with the LabMag Property in Labrador. The northern limit of the proposed mine site of the KéMag Project lies about 18 km north of the northern extremity of the proposed mine site of LabMag.

All the claims were acquired as Map-Designated Claims ("**CDC**") and registered under NML as the 100 % holder. The Property has not been legally surveyed but the location of map-staked claims is defined on the basis of Universal Transverse Mercator ("**UTM**") coordinates.

The information on the claims in Quebec is accessible through the Register of Real and Immovable Mining Rights via the GESTIM geomatics application of the *Ministère des Ressources Naturelles du Québec* ("**MRN**"). On January 14, 2014 Met-Chem accessed the GESTIM system and noted the registration dates for the Property extending from the earliest on January 20, 2005 to the most recent acquisition on May 7, 2008. Expiry dates of the claims range from May 6, 2014 to February 22, 2016. The registered excess assessment work amounted to close to $6 M, while the future required work to renew the claims was about $251,000. All the claims were active and in good standing at the time of writing this Report. A listing of the active claims and details such as expiry dates, fees and required assessment work are provided in Table 4.1.

The information in this section was extracted from the GESTIM system but full and official details on the claim status are available on the website of the MRN.

NML has not applied for a mining license.

Table 4.1 – Summary of Claims Covering the KéMag Property

Map Sheet (SNRC)	Claim From	Claim To	Area (ha)	Issuance Date	Expiry Date	Excess Work ($)	Required Assessment ($)	Required Renewal Fees ($)
23O/03	50761	50808	2,268.44	20/01/2005	19/01/2015	3,786,061.34	83,680.00	5,230.00
23O/03	2001307	2001320	690.24	23/02/2006	22/02/2016	0.00	25,200.00	1,582.00
23O/03	2066137	2066150	646.48	12/03/2007	11/03/2015	1,042,755.16	17,880.00	1,485.00
23O/03	2082473	2082527	1,313.54	04/05/2007	03/05/2015	822,688.47	36,150.00	3,027.00
23O/03	2092019	0	49.21	13/06/2007	12/06/2015	0.00	1,350.00	113.00
23O/04	2092039	0	49.32	13/06/2007	12/06/2015	0.00	1,350.00	113.00
23O/04	2092042	2092043	98.62	13/06/2007	12/06/2015	0.00	2,700.00	226.00
23O/04	2092045	2092049	246.5	13/06/2007	12/06/2015	0.00	6,750.00	565.00
23O/04	2092051	2092085	1,724.17	13/06/2007	12/06/2015	0.00	47,250.00	3,955.00
23O/04	2092087	2092091	246.1	13/06/2007	12/06/2015	0.00	6,750.00	565.00
23O/04	2092095	2092096	98.42	13/06/2007	12/06/2015	0.00	2,700.00	226.00
23O/03	2095143	2095152	492.32	22/06/2007	21/06/2015	340,265.25	13,500.00	1,130.00
23O/04	2116679	2116681	147.76	10/08/2007	09/08/2015	0.00	4,050.00	339.00
23O/03	2148794	2148797	14.08	07/05/2008	06/05/2014	417.80	1,280.00	112.00
TOTAL		**171**	**8,085.20**	**548,358.00**	**589,990.00**	**5,992,188.02**	**250,590.00**	**18,668.00**

4.3 Mineral Tenure in Quebec

Map designation is now the main mean of acquiring a claim in Quebec, and once the map designation notice is accepted, the office of the Registrar of the MRN issues a certificate for the claim. Within surveyed territory, the outline of a claim is the same as that of a land lot, or part of a lot.

The claims give the owner exclusive rights to explore for any mineral substances in the public domain, with a few exceptions like:

- Hydrocarbons;
- Loose deposits such as sand, gravel and clay;
- Land that is also subject to an exploration or mining right for surface mineral substances.

The claims have a validity of two (2) years and can be renewed indefinitely for two-year periods, provided the required exploration work is completed, subject to certain conditions, and renewal fees are paid. The assessment work requirement and renewal fees for the land north of the 52° N latitude (as at January 2014) are listed in Table 4.2 and Table 4.3.

Table 4.2 – Registration Fee of Map Designated Claim (North of the 52° Latitude)

Area of Claim	Number of Map-Designated Claims	
	1 to 150	**Over 150** (Within a single SNRC sheet, by the same person, on the same day)
Less than 25 ha ($)	28.00	Five (5) times the registration fees
25 to 45 ha ($)	101.00	
45 to 50 ha ($)	113.00	
Over 50 ha ($)	127.00	

Table 4.3 – Claim Renewal Fee (North of the 52° Latitude)

Renewal Application	Area of Claim			
	Less than 25 ha ($)	**25 to 45 ha ($)**	**45 to 50 ha ($)**	**Over 50 ha ($)**
More than 60 Days Before Expiry Date	28.00	101.00	113.00	127.00
From 60 Days Before the Expiry Date to the Expiry Date	Twice the registration fees			

The holder is required to carry out assessment work prior to the 60th day preceding the expiry date of the claim, but special conditions apply for property examination work, technical evaluation studies or work preceding the staking date. If the required work was not performed or was insufficient to cover the renewal of the claim, the claim holder may pay a sum equivalent to the minimum cost of work that should have been performed. In addition, the MRN may impose certain conditions and obligations concerning the work to be performed on claims that lie within the boundaries of a town or on territories identified as State Reserves. The Minister also reserves the right to modify these conditions in the public's interest.

Excess work on one (1) claim may be applied to the renewal of other contiguous claims held by the same owner within a radius of 4.5 km from the centre of the claim from which the credits will be used.

Access to the claims is granted to carry out exploration work. However, the claim holder cannot enter land granted for non-mining purposes or land leased for mining surface mineral substances without permission from the current holder of these rights.

A claim holder cannot erect or maintain a construction on lands in the public domain without obtaining the permission of the MRN, unless such a construction is specifically allowed for by ministerial order or consists of temporary shelters that can be easily dismantled and transported.

The information in this Section is only a summary description of the mining rights and the reader seeking full and official descriptions on titles or rights and obligations of the claim holders should refer to the website of the MRN.

Table 4.4 – Minimum Cost of Work to be Carried out on a Claim (North of the 52° Latitude)

Year	Area of claim		
	Less than 25 ha ($)	25 to 45 ha ($)	Over 45 ha ($)
1	48	120	135
2	160	400	450
3	320	800	900
4	480	1,200	1,350
5	640	1,600	1,800
6	750	1,800	1,800
7 and over	1,000	2,500	2,500

4.4 Underlying Agreements and Royalties

On March 6, 2011, NML announced that it had signed a binding Heads of Agreement ("**HoA**") with TS to develop the LabMag and KéMag iron deposits, known collectively as the Taconite Project.

Under the binding HoA, TS shall participate in the development of a Feasibility Study of the Taconite Project and contribute towards 64 % of the costs related thereto. The parties would enter into a binding joint venture agreement upon the successful completion of the Feasibility Study and TS electing to develop one (1) or both of the deposits. After formation of the joint venture, NML is expected to hold a 36 % equity interest in the Taconite Project, including a 20 % free carry equity interest. In addition, should TS exercise its right to invite third-party investors into the Project. NML will have the right of first refusal to acquire an additional four (4) % of paid equity, thereby increasing its ownership in the Project to a maximum of 40 %.

Upon conclusion of the Feasibility Study, TS will have a maximum of four (4) months to make a positive investment decision that could involve the development of either one (1) or both of the deposits. NML will transfer such deposit (s) along with the Property and other related rights to such deposit (s) to a Joint Venture Enterprise ("**JVE**") as defined below. If TS elects to develop only one (1) of the two (2) deposits, NML will retain the Property and related rights in respect of the remaining deposit.

The binding HoA further provides that following a positive investment decision:

- TS will reimburse NML 64 % of the estimated $30 million in expenses that were incurred by NML on the Taconite Project up to the execution of the binding HoA. The $600,000 facilitation fee that TS has paid to NML in exchange for the Taconite Project exclusivity extension from December 31, 2010 to February 28, 2011, will be credited to the payment.
- TS and NML will form a JVE to hold the Taconite Project, where TS and NML would hold shares in the ratio of 80 % and 20 % respectively, the latter being the free carry interest of NML.
- TS will arrange the required equity portion of the financing (excluding NML's optional equity interest) based on a maximum capital expenditure of up to $4.85 billion if both deposits are developed and up to $4.68 billion and up to $3.76 billion respectively, if only the KéMag or LabMag deposits are developed.
- Within 60 days of TS's positive investment decision, NML would also have an option to acquire up to an additional 16 % paid equity, thereby bringing its total equity in the JVE from 20 % to up to 36 %. This additional 16 % equity shall obligate NML to contribute proportionate equity funding to the JVE.
- Arranging debt financing for the Project shall be the responsibility of TS.

The parties have an offtake right on the production in proportion to their ownership interest in the JVE.

4.5 Surface and Access Rights

The proposed mine, concentrator, tailings containment area, waste dump, camp and associated infrastructure at the KéMag Deposit will be located on Crown Land that NML will acquire from the Government of Quebec ("**GoQ**").

The slurry transportation system between the concentrator and the pellet plant will, for most of its length, be located on Crown Land and NML will acquire that land from GoQ. At certain points, the slurry transportation system will cross below the railway owned by *Chemin de Fer Cartier*. Such crossings will conform to the "Standards Respecting Pipeline Crossings under Railways" published by The Railways Association of Canada, and relevant sections of the Canada Transportation Act will apply.

The currently planned power transmission lines between Brisay hydro-electric power station and the Property site, and between Hart-Jaune hydro-electric power station and the slurry transport booster pumping station lie on Crown Land. NML will obtain the necessary permission from GoQ and Hydro-Québec for the construction and installation of the lines or cables, after which the lines or cables will be handed over to Hydro-Québec for operation and maintenance.

4.6 Permitting

Permits will be required for additional activities that may be completed on the Project, such as exploration or construction work.

Several environmental permits will be necessary, as described under Sections 5.5 and 20.1.2 of this Report. A mining license will be required before the Deposit is placed under production.

4.7 Factors that May Affect Mineral Titles

Met-Chem is not aware of any royalties or back-in right agreement to which the Property is subject.

No other significant factors or risks that may affect access, title or the right or ability to perform work on the Property are known to Met-Chem.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

The Property is accessible by well-maintained roads for 25 km northwest of Schefferville, past former open pit mines, and for a further 30 km westward by 4x4 pick-up truck or all-terrain vehicle over a trail that reaches *Lac de la Frontière*, seven (7) km away from the Property. A camp was built at the end of this trail for the 2007 drill program. Access to different parts of the Property is provided by roads that were prepared to move the drilling equipment between the drill sites. No roads connect Schefferville to the populous south, but access is either by airplane or by train.

5.2 Climate and Vegetation

The Project area is under the influence of humid, sub-arctic continental taiga climate conditions experiencing very severe winters and cool summers. Daily average temperatures exceed 0 °C for only five (5) months of the year. Daily mean temperatures for Schefferville average -24.1 °C and -22.6 °C in January and February, respectively, and +12.4 °C and +11.2 °C in July and August respectively. Additional data on the weather in Schefferville are presented in Table 5.1.

Table 5.1 – Schefferville – Historical Weather Data

Month	Daily Temperature (°C)		Total Rainfall (mm)	Total Snowfall (cm)	Average Snow Days
	Maximum	Minimum			
January	5.1	-48.3	0.26	53.72	17
February	5.1	-50.6	0.29	33.26	14
March	9.4	-45.0	1.40	54.65	17
April	13.1	-36.1	9.04	50.49	14
May	28.3	-23.3	26.12	22.38	11
June	34.3	-7.8	69.53	5.76	4
July	31.7	0.0	96.06	0.15	0
August	28.7	-3.3	81.94	0.38	0
September	26.7	-9.4	102.99	11.05	6
October	20.6	-19.4	24.46	50.78	18
November	9.8	-35.6	4.51	62.75	21
December	5.0	-47.2	0.73	53.04	19
TOTAL			**417.33**	**398.41**	**141**

Source: Environment Canada, Weather Station at Schefferville Airport (Data: 1948-1993)

The Project area is located in the Sporadic Discontinuous Zone of permafrost. However, packer tests conducted by ISLLP on the LabMag Property showed relatively low porosity which implies little effect can be expected on rock stability or water influx.

The KéMag Project is located in the broad transitional zone between boreal forest and treeless tundra. Lichen woodlands form a mosaic with treeless ridges, forested valleys and meadows. Sheltered valleys are sparsely occupied by black spruce trees, with some shrubs on exposed ridges and a full ground cover of lichen.

5.3 Local Resources and Infrastructures

Schefferville, an incorporated municipality in the Province of Quebec, is the closest town to the KéMag Property. Schefferville survived despite the closing of the IOCC iron mines in 1982 and the subsequent demolition of a number of houses and public buildings. In 2011, Schefferville had a population of 213 inhabitants (Canada census). However, the town has experienced an influx of workers since 2011 principally triggered by the re-start of iron production in the region. In addition, the 2011 census indicated that some 540 members of the Nation Innu Matimekosh-Lake John lived in the nearby Matimekosh community.

The economy of Schefferville is based on tourism, recreational hunting and fishing and public service administration. The town also provides services such as hotels and restaurants, basic supplies and equipment, contractors and charter flight operators. Although part of the labor force required for a mining operation at KéMag could be found locally, a significant portion would come from other parts of Eastern Canada and training programs will certainly be required.

The region is served by a modern airport that has a 1,525 metres runway and offers scheduled flights to Wabush, Sept-Îles, Quebec City and Montreal, as well as to many destinations in eastern North America or to other Northern communities. Several airlines offer charter service in and out of Schefferville Airport.

Rail service is provided by Quebec North Shore & Labrador Railway ("**QNS&L**") between Sept-Îles and Emeril, and by Tshiuetin Rail Transportation Inc. ("**TSH**") between Emeril and Schefferville. Trains operate twice weekly and accommodate passengers and freight, including large vehicles, gasoline, fuel oil and refrigerated goods.

Schefferville is supplied by a power line from the hydro-electric generating station at Menihek Lake, about 40 km to the south of town.

Kawawachikamach, a community located some 20 km north of the town of Schefferville, is the home of the Naskapi First Nation of Canada. The community was established following the signing in 1978 of the Northeastern Quebec Agreement between the Government of Quebec and the Naskapi Band of Quebec. In 2011, Statistics Canada indicated a population of 586 living in a modern community that has its own school, medical clinic, recreational complex and swimming pool. Kawawachikamach is linked to Schefferville by a gravel-surfaced all-season road.

Potable water supply is abundant and will require minimal treatment. Process water requirements are expected to be large but can be secured year-round by creating fresh water ponds from natural depressions. In addition, some water may be sourced from lakes located adjacent to the KéMag Project.

The Property is relatively large, as compared to area covered by the proposed mine, and it appears that it could accommodate the proposed mine and associated infrastructure.

5.4 Physiography

The Property has an average elevation of 535 m above sea level. It slopes gently from southwest to northeast, away from the height of land representing the Quebec-Labrador border and towards *Lac Harris* and *Lac Gillespie*, more or less parallel to the dip of the rocks. Terrain on the Property is gently rolling to flat, with total relief of 100 m. The streams to the east and west of the height of land in Quebec flow into the Kaniapiskau watershed and then northward into Ungava Bay. *Lac Harris* and *Lac Gillespie* overlie parts of the Deposit.

5.5 Environment and Community Relations

The Project is expected to trigger several regimes of environmental assessment. Federal and provincial regulatory instruments will apply, in addition to the James Bay and Quebec Agreements. After the environmental assessment, applications for the permits required for site preparation and construction, and finally the start of operations must be filed.

The principal permits that will potentially be required from the Government of Canada fall under fisheries (draining water bodies affecting fish habitat), mining effluents, explosives, dangerous goods or radio communication regulations. From the Government of Quebec, permits related to environmental, water or wildlife management, as well as the Mining Act will be required. The Kativik Regional Government, the MRC of Caniapiscau and of *Sept-Rivières* will also require some permitting. NML/TS have also proposed to enter into agreements with the Aboriginal groups that may be affected by the Project.

Essentially complete bio-physical baseline data were collected in 2011 and 2012. The Project description was submitted to the appropriate agencies in March 2013. Collection of socio-economic data will begin after it has been filed and its public consultation program has begun. NML/TS have started the drafting of the Environmental Impact Statement ("**EIS**"). Analyses of the metal leaching and acid rock drainage potential of rocks from the deposit are being conducted. Examination of mineralized samples was done in order to determine whether it contained asbestiform fibers. NML/TS have signed agreements about preferential hiring and contracting among the members of the Aboriginal groups, scholarship program and summer employment of Native students.

Details on environmental matters for the Project are provided under Section 20.0, of the present Report.

6.0 HISTORY

6.1 Prior Ownership

All recorded exploration work prior to staking of the Property by NML in 2004 had been carried out by IOCC. In 1972, IOCC acquired an exploration permit covering the KéMag area but conducted no further work. The KéMag Property was acquired by NML by claim staking between 2004 and 2008.

6.2 Historical Exploration and Development

A summary of recorded exploration and development work on the Property is presented in Table 6.1.

A few figures for the number of drill holes and total meterages in Table 6.1 and in the following Section slightly differ from the figures recorded in the database available to Met-Chem and used for the resource estimation.

Table 6.1 – Summary of Exploration Work

Company	Year	Work Performed
IOCC	1949-50	• Regional aeromagnetic survey (covered the KéMag and LabMag Properties);
	1950	• Field mapping, sampling;
	1958	• Dip-needle magnetic survey (19.5 km^2); • Drilling of 23 holes;
	1968	• Remnant magnetism study of the iron formations within a 64 km radius of Schefferville;
	1971	• Airborne electromagnetic and magnetic survey (518 km^2 in Howells River region);
NML	2005	• Staking of the claims covering the KéMag Deposit (staked in 2004, issued in 2005);
		• Reconnaissance mapping and sampling;
	2006	• Claim staking; • Drilling: 29 holes for 3,585.6 m; • Metallurgical testing;
	2007	• Geostat Systems International: Resources Estimation (for First Public Disclosure); March 20, 2007; • Watts, Griffis and McOuat Limited ("**WGM**"): Technical Report (Certification of Geostat Resource Estimation); September 19, 2007; • Claim staking; • Drilling: 46 holes for a total of 4,979.2 m; • Metallurgical testing;

Company	Year	Work Performed
NML	2008	• Geostat: Update of the Resource Model (May 2008); • Geostat: Update of the Resource Model (September 18, 2008); • Drilling: 15 holes totaling of 2,216.1 m; • Claim staking;
	2009	• BBA; Prefeasibility Study for the KéMag Project (January 2009; NI 43-101 Technical Report on the Pre-Feasibility Study (March 2, 2009);
	2010	• Airborne magnetic and gravity survey;
	2011	• Drilling for metallurgical samples (PQ Core);
	2012	• Metallurgical testing.

6.3 Historical Mineral Resources Estimate

6.3.1 Mineral Resource Estimate, Geostat (2007)

Geostat was retained by NML to carry out a mineral resource estimate of the KéMag deposit. Geostat issued a technical report dated March 20, 2007 to support NML's first public disclosure of Mineral Resources on the Property.

Considering the similarities between the LabMag and KéMag Deposits, a cut-off grade of 18 % Davis Tube Weight Recovery ("**DTWR**") was used based on a Pre-feasibility Study by NML for the LabMag deposit. WGM reviewed the Study and found the costs and assumptions to be reasonable and to support the 18 % DTWR cut-off grade.

Data from 29 drill holes for a total of 3,585.6 m completed in 2006 were used by Geostat for the Resource Estimate. Each seam was modeled separately, using the lithological information from the drill logs to generate the contacts between the different units.

Since most drill sections only contained one drill hole, Geostat extrapolated the seam contacts at an angle of 6° northeast along the dip, based on its similarities with the lithology occurring at the more densely drilled LabMag deposit to the southeast.

A series of density measurements completed at the Midland Research Center ("**MRC**"), Nashwauk, Minnesota, USA, on 43 drill core samples were used to derive an average density for each seam.

Geostat used 3-D block modeling and the Inverse Distance method to interpolate grades within each seam independently (multi-seam model), based on the similarities between the iron formation at KéMag and LabMag.

The Mineral Resources for the KéMag Deposit were defined according to NI 43-101 and the CIM Standards. However, since the drill hole layout forms a line with holes every 500 m, rather than a grid, most of the resources should have been classified as Inferred. However, based on the knowledge of the consistency of grade and geometry of the LabMag deposit, Geostat considered that where a drill hole intersects the iron formation, a classification of Indicated was warranted, provided that a spacing of no more than 500 m separated the drill holes. Yet, Geostat elected to not classify any resources as Measured.

WGM reviewed Geostat's technical report and was satisfied that the work had been conducted professionally and to industry standards.

The resources on a per seam basis are contained in the reports from Geostat (March 20, 2007) and WGM (September 19, 2007), but the figures for the global resources are presented in Table 6.2.

Table 6.2 – Global Mineral Resources, 2007 (Cut-off Grade of 18 % DTWR)

Category	Tonnage (Mt)	DTWR (%)	TotFe (%)	Concentrate	
				Fe (%)	SiO_2 (%)
Indicated	1,349	27.13	30.85	69.09	2.92
Inferred	992	26.91	30.85	68.98	2.97

Although the resources are 43-101 compliant and were validated by WGM, they are mentioned in this Report for their historical interest only but should not be relied upon, since they are no longer current and are superseded by the present Mineral Resource estimate by Met-Chem.

6.3.2 Mineral Resource Estimate, Geostat (May and September 2008 Updates)

The 2008 Mineral Resource estimate of the KéMag Deposit constitutes an update of the estimation done by Geostat in 2007, adding the drill holes completed in 2007 and 2008. Geostat used 88 holes totaling 10,774 m for this Mineral Resource update. The database available to Met-Chem included an additional hole in the 2006 and 2007 drill programs and slightly different total meterages.

The Mineral Resources were estimated using a block model, a cut-off of 18 % DTWR and grade interpolation by the Inverse Distance method. The same average density per seam as in the previous calculations was used, since no additional measurements were available.

A first update of the resources was completed by Geostat and documented in a report dated May 2008 (Table 6.3).

Table 6.3 – Mineral Resources as in May 2008 Report (Cut-off Grade of 18 % DTWR)

Category	Tonnage (Mt)	DTWR (%)	TotFe (%)	Concentrate	
				Fe (%)	SiO_2 (%)
Measured	991	27.41	31.00	69.01	3.02
Indicated	1,323	26.09	31.48	69.80	2.48
Measured + Indicated	**2,314**	**26.65**	**31.27**	**69.46**	**2.71**
Inferred	1,034	26.99	31.35	69.30	2.82

A second update of the resources was completed by Geostat and documented in a report dated September 2008 (Table 6.4).

Table 6.4 – Mineral Resources as in September 2008 Report (Cut-off Grade of 18 % DTWR)

Category	Tonnage (Mt)	DTWR (%)	TotFe (%)	Concentrate	
				Fe (%)	SiO_2 (%)
Measured	1,538	26.26	31.20	69.27	2.71
Indicated	911	26.45	31.38	69.59	2.60
Measured + Indicated	**2,448**	**26.33**	**31.27**	**69.39**	**2.67**
Inferred	1,014	26.73	31.15	69.17	2.81

The reader should not rely on the 2008 Resource Estimates that are mentioned in this Report for their historical interest only. The 2008 Mineral Resource estimates are no longer current and are superseded by the present Mineral Resource estimate by Met-Chem, relying on additional information that was not available to Geostat. Indeed, in 2011, Met-Chem was retained by NML to review the Resource Estimate completed by Geostat in 2008.

Although no additional drilling had been performed on the Property since this estimate, Met-Chem performed a new resource estimation, as requested by NML in 2012. The new calculations took into account some inconsistencies noted in the drill hole database, the geological interpretation, as well as the secondary porosity and the head iron content (TotFe) affecting the density of the Geostat's resources. The new Resource Estimate supersedes all the previous estimates and is discussed in this Report under Section 14.0.

6.4 Historical Drilling

Historical drilling on the Property consisted of 23 shallow holes testing targets defines by the dip-needle magnetic survey completed by IOCC in the winter of 1958. Sixteen (16) holes were drilled on Harris, Gillespie and Jacques Lakes for a total of 246 m (807 ft) but only three (3) holes intersected unleached UIF. Those samples were not analyzed and these historical holes were not used in any of the Mineral Resource estimations.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The LabMag Property is located in the western margin of the Labrador Trough, adjacent to Archean basement gneisses. The Labrador Trough, also known as the Labrador-Quebec Fold Belt, extends for more than 1,000 km along the eastern margin of the Superior Craton, from Ungava Bay to Lake Pletipi in Quebec. The belt is about 100 km wide in its central part and narrows considerably to the north and south. The Grenville Front crosses the southern part of the Labrador Trough.

The rocks in the Labrador Trough are subdivided into a lower sedimentary sequence known as the Knob Lake Group and an upper mafic volcanic-dominated succession known as the Doublet Group. These rocks are collectively referred to as the Kaniapiskau Supergroup (Frarey and Duffell, 1964; Wardle, 1981). The Knob Lake Group is subdivided, from oldest to youngest, into the Seward, *Lac Le Fer*, Denault, Fleming, Dolly, Wishart, Sokoman and Menihek Formations. The iron formations found in the region are within sub-units of the Sokoman Formation.

Metamorphic grade increases from sub-greenschist facies in the west to upper amphibolite facies in the eastern part of the Labrador Trough.

7.2 Property Geology

The Property is overlain for the most part by deep overburden which is locally boggy. However, the extension of the stratigraphic sequence on the LabMag property is well established based on:

- Sporadic exposures of the lower Sokoman Formation overlying the continuous outcrops of the lowermost unit of the Knob Lake Group along the south western margin of the deposit;
- Aeromagnetic response;
- Drilling results.

Drilling by NML has shown that units of the Knob Lake Group, including the Sokoman Formation, underlie a major part, if not all, of the Property, and comprise a NNW striking sequence that dips shallowly to the northeast. Drilling by NML on the KéMag Property indicates that the stratigraphic sequence and type descriptions developed for the Howells River area (LabMag deposit) are applicable to the KéMag Property, with only slight variations of the unit thicknesses.

Archean granitic gneiss of the Ashuanipi Complex occupies several rows of claims lying along the southwestern Property boundary (Figure 7.1). A description of the rock types and the stratigraphy of the Property are summarized in Table 7.1.

Figure 7.1 – Property Geology, with Drill hole Location

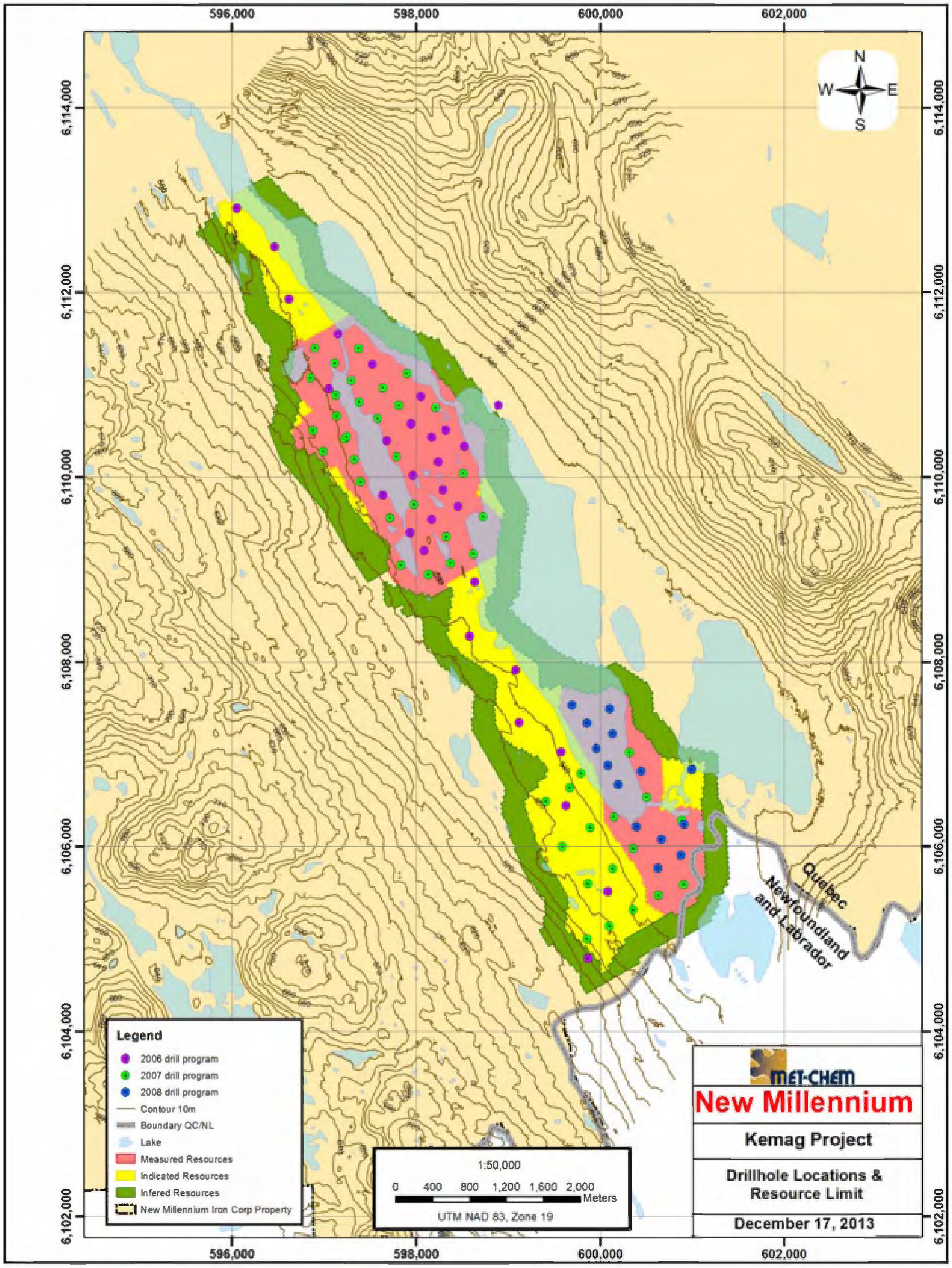


Table 7.1 – Stratigraphy of the Property

Formation	Member	Sub-Member	Facies	Average Thickness and (Range) (m)	Description
PROTEROZOIC					
MENIHEK				Over 79.2	Shale, minor greywacke and carbonate, carbonaceous pyritic shale
Thrust fault					
SOKOMAN	UIF	Lean Chert (LC)	Silicate	25.0 (18.4-32.5)	Magnetite-chert, local shaley (siderite-magnetite) chert IF and stromatolite band;
		Jasper Upper IF (JUIF) **Fe Mineralization**	Magnetite-carbonate	26.2 (20.7-30.8)	Magnetite-chert IF
		Green Chert (GC)	Magnetite-carbonate	3.8 (1.2-9.4)	**Marker Horizon;** green chert
	MIF	Upper Red Cherty (URC)	Hematite-carbonate	8.1 (4.4-16.8)	Jasper-magnetite-chert
		Pink-Grey Cherty (PGC) **Fe Mineralization**	Magnetite-carbonate	12.6 (4.0-22.9)	Disseminated magnetite-chert IF
		Lower Red Cherty (LRC) **Fe Mineralization**	Hematite-carbonate	8.6 (0-18.6)	Layered magnetite-chert IF; gradational lower contact
	LIF	Lower Red-Green Cherty LRGC	Magnetite-carbonate	21.2 (0-46)	Silicate-magnetite-carbonate, magnetite-chert IF gradational lower contact
		Lower IF LIF	Silicate	8.2 (1.4-32.8)	Silicate-carbonate-magnetite-chert
	RUTH		Sulphide	5.2 (2.9-8.7)	Laminated chert-siderite-shale
WISHART				17.7 (14.6-20.4)	Black chert quartzite
Unconformity					
ARCHEAN					
ASHUANIPI COMPLEX		Granitic dykes and mafic intrusives			

The Sokoman Formation has been broken down into members and individual stratigraphic units (sub-members) and all the geological work on the Property has used this classification. The iron formation sub-members within the Sokoman Formation have been classified on the basis of facies [sulphide, silicate, magnetite-carbonate and hematite-carbonate (Table 7.1)]. Most of the contacts between the sub-members of the Sokoman Formation are gradational, except for the Green Chert that has two (2) sharp and easily discernible contacts, making it an excellent marker horizon.

The Sokoman Formation overlies the Wishart Formation, which is a fine to medium grained quartzose sandstone with varying amounts of feldspar grains. In turn, the Sokoman Formation is overlain by the Menihek Formation and is in probable fault thrust contact that generally manifests itself as an intensely deformed and brecciated zone. The shale of the Menihek Formation is exposed along the northeastern margin of the Property. The granite of the Ashuanipi Complex occurs within the southwestern side of the Property. The KéMag taconite Deposit is part of the Sokoman Iron Formation, is approximately 120 m thick and all the sub-member units show variation in thickness as observed from drilling.

The Sokoman Formation in the Property area has undergone very low-grade metamorphism. Furthermore, it has been subjected to minimal post-depositional leaching or weathering. The deposit is narrower and shallower in the central part than in the southern and northern parts, indicating the presence of two (2) shallow basins separated by a plateau.

7.3 Structure

Drilling results indicate that the Wishart and Sokoman Formations on the KéMag Property dip at about five (5) to seven (7)^{o} to the northeast, although few cross sections in the drilled area contain multiple drill holes. Core angles for bedding appear to be in accordance with this interpretation of shallow dip, which is similar to that delineated on the much more densely drilled LabMag property to the southeast.

7.4 Mineralization

The taconite at KéMag consists mostly of alternating small-scale (mm to cm) beds of recrystallized chert or jasper and massive or disseminated magnetite. Magnetite is the predominant iron oxide mineral, while hematite and martite occur in subordinate amounts. Gangue minerals are represented by iron silicates (minnesotaite and stilpnomelane), iron carbonate (siderite) and manganese carbonates (rhodochrosite and kutnahorite).

The iron formation at KéMag has been explored by diamond drilling over a strike length of 9.5 km and it extends beyond the northwest and southeast Property boundaries.

Alumina, carbon and sulphur are only concentrated at the base of the Sokoman Formation (Ruth Member) that would not be mined. Phosphorus generally occurs at low levels in the oxide iron formation.

8.0 DEPOSIT TYPES

The KéMag Deposit consists of magnetite Banded Iron Formation ("**BIF**") of the Lake Superior type. BIFs are sedimentary rocks composed of alternating mm- to cm-scale beds of quartz (chert or jasper) and iron oxides (predominantly magnetite and hematite). Variable amounts of gangue minerals, mostly silicates, carbonates and sulphides are present. Banded iron formations have greater than 15 % iron content and have been the principal sources of iron throughout the world and host many gold deposits (Gross, 1996).

The BIFs can be generalized and classified in two (2) main types, the Lake Superior and Algoma BIFs, on the basis of tectonic systems and depositional environments (Gross, 1965, 1983, 1986). Oxide, silicate and carbonate lithological facies are common to both BIF types.

The Lake Superior-type BIF formed in passive margins settings, in near-shore continental shelves and platform basins. They are associated with typical shelf-type sedimentary rocks with minimal volcanic input (James, 1954, Gross, 1965). Most Lake Superior-type banded iron formations formed during the Paleoproterozoic (2.5 - 1.8 Ga). The Superior-type BIFs represent a vastly more abundant source of iron than the Algoma type and are a major part of the iron mined in the Great Lakes region of the United States.

Additional information, like the salient characteristics of the Lake Superior-type iron deposit model described by Eckstrand (1984) are presented in Met-Chem's Pre-Feasibility Study report dated June 2006.

The BIFs in the Labrador Trough were variably affected by metamorphism and alteration. The taconite is a lithofacies represented by hard, unoxidized BIF, little affected by metamorphism or alteration. Strongly metamorphosed taconites are known as meta-taconite, as those found in the iron deposits in the Grenville part of the Labrador Trough, in the vicinity of Fermont and Wabush.

The exploration model used to design the exploration activities and the drilling at KéMag is principally based on the interpretation of the drill data indicating the presence of gently dipping, Superior-type Iron Formation.

9.0 EXPLORATION

Historical exploration in the KéMag region has consisted principally of field mapping, sampling, drilling, as well as geophysical surveying including ground magnetic and airborne magnetic and EM surveys.

Resources estimates have been completed following each phase of drilling. Cumulatively, the exploration and development work has allowed estimating NI 43-101 compliant Mineral Resources in the drilled area. Table 6.1 provides a summary of the main exploration and development work that has been carried out on the Property.

10.0 DRILLING

10.1 2006 Drilling Program

The 2006 drilling program was initiated by NML to test airborne anomalies outlined during the 1950s and in 1971, since there are no exposures of iron formation on the Property.

Drilling started on June 9, 2006 and concluded on October 14, 2006 but was suspended from August 14 to September 7. A total of 3,585.6 m was drilled in 29 holes at KéMag (Table 10.1).

Table 10.1 – Summary of Drilling by NML, by Year

Year	Holes	Drilled Length (m)
2006	29	3,585.6
2007	46	4,979.2
2008	15	2,216.1
Total	**90**	**10,780.9**

All of the holes were drilled vertically and core size for most drilling was BTW (42 mm diameter) and BQ (36.4 mm diameter). The holes were spotted using a Global Positioning System ("**GPS**") receiver and surveyed at the end of the program. No downhole directional or geophysical surveys were carried out.

The drilling done in 2006 indicated that the seven (7) economic stratigraphic horizons are similar to those occurring at the LabMag deposit with some minor changes in the individual thicknesses and magnetite content. The taconite beds dip at the same average six (6)° towards the northeast.

10.2 2007 Drilling Program

The 2007 program, during which 46 holes were drilled for a total of 4,979.2 m (Table 10.1), was a follow-up of the 2006 program, with the objective of completing the drilling in the regular adopted pattern of 250 m by 300 m grid. This was accomplished in the northern part of the deposit but the drilling continued in the southern part in the same pattern as used in 2006. Drilling started on July 18 and concluded on October 17, 2007. As in 2006, all the drill holes were surveyed at the end of the program.

Logging and subsequent analytical results of the drill core samples indicated no major changes in stratigraphy or the mineralogical characteristics of the seven (7) economic units. The deposit is narrower and shallower in the central part than in the southern and northern parts.

10.3 2008 Drilling Program

From March 5 to April 30, 2008, drilling continued on the southern part of the KéMag deposit, to confirm that the eastern extension of the deposit lies under *Lac Harris, Lac de la Frontière* and the swampy grounds to the south. Fifteen (15) holes were drilled on lines spaced 250 m apart for a total of 2,216.1 m (Table 10.1). The holes were spotted using a GPS receiver.

The results of the drilling confirmed that the deposit continues beyond the western shores of *Lac de la Frontière*, dipping at angles of six (6)° to eight (8)° towards the northeast, and that the stratigraphy, mineralogy and structure are similar to the other parts of the KéMag Deposit.

11.0 SAMPLE PREPARATION, ANALYSIS AND SECURITY

11.1 Core Logging and Sampling – 2006 to 2008 Drill Programs

The core was photographed and logged at the core storage facility in Schefferville. Rock Quality Designation ("**RQD**"), core recovery and magnetic susceptibility measurements were recorded and the core was split using a hydraulic splitter and sampled for assaying. All logging and sample descriptions were recorded on paper forms for later transfer to Microsoft Excel spreadsheets.

Each stratigraphic unit was sampled separately, except for MS and RF, with sample lengths varying from 1.6 m to a maximum of 9.05 m. LIF was sampled as well, even though it is considered as waste. The sampling intervals within the units were based on the extent of magnetite/hematite mineralization. The lean cherty zones exceeding three (3) m were sampled separately, although individual sample lengths seldom exceeded six (6) m in both the mineralized and waste zones.

The split half (½) core was placed on the original core tray and returned to the core storage building to serve as reference samples.

The same method was used for sampling the core produced during the 2006, 2007 and 2008 drilling programs.

11.2 Sample Preparation and Analysis

All the 1,570 split core samples from the 2006, 2007 and 2008 drilling programs, including 88 check samples, were sent to MRC for chemical and Davis Tube ("**DT**") analyses, as follows:

- Head assay for total iron;
- %DTWR on -325 mesh concentrates;
- Assay of the concentrates for iron and silica.

In addition, 21 samples on three (3) fractions (Crude, DT Concentrate and DT Tails) were analyzed for trace elements and sulphur.

MRC's sample preparation and analytical procedure consisted of the following steps:

- Core samples crushed to $^3/_8$" with a jaw crusher;
- Split 1,500 g for test work, save the balance;
- Roll crush 1,500 g to 100 % passing 10 mesh;
- Split 50 g for DT test and crude ore sample analysis, save the balance;
- Stage grind 50 g to -325 mesh, as per MRC procedure (Hanna Procedure);
- %DTWR test on 25-30 g samples, as per the procedure provided by MRC (Hanna procedure);

- Analyze DT concentrate samples for Fe and SiO_2 (non-mercury titimetric method for total iron; SiO_2 determination using hydrofluoric acid);
- Analyze crude ore sample for Fe, save the balance.

11.3 Quality Assurance and Quality Control Programs

11.3.1 NML's QA/QC Protocols (2006-2008)

In 2006, NML monitored the laboratory performance with a total of 13 samples from selected drill hole intersections re-assayed at MRC. The samples consisted of second half core bearing new drill hole names and sample numbers and submitted blindly to the laboratory.

The 13 check samples of 2006 reported consistent values, with the exception of one (1) sample that was re-assayed and yielded an acceptable second result of the 2006 to 2008 drill programs.

Sixty-five (65) duplicate samples were inserted into the sample stream of the 2007 drill program. In addition, seven (7) samples were collected in Hole 07HL1014A drilled at the same location as 7HL1014D and were used as twin hole check samples. Sixty-four (64) duplicate samples and the seven (7) twin-hole check samples were sent to MRC, while one (1) duplicate sample was sent to an external laboratory, Lerch Brothers Inc. ("**LBI**") of Minnesota.

Met-Chem reviewed the analytical results and found a high correlation between the pairs of duplicate samples. Four (4) samples yielded a significant difference between the original and duplicate TotFe %, ranging from 14.37 to 4.22. However, even if these four (4) samples are included in the calculations, the respective averages and standard deviations were 30.71 vs. 30.84 and 6.08 vs. 5.96. The same applies for the head silica analyses. Correlation of DTWR % in the pairs is very high, with a correlation coefficient of 99.3.

Four (4) duplicate samples were collected by NML in 2008. Met-Chem reviewed these data and found a slight positive bias in the duplicate sample analyses for TotFe. However, no conclusion can be drawn from such a small set of data.

NML's QA/QC system did not provide for insertion of standard and blank materials into the sample stream of the 2006 to 2008 drill programs.

11.3.2 MRC's Internal Laboratory Protocols

MRC had its own internal QA/QC program of random selection of samples for re-assay, as follows:

- Analysis of Standards at the start of procedure to calibrate the instruments;
- Submittal of four (4) % of the samples by management to monitor the analytical accuracy of the work. The samples were selected and submitted according to the following procedure:

1. Randomly pick pulp to be assayed and place in an envelope;
2. Assign a new number;
3. Record old and new numbers in a folder that is not in the laboratory;
4. Submit as blind samples for analysis;
5. Record old and new assays for comparison purposes.

In addition, MRC sent 60 selected samples to be checked by LBI.

In 2006, a total of 20 samples were selected for check assays, ten (10) for head assays and ten (10) for DT concentrates. MRC concluded that all check assays returned values well within normal ranges and that no significant bias was observed.

Of the 60 sample pulps sent in 2006 by MRC to the LBI for external check assay, 29 were assayed for TotFe and 30 were assayed for Fe in concentrate. Geostat reviewed the results of the LBI assays and found that:

- TotFe % did not present a statistical bias;
- Fe % and SiO_2 % in concentrate did present a bias.

However, although a bias seems to exist, it is small, amounting to -0.6 % Fe on an average of 69 % for Fe in concentrate and +10.9 % SiO_2 on an average of three (3) % for SiO_2 % in concentrate. Actually, the bias has no impact on the Mineral Resource estimates because they are based solely on DTWR %. Although the small number of pairs available for comparisons could not lead to a statistically significant conclusion, Geostat recommended increasing the number of check samples and further investigating the potential bias by the laboratories concerned in future sampling campaigns.

Geostat also noted that DTWR was not internally checked at MRC nor was it at LBI. Only the NML blind half-core samples were checked. Since DTWR is a critical component of the Mineral Resource, owing to the fact that it is used as the cut-off grade, Geostat recommended adding DTWR checks to its QA/QC program.

Geostat concluded that the quality of the samples used was sufficient to support the estimation of a Mineral Resource at the Indicated level but that a more extensive QA/QC program will be required to support the estimation of Measured Resources.

12.0 DATA VERIFICATION

12.1 Data Verification by Geostat (2007)

12.1.1 Drill hole Database

Geostat checked the database, compared a limited number of assay values in the database against the original assay certificates, as well as some of the lithological descriptions from the geologist logs against the database contents and found no errors.

Geostat reviewed the results of the QA/QC program and considered that the amount of control samples should be increased, but did not find any issue that could significantly impact on the Mineral Resources estimate.

12.1.2 Site Visit

During a three-day site visit starting on February 20, 2007, Geostat visited the site and NML offices in Labrador City, examined drill core and collected 27 samples from the remaining core halves.

Four (4) hole collars were found in the field where expected using a handheld GPS instrument.

12.1.3 Control Sample Results

Geostat sent the 27 samples to LBI, where the original samples (half core) had been prepared and analyzed. The head and concentrate portion of the check samples were analyzed as follows:

- DTWR %: LBI's results were over MRC's by four (4) %, with a systematic bias. Geostat's rationale was that, since the bias is positive, its impact on the Mineral Resource would be conservative;
- TotFe: LBI's results systematically higher than MRC's by 5.5 %. Since the bias is positive, its impact on the Mineral Resource would be conservative;
- Fe in Concentrate: no observed bias;
- SiO_2 in Concentrate: LBI's results systematically higher than MRC's by 15.7 %. Geostat concluded that, although high concentration of SiO_2 in concentrate adversely affects Mineral Reserves, it had no impact on its estimated Mineral Resources. However, Geostat recommended addressing this bias, since it is systematic and large.

Geostat did not study the source of the observed bias as their primary objective was to confirm the iron mineralization in the control samples.

The origin of the bias in TotFe % yielded by the Geostat's check samples is unclear. Indeed, Met-Chem's check samples (coarse rejects) submitted to XRF analysis showed a high correlation between the original and the check samples. Met-Chem believes the bias observed between the concentrates from Geostat's original and duplicate samples may originate from an inconsistent grind size achieved by LBI at the different times at which the samples were processed.

12.2 Data Verification by WGM (2007)

WGM completed a site visit on July 10-11, 2007. The field offices and several drill sites and outcrop were visited and drill core was reviewed. WGM noted that drill core was in excellent condition, core splitting was well done and corroborated that the observed core corresponded to that described in the drill logs. WGM was not required to complete any validation drill core sampling because independent validation had already been completed by Geostat for its Mineral Resource estimate.

WGM noted that, although drill core descriptions in logs are generally adequate, some description of contact relationships between units would be helpful. Contacts are generally transitional. Some are more definite, being gradational over 0.1 m, while others are transitional over ten (10) m. WGM also noted that some samples include significant core loss.

GPS measurements were taken on four (4) drill hole collars. For one (1) of the hole collar (DDH HL1042), coordinates were considerably different than those recorded in the drill hole database. WGM's preliminary conclusion is that this hole is mis-located by about 100 m. WGM also noted that several of the collar markers had no drill hole identifiers. WGM recommended that all of the collars be re-surveyed.

12.3 Data Verification by BBA (2008)

BBA Senior Metallurgist, John Dinsdale, visited the site in November 2008. It was not possible to access the proposed mine site but an inspection was made of the drill core still in storage in Schefferville. IOCC's abandoned mines in the area, the electrical substation, the northern terminal of the Tshiuetin Railway and other infrastructure and facilities in the town of Schefferville were also visited.

BBA was not required to complete any validation drill core sampling because independent validation had already been completed by Geostat for its Mineral Resource estimate.

12.4 QP Visit by Met-Chem (2012)

12.4.1 Field Visit

Yves A. Buro, Eng., and Schadrac Ibrango, P. Geo., Ph.D., both Senior Geologists, Met-Chem, visited the site between September 18 and 19, 2012. They were accompanied in the field with Mr. Henry Simpson, NML's Senior Geologist and Mr. Rabi Mohanty, TS's Chief Geologist. A helicopter was available for transportation from Schefferville to the field and the visit of a few outcrops and a second trip was taken the same day with a pick-up truck to stop at more outcrops.

A series of outcrops representing the major units of the Sokoman Formation were examined in the field and six (6) readings of hole collars were taken with a hand-held

GPS instrument. The office and core handling (logging and sampling) facilities in Schefferville were visited, as well as the warehouse in Wabush where the core was stored in the original marked boxes stacked on pallets in a closed building. The core from selected holes was examined by Met-Chem, with Rabi Mohanty and Alex Howe (Junior Geologist.)

Met-Chem did not find any major errors, in the core examined, relating to the description and contacts of the lithological units and sample intervals that had been selected while logging and sampling the core.

The GPS measurements of the drill hole collars corresponded well with the entries into the database and the plot on the maps. No issues that may have a significant impact on the reliability of the data collected by NML were observed during the visit.

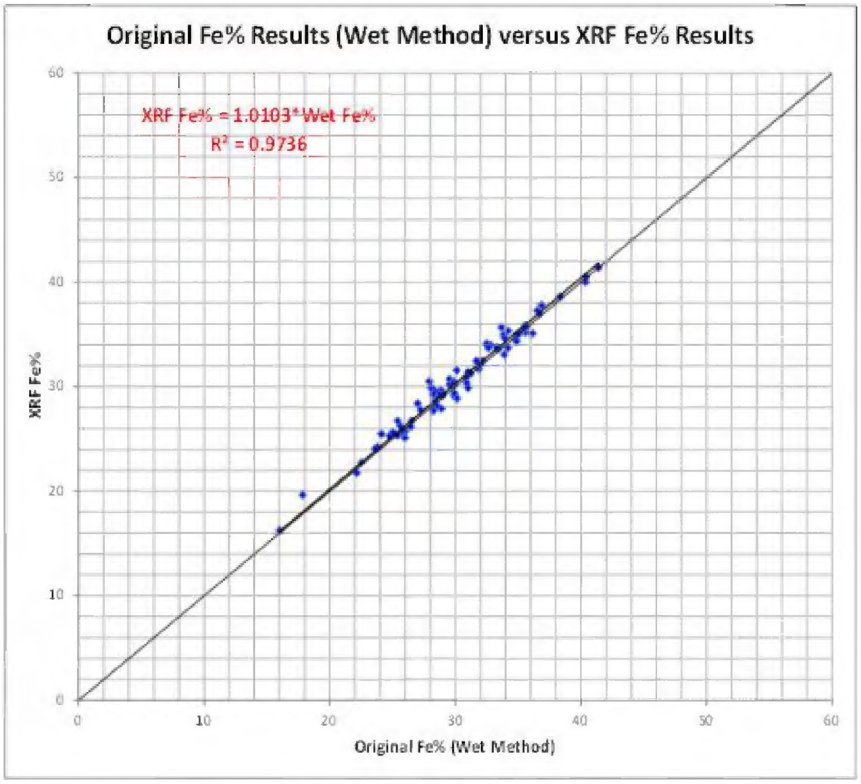
12.4.2 Independent Check Samples

During the QP site visit, Met-Chem independently selected 56 samples from the LabMag deposit and 26 from KéMag for check analysis. The samples represented a fair geographical distribution within the deposit, strike- and depth-wise, and a range of iron values. The samples consisted of coarse rejects from the original samples and were analyzed by SGS Lakefield using the XRF analytical technique. The major oxides, including Fe_2O_3, were analyzed, as well as sulphur by Leco, Loss on Ignition ("**LOI**") and sum of oxides. Four (4) Certified Reference Materials and two (2) duplicate samples were inserted into the sample sequence. Met-Chem prepared the sample bags and tags, as well as the standards and the list of duplicate samples and sent them to LabMag who sent the samples directly to SGS.

The analytical results from the duplicate samples showed a high correlation between the original and the check samples. The maximum differences of Fe % between the pairs of samples was -1.24 % and 2.58 %, with an average of 30.44 % Fe for the original samples as compared to 30.78 % Fe for the check samples. A slight positive bias is visible in the XRF analyses, which can be expected since the XRF technique using fused discs allows full dissolution of the total iron, including the iron locked in silicates, as opposed to the soluble iron determinations on the original samples. The square of the correlation coefficient between the two (2) populations was calculated as 0.9736, which is high (Figure 12.1). Using the relative difference to describe the differences between pairs of duplicate analyses shows a maximum of 9.73 %, but with 75 pairs below five (5) %, which is excellent.

In the absence of appropriate Standard Reference Material, Met-Chem used two (2) available standards, but they did not perform well owing to the presence of sulphides in them, which prevented accurate XRF analysis. No blanks were included in the check samples.

Figure 12.1 – Scatter Diagram Showing the Correlation Between the Original and the Met-Chem's Check Samples



The two (2) pairs of duplicate samples inserted by Met-Chem did not perform very well, exhibiting differences of -2.37 % and -1.40 % Fe in the duplicate versus the first analysis. Although the differences are high, no conclusion can be drawn from only two (2) sets of data. Silica determination was included in the XRF analyses but was not part of the wet chemistry method applied to the original samples.

12.4.3 Bulk Density

The previous resource estimates were completed using the density determined by the pycnometer method, without correction for the effects of porosity and permeability. At Met-Chem's recommendation, NML had the technicians from a sample preparation laboratory in Chibougamau, the *Table Jamésienne de Concertation Minière*, to perform bulk density determination on 167 samples from the KéMag Deposit. The work was completed in 2012 at the NML's warehouse in Schefferville. The results were used to build a regression function for each seam in the LabMag deposit. This matter is discussed in Met-Chem's technical note on the database validation and Resource Estimate and in the Section 14.0 – Mineral Resources Estimates of this Report.

12.5 Geological Review and Audit by Met-Chem

In 2012, Met-Chem was requested to provide an audit of the resources by Geostat and to carry out a new resource estimate of the LabMag and KéMag Deposits. During the course of the work, some issues were investigated that led to a few corrections or changes to the Geostat's resource models.

The primary concerns raised by Met-Chem dealt with the validation of some entries into the drill hole database, weaknesses in the geological interpretation on some sections, the mesh size at which the samples were pulverized before being analyzed and the conversion of DT results at different mesh sizes, the use of Specific Gravity versus In Situ density to convert volumes to tonnes, the sample compositing and the spatial continuity of the mineralization.

These issues and the actions taken to address them are discussed in Section 14.0 of this Report, with the revised resource estimate by Met-Chem.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

This Section addresses itself to the mineral processing and metallurgical test work that has been carried out for both the LabMag and the KéMag taconite deposits that comprise the New Millennium Taconite Project, located in the Labrador Trough. These taconite deposits contain several billion tonnes of taconite iron ore with the two (2) main taconite projects, LabMag and KéMag, situated about 25 km apart in the same valley, LabMag in Newfoundland - Labrador and KéMag in Quebec. The ore characteristics and beneficiation properties in each deposit, though not identical, are similar enough to allow NML to use identical flow sheets and process equipment for each orebody. Blending of the mined ore will ensure a constant quality of feed to the concentration plant. NML pilot plant work started with a pre-feasibility for LabMag in 2006 and continued with a pre-feasibility for KéMag in 2009.

The test work carried out by NML since the start of the initial pilot plant tests in 2005, sought to characterize the ore and establish the process flow sheet and design criteria that would allow development of an efficient and economical process. The process has evolved through various stages of development to prepare an optimized feed for the production of quality pellets with a low silica grade for both blast furnace ("**BF**") and direct reduced iron ("**DRI**") based steelmakers.

13.2 Process Development Stages

This Section of the Report outlines the evolution of metallurgical test work programs undertaken to date and the concurrent process development which has established the process flow sheet and design data for both LabMag and KéMag ores.

Originally, two (2) separate pre-feasibility studies were prepared. One (1) was published in 2006 for the LabMag deposit in Labrador, and the other in 2009 for the KéMag deposit in Québec.

The LabMag pre-feasibility study was based on the results from the first pilot plant run using LabMag Block A bulk sample and was based on a conventional taconite flow sheet consisting of a SAG mill, ball mills and magnetic separators, but also included an option to use HPGRs in place of SAG mills to reduce energy consumption and costs.

The subsequent KéMag pre-feasibility study was based on a second series of pilot plant test results. This time, the SAG mill was replaced by HPGRs in the pilot plant and the flow sheet was developed using a LabMag Block B bulk sample. A confirmation test was run later with the same circuit using a KéMag bulk sample.

The forecasted product portfolio would typically be comprised of high quality direct reduced ("**DR**") grade pellets, BF grade pellets and a certain tonnage of pellet feed concentrate.

The initial goal of the test work was to characterize the ore and determine if it could be concentrated to reach the desired levels of silica for DR applications without grinding to excessive fineness (Blaine value of 2,200 or more). Excessive fineness was a constraint because it was determined during pelletizing tests, that a too fine concentrate poses challenges in achieving the required quality of the pellets and might reduce the quantity of pellets produced. It quickly became clear that the target grades could not be reached at Blaine value below 2,000 without a flotation circuit.

The circuit was modified to limit the final concentrate fineness to 1,800-2,000 Blaine value in order to meet the requirements of a high-capacity induration machine. While the lower Blaine value will increase the silica grade of the final concentrate, the flotation circuit will be designed to reduce the silica to ≤1.5 %. An organic binder will be used instead of bentonite, if necessary, to ensure that the pellets are capable of meeting the requirements of DRI producers.

The 2014 preliminary concentrator design for the Feasibility Study was developed as a single stage HPGR/ball mill concentrator flow sheet based on a tested and proven design to process both LabMag and KéMag ores. The deposits have been shown to be very similar in composition with minor differences in the liberation characteristics of silica. The following Sections provide detail on the ore characterization and pilot plant tests to support the process design.

13.3 Historical Ore Characterization Test Work

In the early 1960's, sampling and test work, including DT tests and liberation studies, were primarily conducted by IOCC's Ore Testing and Research Laboratory ("**OT&R**") in Schefferville. Surface samples, drill core and bulk samples of Howells River taconite were also sent to Hanna Mining Company's ("**HMC**") research laboratory in Hibbing, Minnesota for grindability testing.

The DT test is a standard test procedure to determine the recoverable magnetite content in a sample. Liberation studies were conducted at various grinds to determine the size required to produce a concentrate in the 68 to 70 % Fe range.

In 1971, magnetic separation tests were conducted using a Stearns magnetic drum to more closely simulate commercial plant results. Results showed that concentrate grades with 3.5 % silica with a high percentage of TotFe were produced.

The tests demonstrated that a high quality concentrate can be produced from the Howells River taconite but more detailed analyses would be required for the total grinding power requirement or the split between the primary and secondary grinding in a commercial plant.

13.4 Ore Characterization

The liberation index of a taconite ore is a measure of the silica grade in the DT concentrate for a given grind fineness. While the mineralogy of the LabMag and KéMag orebodies is very similar, the liberation indices are slightly different. NML began DT testing of the drill cores and liberation grinding tests on selected samples in order to characterize the orebody and then correlate the results with the performance of a pilot plant.

As described in the following Sections, this knowledge will be applied to blend the plant feed in order to meet the market's pellet grade requirements.

13.5 Liberation Grinding Tests

Liberation grinding tests are common laboratory methods for determining the extent of grinding required to reach a desired product grade. Because the controlled conditions are repeatable and very reliable, multiple liberation grinding tests were used to characterize its ore. The test procedure is summarized as follows.

Ore is placed in a small rod mill and ground for fixed time intervals. After each interval, the ore is sampled, DT tested, and the products are assayed. The laboratory sized mill is typically benchmarked and the run time correlates directly with power input. This information can be used for grinding mill power requirement predictions and assayed samples demonstrate the liberation behavior of the ore.

MRC in Minnesota carried out tests on a selected number of drill holes in both deposits (LabMag and KéMag) to characterize their liberation. Each grinding campaign contained all seven (7) geologic strata. Grinding was carried out at 10, 14, 18, and 22 minutes and the products were DT tested to determine the magnetic iron recovery and silica content.

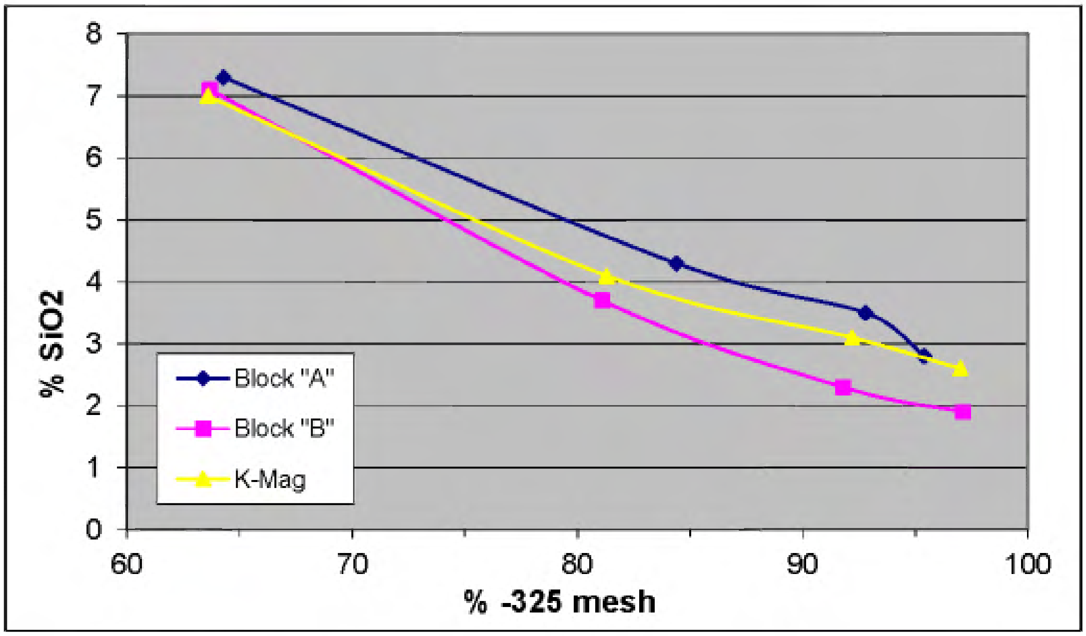
The benchmarked power consumption was 8.3, 11.5, 14.8, and 18 kWh/t respectively. This data was plotted on a curve of % SiO_2 vs. % passing 325 mesh. The three percent (3.0 %) SiO_2 target could be observed at the percent passing 325 mesh indicating the power requirement to sufficiently liberate the silica grains.

It was observed that the LabMag Block A zone is the most difficult to liberate with 95 % passing 325 mesh while LabMag Block B is the easiest at 86 %. The liberation of the KéMag deposit lies between LabMag Blocks A and B at around 93 % but is closer to LabMag Block A, which confirms that they possess very similar liberation characteristics. The results are shown in Table 13.1 and Figure 13.1 below.

Table 13.1 – Liberation Grinding Data

Deposit	% -325 mesh				% Silica			
	10 min	14 min	18 min	22 min	10 min	14 min	18 min	22 min
LabMag Block A	64.3	84.4	92.8	95.4	7.3	4.3	3.5	2.8
LabMag Block B	63.7	81.1	91.8	97.1	7.1	3.7	2.3	1.9
KéMag	63.6	81.3	92.2	97.0	7.0	4.1	3.1	2.6

Figure 13.1 – MRC Liberation Summary



It should be noted that while liberation grinding tests were based on fewer samples than the drill core DT characterization test representing the entire deposit, the results were nonetheless quite similar. It should also be noted that liberation grinding tests only show a trend.

Additional tests were carried out using larger bulk samples with a blended feed. The procedure was further refined to correlate the DT silica of the feed with the actual grade achieved for a given fineness in an operating plant as described in the following sections.

13.6 Pilot Plant Test Work Programs

Over the years, NML has performed many tests on the various ores from the LabMag and KéMag deposits. The results of these tests led to changes in the adopted processing flow sheet. The flow sheet evolved from SAG/Ball Mill to HPGR/Ball Mill and the main driver for this was the lower operating costs due to reduced power requirements with HPGR and the reduced grinding media consumption.

Samples representing the two (2) orebodies were subjected to a series of metallurgical tests to determine various ore characteristics and to test the flow sheets.

13.6.1 SAG/Ball Mill Testing on LabMag Block A Zone (MRC 2005)

Davis Tube work and iron grade determinations on drill core, along with liberation studies, have shown that the LabMag deposit is a typical iron taconite similar to those now being exploited on the Mesabi Range in Minnesota, USA. As such, a wealth of knowledge is available and the flow sheet initially proposed for the upgrading of the LabMag orebody reflects that experience.

LabMag Block A samples totaling 237 tonnes from each of the seven (7) units were collected in 2005. Material was blasted and collected from equal-sized trenches in each stratigraphic unit whose locations were selected for easy access. Three (3) trenches from each unit were chosen to improve the representativeness. The samples were then crushed and blended based on the proportion of individual lithologies in the orebody.

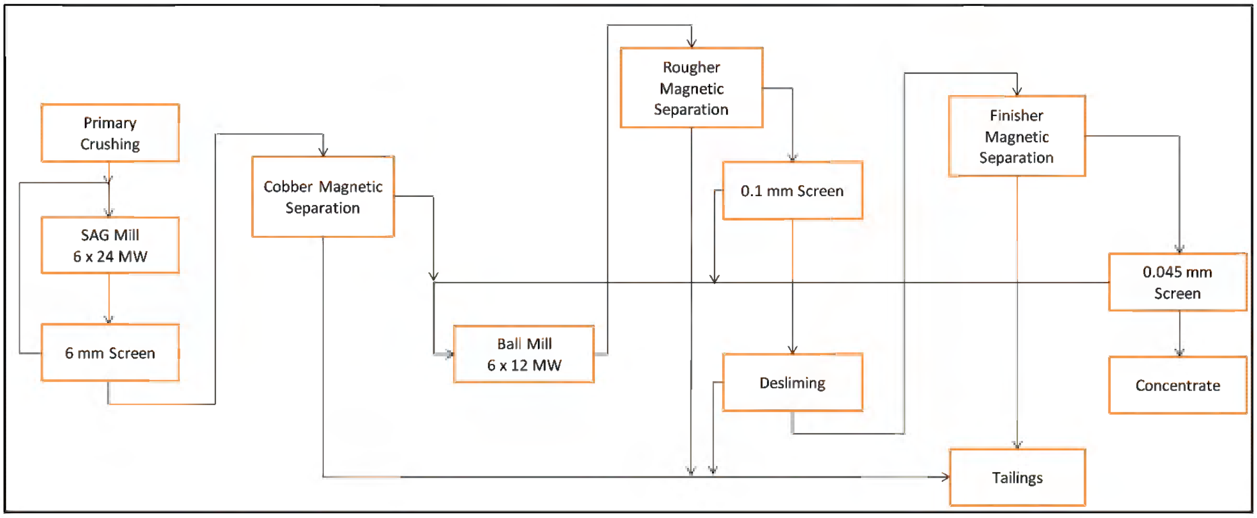
To size the SAG and ball mills, SAGDesign Consulting Group ("**SAGDesign**") from Ontario was selected. SAGDesign performed a testing program in 2005 to characterize the seven (7) lithological layers. They also tested three (3) composite ore blends to represent harder and softer blending scenarios feeding the SAG and ball mills for equipment sizing and power determination.

Based on these results, a pilot plant test was arranged at MRC. A bulk sample of 237 tonnes was sent from Labrador to MRC and testing took place during November and December 2005. In order to provide samples in proportions that reflected the distribution of material on a mining basis as opposed to stratigraphic basis, a further 25 tonnes of PGC material was sent to MRC early in November 2005.

a) Pilot Plant Circuit and Testing

The pilot plant consisted of a six (6) feet (1,829 mm) diameter by three (3) feet (915 mm) long SAG mill with an eight percent (8 %) ball charge, a SAG mill discharge screen with ¼" (6.3 mm) openings, a cobber wet drum magnetic separator, ball mill grinding in closed circuit with a rougher wet drum magnetic separator and a 200 mesh (75 µm) Derrick screen, a multi-stage finisher wet drum magnetic separator and a final screen. The final screen made a separation on 325 mesh (44 µm) and the screen oversize was returned to the ball mill. The screen undersize was final product. The MRC flow sheet is shown in Figure 13.2.

Figure 13.2 – MRC Pilot Plant Flow Sheet



The results of the three (3) campaigns are shown in Table 13.2. A quality concentrate can be achieved from the tested flow sheet with total iron of over 69 % and silica between 3.0 % and 4.0 %. This was done while achieving good weight recovery, averaging 30.1 % for the blends used in the testing. NML was targeting a product with 3.0 % silica or less and this was not achieved even with a Blaine value up to 2,300. Based on the Block A zone DT concentrate silica, the expected magnetic circuit product is 3.5 % SiO_2, which is what was obtained. Vendor test samples were collected to size various process equipment such as thickeners, filters, screens, etc.

Because the energy requirement to grind the ore by SAG mill was high, it was decided to investigate the use of HPGR to process the ore. It was believed that they would require considerably less electrical energy to treat this particular ore than would SAG mills. Discussions with KHD and SGA, both of Germany, confirmed that this belief was well-founded.

Table 13.2 – Summarized Results of Test Runs for MRC Pilot Plant

	Nov. 14-22	Dec. 5-9		Dec. 19 and 20	
	Range	**Range**	**Average**	**Range**	**Average**
Crude Feed Rate – Mt/h	0.9-1.0	0.9-1.0	0.97	0.975	0.975
Concentrate Range – (% Total Fe)					
Without Flotation (Fine Screen U'Size)	68.45-69.65	69.13-69.96	69.63	68.75-68.94	68.83
With Flotation (November/22)	69.85				
Concentrate Range – (% SiO_2)					
Without Flotation (Fine Screen U'Size)	3.27-4.24	2.66-3.77	3.38	3.91-4.13	4.03
With Flotation (November/22)	2.81				
Weight Recovery without flotation *	27.82-31.80	28.4-32.1	30.0	29.9-32.9	30.8
Concentrate Grind – (% - 325 mesh)	76.7-93.3	91.7-98.3	96.1	94.0-95.4	94.5
SAG Mill Grinding Power – (net kWh/MT)	11.1-12.4	11.5-13.6	12.8	12.3-12.9	12.6
Ball Mill Grinding Power – (net kWh/MT)	9.7-10.4	9.5-9.7	9.6	9.4-9.9	9.7
Blaine Surface Area – (cm^2/gram)		1,684-2,800	2,218	2,164-2,337	2,281

*Note that the Weight Recovery with flotation was not done.

13.6.2 HPGR/Ball Mill Testing on LabMag Block B Zone (SGS 2006)

In 2006, about 187 tonnes of material from the LabMag Block B zone was sent to SGS Lakefield to test the new HPGR flow sheet, replacing the SAG mill and adding a flotation circuit to produce DR grade pellets. The Block B ore was selected as it showed better liberation than the Block A zone. At that time, the intention was to focus on mining this part of the deposit at the beginning of the operation.

a) Pilot Plant Circuit and Testing

The bulk sample was first crushed by lithology in two (2) stages with jaw and roll crushers from a top size of about eight inches (8") to 100 % passing ½". This material was then blended to approximate the 15-year mine plan and this formed the HPGR feed. The chemistry of the different bulk samples processed as feed by the different HPGR/Ball Mill pilot plant tests between 2006 and 2012 is shown in

Table 13.3. In the earlier testing, DTWR was not performed to characterize the sample. Table 13.4 presents the characteristics of the blended pilot plant feed.

Table 13.3 – Run-of-Mine Ore Analysis

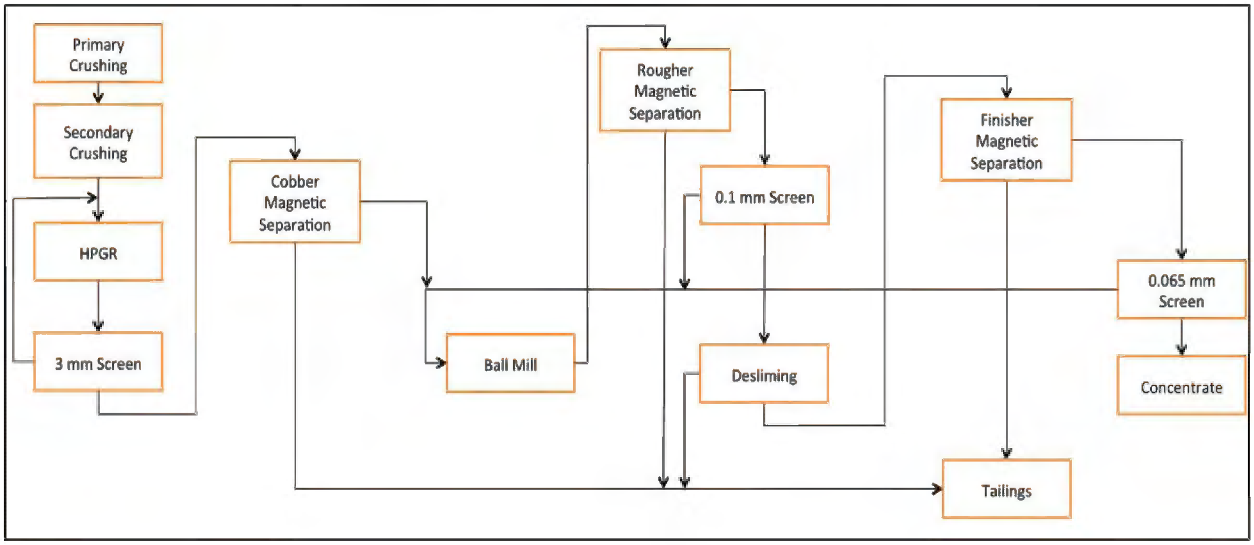
Component (%)	2006/2008 Tests		2012 Tests
	LabMag	KéMag	LabMag B
TotFe	29.50	32.5	30.9
Fe_3O_4	29.00	27.88	27.91
SiO_2	51.60	42.45	49.79
Al_2O_3	0.42	0.22	0.44
K_2O	0.09	0.07	0.09
Na_2O	<0.05	0.03	0.05
CaO	1.22	2.79	1.33
MnO	0.65	1.29	0.74
MgO	0.89	1.48	0.85
TiO_2	0.03	0.02	0
S	0.02	0.02	<0.01
P	0.04	0.01	0.017
LOI	2.49	5.08	2.32 (calc)

Table 13.4 – Blended Pilot Plant Feed (LabMag Block B Zone)

HPGR Feed Characteristics	Value
F_{50}, (microns)	5,865
F_{80}, (microns)	9,228
Percent Passing 150 microns	7.85
Percent Passing 53 microns	5.04
Specific Gravity (kg/L)	3.24
Wet Bulk Density (kg/L)	2.04
Moisture 'as received' (% H_2O)	3.0
Ball Mill Work Index -75µm (kWh/t)	14.8

To select the proper operating conditions, SGS Lakefield first carried out bench-scale HPGR tests under varying conditions followed by locked cycle testing with a 2.8 mm screen. Low-intensity magnetic cobber separation was investigated to determine the weight rejection as well as grindability tests to determine a Bond Work Index of the ball mill feed. The findings were implemented in the larger pilot scale tests which are outlined below. The simplified HPGR flow sheet is depicted in Figure 13.3.

Figure 13.3 – Single stage HPGR circuit



The crushed sample was pressed in the Polysius Labwall laboratory scale HPGR machine, operating in closed circuit with a 2.8 mm screen and a wet cobber separator. The results are presented in Table 13.5. The net power required for the HPGR was 2.81 kWh per tonne of fresh feed. The recycle load was 121 % with the small unit which has a large edge effect. The cobber separation stage recovered 58.1 % of the mass and 97.3 % of the magnetite.

Table 13.5 – HPGR-Cobber Circuit Results

Item	Value
Pressure	52 Bar
Net Power Requirement	2.81 kWh/t
Specific Grinding Force	2.59 N/mm^2
Specific Throughput	262 ts/hm^3
Recycle Rate	121 %
P_{80}	1,326 µm
Cobber Weight Recovery	58.1 %
Cobber Concentrate Fe Grade	41.5 %

The ball milling was carried out in several campaigns, requiring about 22.6 kWh per tonne of crude ore feed and the products were blended to reach a final magnetic concentrate grade of 2.83 % silica with a calculated weight recovery of 26.3 %. The results are presented in Table 13.6.

Table 13.6 – Ball Mill-LIMS Circuit Results

Item	Value
P_{80}	46 µm
% Passing 325 mesh	78 %
BM - LIMS Weight Recovery	45.3 %
Fe Grade	69.6 %
SiO_2 Grade	2.83 %
Product Blaine*	2,110 cm^2/g
Overall Circuit Weight Recovery	26.3 %

*Measured later at SGA

The above results met NML's desired product quality and demonstrated that the LabMag Block B ore had a coarser liberation size and thus reached the target of <3.0 % SiO_2 with magnetic separation alone.

However, the concentrate grade results from this SGS pilot plant are not expected to be reproduces by the commercial plant since the Block B zone has better liberation characteristics than the average LabMag deposit. The average planned feed to the process plant should be about 0.3 to 0.4 % higher in silica and will result in a proportional increase in the product so a product with 3.1 % silica is predicted. Also, since the cobber feed F_{80} from this smaller HPGR unit (1.33 mm) is finer than expected from larger industrial scale units (2.0 mm approximately), the cobber magnetic separator tailings rejection is believed to be optimistic (see section 13.6.4).

13.6.3 HPGR/Ball Mill Testing on KéMag Samples

In order to verify the results of the liberation studies, it was decided to test the KéMag ore in the HPGR/Ball Mill circuit. Unlike the LabMag deposit, it is difficult to collect a bulk sample from trenches in KéMag because of the thick overburden layer. The decision was made to use the saved half (½) core samples from the DT characterization tests at MRC from exploration drilling.

The KéMag bulk sample consisted of 3.9 tonnes of crushed drill core. Table 13.7 shows the composition of this sample as calculated from the drill core analysis. The average DT silica at 2.85 % is slightly higher than the deposit average of 2.59 % SiO_2.

Table 13.7 – KéMag Bulk Sample Analysis

Ore Type	Percentage (%)	Length (m)	DTWR (%)	SiO_2 (%)
LC	25	511	28.23	2.97
JUIF	12	246	27.60	2.56
GC	5.4	111	11.48	1.96
URC	6.8	139	27.98	2.12
PGC	13.9	284	34.54	2.62
LRC	1.4	28	25.56	2.93
LRGC	35.5	725	26.89	3.23
Average	100	2,044	27.59	2.85

a) Pilot Plant Testing KéMag Ore (SGA)

SGA used a flow sheet similar to SGS Lakefield (Figure 13.3) to process the material received from MRC using a pilot scale HPGR unit but investigated two (2) closing screen sizes of 2.8 mm and 1.0 mm to better understand the impact of the closing screen aperture on cobber rejection in the HPGR circuit. The aim was to produce iron ore concentrate suitable for blast furnace use with about 3.0 % SiO_2 and for direct reduction with less than 2.0 % SiO_2 +Al_2O_3.

Results indicate that the grind and liberation characteristics of KéMag material and the quality of the produced concentrate are very similar to those of the LabMag Block A tested by MRC.

The results of the HPGR testing are summarized in Table 13.8 and Table 13.9. The provided sample was delivered pre-crushed to < 9.5 mm and showed a D_{80} of 6.8 mm which is low for a pilot size HPGR feed sample. This leads to optimistic recirculation estimates.

The energy required for the HPGR to reach 2.8 mm was only 2.0 kWh/t of crude ore with a recirculation of 78.9 %. The recirculation increased to 183.3 % when the screen opening was reduced to 1.0 mm and the power rose to 3.1 kWh/t of crude ore, which shows a very significant impact of the closing screen on the circulating load of the HPGR circuit.

A cobber magnetic separation stage was performed on the HPGR products. With the <2.8 mm screen, a tailings weight rejection of 42.1 % was achieved with a magnetite loss of 7.7 %. The <1.0 mm HPGR product achieved a rejected tailings weight of 47.7 % and the magnetite loss decreased to 6.5 %. The decrease in magnetite loss is a result of the better performance of the magnetic separator with finer feed.

Table 13.8 – Summary of Results for HPGR

Roller Press Grinding in Closed Circuit (last Cycle)					
Cut Size (mm)	Spec. Press Force (kN/mm^2)	Specific Throughput (Ts/hm)	Energy Demand Crude Ore (kWh/tnet)	Recirculation Load (%)	Maximum H_2O Feed (%)
2.8	2,300	254	2.0	78.9	1.4
1.0	2,300	247	3.1	183.3	2.9

Table 13.9 – Summary of Results for Cobber Separation

Magnetic Separation on Screen Underflow					
Weight		Magnetite		Concentrate	
Concentrate (TotFe%)	Tails (%)	Content Tails (%)	Recovery Tails (%)	Fe Content (%)	Fe Recovery (%)
57.9	42.1	5.4	7.7	41.3	73.6
52.3	47.7	3.6	6.5	45.0	72.6

To perform the second phase of the test work, a cobber concentrate was produced from the HPGR with a closing screen of 2.8 mm. Because of the small sample provided, a low quantity of cobber concentrate was available for the ball mill stage testing and ball mill test runs could not reach equilibrium. The results of the ball mill runs are summarized in Table 13.10 and Table 13.11.

Table 13.10 – Summary of Results for Ball Mill Test Runs – Part 1

Pilot Plant Test (PPT)	Grinding Energy Demand (kWh/t net)	Rougher Magnetic Separation Tailings			Deslimer Overflow		
		Weight (%)	Magnetite Content (%)	Magnetite Recovery (%)	Weight (%)	Magnetite Content (%)	Magnetite Recovery (%)
1	20.0	49.2	1.0	1.2	1.4	7.1	0.2
2	15.4	54	1.2	1.5	1	8.2	0.2
3	26.7	0	1.4	0	0	8.7	0

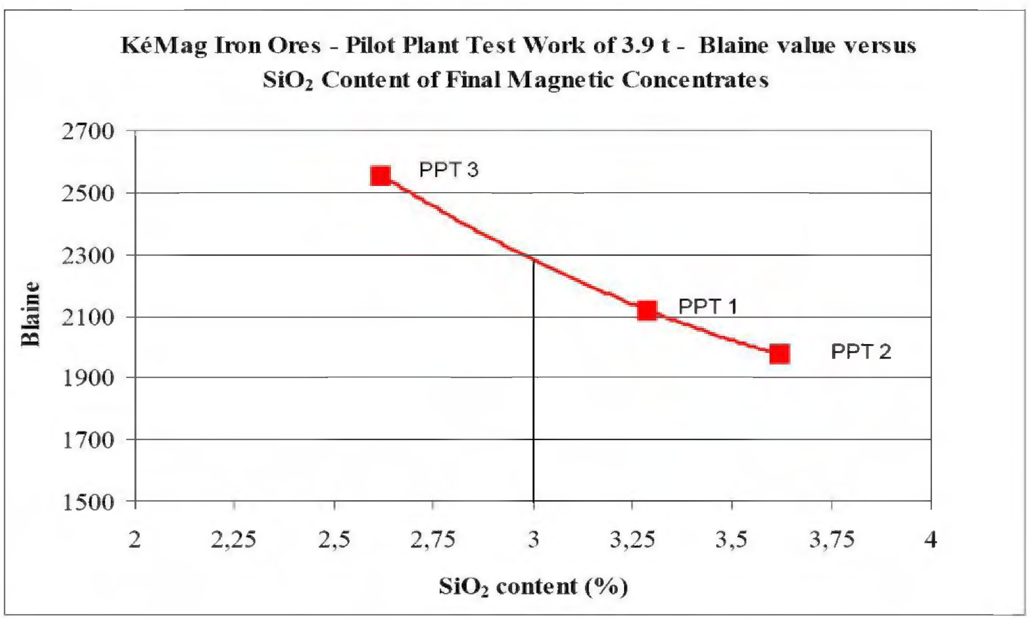
Table 13.11 – Summary of Results for Ball Mill Test Runs – Part 2

PPT	Cleaner Magnetic Separation Tailings			Final Concentrate				
	Weight (%)	Magnetite Content (%)	Magnetite Recovery (%)	Weight (%)	Fe Content (%)	SiO_2 Content (%)	Magnetite Recovery (%)	Blaine (cm^2/g)
1	8.8	25.1	5.2	40.6	68.95	3.29	93.3	2,118
2	2.9	23.8	1.6	42.1	68.6	3.62	96.6	1,976
3	0	26.1	0	0	69.65	2.62	0	2,555

Three (3) ball mill tests were performed at different energy inputs. Test #1 was performed using 20.0 kWh/t net on ball mill feed basis. A final concentrate at 68.95 % Fe and 3.29 % SiO_2 was produced. Test #2 was performed at an energy input of 15.4 kWh/t net which resulted in a coarser concentrate with a silica content of 3.62 %. Finally, test #3 was done at 26.7 kWh/t net, achieving a concentrate with a silica content of 2.62 %.

Using test #1 to calculate the mass balance, an overall weight recovery of 26.5 % was achieved at a Blaine value of 2,118. The power consumption relative to the crude ore feed is estimated at 15 kWh/t. The relationship of Blaine value versus SiO_2 content of the magnetic concentrates is presented in Figure 13.4.

Figure 13.4 – Blaine Value vs SiO_2 Content – SGA Pilot Plant KéMag 2007



The Pilot Plant test work showed that a final concentrate for blast furnace pellet feed is attainable with about 3.0 % SiO_2 at Blaine values of about 2,300 and a total weight recovery of approximately 26 %.

13.6.4 Feasibility Engineering Pilot Plant Test

The market studies undertaken by NML determined that the main products sought out by steel producers were low silica fluxed pellets (2.5 % SiO_2) and DR grade pellets (< 1.8 % SiO_2). NML was faced with the challenge of lowering the silica level in the product without grinding the ore too fine. NML engaged pelletizing technology supplier Outotec to design a very high capacity (8.5 Mt/y) pelletizing machine in order to optimize the operating cost.

During discussions on design parameters for the machine, it was agreed that the ore should be ground to a Blaine value of 1,800 cm^2/g in order to achieve the required physical and metallurgical properties of the pellets at such a high design capacity. To meet these targets, the concentration plant and mine had to coordinate and decide on how to feed the plant with the lowest possible DTC silica without losing reserves.

Based on the DT tests, using a 50-50 blend of LabMag Block A and Block B ore and using 18 % DTWR cut-off, the average DTC silica was determined to be 2.2 %. Since the DTC silica was based on stage grinding the samples to 100 % passing -325 mesh (45µm), the expected concentrate silica in the plant at a fineness of 1,800 Blaine value would be approximately 0.8 to 1.0 % higher in an actual operating plant.

While flotation would be required to lower the concentrate silica to 1.5 %, as in the case of DR grade pellets, there would be high losses in the circuit particularly for the KéMag ore. It was therefore decided that in addition to the DTWR cut-off, the following cut-offs for DTC silica was applied for the LabMag and KéMag deposits respectively, 4.0 and 4.2. The mine plans are based on those cut-offs.

The final DTC silica was determined to be 2.1 % for both deposits with some reduction in the reserves and this would ensure that the plant receives a consistent feed which will meet the concentrate and subsequent pellet specifications.

a) Bulk Sample Collection and Preparation

It was decided to undertake pilot plant tests of the selected flow sheet to determine plant design parameters and to produce sufficient amounts of concentrate with varying grades for pot grate testing to confirm the design of the indurating furnace and to produce sample pellets for interested parties.

A LabMag Block B sample was retrieved from storage at SGS Lakefield and shipped to MRC for homogenization. The sample was previously crushed to -12 mm.

b) Pilot Plant Testing

Fifty-one (51) tonnes of homogenized LabMag Block B bulk sample was sent to the Coleraine Mineral Research Lab ("**CMRL**"), part of the Natural Resources Research Institute ("**NRRI**") of the University of Minnesota at Duluth ("**UMD**"). The purpose of the test was to produce concentrates at 1,800 Blaine value.

The concentrate sample produced by the pilot plant was sent to SGA in Germany to reduce silica to 2.3 % and 1.5 % in a flotation pilot plant to produce low silica BF fluxed pellets and DR pellets respectively. A Blaine value vs silica graph was also developed to compare the DTC silica used to characterize the orebody with the silica values obtained from the pilot plant tests. This difference would provide an indication of the value of silica that could be expected from an operating plant.

Initially, the sample was characterized by NRRI to determine the DTC silica which was found to be 2.2 % (based on MRC characterization which is standard for all tests). Since the actual plant feed would be blended to a value of 2.1 %, the expected silica grade in the pilot plant would be higher than what could have been obtained had the sample been at the average DTC silica of the deposit. In addition to DTC silica, laboratory grinding curves were established for the sample using the same procedure as established at SGA.

At first, the sample was pressed in a pilot sized HPGR to produce a -3.0 mm cobber concentrate as per the circuit in Figure 13.5.

The same flow sheet was used to prepare a -0.7 mm cobber concentrate for comparison, similar to the HPGR and fineness used by Arrium Steel's Whyalla operation. Figure 13.6 presents the ball milling circuit used at NRRI to reach a final concentrate on both feed products (-3.0 mm cobber concentrate and -0.7 mm cobber concentrate) and Table 13.12 shows the results of the eight (8) ball mill runs. Minor changes were made to the circuit to overcome operational challenges, particularly mill dilution.

Figure 13.5 – Batch HPGR / 3 mm Screen / Magnetic Cobber Flow Sheet HPGR

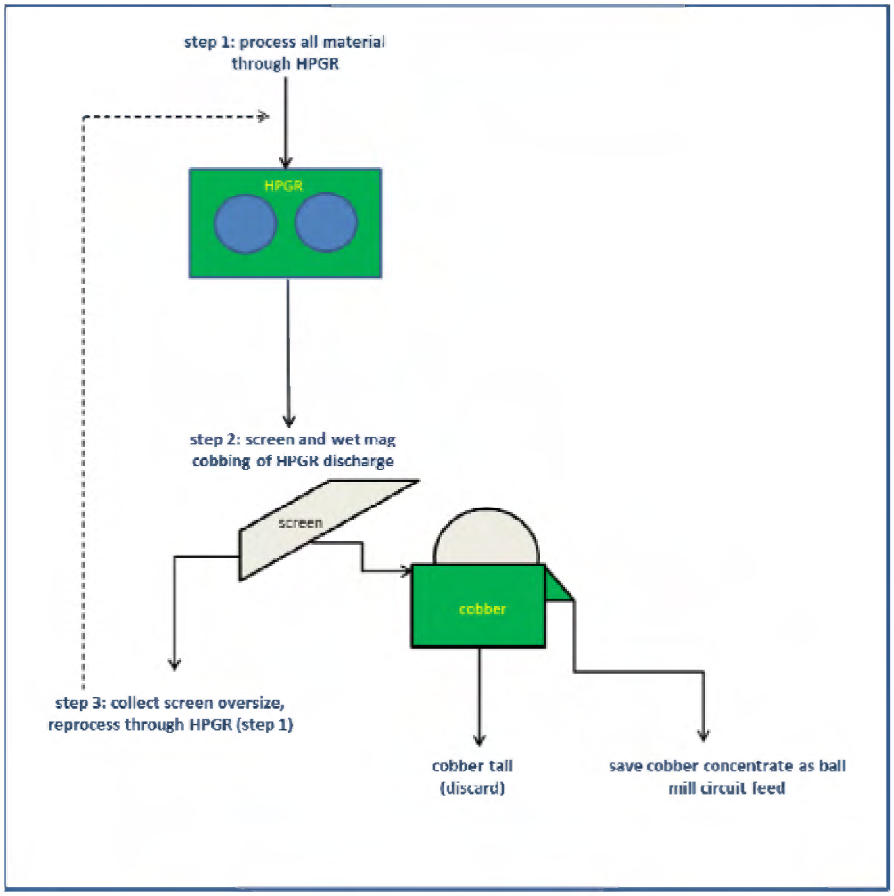


Figure 13.6 – Ball Mill Circuit Used at NRRI

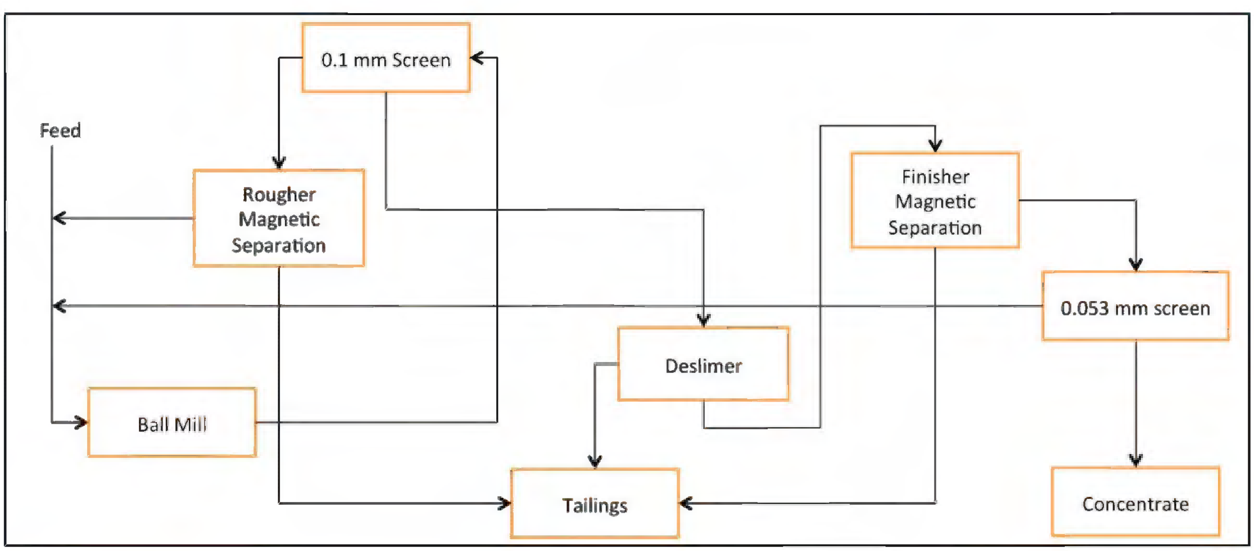


Table 13.12 – Ball Mill Pilot Plant Results From NRRI

Runs	1	2	3	4	5	6	7	8
Feed Type	3.0 mm					0.7 mm		
Date	Aug. 25	Sept. 5	Sept. 7	Oct. 2	Oct. 4	Sept. 10	Sept. 12	Sept. 13
	Block B	Block B	Block B	Block B	Block B	Block B	Block B	Block B
Target Feed Rate (kg/h)	80	100	80	80	80	130	100	90
Ball Mill Solids (%)	73	52	42	51	70	55	38	50
kWh/t	17.66	14.72	13.73	15.14	17.81	8.78	11.63	12.53
Fe (%)	68.26	69.42	69.56	68.09	69.5	70.35	69.9	70.78
SiO_2 (%)	5.03	3.37	3.34	3.22	3.42	3.00	2.82	2.78
Blaine Value	1,518	1,875	1,831	1,939	2,114	1,633	1,666	1,856
DT Fe Concentrate (%)	69.8	70	70.3	70.4	70.1	71	70.6	70.8
DT SiO_2 Concentrate (%)	3.62	2.67	2.54	2.19	2.64	2.33	2.28	2.28
Remark	Variable Feed Urethane Deck	Wire Mesh Installed	Mill Dilution		Manual Dewatering		Mill Dilution	O/S to RGHR Incr. Mill Dens.

For the five (5) ball mill runs with 3.0 mm feed, the results in the range of 1,800 Blaine value (Runs 2 and 3 from September 5 and 7 respectively) reached about 3.3 % SiO_2 in the concentrate. This is very close to the expected 1.0 % higher than DTC silica for a commercial plant. Ball mill runs six (6) to eight (8) which used -0.7 mm feed reached about 2.8 % SiO_2 in the concentrate, or 0.5 % lower silica than the 3.0 mm feed.

This demonstrates the advantage of using HPGR to grind finer before transferring to the ball mill. To simulate a flotation feed which would represent the DTC silica of the average resources, a blend of the various ball mill runs product was prepared using a combination of -3.0 mm and -0.7 mm concentrates to target 3.0 % silica at 1,800 Blaine value.

13.7 Flotation Test Work

Both SGA and SGS Lakefield performed bench-scale flotation tests. Based on its experience with northern Québec and Labrador hematite ore, SGS uses starch depressants to make the flotation more selective, but this significantly reduces the kinetics. On the other hand, SGA targets lower reagent consumption and does not use depressants which it considers are too inefficient for magnetite depression.

13.7.1 LabMag Block B – Flotation Test Program (SGS Lakefield 2006)

a) Mineralogical Characterization of the Magnetite Concentrate

The objectives of this characterization were the estimation of different Fe-bearing phases and associated gangue minerals and liberation characteristics.

Modal analysis of the sample for different phases was carried out by counting grains under optical microscopy and indicated that the sample was composed of 96 % magnetite and 2.8 % quartz (see Table 13.13).

Table 13.13 – Modal Analysis Data of Magnetite Concentrate

Minerals	Weight (%)
Magnetite	96.1
Quartz	2.8
Hematite	0.6
Others*	0.5

* Including pyrite, lepidocrocite, limonite and maghemite.

The liberation characteristics of different phases of magnetite concentrate samples were studied by point counting grains under optical microscopy and the data is provided in Table 13.14. Liberation data revealed that the majority of magnetite was liberated (~93 %).

Table 13.14 – Weighted Liberation Data of Magnetite Concentrate

Mode of Association	Minerals	Weight (%)
Liberated	Magnetite	92.5
	Quartz	2.2
	Others*	0.7
Composite/Locked	Magnetite + Quartz	2.2
	Magnetite + Hematite	1.2
	Composite phases **	1.2

*including hematite, maghemite, limonite, pyrite and lepidocrocite.
**Including magnetite, hematite, quartz, pyrite, lepidocrocite and maghemite.

To produce DR grade pellet feed with < 1.5 % SiO_2, a 2-step reverse flotation process is recommended to float the silica and recover magnetite. Around 50 % of the SiO_2 in the concentrate is locked as middlings and a froth regrind mill followed by magnetic scavenger was investigated to further reduce the silica.

b) Bench Scale Tests

The bench scale flotation test consisted of nine (9) batch tests as well as a 6-cycle locked cycle test. The basic flow sheet tested in the batch tests was as follows:

- Stage 1: Rougher flotation;
- Stage 2: Regrinding of froth;
- Stage 3: 1st cleaner of reground froth;
- Stage 4: 2nd cleaner of reground froth.

The earlier mineralogical studies indicated that the removal of silica in the flotation circuit would be more effective if the middlings were reground to liberate magnetite particles from the silica. In all the tests, dextrin was used as a depressant in varying amounts and the pH was set at 11. Caustic starch was used as pH modifier and very significant quantities of reagents were needed. At pH of 11, the best results were achieved using 500 g/t of depressant.

Based on the batch test conditions, a locked cycle test was carried out. The magnetic circuit concentrate at 2.72 % silica was used for the locked cycle test and showed that a DR grade concentrate could be obtained by reverse flotation of silica. The results projected a 1.45 % SiO_2 DR concentrate at 97.7 % Fe recovery and the silica tails represented only 3.6 % of the mass.

c) Pilot Plant Test

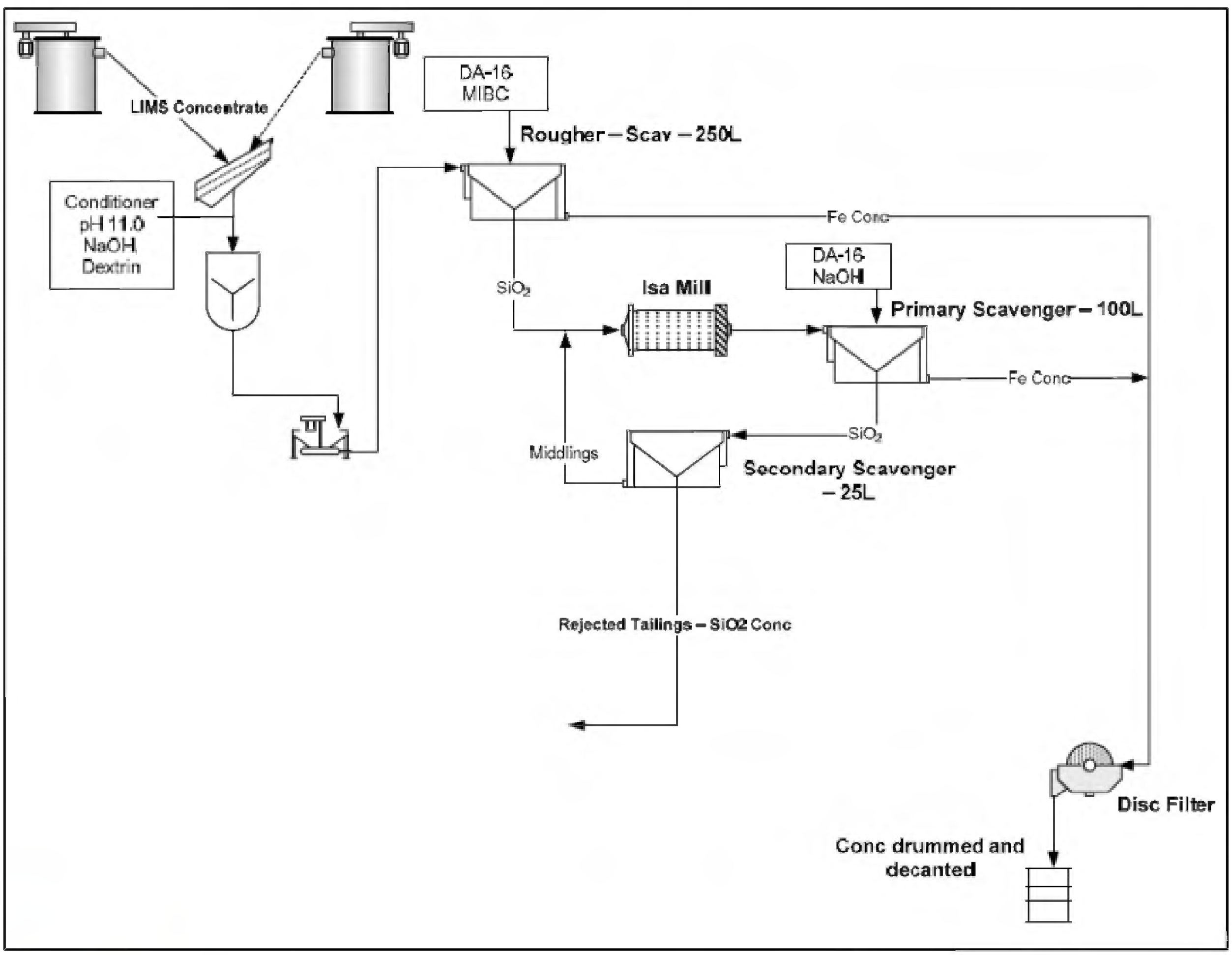
The test program performed by SGS Lakefield in 2006 included a pilot scale flotation test program for the production of a large DR pellet feed sample for pelletizing tests.

Following the bench scale test program, the basic flow sheet shown in Figure 13.7 was used in the pilot plant.

Approximately five (5) tonnes of ball mill circuit concentrate with 2.7 % silica was fed to the pilot plant flotation circuit with the aim of producing DR concentrate at about 1.5 % SiO_2. Approximately 3.74 tonnes of DR concentrate was produced with a grade of 1.72 % SiO_2 and 98.1 % Fe_3O_4 and the target 1.5 % silica was not reached.

Under normal circumstances, it should be possible to match or improve upon the locked test results under optimized full-scale continuous operation.

Figure 13.7 – Flotation Pilot Plant Flow Sheet (SGS 2006)



13.7.2 Flotation Test Program (SGA 2007)

A short bench scale flotation test program was performed on the KéMag ore tested in 2007 by SGA. The small amount of concentrate available did not allow for a full continuous pilot plant test.

The magnetic circuit concentrate between 2.9 % and 3.3 % SiO_2 was used to examine the rougher flotation stage conditions. Table 13.15 shows a summary of the flotation conditions and performance.

Table 13.15 – Summary of Laboratory Rougher Flotation Tests on KéMag

Test	Feed Silica (%)	Concentrate	
		Wt.Rec. (%)	Silica (%)
1	3.33	62.4	1.58
2	3.04	63.5	1.39
3	3.08	70.0	1.46
4	2.92	77.3	1.86
5	2.96	76.0	1.56
6	3.02	88.3	1.70
7	3.08	94.8	2.11
8	3.18	83.1	1.36
9	3.01	88.8	1.35

The flotation test program started with no depressant (caustic starch) at natural pH and the flotation was not selective. Low weight recoveries of 63 % were achieved. Changing the collector improved this selectivity, slightly increasing the weight recovery to 70 % at similar product silica.

The pH was then increased and selectivity improved slightly but the concentrate grade did not reach the target 1.4 % silica at the rougher.

The best results were achieved in Test #9 with the addition of starch. Since starch is cooked in caustic, no further pH adjustment was needed. Total weight recovery increased to 88.8 % with a SiO_2 content of 1.35 %, which is considered excellent.

13.7.3 LabMag Block B – Flotation Test Program (SGS Lakefield 2008)

a) Flotation Test Program

During spring of 2008, a new flotation test program was carried out at SGS Lakefield to optimize the DR concentrate production based on the findings of the previous SGA study. LabMag Block B concentrate leftover from the previous work in 2006 was used. The program aimed to define the best potential flow sheet for flotation at the bench scale and then to validate this flow sheet at a pilot plant scale in continuous operation.

b) Bench Scale Tests

The bench scale flotation test was extensive and explored various reagent levels. All tests, except for one, used both pH adjustment as well as depressant (caustic starch or dextrin WW-82). The collector dosages were quite high but satisfactory grades and weight recoveries were achieved at this stage. Compared to the first testing program of 2006, the main difference is the use of lime as pH modifier and the cleaner stage incorporated single stage magnetic separation after regrinding prior to cleaner flotation. This was also implemented in the pilot plant.

c) Pilot Plant Test

Under optimized pilot plant conditions, with a combination of flotation, magnetic separation and regrinding, the final direct reduction concentrate reached 1.45 % SiO_2 at 95.1 % weight recovery and Fe recovery of 96.5 % with reference to the magnetite plant product feeding the flotation circuit. The flow sheet used to upgrade the concentrate to DR pellet feed is presented in Figure 13.8 and the results from the two (2) confirmatory runs are presented in Table 13.16.

Figure 13.8 – Flotation Flow Sheet for DR Grade Concentrate

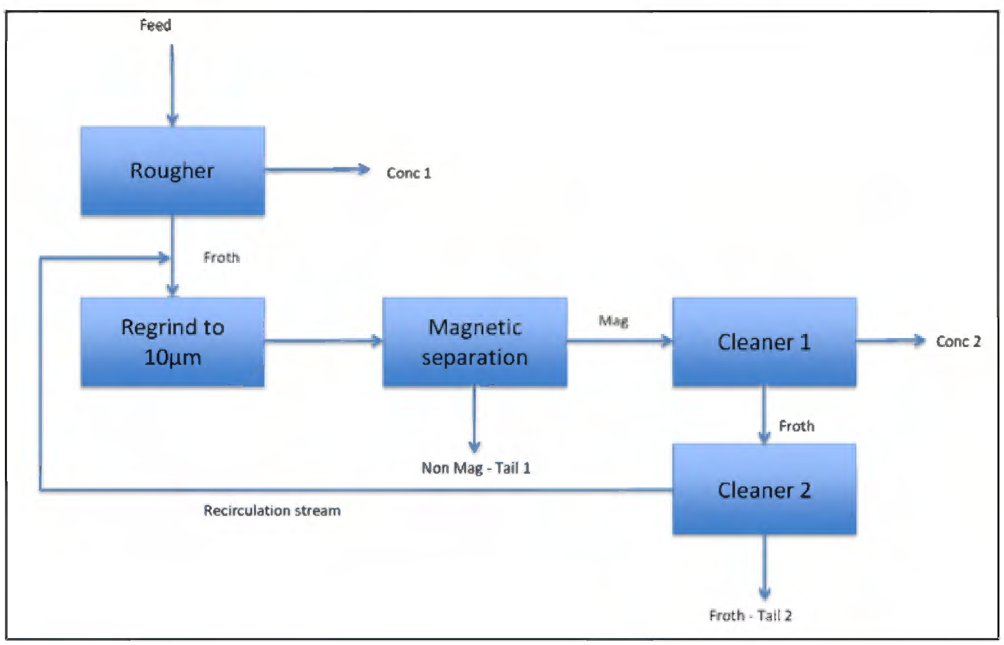


Table 13.16 – Flotation Pilot Plant Results (Confirmation Runs)

Test #	Feed rate (g/L)	Starch (g/t)	Amine (g/t)	Product	Wt (%)	Fe (%)	SiO_2 (%)
PP-05	1,390	352	246	Feed	100	69.9	2.66
	0	0	0	Combined Concentrate	94.5	70.9	1.49
PP-06	1,395	357	261	Feed	100	68.9	2.60
	0	0	0	Combined Concentrate	95.7	69.9	1.40

13.7.4 Flotation Test on CMRL Concentrate (SGA 2012)

Following the pilot plant testing of HPGR and ball mill circuits at CMRL in 2012, concentrate drums were combined for about 1.8 tonnes of flotation feed which were shipped to SGA for pilot plant flotation and for pot grate test feed preparation. The blended feed was characterized and the results are shown in Table 13.17. The concentrate had 3.09 % silica and 1,769 Blaine value, both of which were considered very close to the target values.

Table 13.17 – Flotation Feed Results

	Fe (%)	SiO_2 (%)	Blaine Value
NML Mag Separation Concentrate	69.27	3.09	1,769

Flotation was performed in two (2) stages. The circuit is presented in Figure 13.9. Some changes were made from the previous flotation work at SGS Lakefield. The results are shown in Table 13.18.

Because of the low weight available, this test was performed in two (2) phases:

- Stage 1: Rougher flotation operated continuously;
- Stage 2: Cleaners operated in locked cycle operation.

The flotation process reduces the Blaine value of the concentrate as finer particles become entrained with the froth, which is rejected. Both the BF grade and DR grade concentrate targets were met and the concentrates were forwarded to Outotec for pelletizing tests. Due to the lower Blaine value, a touch-up grind may be implemented at the pellet plant.

Figure 13.9 – Pilot Plant Flotation Circuit (SGA 2012)

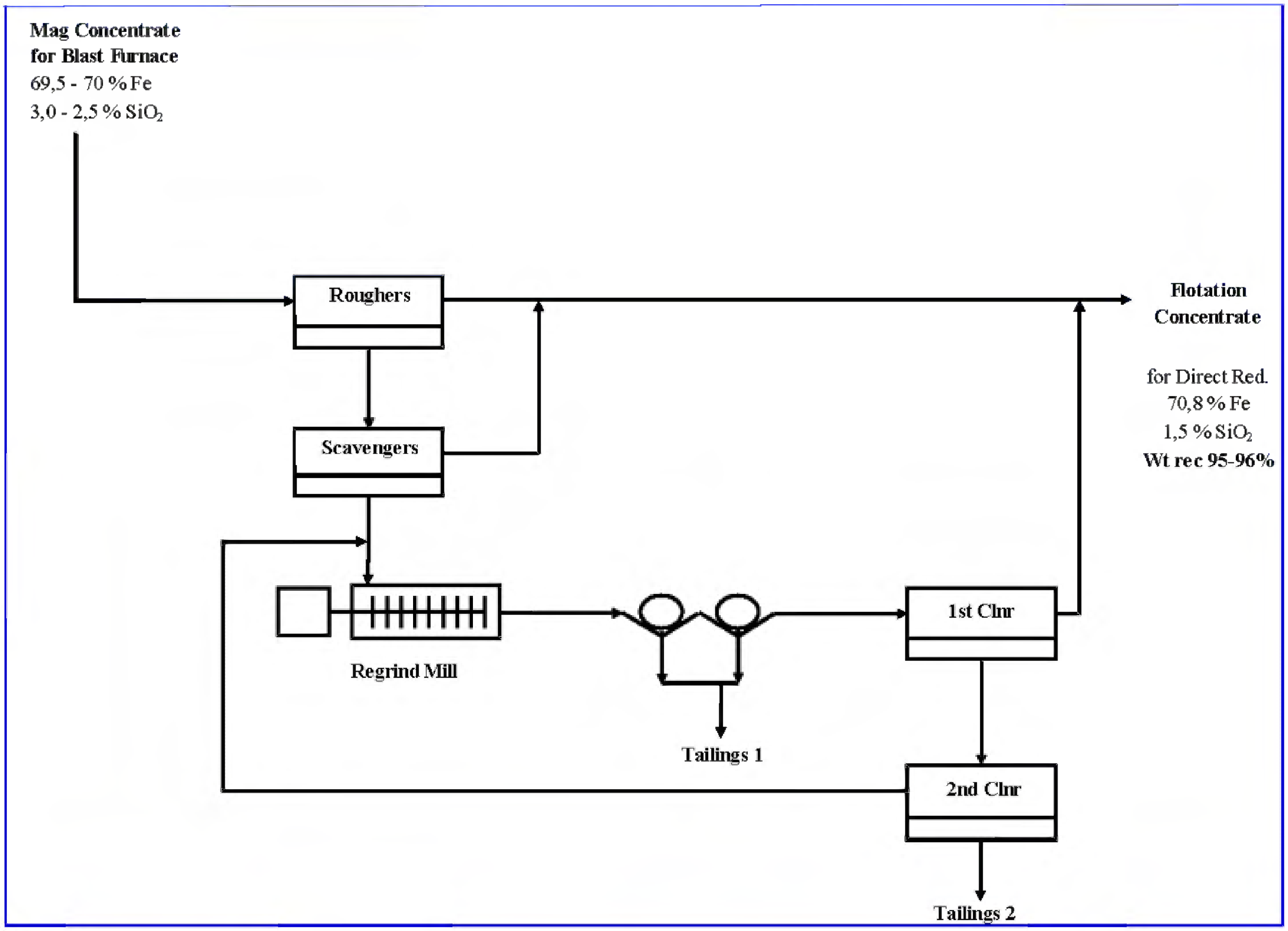


Table 13.18 – Flotation Results at SGA

Product	Weight Recovery (%)	Fe (%)	Fe Recovery (%)	SiO_2 (%)	Mag. (%)	Mag. Recovery (%)	Blaine Value
Rougher Concentrate	69.7	70.6	71.0	1.44	97.8	71.9	1,043
Scavenger Concentrate	3.7	70.4	3.8	1.73	96.9	3.8	1,625
Cleaner Concentrate	20.5	70.6	20.8	1.84	93.3	20.6	2,953
Magnetic Separator Tailings	2.2	25.0	0.8	57.50	11.3	0.3	
Secondary Cleaner Froth	3.9	63.3	3.6	11.35	81.1	3.4	
	100.0	**69.29**	**100.0**	**3.15**	**94.3**	**100.0**	
Combined Concentrate	93.9	70.58	95.6	1.54	96.8	96.3	1,483

13.8 Equipment Suppliers Samples

Vendor test samples were collected and provided to equipment suppliers, such as Metso, Outotec, etc. for development of their own design criteria for their proposals and equipment sizing:

- Crusher sizing test work (Metso, 2006);
- HPGR design test work (SGA, KHD, Polysius, Köppern);
- Screening design test work (Derrick);
- Thickener design test work;
- Final concentrate filtration test work;
- Material handling test work;
- Bulk density (total sample).

13.9 Recommended Test Work for the Detailed Engineering Phase

The following is a summary of recommended additional test work to be performed prior to detail engineering:

- The finer HPGR product flow sheet should be investigated further as the benefits were clear in the flow sheet comparison done by CMRL. HPGR grinding to produce a -0.7 mm cobber concentrate reached the target silica after magnetic separation while the 3.0 mm pressing did not;
- Flotation optimization to investigate the reduction of reagents and the possible elimination of the cleaner circuit;
- Tower mill re-grinding capacity for the flotation circuit must be better evaluated to confirm the number of units and power required;
- Additional vendor test work to confirm the design performance of equipment such as final screens and concentrate filters.

14.0 MINERAL RESOURCES ESTIMATES

14.1 Mineral Resource Estimates Statement

Met-Chem was mandated by NML to carry out a Mineral Resource estimate of the KéMag Deposit located at Lake Harris, Nunavik, approximately 50 km north of Schefferville in the Province of Québec. The KéMag Deposit is situated 18 km to the north of the LabMag Deposit. No newer drilling campaign was performed on the Property since the last resource estimate performed by SGS-Geostat Ltd. in September, 2008. However a re-estimation of the resources was deemed relevant in order to take into account changes introduced in the drill hole database and the geological interpretation after inconsistencies were highlighted and corrected.

Most important changes introduced are related to implementing a correction factor on the density in order to take into account the secondary porosity and also the building of regression function for each seam in order to model the density depending of the head iron content (TotFe). Such regression functions allow a more accurate definition of density, which is correlated to the TotFe, instead using an average density per seam as performed for previous resource estimation.

The effective date of the mineral resource estimate is December 4th, 2012 which is the date for the completion of the core density measurements, using the weight in water and in air method, performed on both LabMag and KéMag core samples at NML's core shack in Labrador City. Met-Chem has not visited the Property since that date. However NML confirmed that no field work or drilling that may have any impact on the geometry, geology or grade of the Deposit has been carried out after the Resource Estimation. Since none of the parameters used in the 2012 Resource Estimates have changed, Met-Chem's QP is satisfied that the 2012 resource figures are still valid and current at the time the present Technical Report was prepared.

The entire database used contained 90 records resulting from exploration work performed between 2006 and 2008. Eighty-nine (89) holes were used to interpolate blocks constrained within surfaces (top and bottom) related to each geological seam. Variogram parameters defined for the more drilled LabMag Deposit were used in resource interpolation. A block modeling approach was used and blocks size was changed comparatively to the previous estimate. The resource interpolation was performed using the Inverse Distance Weighted ("**IDW**") method at a power of two (2) ("**IDW2**"). The resource estimate was performed by Schadrac Ibrango P. Geo., Ph.D., the QP responsible for this Section, or performed under his direct supervision.

The resource classification follows the guidelines adopted by the Canadian Institute of Mining Metallurgy and Petroleum ("**CIM**") through the NI 43-101 standards and guidelines. The criteria used by Met-Chem for classifying the estimated resources are based on certainty of continuity of geology and grades. The CIM standards for resource classification are provided in Section 14.2. Mineral Resources are stated using a DTWR cut-off of 18 %. A summary of the Mineral Resources is provided Table 14.1.

Table 14.1 – Summary of the Mineral Resources (Cut-Off of 18 % DTWR)

Resource by Category	Tonnage (Mt)	In Situ Grades			
		TotFe (%)	DTWR (%)	Concentrate	
				Fe (%)	SiO_2 (%)
Measured	1,507	31.45	26.97	69.69	2.56
Indicated	876	31.95	27.32	69.83	2.51
Measured + Indicated	**2,383**	**31.63**	**27.10**	**69.74**	**2.54**
Inferred	1,007	31.56	26.97	69.31	2.65

14.2 Definitions

According to the final version of the CIM Standards/NI 43-101 which became effective on February 1, 2001 and was revised on June 30, 2011:

A **Mineral Resource** is a concentration or occurrence of diamonds, natural, solid, inorganic or fossilized organic material including base and precious metals, coal, and industrial minerals in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge.

An **Inferred Mineral Resource** is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes.

An **Indicated Mineral Resource** is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough for geological and grade continuity to be reasonably assumed.

A **Measured Mineral Resource** is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough to confirm both geological and grade continuity.

14.3 Mineral Resource Estimate Estimation Procedures

The estimation of the KéMag Mineral Resource includes following procedures:

- Detailed analysis and audit of the drill holes database received from NML;
- Importation of the database into MineSight® v. 7.80-2;
- Generation of basic descriptive statistics of the entire DB to analyze and compare the different statistical parameters in order to provide a general representation of grades behavior;
- Decision on the compositing length and if need of grade capping;
- Importation, audit and necessary adjustments of the sectional geological interpretation provided by NML;
- Re-generation of basic descriptive statistics on a seam by seam basis;
- Construction of geological surfaces representative of top and bottom of each geological layers;
- Analysis the drilling density in order to determine block size parameters;
- Generation and setup of a block model;
- Analysis the relationship between TotFe % and density on a seam by seam basis. Development of a linear regression model for each seam in order to automatically calculated the density of each block depending its iron content;
- Resource interpolation and validation;
- Classification of the Mineral Resource according to CIM/NI 43-101 standards;
- Mineral Resource statement.

14.4 Drill Hole Database, Data Verification and Validation

14.4.1 Data Verification and Validation

One (1) Microsoft Access format DB containing both the LabMag and KéMag Deposits data was provided by NML on December 2, 2011. A second version was supplied on December 15, 2011 which clarified some ambiguities highlighted by Met-Chem. Data specifically related to KéMag was extracted in a Microsoft Excel file for audit, validation purposes and subsequent use. The following issues were highlighted when auditing the DB and discussed with NML prior to corrective actions being taken:

a) Three Holes without Coordinates

Three (3) holes in the KéMag database did not have coordinate. These holes are 07HL1003A, 07HL1007A and 07HL1069A. They were noted by NML as having been abandoned due to technical difficulties. Replacement holes were restarted in the nearby vicinity and are already present in the database. The related holes were removed from the resource database.

b) Duplicate Intervals with Different Lithological Codes

Duplicate intervals with different lithological codes were found in the database. NML reported such intervals as having been introduced by SGS for the previous resource estimate. The duplicated intervals were removed from the database.

14.4.2 Drill Hole Database

The corrected and updated database contains 90 records of holes drilled during the 2006, 2007 and 2008 drilling campaigns. A summary of drilling work performed during these drilling campaigns is labeled in Table 14.2. Fields contained in the DB are summarized in Table 14.3.

Table 14.2 – Drilling Statistics by Year

Company	Drilling Year	Holes	Drilled Length (m)	Sampling Length (m)
NML	2006	29	3,585.60	2,493.9
NML	2007	46	4,979.20	3,875.44
NML	2008	15	2,216.10	1,503.5
Total		**90**	**10,780.90**	**7,872.84**

Table 14.3 – KéMag Database Content

File	Fields
Collar	Easting, Northing, Elevation, Azimuth, Dip, Length
Assays	Hole ID, From, To, Fe%h, DTWR%, Fe%c, SiO_2%c, SiO_2%h, SMG%h, SMG%c, P%h, Mn%h, Al_2O_3%h, TOx%h, SpGrav, Fe++, P%c, Mn%c, Al_2O_3%c, tOx%c, CaO%c, MgO%c, TiO_2%c, Na_2O%c, K_2O%c, MagSus, Cr_2O_3%c, V%c, LOI%h, LOI%c, Zone
Lithology	Hole ID, From, To, RCODE, GCODE

Table 14.4 presents the different lithological units as logged by geologists and their classification by rock group. Basic statistics by seam basis and calculated from the entire DB for iron formation are presented in Table 14.5.

Table 14.4 – KéMag Lithological Units

RCODE	GCODE	Rock Type	Rock Group
1	OB	Overburden	Stripping
2	MS	Menihek Shale	Stripping
3	LC	Lean Chert	Iron Oxide
4	JUIF	Jasper Upper Iron Formation	Iron Oxide
5	GC	Green Chert	Iron Oxide
6	URC	Upper Red Chert	Iron Oxide
7	PGC	Pinky-Green Chert	Iron Oxide
8	LRC	Lower Red Chert	Iron Oxide
9	LRGC	Lower Red-Green Chert	Iron Oxide
10	LIF	Lower Iron Formation	No Ore

Table 14.5 – Basic Statistics by Seam of Main Quality Elements

		TotFe (%)	$TotSiO_2$ (%)	DTWR (%)	Concentrate	
					Fe (%)	SiO_2 (%)
LC	**Average**	29.50	27.05	68.65	2.73	29.50
	Minimum	4.21	0.00	0.00	0.00	4.21
	Maximum	40.28	45.00	72.10	10.80	40.28
	St. Dev.	5.04	8.31	8.17	2.12	5.04
	COV	0.17	0.31	0.12	0.78	0.17
	Samples	385	385	385	385	385
JUIF	**Average**	34.04	22.63	69.85	2.31	34.04
	Minimum	14.41	8.00	64.32	1.16	14.41
	Maximum	38.18	38.50	71.29	9.50	38.18
	St. Dev.	3.86	8.54	1.07	1.11	3.86
	COV	0.11	0.38	0.02	0.48	0.11
	Samples	80	80	80	80	80

		TotFe (%)	TotSiO$_2$ (%)	DTWR (%)	Concentrate	
					Fe (%)	SiO$_2$ (%)
GC	Average	21.16	12.78	70.06	1.75	21.16
	Minimum	12.19	2.50	66.70	0.00	12.19
	Maximum	36.82	36.00	71.60	4.30	36.82
	St. Dev.	5.50	6.81	0.94	0.65	5.50
	COV	0.26	0.53	0.01	0.37	0.26
	Samples	80	80	80	80	80
URC	Average	33.90	26.71	70.35	1.87	33.90
	Minimum	16.38	13.50	64.32	0.00	16.38
	Maximum	39.39	41.00	72.00	9.50	39.39
	St. Dev.	3.54	4.72	1.06	1.06	3.54
	COV	0.10	0.18	0.02	0.56	0.10
	Samples	79	79	79	79	79
PGC	Average	33.20	30.53	70.11	2.21	33.20
	Minimum	4.66	1.50	64.07	0.00	4.66
	Maximum	44.04	46.00	71.70	8.28	44.04
	St. Dev.	3.58	9.57	1.11	1.14	3.58
	COV	0.11	0.31	0.02	0.52	0.11
	Samples	172	172	172	172	172
LRC	Average	33.48	19.68	70.71	1.75	33.48
	Minimum	31.22	7.50	61.30	0.66	31.22
	Maximum	37.27	39.50	72.30	12.60	37.27
	St. Dev.	1.27	9.51	1.82	2.00	1.27
	COV	0.04	0.48	0.03	1.14	0.04
	Samples	42	42	42	42	42
LRGC	Average	31.46	25.14	69.29	2.97	31.46
	Minimum	19.06	6.50	51.70	0.44	19.06
	Maximum	39.53	49.00	72.00	22.40	39.53
	St. Dev.	3.09	9.06	2.91	3.00	3.09
	COV	0.10	0.36	0.04	1.01	0.10
	Samples	461	461	461	461	461

The statistics on DTWR % show PGC as being the most magnetic rich layer. The highest iron content in head is observed on JUIF and URC. The high head iron content in URC could be explained by its mixed nature (hematite and magnetite). The lowest DTWR % and Fe % are observed on GC which appears being less rich at KéMag than at LabMag.

14.5 Geological Modeling Procedures

The original geological modeling was performed by the NML's geological team and transmitted to SGS for the previous estimates. The KéMag deposit is composed of a series of sub-horizontal layers (or seams) slightly dipping at -6° to the northeast without any significant deformation. Each layer is modeled separately. The methodology used for the geological modeling is based on the generation of vertical sections where digitalized lines are snapped between seam contacts from hole to hole. These sections lines are then joined together by triangulation to create geological surfaces representing each seam. NML supplied Met-Chem with the section lines and geological surfaces triangulated by SGS. Inconsistencies were found when Met-Chem audited the data received. They were discussed and agree with NML before necessary actions have been taken. Relevant inconsistencies found are related to:

- Undulating and no realistic digitalized lines that did not reflect the actual conditions or seam contacts;
- Some seams extension that did not respect the principle of layers parallelism;
- Discrepancies observed on some seams between NML's original interpretation (sections in PDF format) and SGS's digitalization. It was highlighted that SGS has not strictly considered the sectional interpretation provided by NML for digitalization but rather has generated its own digitalization using the numerical DB provided. However, it appeared that minor modifications and readjustments were added on the sections where some geological intervals were recoded to have a better consistency of the seam thickness from hole to hole. These modifications were not reflected in the DB transmitted to SGS. Met-Chem has taken these modifications into account in the present resource modeling.

The geological interpretation of first and last cross sections, at the northwest and southeast ends, was extrapolated for an additional 250 m which represents in average half of the drilled sections spacing. In order to complete the surfaces and properly constrain the geology, the surfaces at the ends of cross sections (i.e. up and down dip) were extended at an angle of -6° to cover the lateral extend of the deposit. Thus, the surfaces were extrapolated along the natural slope of seams for an additional 250 m beyond the last drill hole on the down dip slope and until the seam intersected the surface topography on the up dip slope. Figure 14.1 and Figure 14.2 show drill holes location and a typical vertical cross section.

Figure 14.1 – Drill hole Locations

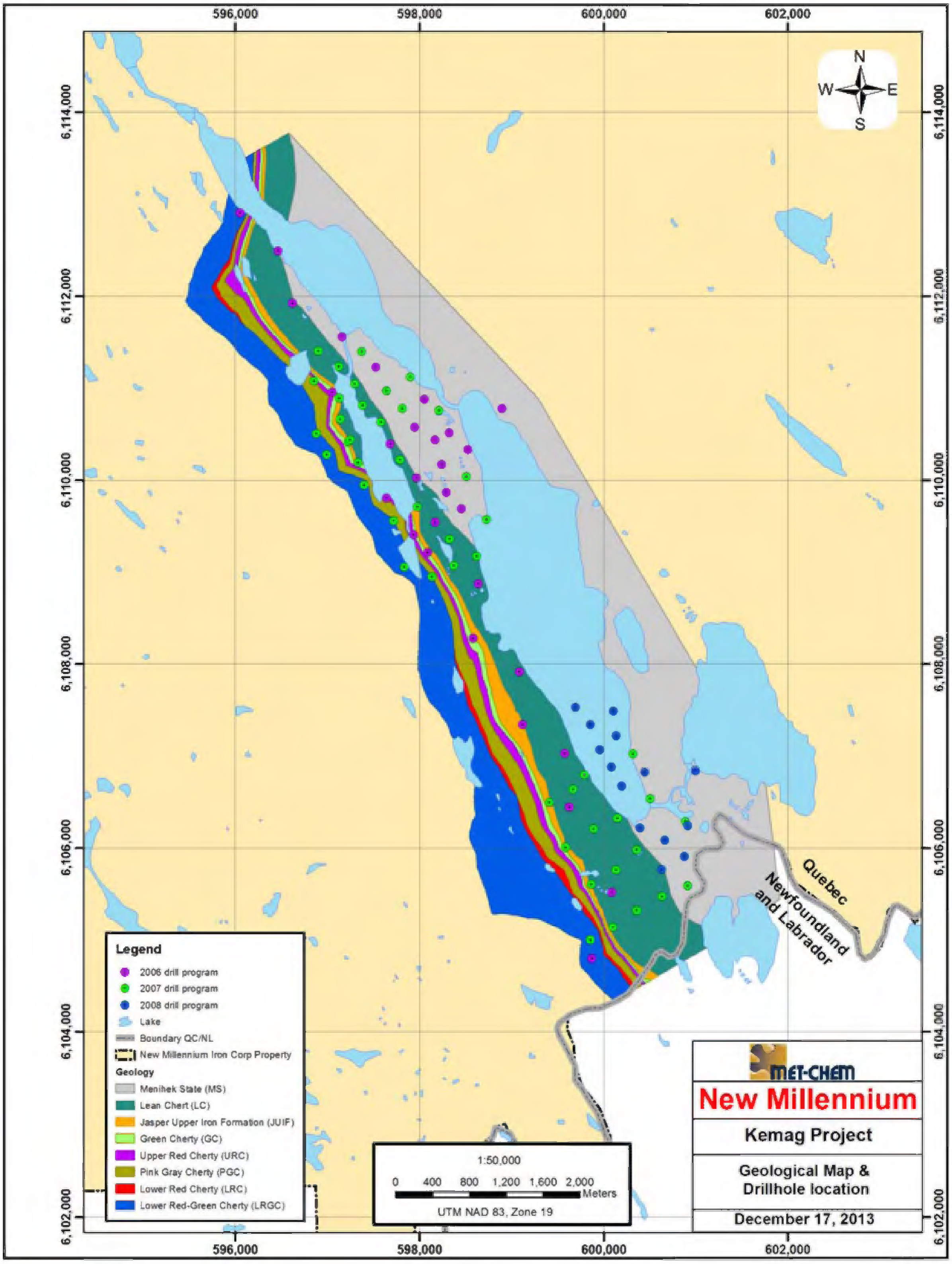
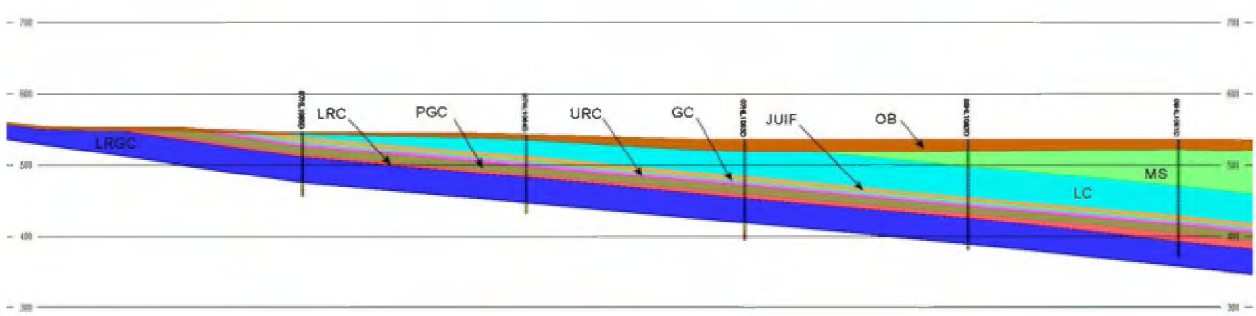


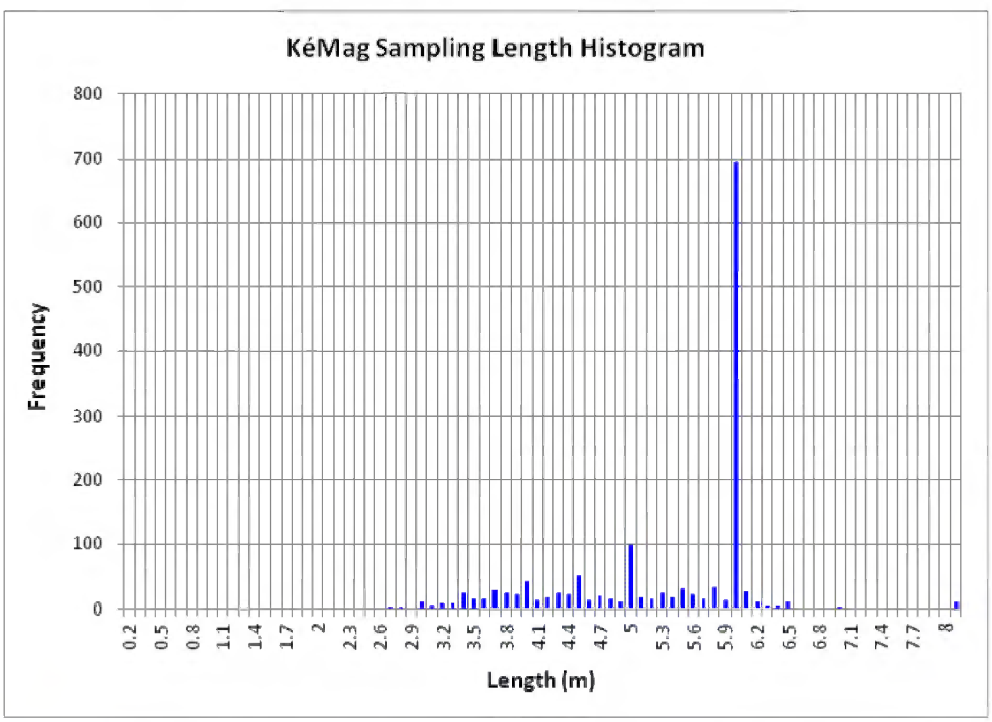
Figure 14.2 – Typical Vertical Cross-Section



14.6 Statistical Analysis of Sampling Length and Compositing

In the resource estimate of 2008, SGS composited the assays at a uniform length of three (3) m prior to blocks interpolation. The reason for this was to find a better match with the thin thickness of some seams such as GC. A statistical analysis of the sampling length at KéMag shows that most samples were taken at a six (6) m interval. The sampling length histogram is presented in Figure 14.3. Met-Chem has a concern with this approach, which consists of compositing samples at a length that is half (½) of the most frequent sampling length. Such an approach leads to splitting most samples into two (2) parts with the un-altered grades from the original sample then being used twice, once for each part of the split sample.

Figure 14.3 – Sampling Length Histogram



This leads to the sample population being duplicated/doubled after having been composited. The repeated values do not provide an accurate reflection of the samples' local and natural variability. A well-accepted industry standard is to consider the statistical mode of sampling intervals as being the best compositing length. Using this statistical mode allows most assayed intervals to remain unchanged after compositing. If the statistical mode is not used, the compositing should at least remain a weighted aggregation process rather than a disaggregation process. For variograms analysis much disaggregation (over-splitting) of assays into small units could lead to a false measure of spatial continuity in the downhole direction and could also affect the nugget effect and short range structures. Met-Chem elected to consider the statistical mode of the sampling length, being six (6) m, as compositing length for the resource estimate of the KéMag Deposit. The GC seam was composited at three (3) m in order to take into account its particularly thin thickness.

14.7 Variogram Modeling

Variograms generated for the KéMag Deposit were of poor quality and one (1) main reason explaining this is the sparser drilling density compare to the LabMag deposit. Even variograms generated for LabMag, on a seam by seam basis, were mostly of poor quality and the best results were obtained using a "One Seam" model. For the present estimate variograms parameters defined for the LabMag deposit on a "One Seam" model were applied to the KéMag Deposit which has similar geology. Such parameters are related to a range of 900 m on the strike direction and a range of 450 m on the dip direction. The third parameter (Z vertical direction) is set equal to the thickness of each seam.

14.8 Density/Specific Gravity

An average specific gravity per seam was previously used for the conversion of volumes into tonnes. The specific gravity of each seam was determined using the pycnometer method and no correction was made to account for rock secondary porosity or permeability. Met-Chem recommended that investigations be undertaken in order to quantify the effect of the secondary porosity (also called permeability or fractures porosity). SNC-Lavalin was requested to carry-out Packer Tests on five (5) selected holes of LabMag. The results show a secondary porosity varying between 0.1 % and 0.3 % in the Menihek Formation, 0.1 % and 0.5 % in the Upper Iron Member (LC, JUIF and GC) of the Sokoman Iron Formation, 0.1 % and 1.2 % in the Middle Iron Member (URC, PGC and LRC) of the Sokoman Iron Formation and between 0.1 % and 0.8 % in the Lower Iron Member (LRGC and LIF) of the Sokoman Iron Formation. The results are tabulated in Table 14.6. It was agreed to increase the values obtained to a single average factor of three percent (3 %) in order to account for the vertical component of the secondary porosity missed during the Packer Tests since all drill holes tested were drilled vertically.

Table 14.6 – Secondary Porosity Estimation of Rock Units for Tested Boreholes

Unit	06HR1266D n (%)	06HR1267D n (%)	06HR1270D n (%)	06HR1275D n (%)	06HR1278D n (%)
MS	0.3-1	0.1-0.2	0.1-0.3	0.1-0.2	0
LC	0.1-2.4	0.1-?	0.4-?	0	0.5
JUIF	1.3-1.8	0	0	0	0
LC-JUIF-GC	0	0.2	0.1	0	0.5
URC-PGC-LRC	0.6	0.1	0.2	1.2	0.1
LRGC	0.2	0.1	0	0.8	0
LRGC-LIF	0	0	0	0	0.7

In addition to the Packer Tests, 167 bulk density measurements were performed on half (½) core in December, 2012 in NML's core warehouse located in Labrador City. The method used is the classical Archimedes' water displacement technique where a sample is weighted in air and in water and the bulk density calculated as the ratio between the weight in air and the difference of the weight in air and the weight in water. This work was recommended by Met-Chem in order to build regression functions, per mineralized seam basis, between density and TotFe%. Such regression functions give a better estimate of the density than using an average density per seam as done in previous estimates. Furthermore, this complete exercise was necessary since the average density calculated for each seam was not the same for LabMag and KéMag.

A preliminary evaluation of the results from the density measurement campaign was to superimpose the pair data, TotFe% against density, for each seam of LabMag and KéMag using different colors. It was noted that there is no systematic bias between the populations. An example is shown for the LC seam in Figure 14.4. The lack of bias for any seam demonstrates that the LabMag and KéMag pair data originated from the same statistical population. Consequently, subsequent seam-wise regression functions should be built by combining the available data for both LabMag and KéMag.

Due to difficulties in retrieving selected boxes containing GC samples in the case of LabMag during work performed on December, 2012, the bulk density measurements were only performed on samples originating from KéMag. Figure 14.5 to Figure 14.11 display scatter plots of density against TotFe% and regression equations set for each seam to convert the volume of each block into tonnage.

Figure 14.4 – Scatter of LabMag and KéMag Pair Data for LC Seam

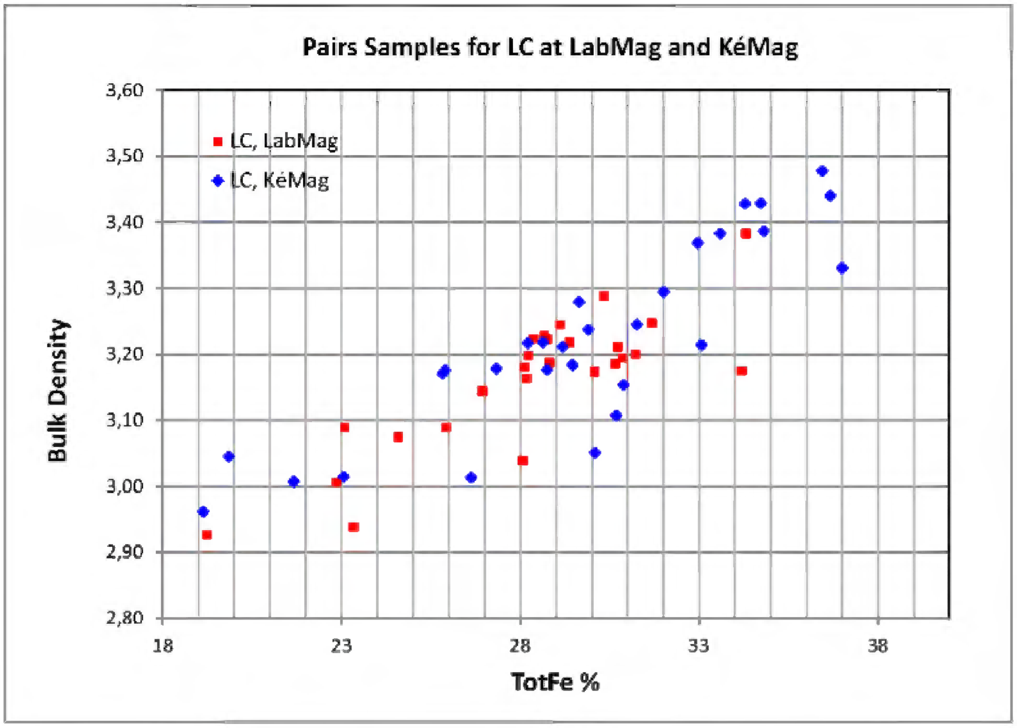


Figure 14.5 – Regression Model for LC Seam

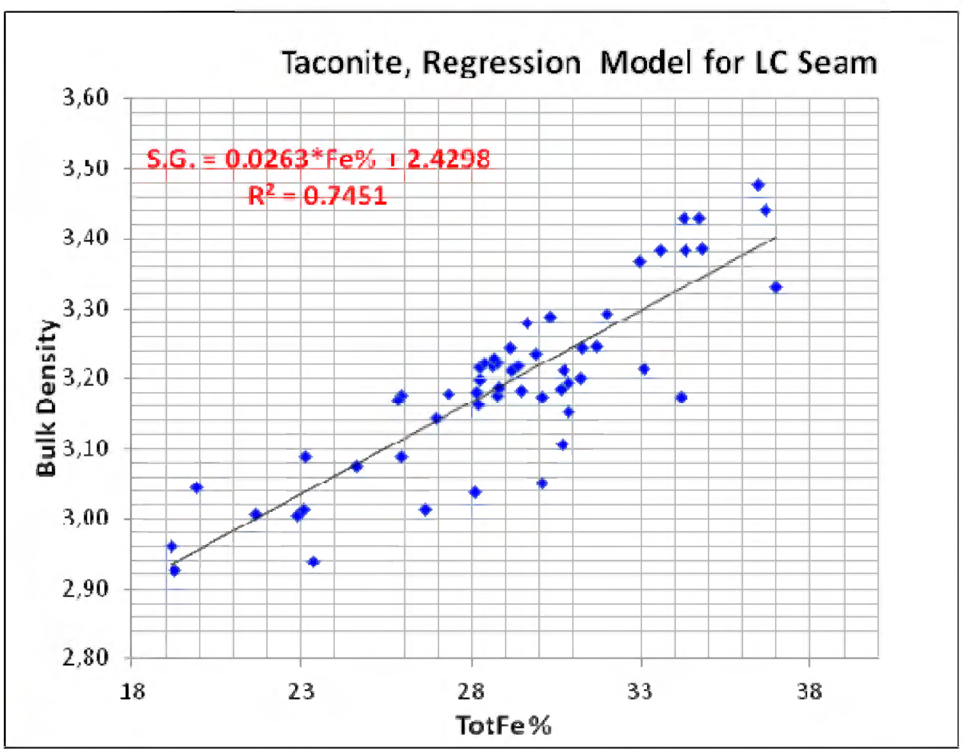


Figure 14.6 – Regression Model for JUIF Seam

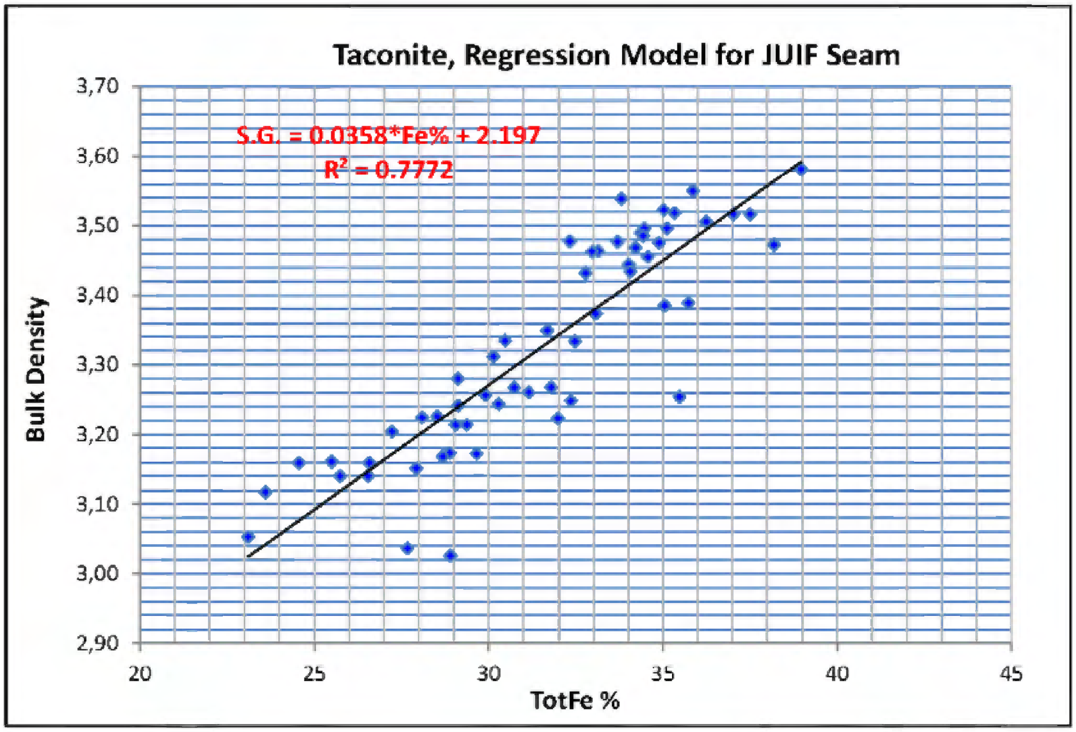


Figure 14.7 – Regression Model for GC Seam

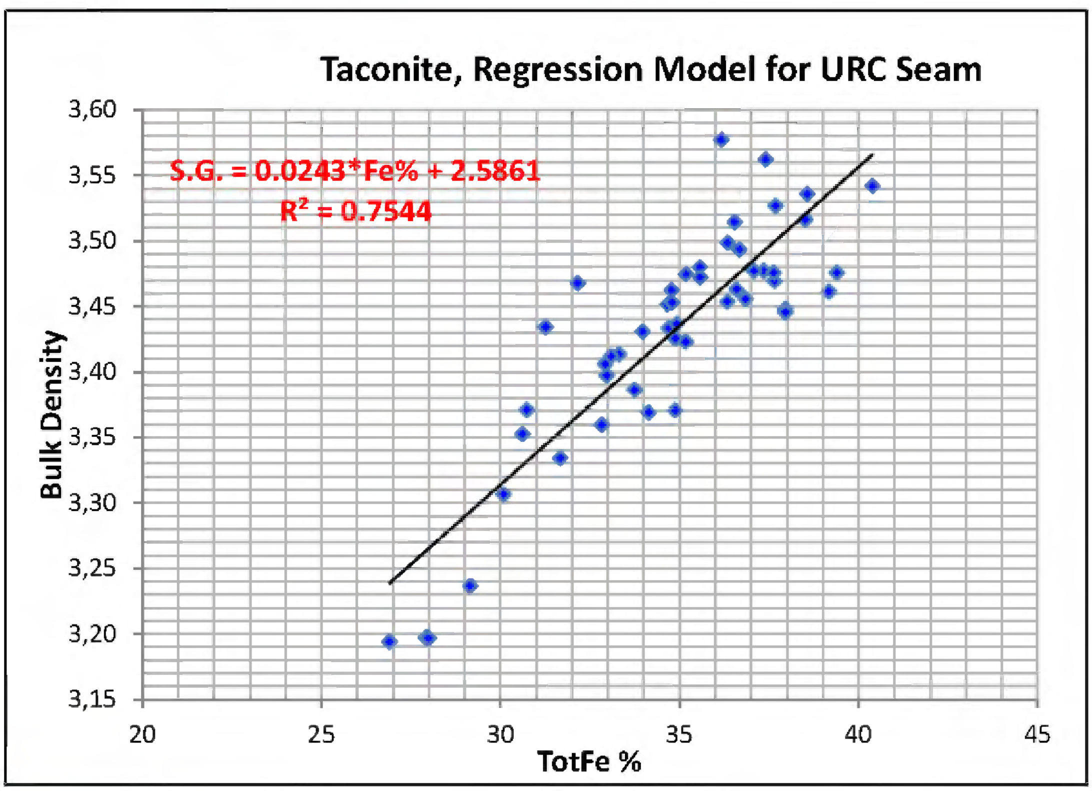


Figure 14.8 – Regression Model for URC Seam

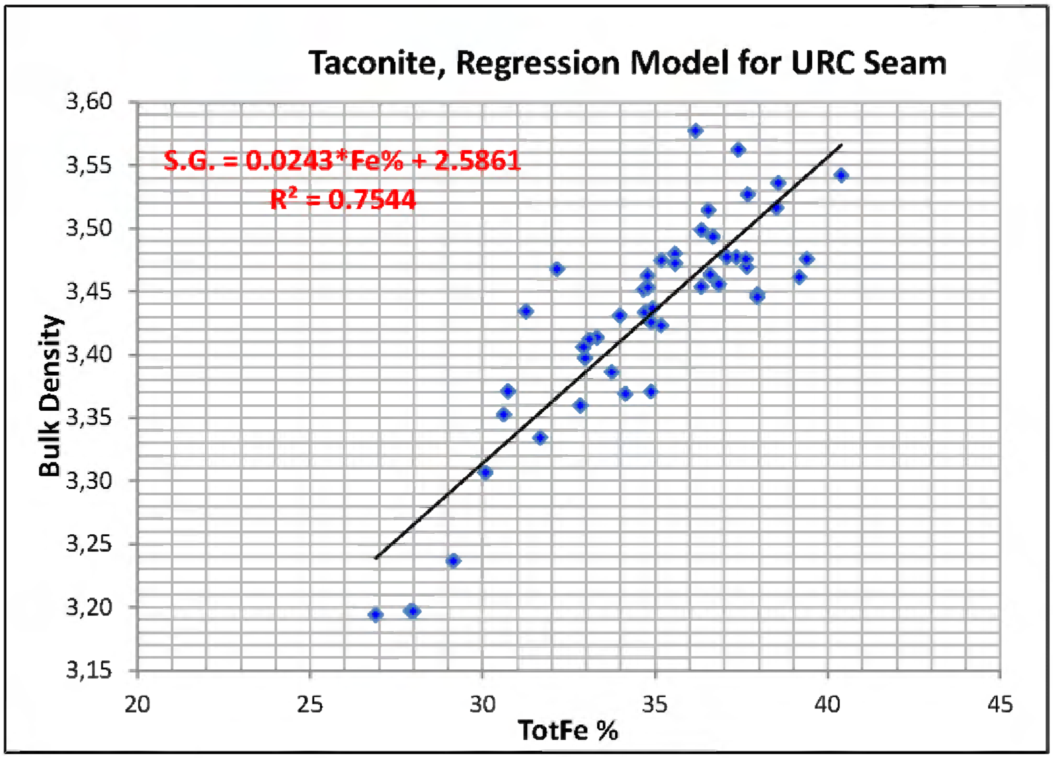


Figure 14.9 – Regression Model for PGC Seam

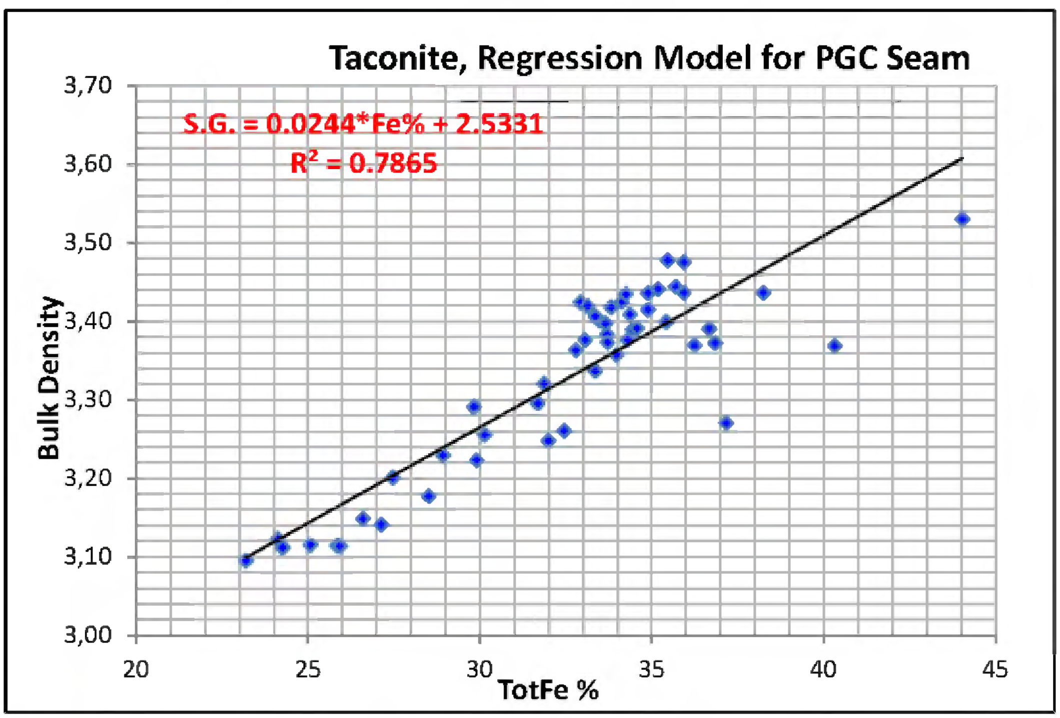


Figure 14.10 – Regression Model for LRC Seam

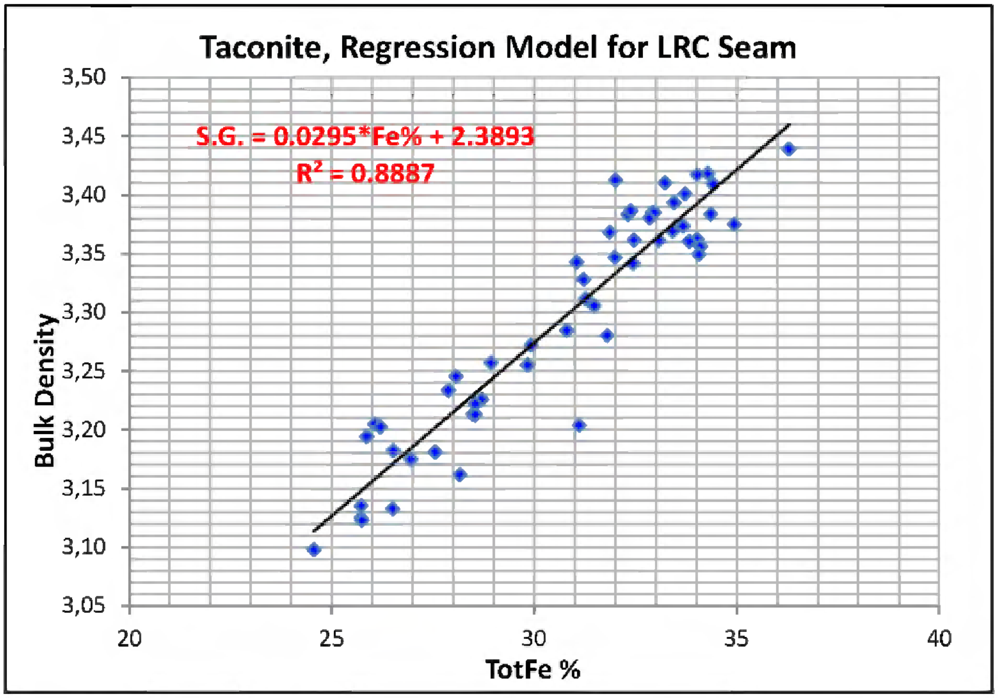
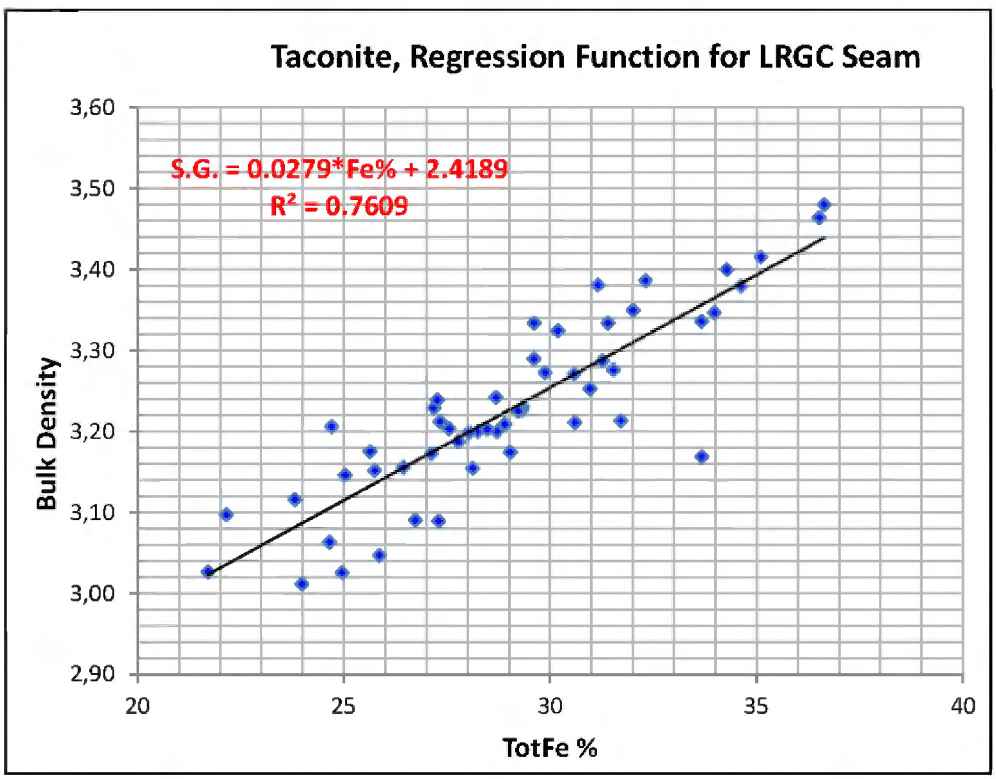


Figure 14.11 – Regression Model for LRGC Seam



14.9 Block Model Setup/Parameters

A new block model was created using the MineSight® software package to generate a grid of regular blocks for the estimation of tonnages and grades. The block size was changed from that used in previous estimate in order to take the drilling density into consideration. Industry standards recommend that blocks size (in the X and Y directions) not be smaller than one half (½) to one fourth (¼) of the drilling spacing. Block size is a particularly sensitive parameter and plays a key role when using geostatistical estimation methods such as kriging. In this case, the kriging variance, which is used for evaluating the interpolation results and for resources classification, is related to the block size and the spatial distribution of holes. The closer the composites used to interpolate a block are, the lower will be the kriging variance. Even for estimation approaches other than geostatistical, such as IDW2, a too small block size would lead to false impressions of the grades of blocks located far from drill hole intercepts. In this case, the representativity of local estimates would be questionable, even if the global statistics appear to reflect assay and composite average statistics.

The average drilling spacing estimated for the KéMag Deposit is 410 m. Taking this average spacing into account, Met-Chem elected to consider blocks having 120 m in the X direction by 120 m in the Y direction. A vertical bench height of 15 m was considered to better fit with the projected type of mining equipment.

Due to the oblique pattern of the mineralization comparatively to the UTM coordinate system Met-Chem kept a rotated model as defined by SGS in previous estimate. The rotation angle is 330° in the manner that the north grid is oriented according the seam strike direction. Block model parameters are summarized in Table 14.7.

Table 14.7 – Block Model Parameters

Direction	Minimum (UTM)	Maximum (UTM)	Block Size	Number of Blocks	Model Origin (UTM)
Easting (X)	593,722.44	602,475.81	120	34	597,646.63
Northing (Y)	6,103,480	6,114,561.5	120	87	6,109,804.5
Elevation (Z)	290	815	15	35	0
Rotation	330° along the UTM North				

14.10 Resource Interpolation

All seams, exclusion made of GC, were composited at a fixed length of six (6) m. The GC seam was composited at a fixed length of three (3) m in order to better match with its thin thickness. Normal down-the-hole compositing approach was used. For the six (6) m composites, all composites shorter than three (3) m were discarded in order to preserve sample representativeness and avoid bias introduced by shorter interval composites. For the three (3) m composites, all composites shorter than 1.5 m were discarded in order to preserve sample representativeness and avoid bias introduced by shorter intervals composites. Each seam was constrained and interpolated separately. Table 14.8 to Table 14.14 display the composite statistics for each seam.

Table 14.8 – Composite Statistics for LC Seam

LC	Average	Median	Mode	St. Dev.	Variance	Range	Minimum	Maximum	Samples
TotFe %	29.58	29.99	30.96	4.93	24.29	34.34	4.21	38.55	366
DTWR %	68.60	70.32	70.90	8.33	69.43	71.97	0.00	71.97	366
Fe % Concentrate	27.08	27.54	27.50	8.09	65.40	45.00	0.00	45.00	366
SiO_2 % Concentrate	2.73	1.92	0.00	1.98	3.94	10.09	0.00	10.09	366

Table 14.9 – Composite Statistics for JUIF Seam

JUIF	Average	Median	Mode	St. Dev.	Variance	Range	Minimum	Maximum	Samples
TotFe %	33.97	34.86	33.07	3.84	14.77	23.77	14.41	38.18	74
DTWR %	69.83	70.10	70.52	1.09	1.18	6.97	64.32	71.29	74
Fe % Concentrate	22.33	22.00	13.00	8.49	72.16	30.50	8.00	38.50	74
SiO_2 % Concentrate	2.31	2.08	1.70	1.13	1.28	8.34	1.16	9.50	74

Table 14.10 – Composite (3 m) Statistics for GC Seam

GC	Average	Median	Mode	St. Dev.	Variance	Range	Minimum	Maximum	Samples
TotFe %	20.84	19.68	18.04	4.99	24.86	24.63	12.19	36.82	103
DTWR %	70.00	70.16	69.89	0.91	0.83	4.90	66.70	71.60	103
Fe % Concentrate	12.80	10.50	10.50	6.60	43.51	33.50	2.50	36.00	103
SiO_2 % Concentrate	1.80	1.58	1.44	0.64	0.41	4.30	0.00	4.30	103

Table 14.11 – Composite Statistics for URC Seam

URC	Average	Median	Mode	St. Dev.	Variance	Range	Minimum	Maximum	Samples
TotFe %	34.08	34.75	34.78	3.33	11.09	23.01	16.38	39.39	72
DTWR %	70.38	70.50	69.70	1.07	1.14	7.68	64.32	72.00	71
Fe % Concentrate	26.65	26.00	23.50	4.75	22.55	27.50	13.50	41.00	72
SiO_2 % Concentrate	1.88	1.61	1.98	1.11	1.22	9.50	0.00	9.50	71

Table 14.12 – Composite Statistics for PGC Seam

PGC	Average	Median	Mode	St. Dev.	Variance	Range	Minimum	Maximum	Samples
TotFe %	33.25	33.81	33.26	2.99	8.92	27.46	10.16	37.62	154
DTWR %	70.15	70.36	70.80	0.99	0.97	5.89	65.81	71.70	154
Fe % Concentrate	30.43	33.30	38.00	9.05	81.94	38.17	6.16	44.33	154
SiO_2 % Concentrate	2.19	1.95	2.14	1.02	1.04	6.67	0.82	7.49	154

Table 14.13 – Composite Statistics for LRC Seam

LRC	Average	Median	Mode	St. Dev.	Variance	Range	Minimum	Maximum	Samples
TotFe %	33.53	33.45	34.02	1.18	1.38	6.05	31.22	37.27	39
DTWR %	70.74	71.10	71.12	1.87	3.48	11.00	61.30	72.30	39
Fe % Concentrate	19.71	15.58	13.50	9.64	92.99	32.00	7.50	39.50	39
SiO_2 % Concentrate	1.75	1.26	1.20	2.06	4.23	11.81	0.79	12.60	39

Table 14.14 – Composite Statistics for LRGC Seam

LRGC	Average	Median	Mode	St. Dev.	Variance	Range	Minimum	Maximum	Samples
TotFe %	31.49	31.90	31.56	2.81	7.92	18.29	19.06	37.35	429
DTWR %	69.25	70.26	70.30	2.72	7.40	17.71	54.29	72.00	429
Fe % Concentrate	25.18	25.00	27.50	8.18	66.93	39.43	6.50	45.93	429
SiO_2 % Concentrate	3.02	2.02	1.88	2.79	7.78	19.69	0.78	20.47	429

The resource interpolation was performed using the IDW method at a power of 2 (IDW2). In Met-Chem's opinion, this method gives similar results to those of geostatistical approach, such as Ordinary Kriging ("**OK**"), in the case of uniform and relatively continuous geological formations such as BIF. In the case of IDW, each weighting factor is inversely proportional to the distance to the center of the block and the composite considered. Met-Chem has also taken into account the anisotropy of the mineralization, which is defined in variograms analysis.

A second interpolation was also performed using OK in order to be compared with the results provided by IDW2 method. In OK approach a block is estimated by combining linearly (adding) selected composites that are weighted with factors resulting from its geometrical position and the quality of the variogram, range and nugget, in the considered direction. Both estimation approaches gave similar results. Three (3) interpolation passes were used in the resource estimation. The basis search ellipse was the kept the same during the first and second passes and only the minimum number of composites to interpolate a block was reduced from nine (9) to six (6) in the case of the second pass.

The maximum number of composites to interpolate a block was set at 15 while the maximum number of composites belonging to a single hole was fixed at three (3). This ensures that at least three (3) holes are required to allow a block to be interpolated during the first pass while at least two (2) holes are necessary for interpolating a block in the second pass. In the third pass, the basis search ellipse was relaxed using a multiplicative factor of 1.5.

The minimum number of composites is set at three (3) while the maximum number of composites to interpolate a block and the maximum number of composites allowed for a single hole were kept same as in the first and second passes. Consequently, at least one (1) hole is required for interpolation during the third pass. An octant search method, where the maximum number of composites per octant was set equal to four (4), was used as declustering method. Interpolation parameters are summarized in Table 14.15.

Table 14.15 – Interpolation Parameters for LabMag

Items	Description		
Grade Interpolation Method	Ordinary IDW2, seam by seam interpolation		
Composites	By fixed length of six (6) m [three (3) m for GC]		
Search Method : Octant	Maximum of four (4) composites per Octant		
Ellipse Orientation	Azimuth 315°, dip -6°		
Resource Category	**Pass 1**	**Pass 2**	**Pass 3**
Minimum Number of Composites for a Block	9	6	3
Maximum Number of Composites for a Block	15	15	15
Maximum Number of Composites per Hole	3	3	3
Ellipse Size Along Strike (N315°)	900 m	900 m	1,350 m
Ellipse Size Across Strike (N45°)	450 m	450 m	675 m
Ellipse Size on the Minor Axis (-90°)	Seam Thickness	Seam Thickness	Seam Thickness

14.11 Estimates Validation

14.11.1 Visual Comparison

Estimates blocks were compared with composite and raw assay grade, on section, plan and 3D basis, as first step of the mineral resource validation. The correlation was good and no evident discrepancies were found. Blocks interpolated were well constrained between the top and bottom of each geological unit. The search ellipse was also well oriented blocks grades pattern follows the directions of best continuity namely the strike and dip direction.

14.11.2 Descriptive Statistics

Met-Chem generated basis descriptive statistics based on the whole raw of assays, composites and interpolated blocks in order to validate the soundness of the Mineral Resources estimates. In this case, no cut-off was applied to the blocks model. The results are presented in Table 14.16. They show that statistics of assays and composites are well repeated in the block model after interpolation. However it appears that DTWR % and SiO_2 %c averages for blocks are reasonable but slightly lower than the averages of assays and composites. Both quality elements show pretty high standard deviation and coefficient of variation. This high scattering has somehow an impact on the quality of the estimate. However these differences are not considered to be significatives.

Table 14.16 – Descriptive Statistics for Comparison of Assays, Composites and Block Grades

		TotFe (%)	DTWR (%)	Concentrate Fe (%)	Concentrate SiO_2 (%)
Assays	W. Average	31.04	25.71	69.63	2.64
	St. Dev.	4.97	9.43	4.86	2.27
	CV	0.16	0.37	0.07	0.88
	Samples	1,300	1,300	1,299	1,299
Composites	Average	30.93	25.59	69.42	2.57
	St. Dev.	4.66	8.81	4.76	2.09
	CV	0.15	0.34	0.07	0.81
	Samples	1,315	1,315	1,314	1,314
Blocks	Average	30.59	24.27	69.61	2.39
	St. Dev.	4.74	7.63	2.26	1.40
	CV	0.16	0.31	0.03	0.58
	Samples	15,983	15,983	15,983	15,983

14.12 Resource Classification

Mineral Resource is sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories. Two (2) key parameters, that are geological continuity and grades continuity, play a key role in increasing the confidence level. They are generally directly related to the drilling density and areas more densely drilled are usually better known and understood that areas having a sparser drilling.

The methodology used by SGS for classifying the Mineral Resources estimated in 2008 is based on a visual appreciation of the drilling density, geological continuity and iron evidence on mapped outcrops rather than on an automatic and computerized classification approach that are now more widely used. Met-Chem traditionally used automatic and computerized approach to classify blocks that have been interpolated. In this exercise, a rigorous analysis is first performed upstream to make sure that all geology related works completed in previous steps of the project development, such as holes location/surveying and the QA/QC, with respect to industry standards. If it is the case, a resource category is automatically allocated to each block after its interpolation. The resource category is dependent on the following:

- The search ellipse size (or a defined ratio of this size) which determine the area of search for selecting the composites;
- The pass used to interpolate the block which determine the minimum number of holes involved and then the drilling density around that block.

The search ellipse size, determined by geostatistical analysis, plays a key role for the use of an automatic computerized classification approach. However, the resource interpolation of the KéMag Deposit was performed on a seam by seam basis, whereas the geostatistical analysis was made on a "One Seam" basis. As previously mentioned, geostatistical analysis based on "One Seam" was done because most of the variograms generated on a single seam basis, except for LRGC which is the thicker layer, were erratic with extreme nugget effects and no evident structures. Taking this fact into account, Met-Chem elected to keep the more conservative resource classification scheme as defined by SGS instead using the large search ellipse developed using variograms defined on the basis of a "One Seam" model. Figure 14.12 displays a plan view of resource zones while Figure 14.13 presents a typical cross section with categorized resources.

14.13 Mineral Resource Statement

Mineral Resources are stated using a DTWR of 18 %. Table 14.17 presents the cumulative resources while Table 14.18 presents the Mineral Resources on a seam basis.

Met-Chem is unaware of any legal, political, environmental, or other risks that could materially affect the potential development of the Mineral Resources.

Due to the uncertainty attached to Inferred Mineral Resources, it cannot be assumed that all or part of an Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration. Mineral Resources that are not Mineral Reserves do not demonstrate economic viability.

Figure 14.12 – Plan View of Resource Zones

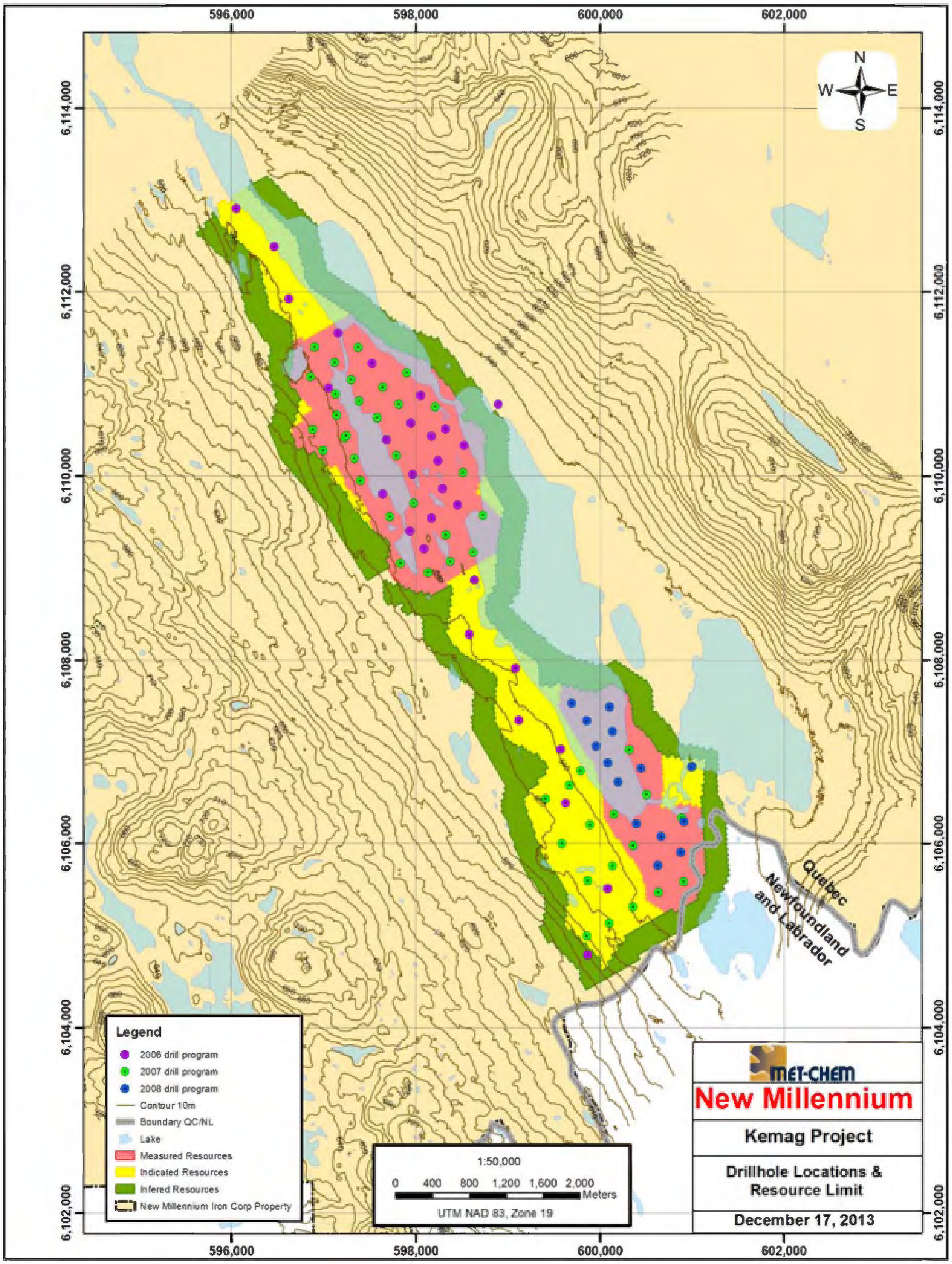


Figure 14.13 – Typical Cross-Section with Categorized Resources

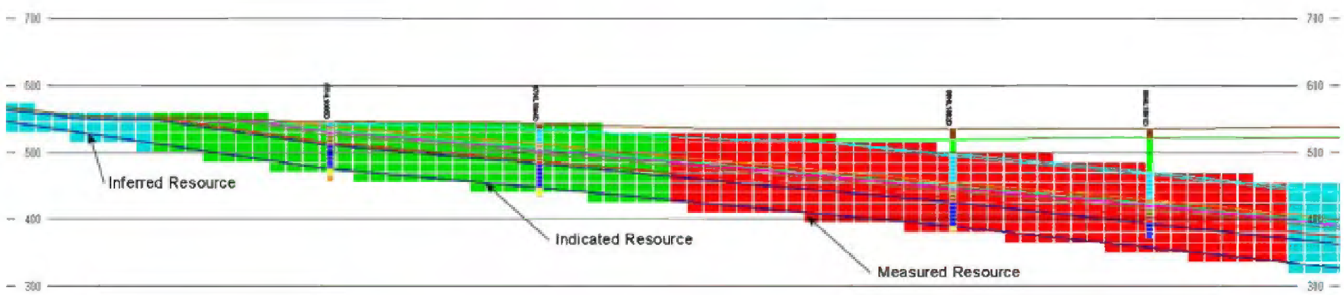


Table 14.17 – Cumulative Mineral Resources – In Situ Grades

Resource by Category	Tonnage (Mt)	TotFe (%)	DTWR (%)	Concentrate	
				Fe (%)	SiO_2 (%)
Measured	1,506.87	31.45	26.97	69.69	2.56
Indicated	876.24	31.95	27.32	69.83	2.51
Measured + Indicated	2,383.11	31.63	27.10	69.74	2.54
Inferred	1,006.84	31.56	26.97	69.31	2.65

Table 14.18 – Mineral Resources by Seam Basis

	Seam	Tonnage (Mt)	TotFe (%)	DTWR (%)	Concentrate	
					Fe (%)	SiO_2 (%)
Measured	LC	529.11	30.17	28.10	69.82	2.38
	JUIF	60.58	33.54	28.02	69.39	2.75
	GC	3.29	22.16	20.32	69.57	2.31
	URC	84.42	34.20	27.56	70.43	1.82
	PGC	222.24	33.33	31.20	70.25	2.08
	LRC	7.74	33.32	20.34	71.07	1.27
	LRGC	599.49	31.31	24.33	69.27	3.00
	Total	**1,506.87**	**31.45**	**26.97**	**69.69**	**2.56**

	Seam	Tonnage (Mt)	TotFe (%)	DTWR (%)	Concentrate	
					Fe (%)	SiO_2 (%)
indicated	LC	166.36	31.14	29.55	70.32	2.22
	JUIF	46.35	32.75	24.33	69.70	2.47
	GC	2.85	26.25	20.34	69.55	2.72
	URC	59.86	33.79	25.43	70.30	1.87
	PGC	126.28	33.32	29.40	69.80	2.58
	LRC	31.00	34.03	26.54	71.12	1.41
	LRGC	443.54	31.42	26.56	69.51	2.77
	Total	**876.24**	**31.95**	**27.32**	**69.83**	**2.51**
Measured + Indicated	LC	695.47	30.40	28.45	69.94	2.34
	JUIF	106.93	33.20	26.42	69.52	2.63
	GC	6.14	24.06	20.33	69.56	2.50
	URC	144.28	34.03	26.68	70.38	1.84
	PGC	348.52	33.33	30.55	70.09	2.26
	LRC	38.74	33.89	25.30	71.11	1.38
	LRGC	1,043.03	31.36	25.28	69.37	2.90
	Total	**2,383.11**	**31.63**	**27.10**	**69.74**	**2.54**
Inferred	LC	298.63	30.02	26.76	68.76	2.57
	JUIF	37.54	34.02	27.86	69.57	2.51
	GC	0.07	24.79	18.56	69.6	2.5
	URC	56.87	34.22	27.14	70.49	1.71
	PGC	130.38	33.46	31.03	70.06	2.25
	LRC	8.81	33.46	23.54	70.98	1.44
	LRGC	474.54	31.47	25.96	69.26	2.96
	Total	**1,006.84**	**31.56**	**26.97**	**69.31**	**2.65**

15.0 MINERAL RESERVE ESTIMATES

The Mineral Reserves for the KéMag deposit were prepared by Jeffrey Cassoff, Eng., Lead Mining Engineer with Met-Chem Canada Inc. and Qualified Person. The Mineral Reserves have been developed using best practices in accordance with CIM guidelines and NI 43-101 reporting.

The effective date of the Mineral Reserve estimate is December 4th, 2012.

The Mineral Reserves were derived from the Mineral Resource block model that was presented in Section 14.0. The Mineral Reserves are the portion of the Measured and Indicated Mineral Resources that have been identified as being economically extractable and which incorporate mining loses and the addition of waste dilution. The Mineral Reserves form the basis for the mine plan presented in Section 16.0.

Table 15.1 presents the Proven and Probable Mineral Reserves for the KéMag deposit. The Mineral Reserves have been categorized into two (2) classes which are based on the following cut-off parameters:

- Class 1 – DTWR ≥ 19.0 % and SiO_2 grade of the concentrate ≤ 4.2 %;
- Class 2 – Remaining mineralized material.

Table 15.1 – Mineral Reserves

Category	Tonnage (Mt)	DTWR (%)	TotFe (%)	Concentrate	
				Fe (%)	SiO_2 (%)
Class 1					
Proven	1,172	27.0	31.2	69.8	2.2
Probable	718	26.9	31.4	70.1	2.1
Proven & Probable	**1,891**	**27.0**	**31.3**	**69.9**	**2.2**
Class 2					
Proven	275	20.0	29.5	64.8	3.5
Probable	218	19.1	26.4	64.8	3.7
Proven & Probable	**493**	**19.6**	**28.1**	**64.8**	**3.6**
Total					
Proven	1,448	25.6	30.9	68.9	2.5
Probable	936	25.1	30.3	68.9	2.5
Proven & Probable	**2,384**	**25.4**	**30.6**	**68.9**	**2.5**

The totals may not add up due to rounding errors.

15.1 Geological Information

The following Section discusses the geological information that was used for the mine plan and Mineral Reserve estimate. This information includes the geological block model, the topographic and lithological surfaces, the iron formation limit and the material properties for ore, waste and overburden.

The mine planning work carried out for the Feasibility Study was done using MineSight® Version 7.05. MineSight® is a commercially available mine planning software that has been used by Met-Chem for over 25 years.

15.1.1 Resource Block Model

The mine plan that was carried out for the KéMag deposit is based on the 3-dimensional geological block model that was presented in Section 14.0. Each block in the model is 25 m wide, 50 m long and 15 m high. The model is rotated 314°.

Since each block in the model can be intersected by several rock types, the percent of the rock type as well as the grade items associated with that rock type are stored in each block. The following items that are included in the model were used for the purposes of mine planning:

- Percent of rock type in block;
- DTWR by rock type;
- Percent TotFe by rock type;
- Percent of Fe in the Davis Tube concentrate by rock type;
- Percent of SiO_2 in the Davis Tube concentrate by rock type;
- Density by rock type;
- Resource classification of the block (Measured, Indicated and Inferred).

15.1.2 Topographic Surface

The mine design for the Feasibility Study was carried out using a topographic surface based on a two (2) m contour interval that was supplied by NML/TS. This surface covers the pit area as well as most of the dump footprints. For mine infrastructure outside of this area, a topographic surface based on 50 ft contour intervals was used. This surface was also supplied by NML/TS.

The two (2) m contour surface is based on the digital elevation model ("**DEM**") acquired by NML in 2011. This DEM was created with elevation readings done at five (5) m spacing, where each reading had a vertical error of ± 1 m. The elevation readings were achieved using an automated process run on high-resolution color air photos acquired by NML in 2008.

The 50 ft contours are part of a federal database provided by Natural Resources Canada. This product is based largely on the information shown on 1:50,000 topographic maps and, where available, more up-to-date information.

15.1.3 Lithological Surfaces

The surfaces representing the lithological contacts between the different rock types were supplied by NML/TS. Met-Chem verified these surfaces with the contacts in the drill hole logs. Included in these surfaces was the contact between the overburden and Menihek Shale. Overburden is defined as loose sand and gravels that can typically be excavated without the need for drilling and blasting.

Since the geological block model focuses only on the mineralized blocks, Met-Chem added the percent of Menihek Shale and overburden to each block based on the topographic and lithological surfaces.

15.1.4 Iron Formation Limit

As the iron formation extends beyond the area that was drilled during exploration, it is important to ensure that no permanent infrastructure is placed on top of potential Mineral Resources. NML/TS supplied a boundary line on the west side of the KéMag deposit that represents where the iron formation daylights at surface. This boundary line is based on a projection of the geological interpretation. The plant site, tailings facilities, dumps and stockpiles were intentionally placed outside of the iron formation limit for the purposes of the Feasibility Study.

Met-Chem recommends that further geological work be carried out to better define the iron formation limit. Additional drilling or trenching in this area will either increase the resource base or allow for the surface infrastructure to be designed closer to the pit crest.

15.1.5 Material Properties

The material properties for the different rock types are outlined below. These properties are important in estimating the Mineral Reserves, the equipment fleet requirements as well as the dump and stockpile design capacities.

a) Density

As was discussed in Section 14.0 of this Report, the density for each block within the iron formation is a function of the grade of iron. Table 15.2 presents the average in-situ dry density for each rock type in the block model. A density of 2.0 t/m^3 was used for the overburden and 2.94 t/m^3 for the Menihek Shale ("**MS**"). Both of these numbers are consistent with similar projects in the region.

Table 15.2 – Density by Rock Type

Rock Type	In-Situ Dry Density (t/m^3)
Overburden	2.00
Menihek Shale	2.94
LC	3.23
JUIF	3.38
GC	3.09
URC	3.41
PGC	3.35
LRC	3.39
LRGC	3.29

b) Swell Factor

The swell factor reflects the increase in volume of material from its in-situ state to after it is blasted and loaded into the haul trucks. A swell factor of 30 % was used for the Feasibility Study, which is a typical value used for open pit hard rock mines.

c) Moisture Content

The moisture content reflects the amount of water that is present within the rock formation. It affects the estimation of haul truck requirements and must be considered during the payload calculations. The moisture content is also an important factor for the process water balance.

Since the Mineral Reserves are estimated using the dry density, they are not affected by the moisture content value.

A moisture content of two (2) % was used for the Feasibility Study. This value is typical for similar projects in the region.

Met-Chem recommends that measurements be conducted to better estimate the density of the overburden and Menihek Shale as well as the swell factor and moisture content of all material types. This will lead to a more accurate determination of the mine equipment requirements and dump and stockpile designs.

15.2 Pit Optimization

The first step in the Mineral Reserve estimate is to carry out a pit optimization analysis. The pit optimization analysis uses economic criteria to determine the cut-off grade and to what extent the deposit can be mined profitably. The pit optimization analysis assumes that the capital for the Project has been spent and analyzes the orebody based on operating costs and revenue.

The pit optimization analysis was done using the MS-Economic Planner module of MineSight® Version 7.05. The optimizer uses the 3D Lerchs-Grossmann algorithm to determine the economic pit limits based on input of mining and processing costs and revenue per block. In order to comply with NI 43-101 guidelines regarding the Standards of Disclosure for Mineral Projects, only ore blocks classified in the Measured and Indicated categories are allowed to drive the pit optimizer. Inferred Resource blocks are treated as waste, bearing no economic value.

Table 15.3 presents the parameters that were used for the pit optimization analysis. All figures are in Canadian dollars. The cost parameters that were used are based on the pre-feasibility study with adjustments made by NML and TS from their current financial models. Adjustments were also made based on recommendations from The Boston Consulting Group. During the pit optimization analysis it was assumed that the final product sold will be an iron concentrate. Pelletizing was treated as a value added process. The parameters used are preliminary estimates for developing the economic pit and should not be confused with the operating costs subsequently developed for the Feasibility Study.

Table 15.3 – Pit Optimization Parameters

Item	Value	Units
Mining Cost (Overburden)	1.70	$/t
Mining Cost (Waste)	2.50	$/t
Mining Cost (Ore)	2.50	$/t
Crushing and Concentrating Cost	3.00	$/t
Slurry Transportation Cost	2.00	$/t (concentrate)
Filtration Cost	0.86	$/t (concentrate)
Port and Ship loading Cost	2.00	$/t (concentrate)
Administration Cost	1.50	$/t (concentrate)
Sales Price (Concentrate)	68.41	$/t (concentrate)
Maximum Overall Pit Slope[1]	45	degree

1 – The overall pit slope is used through the overburden, Menihek Shale and the Iron Formation. This slope is discussed in more detail in the following Section on Pit Design.

In order to maximize the profitability of the deposit and to ensure that the concentrator can process the run of mine ore, certain minimum and maximum cut-offs were applied to the resources. The following method was used to determine the cut-offs for the KéMag deposit:

- The intent of the Project is to produce a concentrate with an average SiO_2 value of 2.1 % and a mine life of 25 years. Since there are not enough resources in the KéMag deposit to achieve both of these criteria, the targeted SiO_2 in the concentrate was increased to 2.2 %. In order to achieve this value of SiO_2, a SiO_2 cut-off of $\leq$ 4.2 % was applied to the resource model. Material above this cut-off is classified as "Class 2";
- The pit optimization analysis was then run at varying weight recovery cut-offs to determine the cut-off that results in the maximum net present value ("**NPV**"). A discount rate of 12 % was used.

The tonnages that result from the pit optimization are not Mineral Reserves since they do not account for dilution and ore loss and are not based on an engineered pit design. Table 15.4 presents the pit optimization results at the varying DTWR cut-offs.

Figure 15.1 presents a chart showing the NPV at the varying DTWR cut-offs. Figure 15.2 presents a chart showing the ore tonnage at the varying DTWR cut-offs.

The pit that resulted with the highest NPV is PIT#16 which was run at a DTWR cut-off of 12 %. This pit has an average DTWR of 25.6 %. In order to compare the KéMag Project with the LabMag Project, the concentrators for both Projects have been designed for the same 27 % average DTWR. Therefore, PIT#22 which has an average DTWR of 27 % was selected to be used for the basis of the detailed pit design for KéMag. PIT#22 was run at a cut-off of 19.0 %. The main difference between PIT#16 and PIT#22 is that roughly 300 Mt of Class 1 Reserves are considered as Class 2 Reserves.

In summary, the pit optimization analysis determined that the ultimate pit limits for the KéMag deposit are based on a minimum DTWR cut-off of 19.0 % and a maximum SiO_2 cut-off of 4.2 %. The tonnages that result from this pit are 1,990 Mt of Class 1 material at an average DTWR of 27.0 % and SiO_2 of 2.19 %, 494 Mt of Class 2 material at an average DTWR of 19.1 % and SiO_2 of 3.55 % and 633 Mt of overburden and Menihek Shale.

Table 15.4 – Pit Optimization Results

Pit	DTWR Cut-off (%)	Class 1 Ore			Class 2 Ore			Waste		Strip Ratio	Mine Life[1] (y)	Net Margin (M$)	NPV[2] (M$)
		(Mt)	DTWR (%)	SiO_2 (%)	(Mt)	DTWR (%)	SiO_2 (%)	OB (Mt)	MS (Mt)				
10	0	2,361	25.2	2.19	190	25.0	5.64	216	422	0.4	27.6	21,890	6,310
11	2	2,361	25.2	2.19	191	25.0	5.63	216	422	0.4	27.6	21,890	6,311
12	4	2,360	25.3	2.19	191	24.9	5.61	216	422	0.4	27.6	21,891	6,311
13	6	2,358	25.3	2.19	193	24.7	5.57	216	422	0.4	27.6	21,892	6,312
14	8	2,347	25.4	2.19	204	23.7	5.37	216	422	0.4	27.6	21,881	6,316
15	10	2,325	25.5	2.19	226	22.3	5.09	216	422	0.4	27.5	21,836	6,321
16	12	2,304	25.6	2.19	248	21.3	4.86	216	422	0.4	27.4	21,769	6,322
17	14	2,280	25.8	2.18	272	20.6	4.67	216	422	0.4	27.3	21,659	6,319
18	16	2,234	26.0	2.17	317	19.8	4.37	216	422	0.4	26.9	21,398	6,305
19	17	2,184	26.2	2.17	364	19.4	4.09	216	422	0.5	26.6	21,075	6,283
20	18	2,106	26.5	2.18	425	19.1	3.80	215	422	0.5	25.9	20,540	6,248
21	18.5	2,050	26.7	2.19	461	19.1	3.66	215	420	0.5	25.5	20,147	6,222
22	**19**	**1,990**	**27.0**	**2.19**	**494**	**19.1**	**3.55**	**214**	**419**	**0.6**	**25.0**	**19,720**	**6,195**
23	20	1,871	27.4	2.21	540	19.2	3.43	213	417	0.6	23.9	18,854	6,137
24	21	1,749	27.9	2.22	580	19.3	3.34	213	416	0.7	22.8	17,916	6,062
25	22	1,614	28.4	2.23	591	19.6	3.32	211	412	0.8	21.4	16,853	5,976
26	23	1,465	29.0	2.24	594	19.9	3.30	209	407	0.8	19.9	15,609	5,854
27	24	1,305	29.7	2.23	557	20.2	3.31	205	401	0.9	18.2	14,265	5,709
28	25	1,162	30.3	2.22	564	20.5	3.22	202	392	1.0	16.6	12,907	5,499
29	26	1,031	30.9	2.21	573	20.8	3.12	196	378	1.1	15.0	11,591	5,255
30	27	915	31.5	2.20	578	21.2	3.04	189	367	1.2	13.6	10,369	4,984

1 - The mine life assumes an annual production of 22 Mt of concentrate and accounts for a ramp-up of 60% in Year-1 and 85 % in Year-2.
2 - The NPV is calculated at a discount rate of 12 %.

Figure 15.1 – Pit Optimization Results (NPV)

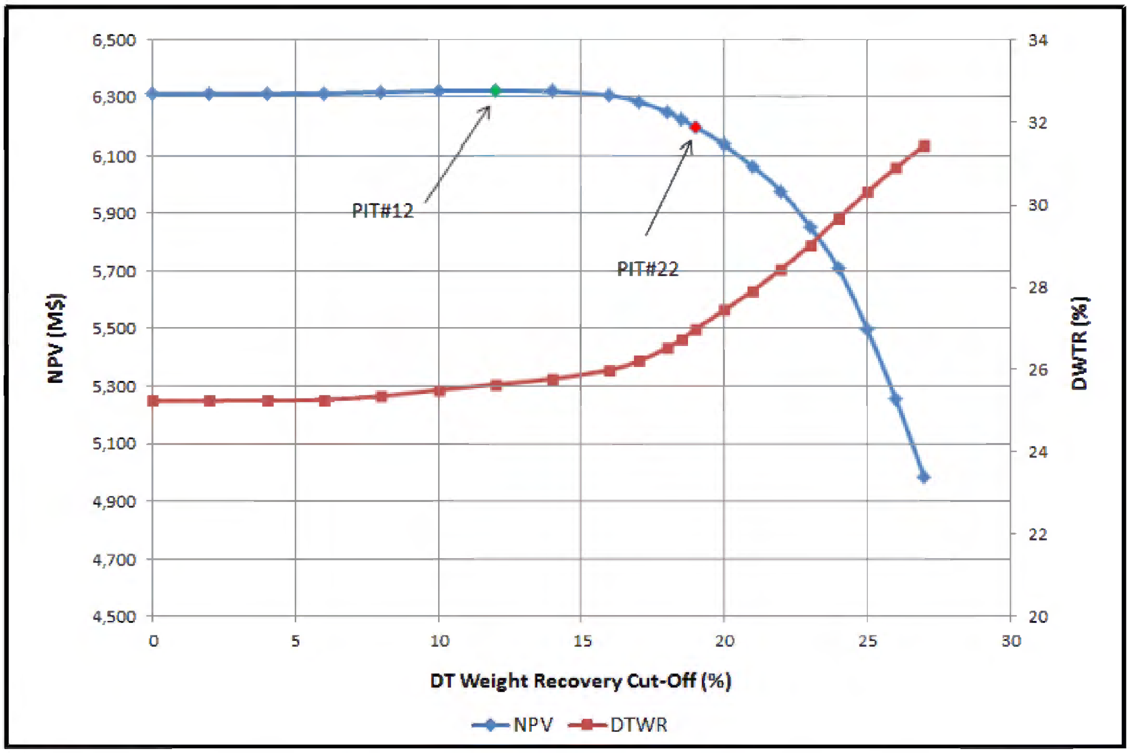
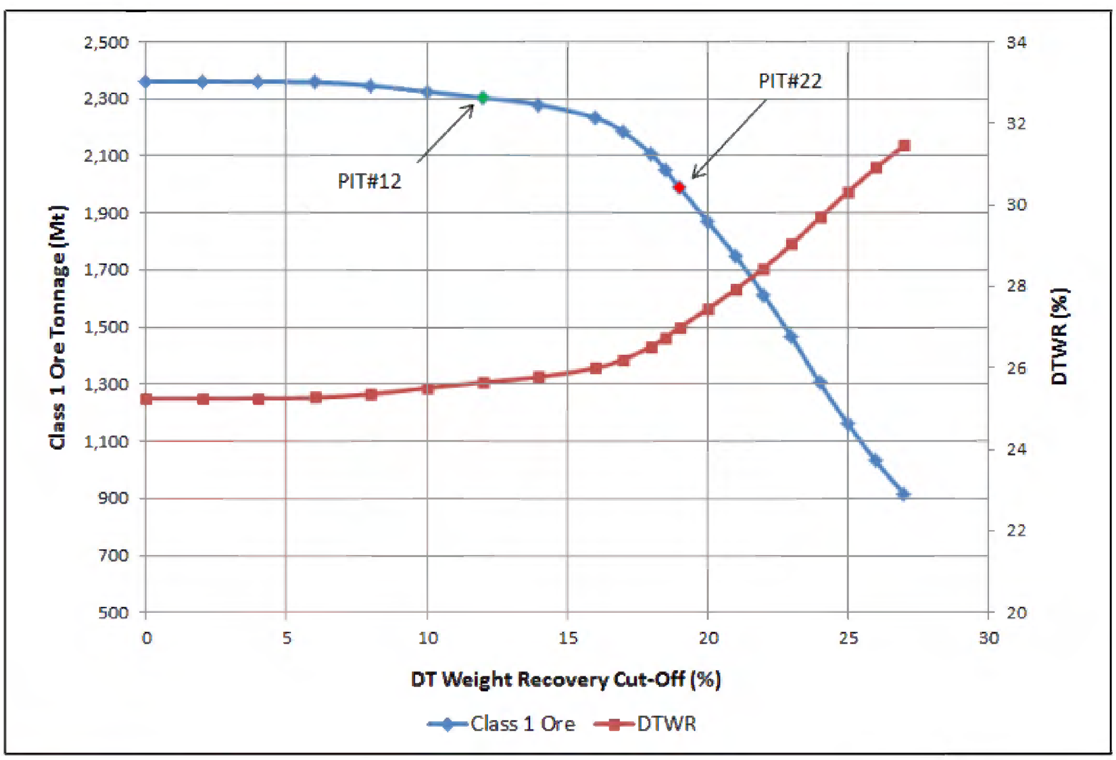


Figure 15.2 – Pit Optimization Results (Tonnage)



15.3 Open Pit Design

The next step in the Mineral Reserve estimation process is to design an operational pit that will form basis of the production plan. This pit design uses the pit shell as a guideline and includes smoothing the pit wall, adding ramps to access the pit bottom and ensuring that the pit can be mined using the initially selected equipment. The following section provides the parameters that were used for the open pit design and presents the results.

15.3.1 Geotechnical Pit Slope Parameters

The KéMag deposit will be developed in hard rock which should lead to competent pit walls. Since no geotechnical work has been carried out to date to engineer the pit wall angles, an overall slope of 45° was selected as a conservative approach for the Feasibility Study. Since the final pit wall does not get established until Year-2 of the operations, the geotechnical analysis to engineer the final pit wall can be postponed to a later date. A change in the pit slope will either increase or decrease the waste stripping within the pit but will not significantly affect the Mineral Reserves. Met-Chem recommends that a geotechnical analysis to determine the final pit wall configuration be carried out before mining begins in this area.

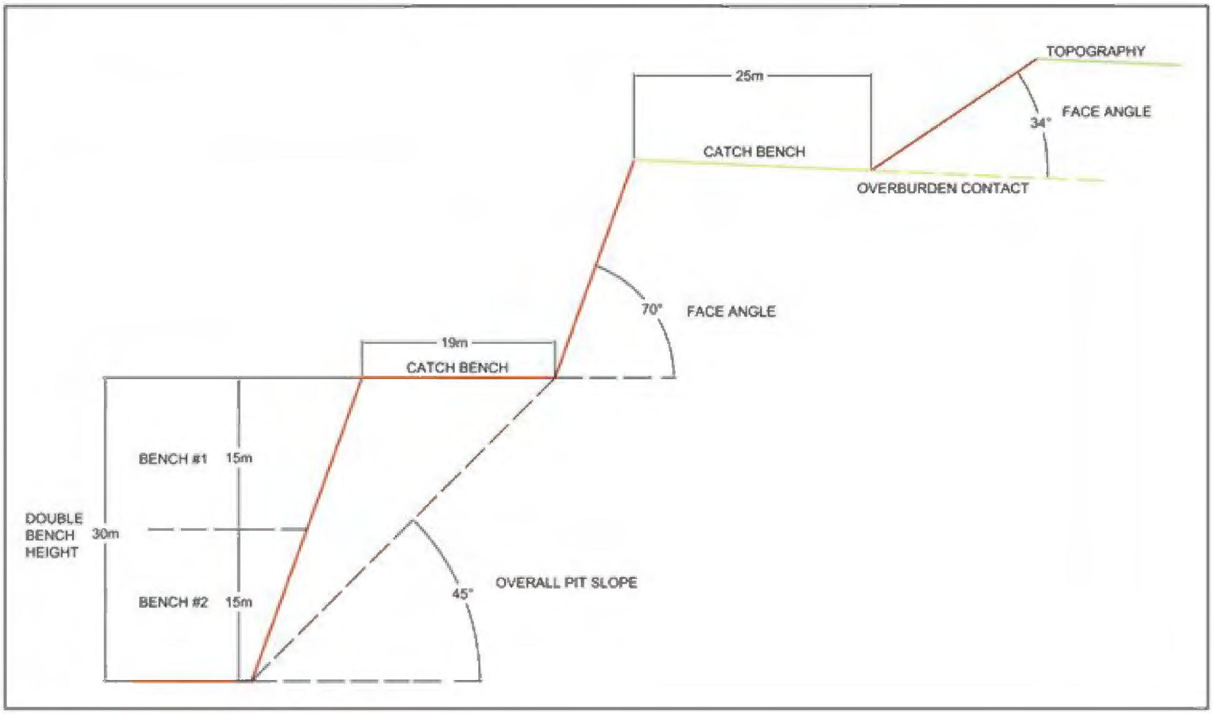
The pit will be mined with 15 m high benches. This bench height is well suited for the size of equipment (shovels and drills) that are planned for the mine. Equipment selection is presented in Section 16.0.

The overall slope of 45° is achieved by incorporating a 19 m catch bench for each 30 m in height [two (2) – 15 m benches], with a face angle of 70°. A width of 19 m for the catch benches is sufficiently wide so that access to these benches for track dozers and pick-up trucks will be maintained at all times.

A slope of 34° has been incorporated through the overburden formation since this material is less competent than the bedrock. A 25 m wide berm has been included at the contact between the overburden and the bedrock for increased stability. The configuration of the final pit wall is presented in Figure 15.3.

The maximum height of the overburden along the final pit wall is 20 m. The maximum overall height of the final wall is 210 m.

Figure 15.3 – Pit Wall Configuration



15.3.2 Haul Road Design

Since the ore body daylights at surface, the ultimate pit does not require the design of a permanent access ramp to the pit bottom. The benches will be mined flat and the pit access will be developed along the floor as the pit wall advances towards the east. The floor of the KéMag deposit dips at approximately six (6)° which translates into a grade of 10.5 %. Although this grade is not optimal for the haul trucks, the truck manufacturers (Liebherr and Caterpillar) have agreed that it is manageable.

Temporary ramps will be required in order to maintain access to the benches in the advancing wall. These ramps will either be cut with the shovels or backfilled with mined out waste rock. The temporary ramps will be built to have a maximum grade of eight (8) %.

A service road for light vehicles has been established around the perimeter of the ultimate pit. This road is designed to be ten (10) m wide and will be used primarily to access the ditch network.

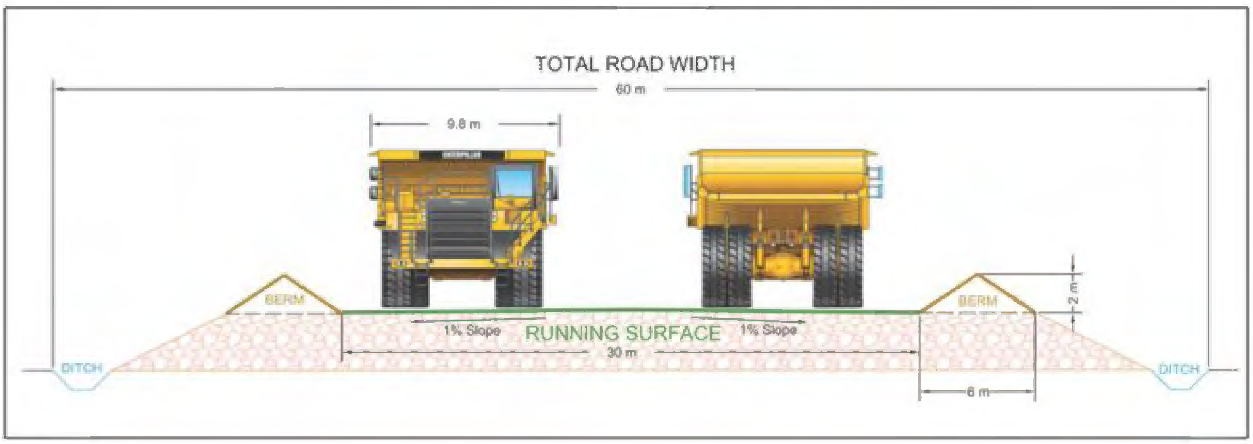
The mine haul roads are designed for a 363-tonne haul truck. For double lane traffic, the industry standard indicates the running surface width to be a minimum of three (3) times the width of the largest truck. The overall width of a 363-tonne haul truck is 9.8 m which results in a running surface of 30 m. The overall width of the haul road must account for safety berms. The following dimensions of the safety berms are based on industry standards:

- The safety berm height should be equal to the radius of the largest truck tire as a minimum, which is two (2) m. The safety berm slopes are 34° and will be built in a triangular shape. The width required for the bottom of each safety berm is six (6) m.

The overall width of the road including safety berms is 42 m. The haul road corridors have been designed for 60 m. This ensures that there is enough room for the ditches and the road base side slopes if fill material is required for construction.

The maximum road grade is eight (8) % and the road design incorporates a crown of one (1) %. The safety berms are interrupted every 25 m in length to allow for water to run-off into the ditches. Figure 15.4 presents a typical section of the haul road.

Figure 15.4 – Haul Road Configuration



15.3.3 Mine Dilution and Ore Loss

For a productive mining operation, it is typical for contamination to occur between the ore and waste at the contact boundary. This is due to the nature of the large size of shovels and the fact that the rock requires blasting. In order to account for this, the following method was used to estimate mining dilution and ore loss for the Feasibility Study.

The following two (2) areas have been identified where mining dilution and ore losses will occur when mining the KéMag deposit:

- Blending of lithological layers within each block in the resource model;
- Dilution and ore losses at the ore/waste contacts.

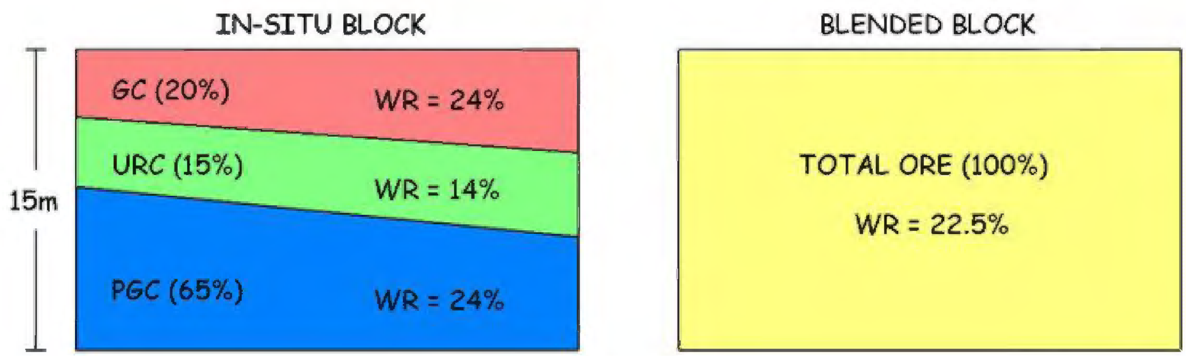
a) Blending of Lithologies

Since the KéMag pit was designed with 15 m high benches that will be drilled, blasted and mined in one (1) pass, it will be difficult to effectively separate the different lithological layers at the shovel face. These layers are often only several metres thick.

To account for this in the mine plan, an average grade was calculated for each block in the resource model based on the individual grades for each lithology within the block. The cut-off grade criteria for DTWR and SiO_2 was then applied on the average grade items for each block to classify it as an ore or waste block.

The blending of lithologies at the shovel face induces a certain amount of dilution since thin zones that would not meet the cut-off criteria on their own are blended with the rest of the block and sent to the crusher as ore. The net result of this type of mining dilution is an increase in ore tonnage with decreased average weight recovery. Figure 15.5 presents a section of a typical block in the resource model. The figure illustrates how the three (3) ore types in the block are diluted to arrive at a weighted average weight recovery of 22.5 %. Prior to dilution, only 85 % of the block meets the cut-off criteria. After dilution is accounted for, the entire block (100 %) is considered as ore.

Figure 15.5 – Blending of Lithologies



b) Mining Dilution at the Ore / Waste Contacts

The second area where mining dilution will occur is at the ore/waste contacts. Due to the fact that the mining operation will incorporate drilling and blasting and that the loading equipment is considerably large, it will be very difficult to perfectly separate the ore and waste at the geological contact. The two (2) main areas where the orebody follows waste contacts are at the top (Overburden and Menihek Shale) and the bottom (Lower Iron Formation):

- Upper Contact (Overburden) – The mining dilution effects from the overburden should be negligible because the overburden is a layer of sand and gravel that does not require blasting and is easily identifiable in the field;
- Upper Contact (Menihek Shale) – The Menihek Shale is not an iron formation and is therefore considered purely as waste. The Menihek Shale is clearly distinguishable from the upper mineralized layers because its color is carbon black. Since this layer may be acid generating, care must be taken

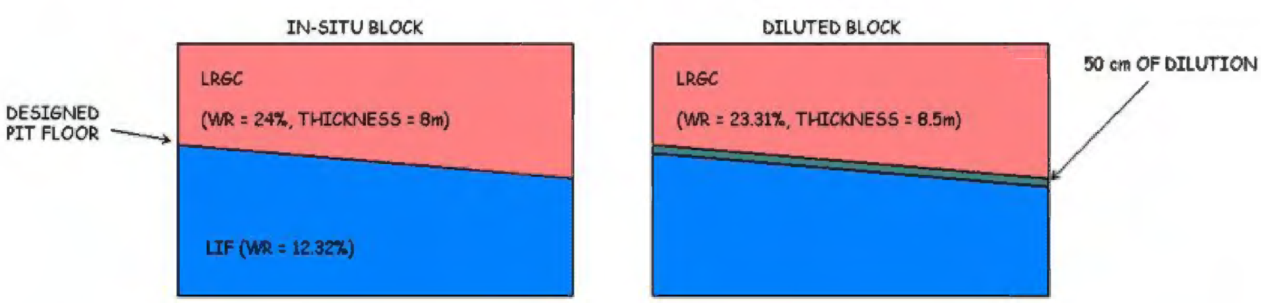
such that the Menihek Shale is not blended with the ore and sent to the plant. In order to avoid dilution, it is assumed that the mining operations will target 0.5 m below the Menihek Shale contact and an ore loss will therefore be incurred;

- Lower Contact (Lower Iron Formation) – The Lower Iron Formation can be visually distinguished from the LRGC layer. The LRGC layer has a visible black magnetite banding while the Lower Iron Formation is lighter in color. However, during blasting, some Lower Iron Formation may mix with the LRGC and be sent to the plant as ore. The Lower Iron Formation contains iron, and does not affect the process in the plant other than the fact that the recovery will be poor. In order to calculate the mining dilution from the Lower Iron Formation, it is assumed that a depth of 0.5 m will be taken from the Lower Iron Formation and sent to the plant. Using the drill hole database, the following grade items were calculated to represent the averages for the Lower Iron Formation:
  - DTWR – 12.32 %;
  - TotFe – 24.53 %;
  - SiO_2 in the concentrate – 2.26 %.

As discussed above, during the mining operation there will be situations when some of the Lower Iron Formation will be sent to the plant as ore. There will also be situations when some of the LRGC will remain in the pit floor. Since it is impossible to predict when each of these scenarios will occur, it was assumed that the tonnage increase as a result of mining dilution will be equal to the tonnage decrease as a result of ore losses.

Figure 15.6 illustrates how the weight recovery is adjusted for a typical ore block along the pit floor in order to account for mining dilution.

Figure 15.6 – Dilution Calculation



15.3.4 Lake Harris

A preliminary evaluation determined that approximately 800 Mt (35 %) of the Measured and Indicated Mineral Resources are contained below Lake Harris. If Lake Harris were to be left intact, the remaining Mineral Resources would only support a ten (10) year mine life which would most likely not support the Project.

In order to mine the resources that lie underneath Lake Harris, a system of dams and ditches will be constructed. Based on the mine plan that is presented in Section 16.0 of this Report, the lake needs to be drained by the start of production.

Firstly, a diversion channel will keep the water from filling lakes Harris and Gillespie. It will also drain the overflow from Boundary Lake and keep runoffs from the hills from reaching the mine pit. Cofferdams and dykes will retain water between the diversion channel and the mine pit. As the mine progresses into Lake Harris in the area that is actually below the water surface, water will be pumped in the main stream and all water from this zone will flow towards the Goodwood River.

15.3.5 Open Pit Design Results

The pit that has been designed for the KéMag deposit is approximately 9.4 km long and 1.5 km wide at surface with a maximum pit depth of 210 m. The total surface area of the pit is roughly 11 km^2. About 90 % of the pit floor follows the top of the Lower Iron Formation while the remaining ten (10) % of the floor is composed of a series of islands. These islands reflect mineralization that does not meet the cut-off criteria and is therefore not worth mining. The islands were designed with an overall slope of 45°. Figure 15.7 presents the final pit design for the KéMag deposit.

In order to access the mine site, a road will be built on the south side of the KéMag deposit along the provincial border. The pit was designed with a minimum offset of 50 m from this access road.

Table 15.5 presents the maximum and average thicknesses of each rock type throughout the KéMag pit.

Figure 15.7 – Pit Layout

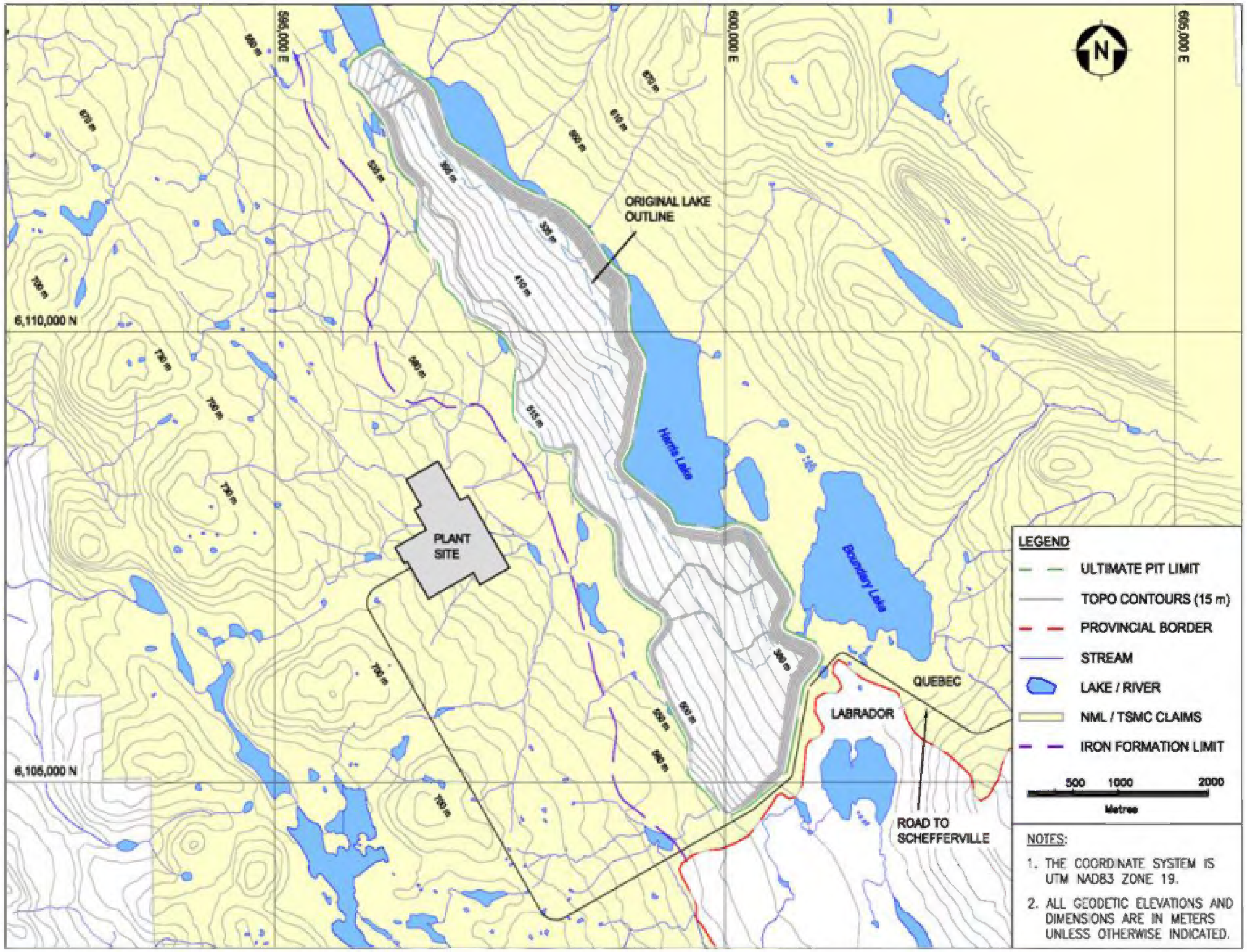


Table 15.5 – Thickness of Rock Types within the Pit

Description	Maximum (m)	Average (m)
Overburden	23.38	10.22
Menihek Shale	96.42	31.98
LC	58.81	42.74
JUIF	14.23	6.10
GC	8.64	4.40
URC	7.68	4.78
PGC	17.34	10.60
LRC	11.50	4.62
LRGC	48.19	28.87
Total Pit[1]	210.59	90.53

1 - The total pit represents the thickest part of the pit as well as the overall average thickness.

The KéMag pit includes 1,172 Mt of Proven Mineral Reserves and 718 Mt of Probable Mineral Reserves for a total of 1,891 Mt. In order to access these reserves, 231 Mt of overburden, 440 Mt of Menihek Shale and 120 Mt of Inferred Mineral Resources must be mined. This total waste quantity of 791 Mt results in a stripping ratio of 0.33 to 1.

The Class 1 Mineral Reserves for KéMag contain an average DTWR of 27.0 %, an average TotFe of 31.3 %, an average Fe in the concentrate of 69.9 % and an average SiO_2 in the concentrate of 2.2 %. The Class 2 Mineral Reserves for KéMag contain an average DTWR of 19.6 %, an average TotFe of 28.1 %, an average Fe in the concentrate of 64.8 % and an average SiO_2 in the concentrate of 3.6 %. These numbers account for the mining dilution and ore loss discussed earlier in this Section of the Report. Even though the mine plan does not demonstrate the processing of the Class 2 material, Met-Chem considers this tonnage as Mineral Reserves since it will be stockpiled close to the plant site for future processing.

Table 15.6 presents the Mineral Reserves for the KéMag deposit by reserve category. Table 15.7 presents the Class 1 Mineral Reserves by rock type and Table 15.8 presents the Class 2 Mineral Reserves by rock type.

Table 15.6 – Mineral Reserves by Category

Category	Tonnage (Mt)	DTWR (%)	TotFe (%)	Concentrate	
				Fe (%)	SiO_2 (%)
Class 1					
Proven	1,172	27.0	31.2	69.8	2.2
Probable	718	26.9	31.4	70.1	2.1
Proven & Probable	**1,891**	**27.0**	**31.3**	**69.9**	**2.2**
Class 2					
Proven	275	20.0	29.5	64.8	3.5
Probable	218	19.1	26.4	64.8	3.7
Proven & Probable	**493**	**19.6**	**28.1**	**64.8**	**3.6**
Total					
Proven	1,448	25.6	30.9	68.9	2.5
Probable	936	25.1	30.3	68.9	2.5
Proven & Probable	**2,384**	**25.4**	**30.6**	**68.9**	**2.5**

The totals may not add up due to rounding errors.

Table 15.7 – Class 1 Mineral Reserves by Rock Type

Category	Tonnage (Mt)	% of Total	DTWR (%)	TotFe (%)	Concentrate	
					Fe (%)	SiO_2 (%)
LC	555	29.4	29.0	30.5	69.7	2.2
JUIF	121	6.4	25.0	33.6	69.8	2.4
GC	87	4.6	12.7	20.5	69.9	1.9
URC	109	5.8	27.6	33.6	70.3	1.9
PGC	276	14.6	31.5	33.4	70.1	2.2
LRC	49	2.6	23.1	33.7	70.9	1.7
LRGC	693	36.7	26.1	31.7	69.9	2.3
Total Class 1 Mineral Reserves	**1,891**	**100**	**27.0**	**31.3**	**69.9**	**2.2**

- The average grades may not add up due to rounding errors.

Table 15.8 – Class 2 Mineral Reserves by Rock Type

Category	Tonnage (Mt)	% of Total	DTWR (%)	TotFe (%)	Concentrate	
					Fe (%)	SiO_2 (%)
LC	218	44.2	16.4	24.1	60.0	3.4
JUIF	37	7.5	16.7	33.6	70.1	2.2
GC	24	4.9	9.7	19.8	70.4	1.6
URC	23	4.8	23.7	34.1	70.4	1.9
PGC	53	10.8	27.9	32.7	69.6	2.7
LRC	23	4.6	17.1	33.2	69.8	2.8
LRGC	115	23.3	24.7	31.6	66.8	5.5
Total Class 1 Mineral Reserves	**493**	**100**	**19.6**	**27.9**	**64.8**	**3.6**

-The average grades may not add up due to rounding errors.

16.0 MINING METHODS

The mining method selected for the Project is conventional truck and shovel. Due to the high tonnages expected to be mined and the relatively long haul distances, the trucks and shovels will be the largest available on the market.

Vegetation and topsoil will be cleared using a mining contractor and be carried out with a fleet of dozers, small excavators and articulated haul trucks ahead of the mining operation. Suitable organic material will be stockpiled for future reclamation use. Overburden will then be stripped using a fleet of excavators and hauled to the overburden dumps. The ore and waste rock will be mined with 15 m high benches, drilled and blasted then loaded with large mining shovels into a fleet of rigid frame trucks which will haul the material either to the waste dumps, the Class 2 ore stockpiles or the primary crushers.

16.1 Geotechnical Pit Slope Parameters

The geotechnical pit slope parameters were presented in Section 15.0.

16.2 Hydrogeology and Hydrology Parameters

Since the mine design for the Feasibility Study was carried out prior to the completion of the fall 2012 hydrogeological field investigation, the mine dewatering calculations and design were done using data that was collected on the neighboring LabMag deposit in 2006. Due to the limited scope of the 2006 hydrogeological study, Met-Chem recommends verifying the mine dewatering calculations from the Feasibility Study with the results from the 2012 study and updating as required.

The five (5) sources of water that affect the mining operation are surface run-off, rainfall, snowmelt, permafrost melt and groundwater. The quantity for each of these sources of water was estimated for each period of the mine plan in order to calculate the dewatering requirements.

16.2.1 Surface Run-off

In order to limit the surface run-off from entering the pit, a perimeter ditch will be established around the limits of the pit to capture the surface water before it enters into the mining area. Water collected in the ditch system will be directed to a system of settling ponds where it will be treated and sampled prior to discharge into the environment. Perimeter ditches will also be established around the dump and stockpile to capture water run-off.

16.2.2 Rainfall and Snowmelt

The amount of rainfall and snowmelt was estimated using historical meteorological data available for the Schefferville area. The rainfall and snowfall quantity averages 90 % of the total mine dewatering requirements.

16.2.3 Groundwater

The amount of groundwater inflow was estimated using data from pumping tests that were carried out as part of the 2006 hydrogeological study. The groundwater quantity averages eight (8) % of the total mine dewatering requirements.

16.2.4 Permafrost Melt

An additional amount of water is expected to enter into the pit when zones of permafrost are encountered. Due to the limited information available on permafrost conditions in the pit, conceptual considerations based on Darcy Law's and the average thickness of the active layer in the area was used to estimate this quantity. The permafrost quantity averages two (2) % of the total mine dewatering requirements.

Water that enters into the pit will be collected in sumps that will be established on the lowest point of the pit floor. This water will be pumped from the sumps to the surface and channeled to settling ponds. More details on the site wide dewatering plan are presented in Section 18.0.

The size of pump selected for the mine dewatering plan has a power rating of 450 kW and can handle flow rates of 900 m^3/h. These pumps will be connected to the mine electrical network. The cost to purchase and operate the pumps as well as piping and other accessories has been included in the mine capital and operating cost estimate.

16.3 Dump and Stockpile Design

The Feasibility Study for the KéMag deposit includes two (2) Class 2 ore stockpiles and three (3) waste rock and overburden dumps. The following Section presents the design parameters that were used for the waste dumps and the ore stockpiles.

In order to comply with mining regulations, organic material within the pit limits, the dump areas and infrastructure footprints must be stripped and stockpiled for future reclamation use. This material known as topsoil will be placed in stockpiles around the Property. Since the topsoil quantities are small and the stockpiles are temporary, they are not shown on any of the site layout drawings. An average topsoil thickness of 0.3 m has been used for volume estimation purposes.

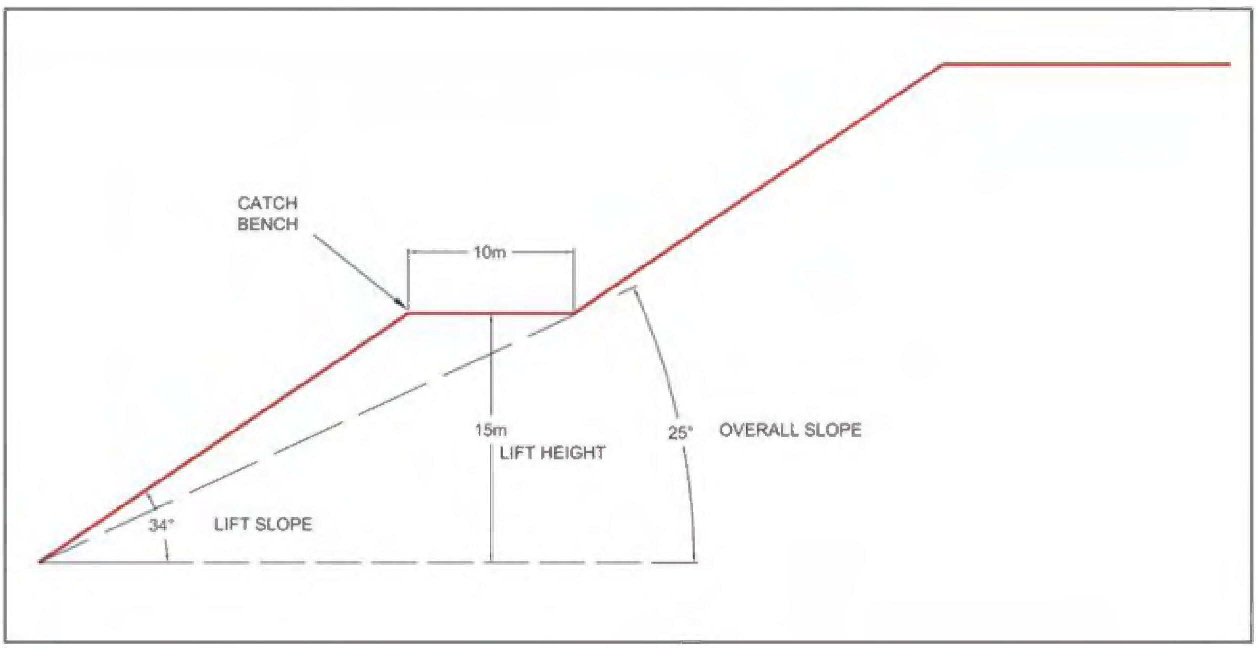
The waste rock and overburden will be managed in the same dump since stockpiling overburden separately would require very shallow slopes due to the nature of the material and would increase the footprint of the altered land use.

The following parameters were used for the design of the waste rock dumps and ore stockpiles. These parameters are based on projects that Met-Chem has completed with similar rock types. Since the dumps and stockpile are very large, Met-Chem recommends that a geotechnical study be carried out to confirm the proposed dump design criteria prior to deposition:

- Swell factor – 30 %;
- Overall slope – 25°;
- Lift slope – 34°;
- Catch Bench – 10 m berm per 15 m lift;
- Maximum dump height – 100 m;
- Setback from pit crests – 100 m;
- Setback from mine haul roads and slurry transport corridor – 100 m;
- Setback from other dumps and stockpiles – 100 m;
- Setback from the provincial boundary – 25 m;
- Setback from the Iron Formation Limit – 150 m.

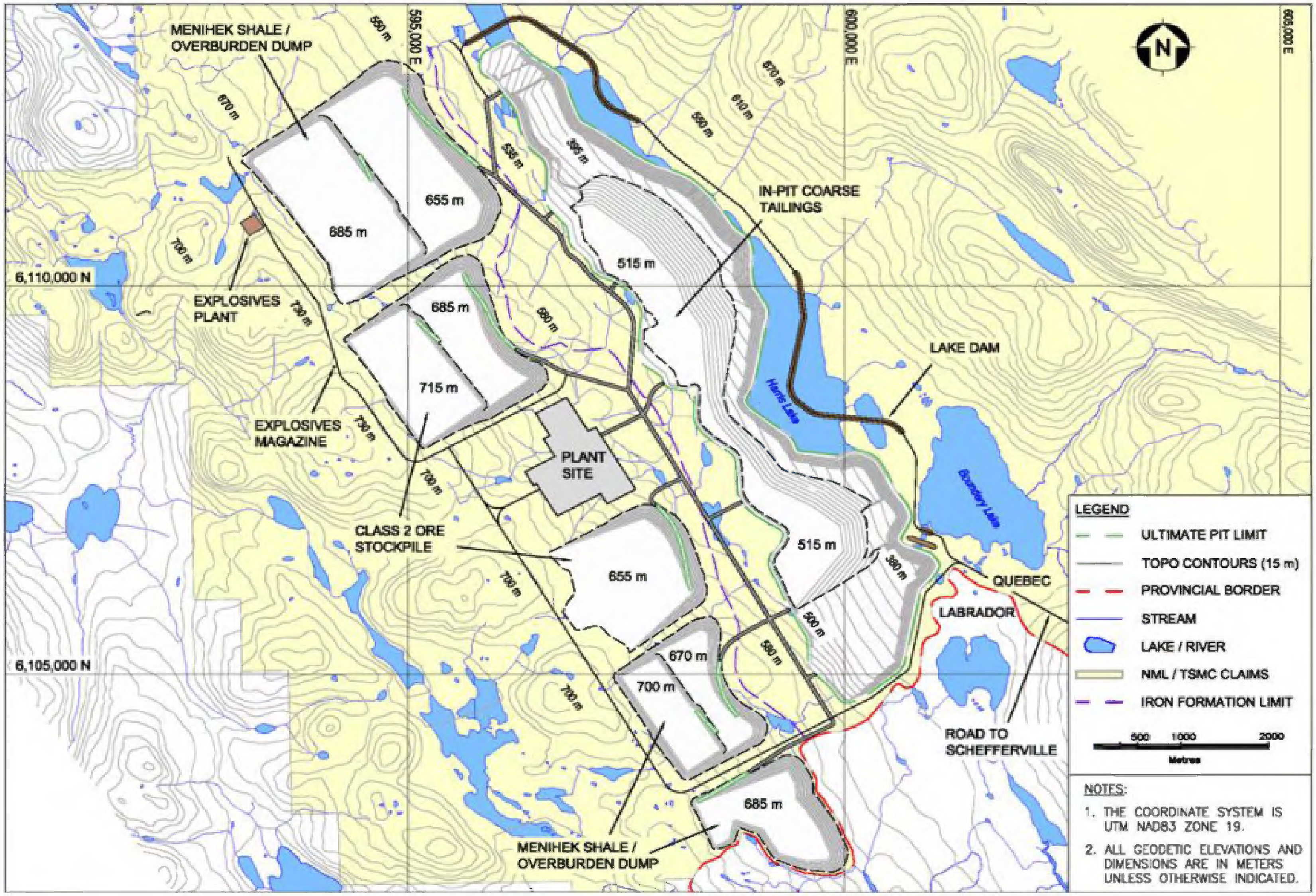
The waste dumps and stockpiles will be constructed in five (5) m high lifts, compacted by a bulldozer. Figure 16.1 presents a typical section of the waste dump. Figure 16.2 presents the dump and stockpile layout for the mine plan.

Figure 16.1 – Waste Dump Configuration



The dumps and stockpile will have a perimeter ditch around their toes to capture water run-off. The dumps and stockpile were designed within claims owned by NML/TS and were placed outside of the iron formation limit that was presented in Section 15.0.

Figure 16.2 – Dump and Stockpile Layout



The Class 2 ore stockpiles are located on each side of the plant site. The north stockpile was designed to contain approximately 122 Mm^3 of material and has a footprint of 310 ha. The stockpile was designed with a terrace in order to take advantage of the sloping topography and maintain the maximum dump height restriction of 100 m. The lower terrace reaches a final elevation of 685 m, while the upper terrace reaches a final elevation of 715 m at the end of the mine life. The south stockpile was designed to contain approximately 73 Mm^3 of material and has a footprint of 220 ha. The final height at the end of the mine life will be 655 m.

The waste dump located on the north side of the plant site was designed to contain approximately 237 Mm^3 of material and has a footprint of 520 ha. The dump was designed with a terrace in order to take advantage of the sloping topography and maintain the maximum dump height restriction of 100 m. The lower terrace reaches a final elevation of 655 m, while the upper terrace reaches a final elevation of 685 m at the end of the life of mine.

The waste dump located on the south side of the plant site was designed to contain approximately 87 Mm^3 of material and has a footprint of 200 ha. The dump was designed with a terrace in order to take advantage of the sloping topography and maintain the maximum dump height restriction of 100 m. The lower terrace reaches a final elevation of 670 m, while the upper terrace reaches a final elevation of 700 m at the end of the life of mine.

The third waste dump is located just north of the provincial border. This waste dump was designed to contain approximately 78 Mm^3 of material and has a footprint of 160 ha. The final height at the end of the mine life will be 685 m.

In order to reduce the size of the tailings facilities, the coarse tailings will be placed in the mined out pit beginning in Year-11. This material will be placed on the final pit floor and has been designed with an overall slope of 17°. Additional details on the coarse tailings are provided in Section 18.0 of this Report.

The dumps and stockpiles presented in this Report respect the volumes generated from the mine plan and are used to calculate haul distances for mine fleet estimation. All environmental aspects related to the dump and stockpile locations and designs are discussed in more detail in Section 20.0 of this Report.

16.4 Mine Planning

The following Section discusses the mine plan that was prepared for the KéMag deposit. This mine plan forms the basis of the mine capital and operating cost estimate presented in Section 21.0. The mine plan was established annually for the first ten (10) years of production and in three (3) year periods for the remaining 12 years.

16.4.1 Mine Planning Parameters

a) Work Schedule

Mining operations for the Project will be 365 days per year, operating around the clock on two (2), twelve (12) hour shifts. The fleet requirements and manpower are based on this work schedule. A total of seven (7) days per year have been accounted for when the mine will be shut down due to severe weather conditions. These conditions may occur during peak rainfall or snowfall periods or at times when the ambient temperature is extremely low.

b) Annual Production Requirements

The mine plan is based on an annual production of 22 Mt of concentrate. The concentrate includes 17 Mt of BF grade pellet feed and 5 Mt of DR grade pellet feed. The mine plan incorporates a ramp up of 60 % in Year-1 (13.2 Mt of concentrate) and 85 % in Year-2 (18.7 Mt of concentrate) before reaching full capacity in Year-3.

c) Weight Recovery Adjustment

In order to correlate the Davis Tube Weight Recovery value in the block model to the expected weight recovery in the plant, the following calculation was performed:

- MagFe (%) = DTFe Concentrate x DTWR;
- Weight Recovery for BF Concentrate = MagFe x 0.95 x 0.975 / 69.5;
- Weight Recovery for DR Concentrate = MagFe x 0.95 x 0.95 / 70.0;
- A weighted average of the two (2) weight recoveries was then used based on the 17 Mt / 5 Mt split of the total concentrate.

d) Blending of Rock Types

In order to minimize the variations of material hardness that is sent to the plant, the different rock types are blended during each period of the mine plan.

e) Blending and Grade Control

The objective of the mine plan is to supply a constant run of mine feed that is as close to 27 % DTWR and 2.2 % SiO_2 in the concentrate as possible. In order to minimize variations from this target for the entire 22-year mine plan, mining begins in the center of the deposit and advances in both the north and south directions. There will typically be four (4) active ore mining faces during the operation to allow for blending.

f) Push-back Strategy

A push-back strategy was applied to the mine plan so that the coarse tailings can be placed in-pit and follow closely behind the mining advance. In order to achieve this, the pit is divided into eight (8) slots that are roughly 1,000 m wide at surface. There is an opening slot, four (4) slots on the north side and three (3) slots on the south side. The slots are mined from west to east and developed down to the final pit floor. Once a slot is completely mined out additional in-pit coarse tailings can be placed in that slot.

16.4.2 Mine Production Schedule

This Section presents the mine production schedule that was established for the KéMag deposit. The mine production schedule is presented in Table 16.1. This schedule meets the targets and objectives that were discussed above. Figure 16.3 presents a plan view showing the pit advance by period. The reason several periods are missing from this figure is because the mining is occurring within the same limits as the previous period.

A pre-production phase of one (1) year has been planned for the KéMag deposit. This period includes clearing and grubbing activities, topsoil and overburden stripping, mine road construction and the development of the pit for ore production. The pre-production activities will be carried out by a mining contractor as is typically done in mining operations. The overburden quantity excavated in the pre-production phase is 10.8 Mt.

The annual DTWR in the mine plan fluctuates between a low of 25.4 % to a high of 32.7 % with an average of 27.1 %. As a result, the annual run of mine feed to the plant when in full production, ranges from 80.9 Mt to 94.3 Mt with an average of 89.0 Mt.

The annual SiO_2 content in the concentrate fluctuates between 2.1 % and 2.4 % with an average of 2.2 %. The annual total material moved ranges from 70 Mt in Year-1 to 154 Mt in Year-4 with an average of 148 Mt per year for the life of mine.

The 1,891 Mt of Class 1 Mineral Reserves are depleted in Year-22 of the mine plan. Class 2 Mineral Reserves will be stockpiled and used when technology permits to yield products of required quality.

Figure 16.4 presents a chart showing the tonnages mined each year. The tonnages shown are annualized for the three (3) year periods. Figure 16.5 presents the annual DTWR and SiO_2 in the concentrate for the mine plan.

Table 16.1 – Mine Production Schedule

Description	Units	Pre-Prod	Year 01	Year 02	Year 03	Year 04	Year 05	Year 06	Year 07	Year 08	Year 09	Year 10	Years 11 - 13	Years 14 - 16	Years 17 - 19	Years 20 - 22	Total
Concentrate	**Mt**	**0.0**	**13.2**	**18.7**	**22.0**	**22.0**	**22.0**	**22.0**	**22.0**	**22.0**	**22.0**	**22.0**	**66.0**	**66.0**	**66.0**	**66.0**	**472**
Class 1 Ore to Plant	**Mt**	**0.0**	**43.8**	**61.6**	**87.3**	**93.3**	**93.6**	**94.3**	**91.2**	**84.8**	**80.9**	**84.8**	**273.3**	**265.5**	**260.9**	**270.1**	**1,885**
LC	Mt	0.0	14.0	13.7	31.8	44.8	22.0	27.1	36.0	22.0	15.3	29.0	112.4	79.4	67.8	38.8	**554**
JUIF	Mt	0.0	3.7	2.8	9.0	8.8	2.5	4.0	4.6	6.1	4.3	4.5	18.0	14.9	17.8	19.9	**121**
GC	Mt	0.0	3.3	2.9	2.7	3.3	1.8	2.8	2.1	3.6	3.2	3.8	14.7	11.5	16.3	15.1	**87**
URC	Mt	0.0	2.7	2.9	4.2	4.4	3.2	4.4	3.4	4.4	4.4	3.6	13.7	14.4	19.6	22.8	**108**
PGC	Mt	0.0	5.5	7.3	12.6	9.4	10.2	15.9	10.8	12.5	14.0	10.6	34.4	38.6	46.6	48.1	**276**
LRC	Mt	0.0	0.0	0.0	2.9	2.5	2.4	1.4	0.1	0.0	2.2	2.3	8.8	6.2	5.3	11.8	**46**
LRGC	Mt	0.0	14.6	31.9	24.0	20.1	51.4	38.7	34.3	36.1	37.6	31.1	71.4	100.6	87.5	113.6	**693**
DT Weight Recovery	**%**	**0.0**	**32.5**	**32.7**	**27.2**	**25.6**	**25.5**	**25.4**	**26.1**	**28.1**	**29.2**	**27.9**	**26.1**	**26.7**	**27.5**	**26.5**	**27.1**
SiO_2 (in concentrate)	**%**	**0.0**	**2.1**	**2.1**	**2.2**	**2.4**	**2.4**	**2.3**	**2.3**	**2.3**	**1.9**	**2.0**	**2.2**	**2.1**	**2.2**	**2.2**	**2.2**
Fe (in concentrate)	**%**	**0.0**	**70.1**	**70.0**	**70.0**	**69.9**	**69.7**	**69.6**	**69.9**	**69.9**	**70.4**	**70.4**	**70.0**	**70.3**	**69.5**	**69.7**	**69.9**
Plant WR	**%**	**0.0**	**30.2**	**30.3**	**25.2**	**23.6**	**23.5**	**23.3**	**24.1**	**26.0**	**27.2**	**26.0**	**24.2**	**24.9**	**25.3**	**24.5**	**25.0**
Class 2 to Stockpile	**Mt**	**0.0**	**1.7**	**1.6**	**21.9**	**25.2**	**16.0**	**18.8**	**19.8**	**7.8**	**2.9**	**7.0**	**83.5**	**38.4**	**95.7**	**135.0**	475
Total Waste	**Mt**	**10.8**	**24.1**	**17.0**	**42.1**	**35.8**	**41.7**	**39.2**	**41.5**	**34.3**	**43.6**	**42.6**	**101.4**	**149.4**	**95.5**	**46.5**	766
Overburden	Mt	10.8	19.6	0.0	25.7	20.2	8.8	10.8	6.6	0.0	22.9	31.7	11.2	3.6	36.2	23.0	231
Menihek Shale	Mt	0.0	0.0	6.8	2.4	8.2	28.6	27.1	31.7	32.0	12.7	9.2	90.2	124.4	40.5	1.4	415
Inferred	Mt	0.0	4.5	10.1	14.1	7.4	4.3	1.4	3.2	2.3	8.1	1.7	0.0	21.4	18.9	22.1	119
Total Material Moved	**Mt**	**10.8**	**69.6**	**80.2**	**151.3**	**154.2**	**151.3**	**152.3**	**152.5**	**126.9**	**127.4**	**134.4**	**458.2**	**453.2**	**452.1**	**451.6**	**3,126**
Stripping Ratio		**n/a**	**0.5**	**0.3**	**0.4**	**0.3**	**0.4**	**0.3**	**0.4**	**0.4**	**0.5**	**0.5**	**0.3**	**0.5**	**0.3**	**0.1**	**0.3**

Figure 16.3 – Pit Advance (Plan View)

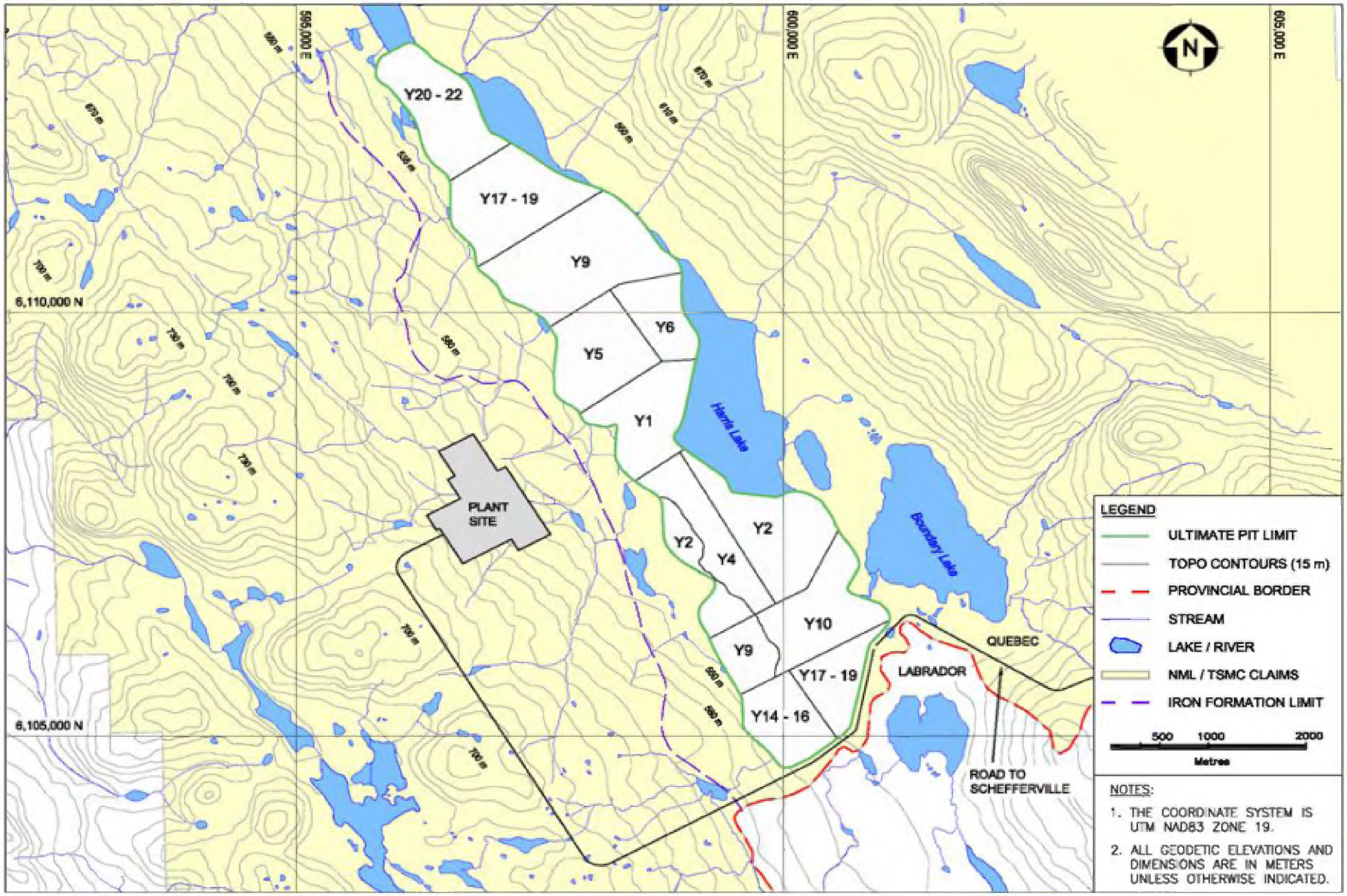


Figure 16.4 – Mine Production Schedule (Production)

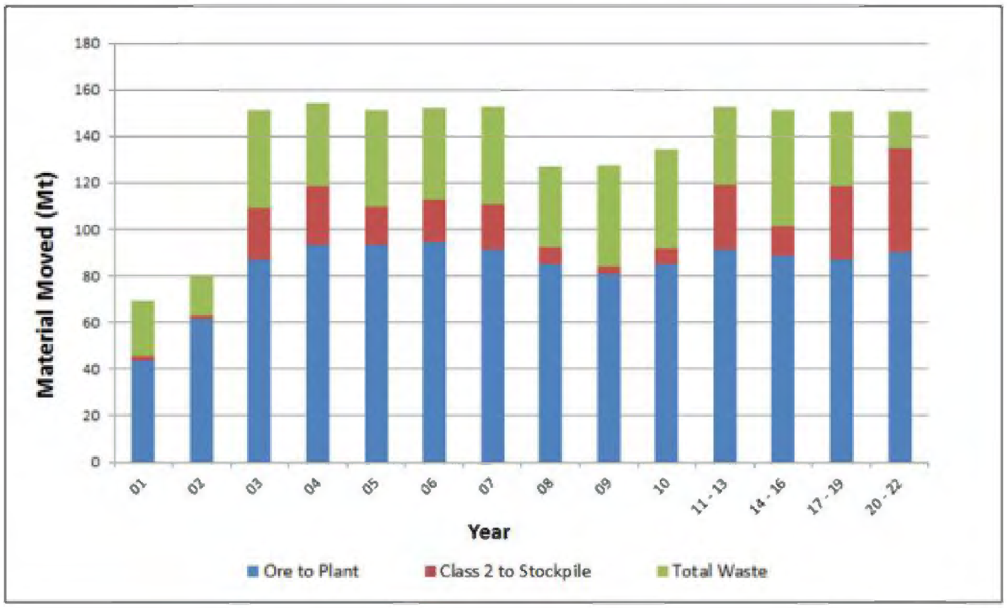
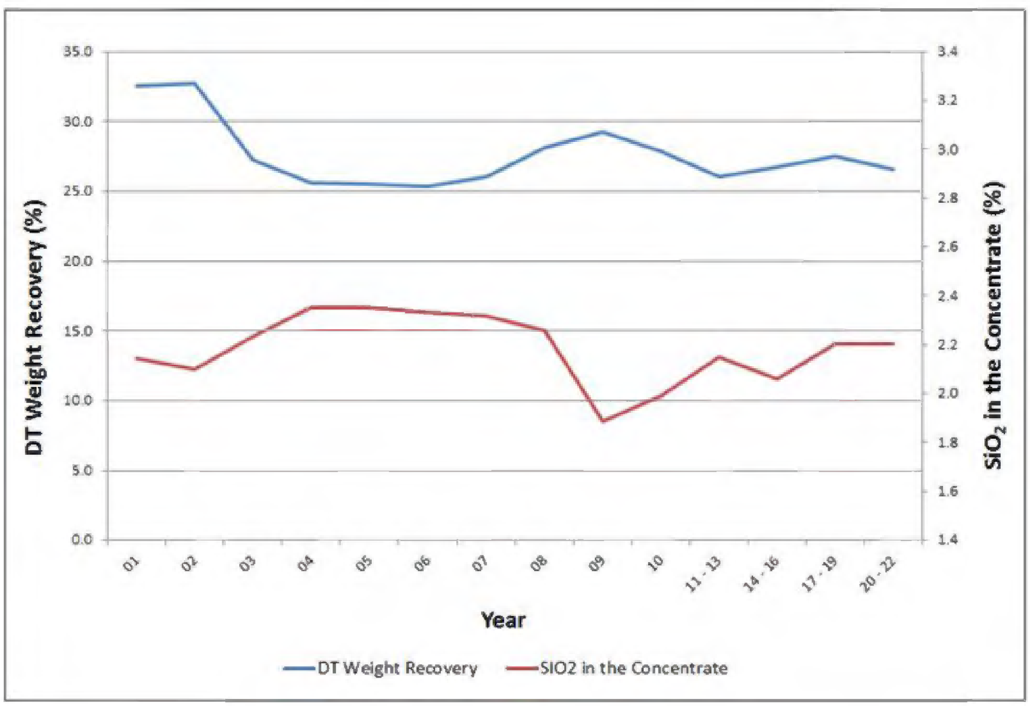


Figure 16.5 – Mine Production Schedule (Grades)



16.5 Mine Equipment Fleet

The following Section discusses equipment selection and fleet requirements in order to carry out the mine plan. Table 16.2 presents the list of major mining equipment on an annual basis.

Table 16.2 – Major Mining Equipment Fleet

Equipment	Description	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
Haul Truck	Payload – 340 t	15	17	30	35	35	35	35	35
Cable Shovel	Bucket – 100 t	3	3	7	7	7	7	7	7
Hydraulic Shovel	Bucket – 90 t	1	1	1	1	1	1	1	1
Wheel Loader	Bucket – 37 t	1	1	1	1	1	1	1	1
Production Drill	Bit Load – 65 t	4	6	9	10	10	10	10	10

Equipment	Description	Year 9	Year 10	Years 11 - 13	Years 14 - 16	Years 17 - 19	Years 20 - 22
Haul Truck	Payload – 340 t	35	35	35	35	35	35
Cable Shovel	Bucket – 100 t	7	7	7	7	7	7
Hydraulic Shovel	Bucket – 90 t	1	1	1	1	1	1
Wheel Loader	Bucket – 37 t	1	1	1	1	1	1
Production Drill	Bit Load – 65 t	10	10	10	10	10	10

16.5.1 Haul Trucks

The haul truck selected for the Project is a rigid frame mining truck with a payload of 340 tonnes. This truck has been selected since it results in a manageable fleet size given the large quantities of material and haul distances expected. The standard payload of 363 tonnes has been reduced to 340 tonnes to account for the liners on the truck boxes that are required for this iron ore application. The following parameters were used to calculate the number of trucks required to carry out the mine plan:

- Mechanical Availability – 85 %;
- Utilization – 90 % (non-utilized time is accrued when the truck is not operating due to poor weather, blasting, shovel relocation and no operator available);
- Nominal Payload – 340 tonnes (240 m^3 heaped);
- Shift Schedule – Two (2), 12 hour shifts per day, seven (7) days per week;
- Operational Delays – 80 min/shift (this includes 15 minutes for shift change, 15 minutes for equipment inspection, 40 minutes for lunch and coffee breaks and

10 minutes for fuelling). Fuelling will be carried out once every two (2) shifts for 20 minutes;

- Job Efficiency – 90 % (54 min/h; this represents lost time due to queuing at the shovel and dump as well as interference along the haul route);
- Rolling Resistance – three (3) %.

Table 16.3 summarizes the annual hours of a haul truck based on the specified parameters. Figure 16.6 displays these hours in a graphical format. Haul routes were generated for ore and waste for each period to calculate the truck requirements. These haul routes were imported in Talpac©, a commercially available truck simulation software package that Met-Chem has validated with mining operations. Talpac© calculated the travel time required for a 340-tonne haul truck to complete each route. These travel times were validated with the truck manufacturers.

Table 16.3 – Truck Hours (h/y)

Total Hours	8,760	7 days per week, 24 hours per day, 52 weeks per year
Scheduled Hours	8,592	Hours available accounting for weather delays
Down Mechanically	1,289	15% of total hours
Available	7,303	Total hours minus hours down mechanically
Standby	730	Ten (10) % of available hours (represents 90% utilization)
Operating	6,573	Available hours minus standby hours
Operating Delays	730	80 min/shift
Net Operating Hours	5,843	Operating hours minus operating delays
Working Hours	5,258	90 % of net operating hours (reflects job efficiency)

Figure 16.6 – Truck Hours

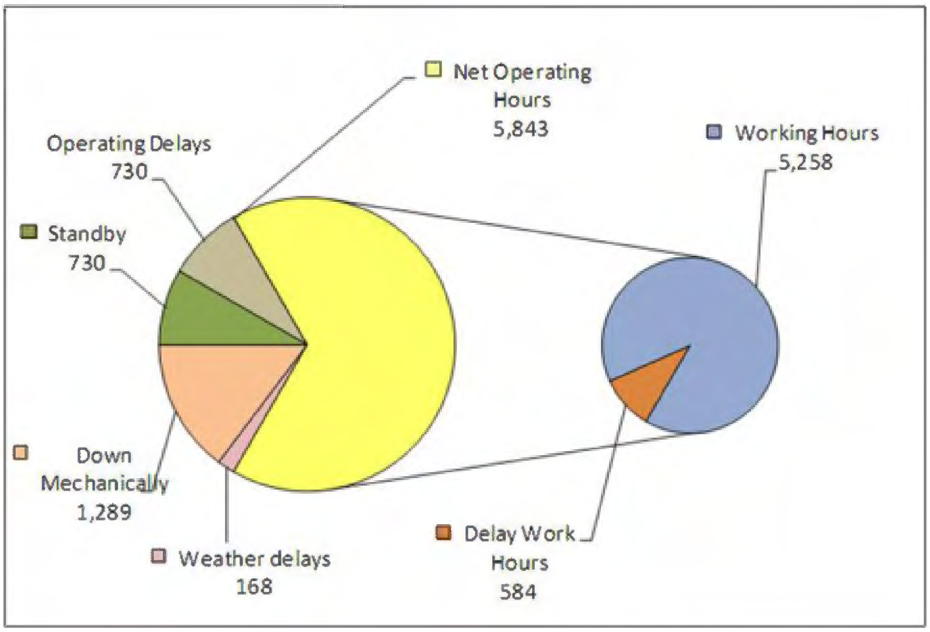


Table 16.4 shows the various components of a truck's cycle time. The load time is calculated using a cable shovel with a 100-tonne bucket payload as the loading unit. This class of shovel which is discussed in the following Section loads a 340-tonne haul truck in four (4) passes.

Table 16.4 – Truck Cycle Time

Activity	Duration (sec)
Spot @ Shovel	45
Load Time[1]	180
Travel Time	Calculated by Talpac©
Spot @ Dump	30
Dump Time	30

1. Four (4) Passes @ 45 sec/pass.

Haul routes were determined for each period of the mine plan based on the centroid of the mining area and the centroid of the dumping location for each material type. Haul productivities (tonnes per work hour) were calculated for each haul route using the truck payload and cycle time.

Truck hour requirements were calculated by dividing the production into the productivity for each haul route. The number of trucks required was calculated assuming each truck has the 5,258 hours available to work in a full year, shown in Table 16.3.

A fleet of 15 trucks is required during Year-1. This number increases to 17 in Year-2, 30 in Year-3 and reaches a peak of 35 from Year-4 to Year-22. The truck fleet required by year is presented in Table 16.2.

16.5.2 Shovels

The main loading machine selected for the Project is an electric cable shovel with a bucket payload of 100 tonnes. This large mining shovel will be suitable to handle the production requirements as well as the face heights expected.

The following parameters were used to calculate the number of shovels required to carry out the mine plan:

- Mechanical Availability – 85 %;
- Utilization – 90 %;
- Bucket Capacity – 100 tonnes (39 m^3);
- Shift Schedule – Two (2), 12 hour shifts per day, seven (7) days per week;
- Operational Delays – 70 min/shift (this includes 15 minutes for shift change, 15 minutes for equipment inspection, 40 minutes for lunch and coffee breaks);
- Job Efficiency – 50% (30 min/h; this represents lost time due to waiting for trucks, cleaning up the loading area and relocating).

The electric cable shovel can load a 340-tonne haul truck in four (4), 45 second passes for a total load time of 3.0 minutes. Assuming there are trucks available to load, the shovel can load 18.7 trucks per hour for a theoretical productivity of 6,800 t/h accounting for mechanical availability, utilization, operational delays and job efficiency. Each shovel has 2,876 available work hours in a full year. Industry benchmarking shows that this size of cable shovel should be able to excavate between 20 to 25 Mt per year. In order to mine the tonnages presented in the mine plan, three (3) shovels are required during Year-1 and Year-2 and reaches a peak of seven (7) shovels in Year-3.

A diesel powered hydraulic shovel with a bucket payload of 90 tonnes has been included in the shovel fleet. This shovel will assist with the blending of ore since it can be relocated faster than a cable shovel and can be more selective when mining a 15 m bench. The hydraulic shovel can load a 340-tonne haul truck in four (4) passes. The hydraulic shovel is planned to excavate 15 Mt per year.

A large front end wheel loader equipped with a 56-tonne capacity bucket has been included in the fleet to be used for stockpile rehandle and as an alternate loading machine.

The shovel and loader fleets required by year are presented in Table 16.2.

16.5.3 Drilling and Blasting

Production drilling will be carried out using a fleet of electric powered rotary drills. The proposed blast pattern is presented in Table 16.5. The drilling and blasting parameters were determined based on benchmarking with taconite operations in Minnesota's Mesabi Range and discussions with explosive suppliers.

The number of drills required was estimated assuming an 85 % mechanical availability, 90 % utilization and a penetration rate of 25 m/h. Each drill will be able to drill blast patterns totaling approximately 13 Mt per year.

In order to drill the tonnages presented in the mine plan, four (4) drills are required in Year-1, six (6) in Year-2, nine (9) in Year-3 and it reaches a peak of ten (10) in Year-4. The drill fleet required by year is presented in Table 16.2.

Table 16.5 – Blasting Parameters

Parameter	Units	Ore	Waste
Bench Height	m	15	15
Blasthole Diameter	mm	381	381
Burden	m	8.3	8.8
Spacing	m	9.5	10.1
Subdrilling	m	1.5	1.5
Stemming	m	8.2	8.2
Explosives Density	g/cm^3	1.26	1.26
Powder Factor	kg/t	0.39	0.39

Blasting will be executed under contract with an explosives supplier who will be responsible for the following services:

- Transportation and storage of explosive manufacturing products and blasting accessories to site;
- Manufacturing of bulk emulsion explosives;
- Loading and priming of blastholes.

Two (2) sites have been selected for the contractor to establish the explosives operation. The site selection has accounted for the required minimum distances as specified by the Canadian explosives regulations. Approvals and permits are required from the government regulating bodies prior to construction. These sites are located on:

- Explosives Plant – this site includes the storage facility for raw materials, the offices and garages as well as the emulsion plant and pumper truck loading area;

- Explosive Magazines – this site includes the magazines to store the blasting caps, primers, detonation cord and packaged explosives.

For the fabrication of the bulk emulsion, explosives suppliers have proposed transporting ammonium nitrate solution from Sept-Îles to Schefferville by rail. The ammonium nitrate solution will be stored in rail cars at a rail siding near Schefferville and transported to the explosives plant as required. The explosives plant facilities will be composed of predesigned/prebuilt modules that are easily transported and assembled.

In order to support the explosives supplier, the mine operator is required to build and maintain the access road to the two (2) sites and to supply electric power, communications and diesel fuel for the manufacturing of the emulsion as well as the operation of mobile equipment. The mine operator is also required to mobilize and house the contractor's workforce.

Based on the blasting parameters presented in Table 16.5, the amount of explosives required per year is approximately 45 million kg. In order to manufacture this amount of explosives per year, the following quantities of consumables are required:

- Ammonium Nitrate – 37,000 t;
- Diesel Fuel – four (4) million litres (includes fuel for mobile equipment fleet);
- Water – seven (7) million litres;
- Electrical Energy – two (2) MWh per year.

During full production there will be roughly three (3) blasts per week each producing approximately 750,000 t of material.

16.5.4 Auxiliary Equipment

The following fleet of support and service equipment was included to carry out the mine plan:

- Ten (10) track dozers (634 - 655 kW) are required for construction of the waste dumps as well as pit and road maintenance;
- Three (3) wheel dozers (637 - 674 kW) are required to clean up the shovel pits;
- Five (5) road graders (350 - 400 kW) are required for mine road maintenance;
- Three (3) wheel dozers (266 - 303 kW) equipped with cable reel attachments are required to manipulate the trailing cable for the shovels and drills;
- Two (2) large utility excavators (358 – 397 kW), and four (4) small utility excavators (109 – 119 kW) have been added to the fleet to establish ditches and sumps for mine dewatering as well as other activities. The large excavators will be equipped with a 4.6 m^3 bucket and the small excavators will be equipped with a 1.2 m^3 bucket;

- Five (5) haul trucks with 60-tonne payloads will be used as part of the utility crew. These trucks will be used for road maintenance, dewatering and other miscellaneous activities;
- Two (2) utility front end loaders (283 – 396 kW) will be used as part of the utility crew. These loaders will be used for road maintenance, dewatering and other miscellaneous activities;
- Three (3) haul trucks equipped with 200,000-litre tanks will be used as water trucks in the summer converted into sand trucks in the winter. These trucks are important to maintain the integrity of the roads to allow for a safe and productive operation;
- Two (2) track mounted secondary drills capable of drilling 165 mm holes will be used for pre-shearing and secondary blasting;
- One (1) soil compactor is required for road construction and maintenance.

The remaining support and service equipment includes fuel/lube trucks, mechanic trucks, tire handlers, large tow trucks, boom trucks, mobile cranes, low boys, transport busses, pickup trucks and light towers.

Table 16.6 provides a summary of the auxiliary equipment.

Table 16.6 – Auxiliary Equipment

Support Equipment	Description	# Units
Track Dozer	634 – 655 kW	10
Wheel Dozer	637 – 674 kW	3
Road Grader	350 – 400 kW	5
Cable Reeler	266 – 303 kW	3
Large Utility Excavator	358 – 397 kW	2
Small Utility Excavator	109 – 119 kW	4
Utility Haul Truck	60 t payload	5
Utility Front End Loader	283 – 396 kW	2
Water / Sand Truck	200,000-litre	3
Secondary Drill	165 mm drill holes	2
Soil Compactor	130 – 160 kW	1
Light Tower	8 kW	20
Service Equipment	**Description**	**# Units**
Fuel / Lube Truck	12,000-litre	3
Mechanic Truck	n/a	4
Tire Handler	n/a	2
Boom Truck	22-tonne	2
Small Mobile Crane	75-tonne	2
Large Mobile Crane	100-tonne	2
Tow Truck / Lowboy	240T Chassis	2
Transport Bus	20 person cap.	4
Pick-up Truck	4 x 4	20

16.5.5 Lead Times for Delivery

The cable and hydraulic shovels are considered as the long lead items for the mine equipment fleet at an estimated delivery time of 18 months. These shovels are required during the first year of mine operations. The estimated delivery time for the major mining equipment is presented in Table 16.7.

Table 16.7 – Mining Equipment Lead Delivery Time

Equipment	Lead Delivery Time (months)
Haul Truck	14
Cable Shovel	18
Hydraulic Shovel	18
Wheel Loader	12
Tracked Dozer	12
Road Grader	6

16.5.6 Mine Dispatch

A mine dispatch system will be installed since the fleet size is large and the ore blending requirements are important. The cost to install a mine dispatch system is included in the mine capital cost estimate.

16.5.7 Equipment Simulator

In order for the mine operations to run safely and efficiently, a truck, shovel and dozer simulator will be required on site to train the operators. The cost to purchase the simulators is included in the mine capital cost estimate.

16.6 Mine Manpower Requirements

The total mine workforce for the Project ranges from 345 employees in Year-1 to a maximum of 535 from Year-4 to 22. This workforce is comprised of management, staff as well as hourly employees. The 25 management and staff employees include the mine superintendent, the maintenance superintendent, the engineering supervisor, six (6) mining engineers, four (4) geologists, four (4) planning technicians, four (4) surveyors and four (4) maintenance planners. The 22 non-management staff employees will work on a two (2) week on, two (2) week off rotation, therefore the number of employees stated accounts for duplication.

The hourly workforce includes four (4) crews in order to provide 24 hour per day coverage, seven (7) days per week. Each crew will be comprised of three (3) pit foremen, two (2) maintenance foremen, a drill and blast foreman, two (2) grade control technicians, a dispatcher, a trainer, equipment operators, a dewatering crew, a power distribution crew, mechanics, electrical technicians, welders, labourers and maintenance attendants. The number of mechanics, electrical technicians and welders was estimated assuming a maintenance ratio of 0.25. The mine workforce also accounts for two (2) % absenteeism and holidays. Table 16.8 shows the mine manpower requirements for Year-7.

Table 16.8 – Mine Manpower Requirements (Year-7)

Description	Management	Staff	Shift
Mine Operations			
Mine Superintendent	1		
Pit Foreman			12
Drill and Blast Foreman			4
Truck Operator			123
Shovel Operator			33
Loader Operator			5
Drill Operator			41
Track Dozer Operator			54
Wheel Dozer Operator			21
Grader Operator			13
Water / Sand Truck Operator			13
Fuel and Lube Truck Operator			25
Labourer			9
Dewatering Crew			9
Power Distribution Crew			5
Dispatcher			4
Trainer			
Mine Maintenance			
Maintenance Superintendent			
Maintenance Foreman	1		8
Maintenance Planner			
Mechanic		4	66
Electrical Technician			13
Welder			17
Wash Bay Attendant			9
Tire Bay Attendant			9
Tool Crib Attendant			9
Mine Engineering			
Engineering Supervisor			
Mining Engineer	1		
Geologist		6	
Grade Control Technician		4	8
Mining Technician			
Surveyor		4	12
Sub Total Mine	**3**	**22**	**510**
Total Mine Workforce			**535**

17.0 RECOVERY METHODS

The process plant will extract magnetite from the taconite reserves of the Labrador Trough, south of the Quebec border and north of Schefferville to produce a concentrate. The operation will mill approximately 88 Mt/y of iron bearing mineralization to produce 22 Mt/y of magnetite concentrate.

The process plant will include multiple comminution stages combined with conventional magnetite recovery techniques. Silica grade can be further reduced in a flotation plant to produce a high grade iron concentrate with low SiO_2 suitable for DR pellets or to be sold as pellet feed concentrate.

Unless otherwise noted, all weight and throughput are expressed in dry tonnes.

17.1 Process Design Criteria

The process plant is designed to treat approximately 88 Mt/y of taconite ore, at a DTWR of 27 % and Fe grade of approximately 30 % that will permit production of concentrate about 2.1 % or 2.2 % of silica. The magnetite weight recovery is critical since any hematite present will not be recovered in the concentration process. The ROM necessary to produce the required tonnage of concentrate is calculate based on the weight recovery and the design factor, part of the design criteria, ensure that the process equipment has enough capacity to take care of the expected feed variation.

Operating 358 days per year, the process plant will recover a nominal 22 Mt/y of concentrate. The concentrate product quality will be such that DR grade and BF grade pellet feed can be produced as needed. The plant design is based on a 22-year plant life.

The process plant design is based on test work performed on representative samples, selected from the LabMag and KéMag deposits, tested in pilot plants from 2005 to 2013 and complemented by bench scale test work and supplier tests for equipment sizing. Minor items included in the basis of design of the process plant have not necessarily covered by test work. In such cases, assumptions have been made based on industry experience as agreed by ISLLP and NML/TS.

The design philosophy is driven by the requirement for known and established processes which provide the most effective means of concentrating the ore. The process selected for the Taconite Project consists of the following steps:

- Fixed primary crushing units located centrally in relation to the pit, on the edge of the resource to avoid unnecessary haulage;
- Conventional secondary crushing and screening with primary stockpile to allow for maintenance down time and weather related delays;
- HPGR grinding and screening followed by cobber Low Intensity Magnetic Separators (“**LIMS**”) concentration;

- Ball mill regrinding with rougher and finisher LIMS concentration as well as screening;
- Flotation as needed to further reduce the silica content for low silica BF and DR grade pellet feed;

The overall operation will operate for 24 hours per day, seven (7) days per week and 51 weeks per year (i.e. 358 days per year). Primary crushing and secondary crushing will have an availability of 66 and 85 %. The process plant and the flotation plant will operate at an availability of 92 and 95 %.

17.2 Process Facilities Location Criteria

The processing plant flow sheet and design criteria are based on the results from the metallurgical test work, program discussed in Section 13.0 of this Report. The LabMag and KéMag ore characteristics are similar though not identical and the same flow sheet and plant design can be used for either ore body and still meet the concentrate quantity and quality criteria. Blending will ensure a constant quality of feed to the concentrator.

The KéMag mine site is located in Québec, north of the Labrador border in the same north/south axis as the KéMag deposit. The concentrator will be centrally located along the edge of the mine pit.

The location and layout of the process plant was subjected to a number of considerations but minimizing the distance from the mining area and avoiding the area designated for tailings disposal site and waste rock dumps were the main criteria.

The following factors were considered in a qualitative assessment of the process plant, tailings and water storage facility sites:

- Site geometry and terrain;
- Process plant and crushed ore stockpile layout;
- Tailings impoundment area (“**TIA**”) facility to have capacity for a 22-year mine life;
- Location of the TIA relative to the water bodies and topography;
- Water storage facility capacity;
- Earthworks requirement.

17.2.1 Plant General Information

The process plant general design basis is shown in Table 17.1 and the coordinates of the sites are shown in Table 17.2.

Table 17.1 – Process Plant General Design Basis

Items	Value	Units
Iron Concentrate Production Target (Dry)	22, 000,000	t/y
Run of Mine Production - Design (Dry)	86, 140,000	t/y
Run of Mine Humidity	2	%
Weight Recovery (Including Flotation)	25.54	%
Scheduled Operating Days per Year	358	days
Scheduled Operating Hours per Year	8,592	hrs
Utilization (Percentage of Scheduled Operation Hours Available)		
Primary Crushing	66	%
Secondary Crushing	85	%
Primary Grinding and Cobbing	92	%
Secondary Grinding and Concentration	92	%

Table 17.2 – Process Plant Coordinates

Location	Northing	Easting
KéMag	6,107,709	597,146
The Coordinates System are: (UTM) NAD 83, ZONE 19		

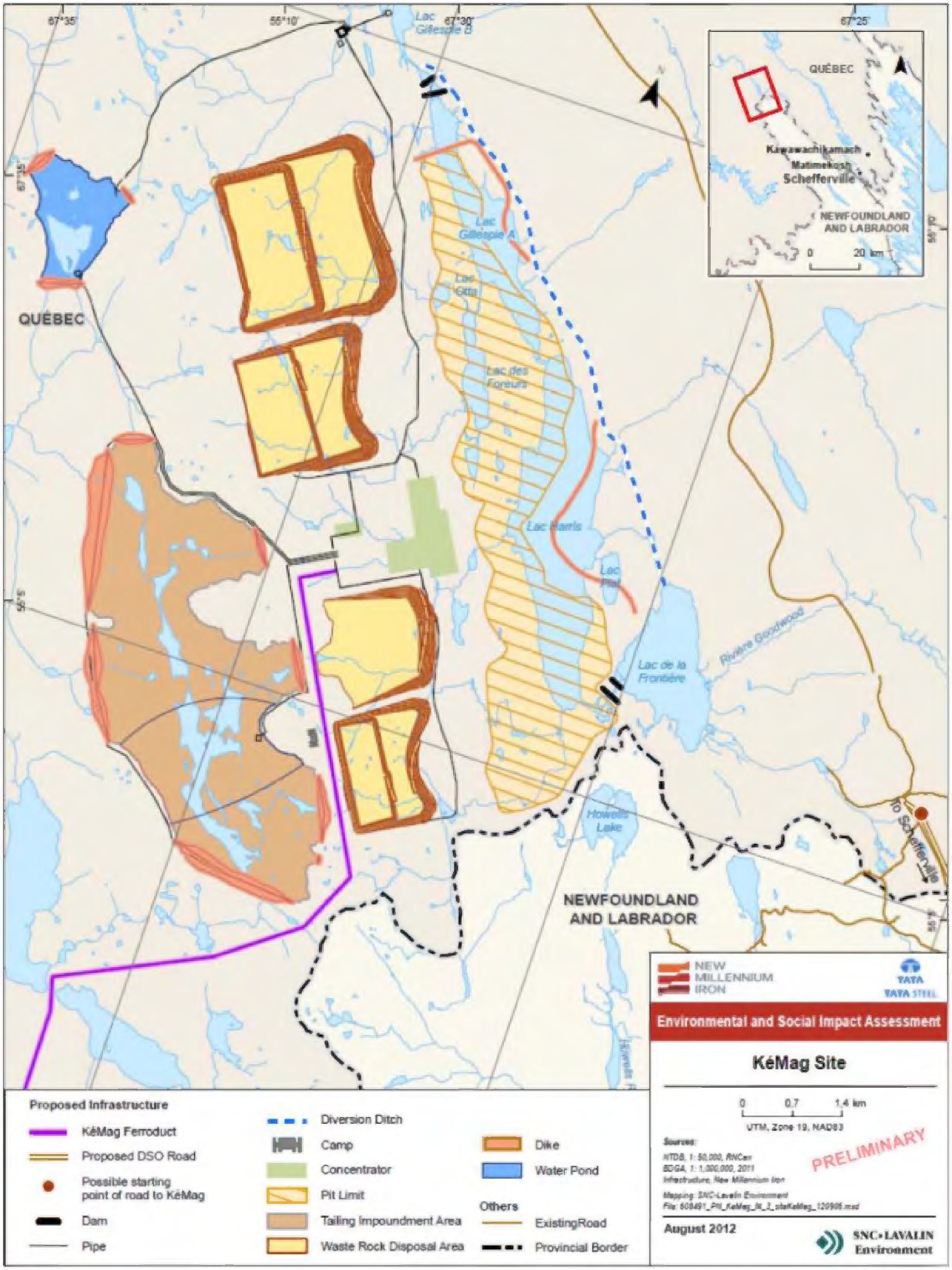
17.3 Description and Geometry

The KéMag process plant is located west of the pit in a rocky area. The topography is entirely downward-sloping to the east at rates of 10 % or less. The site has substantial length and width and provides sufficient room for the processing facilities and structures.

The process plant site is north of and adjacent to a natural valley that forms a depression in the terrain. This valley is the preferred site for the tailings storage facility and its centre is situated approximately four (4) km from the process plant. Figure 17.1 shows the general arrangement of the KéMag site.

The proximity of the tailings storage facility to the process plant offers a major benefit to the operation in terms of operating cost, tailings handling, operational efficiency and control. The tailings storage facility will be able to accommodate 22 years of tailings assuming a conservative settled density. Coarse and fine tailings will be pumped together and the coarse fraction reclaimed to raise the tailings storage dams as needed.

Figure 17.1 – KéMag Process Plant Location



The water storage pond area is situated in another naturally occurring depression about four (4) km west of the plant. It will collect water from Lake Harris, the mine and waste dumps and will act as a safeguard for dry years. Figure 17.2 shows the location of the KéMag water pond.

Figure 17.2 – KéMag Fresh Water Pond

17.3.1 Ore Characteristics

The data used for the Feasibility Study for the ore characteristics are the typical chemical compositions for the bulk sample used in the pilot plant for KéMag ore and are shown in Table 17.3. It is to be noted that the design criteria and equipment sizing are identical for the KéMag orebody even though there are minor differences in the ore characteristics.

Table 17.3 – Ore Characteristics – KéMag

Item	KéMag	Design	Units
Ore Density	3.41	3.41	g/ml
Crushing Work Index	19.0	16.2	kWh/t
Ball Mill Work Index, BWI, Closing 150 µm	_	16.0	kWh/t
Ball Mill Work Index, BWI, Closing 75 µm	13.92	0	kWh/t
Abrasion Index	0.81	0.8	g

Item	KéMag	Design	Units
Typical Chemical Composition of Fresh Ore			
Fe	31.14	31.0	wt %
SiO_2	42.45	47.03	wt %
Al_2O_3	0.22	0.32	wt %
MgO	1.48	1.19	wt %
CaO	2.79	2.01	wt %
Na_2O	0.03	0.02	wt %
K_2O	0.07	0.08	wt %
TiO_2	0.02	0.03	wt %
MnO	1.29	0.97	wt %
P	0.01	0.03	wt %
S	0.02	0.02	wt %
Loss on Ignition	5.08	3.79	wt %

17.3.2 Mineralogy

The dominant iron rich mineral is magnetite which makes up approximately 96 % of the concentrate (by mass). The second iron rich mineral, hematite is non-magnetic material and is not recovered in the concentrate and the majority of the hematite is eliminated in the cobber tails. The mineralogical properties of the ore feed used for the mass balance are shown in Table 17.4.

Table 17.4 – Mineralogical Properties of the Ore Feed Used for the Mass Balance

Mineral	Design Value	Units
Magnetite, Fe_3O_4	27	wt %
Hematite, Fe_2O_3 / Gangue Minerals	13	wt %
Quartz, SiO_2 (Representing Gangue Materials)	60	wt %
Feed Fe Grade	32.5	wt %

17.4 Process Flow Sheet

The basic process flow sheet consists of primary and secondary crushing and screening circuits feeding HPGRs for dry comminution. After cobbing, the liberation of the iron-bearing mineral (magnetite) is completed in a wet grinding circuit. Concentration stages include besides cobbing, roughing and finishing with LIMS.

Depending on the product silica level required at the pellet plant, part or all of the final concentrate will be directed to the flotation circuit. Final concentrate will be thickened and pumped to the slurry storage tanks for transport via the concentrate slurry transportation system.

All tailings size fractions are pumped together to the Tailings Storage Facility ("**TSF**") and the heavier coarse fraction that settles first on the beaches reclaimed to raise the tailings impoundment area dykes. The decanted water will be recycled to the process plant.

17.4.1 Feasibility Study Flow Sheet Development

The Feasibility Study was initiated in November 2011. The feasibility engineering was based on the flow sheet and mass balance from the pilot scale testing conducted between 2006 and 2009 as a part of the pre-feasibility studies. These results were used for equipment selection and sizing calculations. The block flow diagram shown in Figure 17.3 illustrates the overall beneficiation process developed for the Project. This block diagram is based on nominal operation of 358 days per year or 8,792 hours and has to be adjusted for the utilization factor for each area of the process.

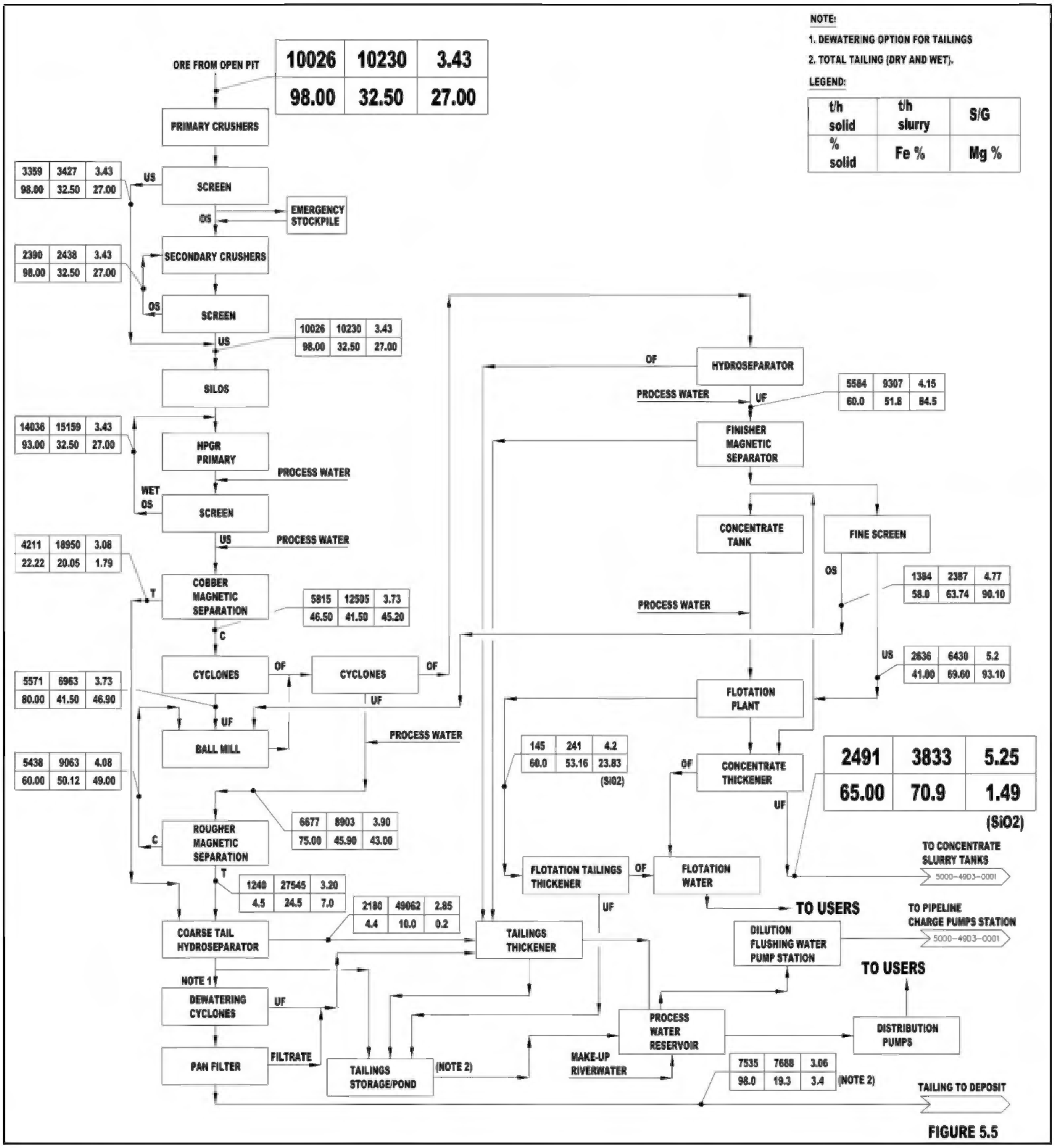
The proposed process flow sheet uses industry proven unit operations in what is essentially a conventional magnetite ore processing circuit to produce DR and BF pellet feed concentrates. The process design is based on test work performed on samples selected from the taconite deposits. Test work was conducted on individual samples, composite samples and includes vendor test work.

Based on analysis of the test work data, the process flow sheets and mass balances were developed.

The design criteria consist of test work data, vendor information, assumptions based on industry experience, recent similar projects and ISLLP's database information.

Overall, the flow sheet utilizes proven mineral processing technology which is typical for a magnetite iron ore beneficiation plant.

Figure 17.3 – Process Plant Process Block Flow Diagram



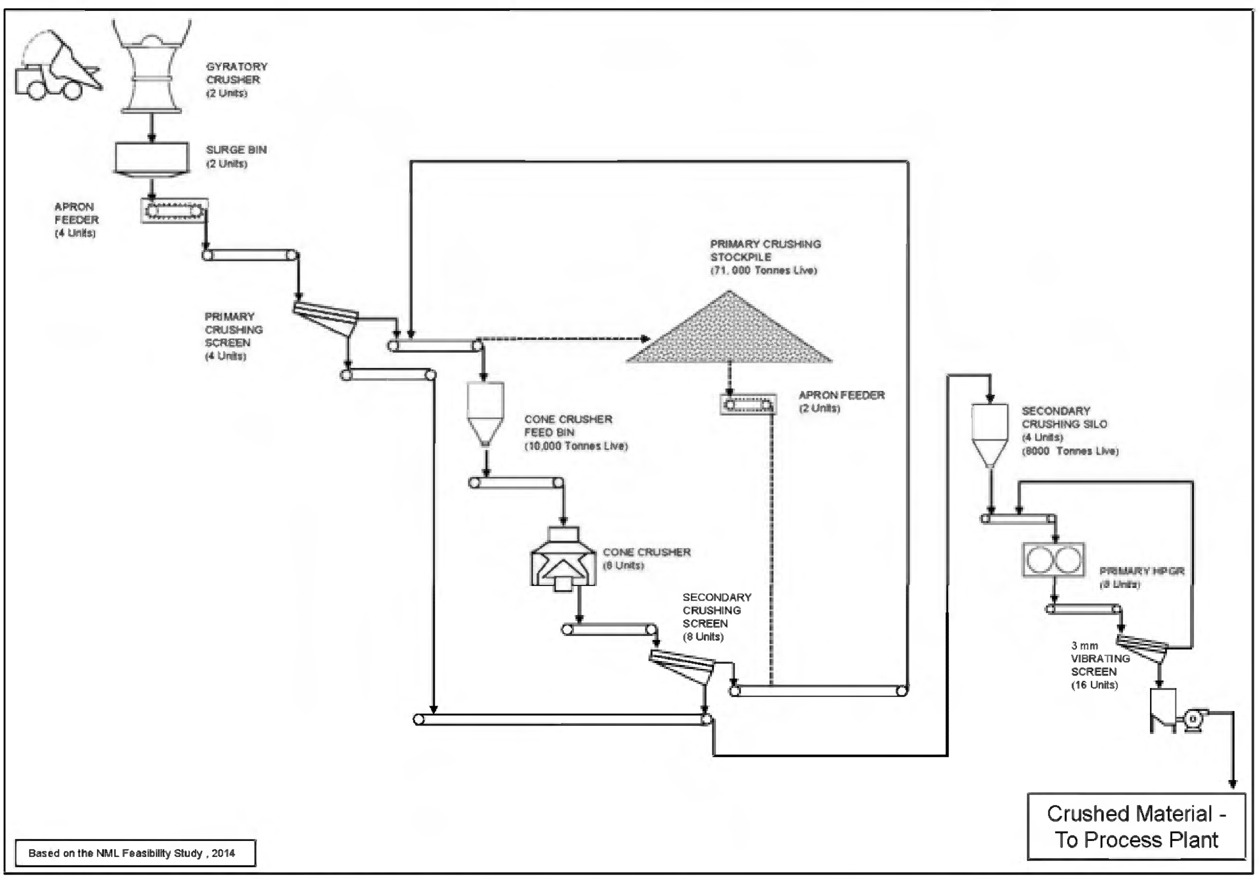
17.4.2 Flow Sheets and Plant Description

The simplified process plant flow sheet is divided in three (3) parts: dry plant comminution, Figure 17.4, the wet comminution and magnetic separation plant, Figure 17.12 and the flotation plant, Figure 17.22. The process is described in the following sub-sections.

a) Dry Plant Comminution (Primary and Secondary Crushing, Stockpiling and HPGR Grinding)

The objective of the dry comminution circuit is to reduce the particle size of the ore as efficiently as possible in terms of energy consumption, capital expenditure, and operating cost. The dry comminution flow sheet consists of two (2) stages of conventional crushing and one (1) stage of HPGR with screening. Figure 17.4 represents the simplified flow sheet.

Figure 17.4 – Dry Plant Comminution Flow Sheet



Two (2) primary crushers, rated at 7,500 t/h each, will process a nominal 10,026 t/h of run of mine. The crushed ore will be screened by four (4) double deck banana screens with an upper screen aperture of 100 mm and a lower deck aperture of 55 mm. The undersize will be conveyed to the secondary ore silos ahead of the HPGR while the oversize will be sent to the cone crushers 10,000 tonnes feed bin or the primary ore stockpile. Removing the finer material at this stage allows one third ($^1/_3$) of the run of mine to bypass the secondary crushers and reduces the fines in the primary stockpile making it less prone to freezing.

The primary crushing building consists of:

- Two (2) opposing dumping positions per crusher;
- Two (2) gyratory crushers (such as Metso 60 - 110E or FLS 60 x 113VD), the largest in operation;
- A common service and control area located between the two (2) dump pockets;
- Crushed material hoppers leading to apron feeders;
- Apron feeders that discharge onto reclaim conveyors which feed the primary screens.

A schematic arrangement is shown in Figure 17.5.

b) Primary Stockpile and Reclaim

The stockpile will provide buffer storage to allow continued operation while major maintenance is being performed or during winter storms, assuming that the stockpile is sufficiently replenished. The primary ore stockpile will have a 346,000 tonnes storage capacity of which 71,000 tonnes will be live. The stockpile will not be covered. The coarse ore in the dead zone will be pushed to the reclaiming tunnel by dozers.

Oversize from the four (4) primary screens will be conveyed to the top of the secondary crushing building where excess tonnage will be diverted and stacked on the primary stockpile.

The ore will be reclaimed via two (2) apron feeders feeding two (2) conveyors in a single tunnel under the pile. Figure 17.6 shows the primary stockpile and reclaim arrangement.

Further crushing is performed by cone crushers until material passing the 55 mm screens is sent to the four (4) 8,000 t silos prior to the HPGR grinding. Eight (8) roller presses are fed from the silos and material is screened by 16 single deck vibrating screens to a maximum size of three (3) mm before being sent to the wet magnetic separators (cobbers).

Figure 17.5 – Schematic Arrangement

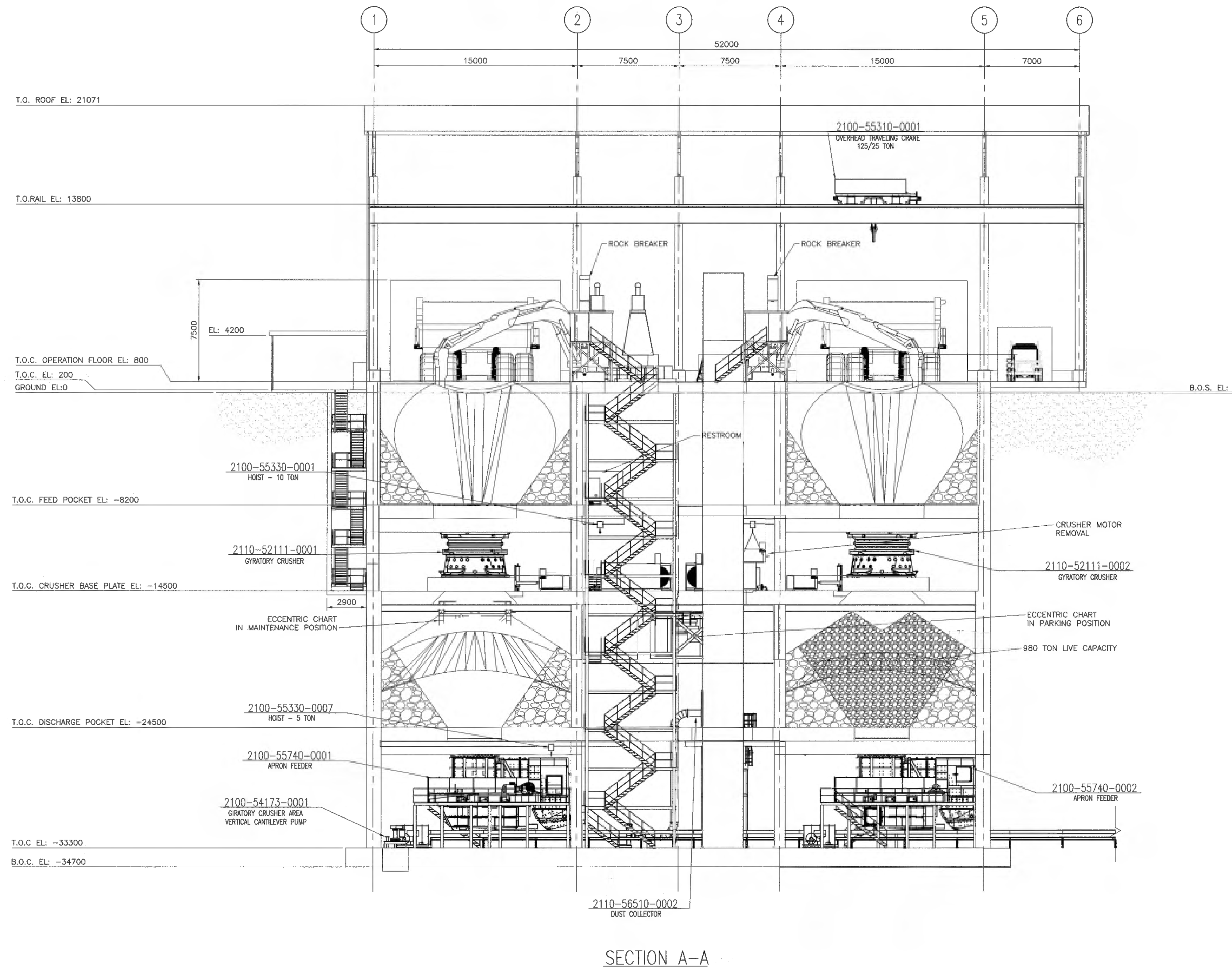


Figure 17.6 – Stockpile / Reclaim

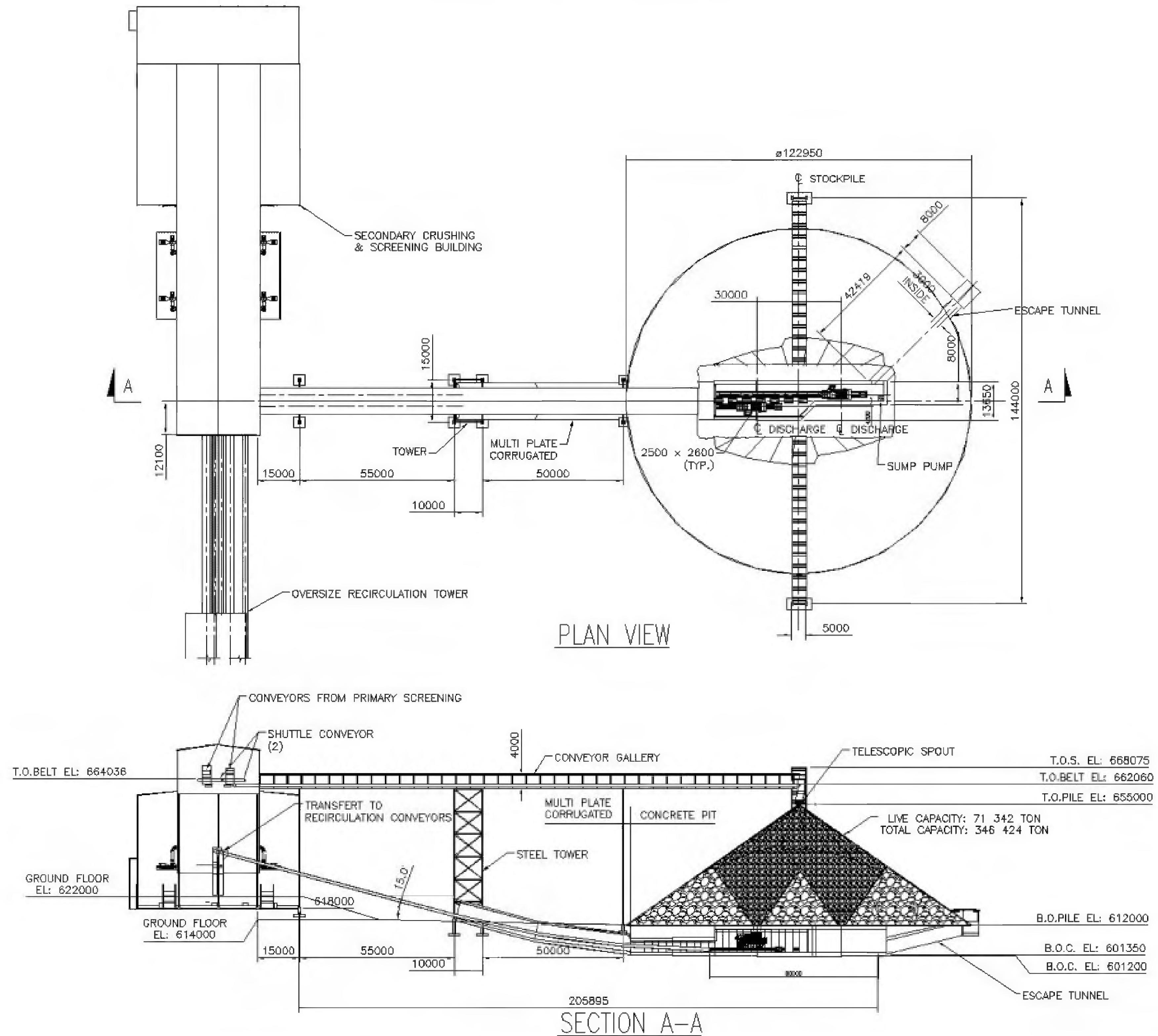


Table 17.5 – Dry Comminution Nominal Flow Rates

Items	Units	Nominal Flow Rate (t/h)	Feed Size
Primary Crushers	2	5,014	< 1 m
Primary Screens Oversize	4	1,667	> 55 mm
Primary Screens Undersize	4	840	< 55 mm
Primary Stockpile	1	346,000 t	> 55 mm
Secondary Crushers	6 + 2	1,510	> 55 mm
Secondary Screens Oversize	6 + 2	398	> 55 mm
Secondary Screens Undersize	6 + 2	1,112	< 55 mm
Crushed Ore Silos	4	8,000 t each	> 55 mm
Roller Presses	8	3,008	> 55 mm
Vibrating Screens Undersize	16	877	> 3 mm
Vibrating Screens Oversize	16	627	< 3 mm

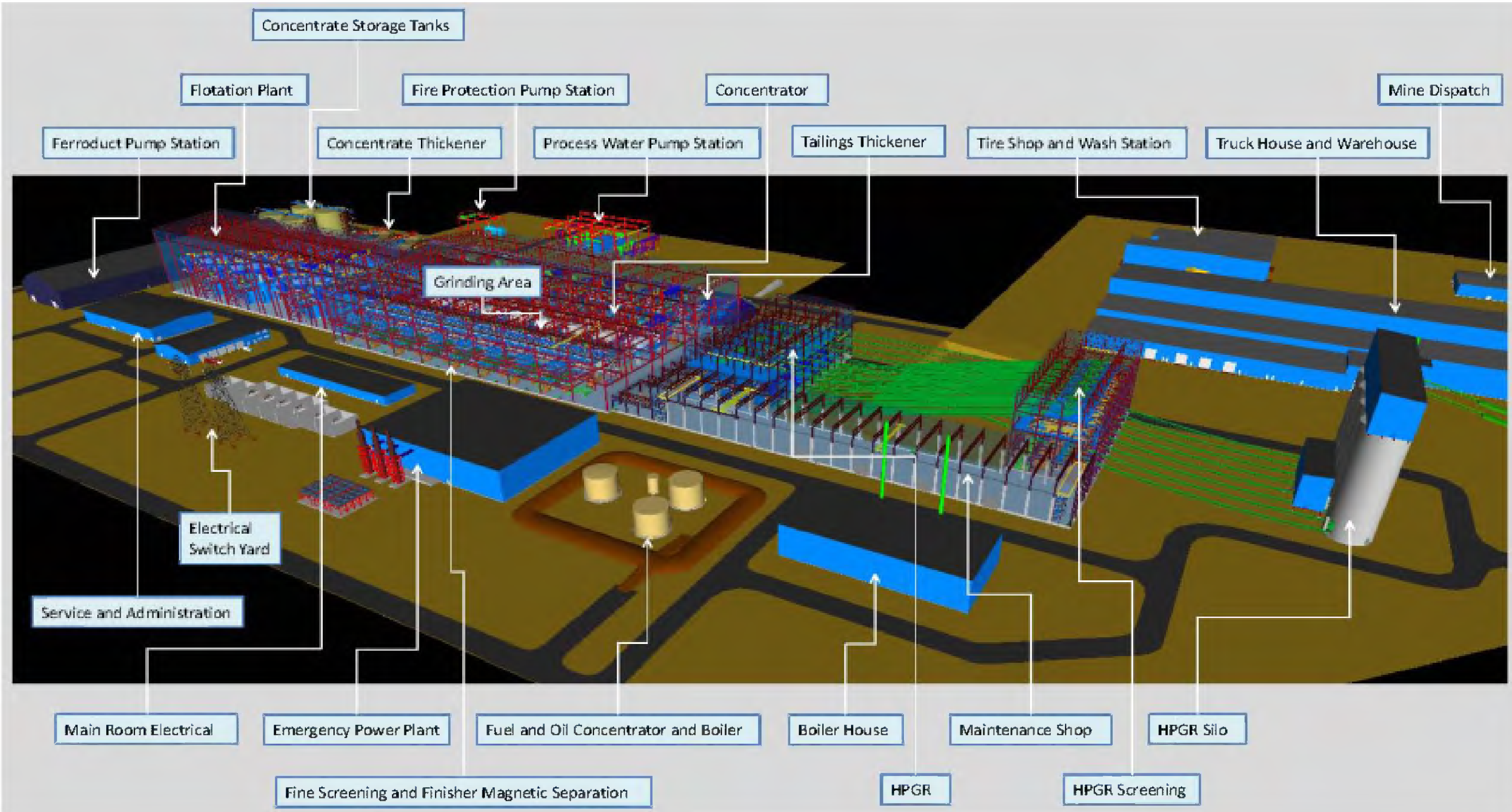
17.4.3 Secondary Crushing and Screening and HPGR Silos

The objective of the secondary crushing circuit is to reduce the size of the ore to a P_{80} of 36 mm prior to grinding with HPGR and cobber magnetic separation. Primary screen oversize and primary stockpile reclaim provide fresh feed to eight (8) cone crushers and secondary screens. The secondary screen oversize will be returned to the primary screens oversize conveyors and the undersize will be sent to four (4) fine ore bins.

The cone crushers will be fed from a 10,000 tonnes live capacity bin through eight (8) apron feeders each feeding a cone crusher. The cone crushers will be the largest proven units currently in operation, each having a 930 kW motor. Six (6) cone crushers will operate on average while two (2) remain on standby or under maintenance. A single-deck, 55 mm aperture banana screen will be under each cone crusher and will re-circulate 26 % of the crusher product to the secondary crusher feed bin. The screen undersize (<55 mm) will be transported by conveyors to the fine ore bins.

Figure 17.7 below shows the plant isometric view.

Figure 17.7 – Isometric View Plan



17.4.4 High Pressure Grinding Rolls

HPGRs were selected for the primary grinding circuit. The main advantages of HPGR over other mills are their low energy consumption, high reliability, simplicity and ability to dry grind. They also require no grinding media. NML has done extensive pilot scale testing of HPGR for grinding its taconite ore, which is a very hard and abrasive material and the three (3) major HPGR suppliers have successfully tested the NML ore.

Fresh feed to the HPGR will be reclaimed from the four (4) fine ore bins via eight (8) conveyors. The HPGR screening plant will be between the fine ore bins and the HPGR building. A set of conveyors will collect the screen oversize and return it to the HPGR feed bin. The three (3) mm screen aperture will generate a recirculating load of 140 %, resulting in the selection of eight (8) HPGRs of the largest proven operating capacity.

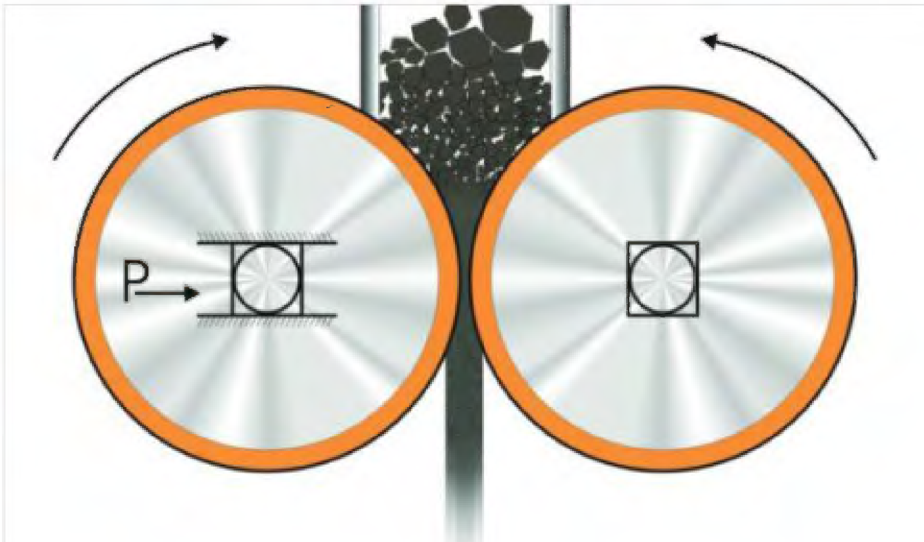
The HPGR variable speed capability will be used to ensure that the level in the feed hopper is maintained at a minimum height for proper choke feeding to prevent the loss of operating gap between the rolls.

Each HPGR unit will consist of two (2) horizontal rollers 2.2 m in diameter and two (2) m wide. One (1) roller is fixed while the other is movable and pressure is maintained by a hydraulic system to maintain an applied force. Ore will be continuously fed from the top between the rollers. After being pressed, a dense cake product is formed and discharged from the bottom. It will contain a high proportion of fines and unbroken particles which contain micro-cracks and fissures which form preferentially along planes of weakness. Figure 17.8 shows a schematic of an HPGR.

The HPGR application benefits include:

- Lower energy consumption than other comminution mills;
- High single machine throughput capacity;
- Resultant products have micro-cracks that improve the materials' grindability;
- High availability;
- Small footprint, lower construction investment;
- Low vibration and noise.

Figure 17.8 – HPGR Schematic



Dust extraction from the HPGR bins is via a bag house with the dust being returned to the HPGR product conveyor. Moisture in the blended feed will assist in dust suppression [two (2) % in the fresh feed mixed with six (6) % in the recirculation].

The main maintenance item of HPGR is the replacement of the worn studded roll liners (tyres). During this operation the rolls with worn out liners are removed from the HPGR and sent for re-lining and rebuilt rolls with new liners are installed.

In the current design, two (2) rows of four (4) HPGR's are aligned back to back and roll removal can be achieved along the centre aisle. Rolls are pulled out by block tackles onto the carts and moved to the last bay where a 125 t bridge crane can load them on flatbed trucks.

With eight (8) HPGRs, on average, one (1) set of tyre replacement is expected per month assuming a conservative 6,000 hours liner life. Figure 17.9 and Figure 17.10 show the HPGR building details.

Figure 17.9 – HPGR Plant Detail 001

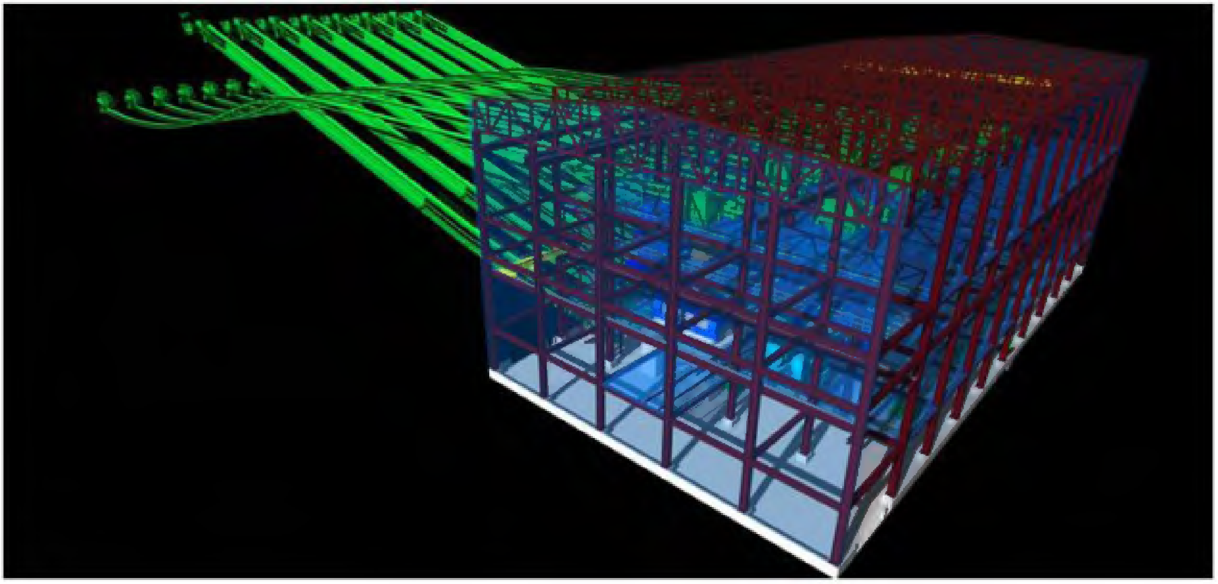
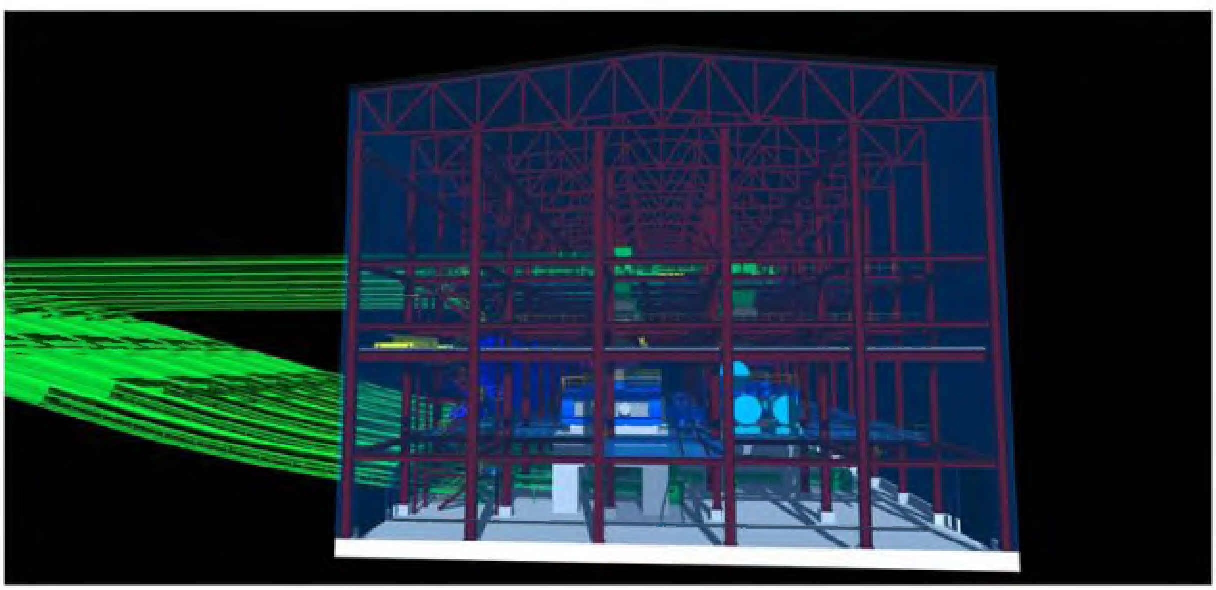


Figure 17.10 – HPGR Plant Detail 002

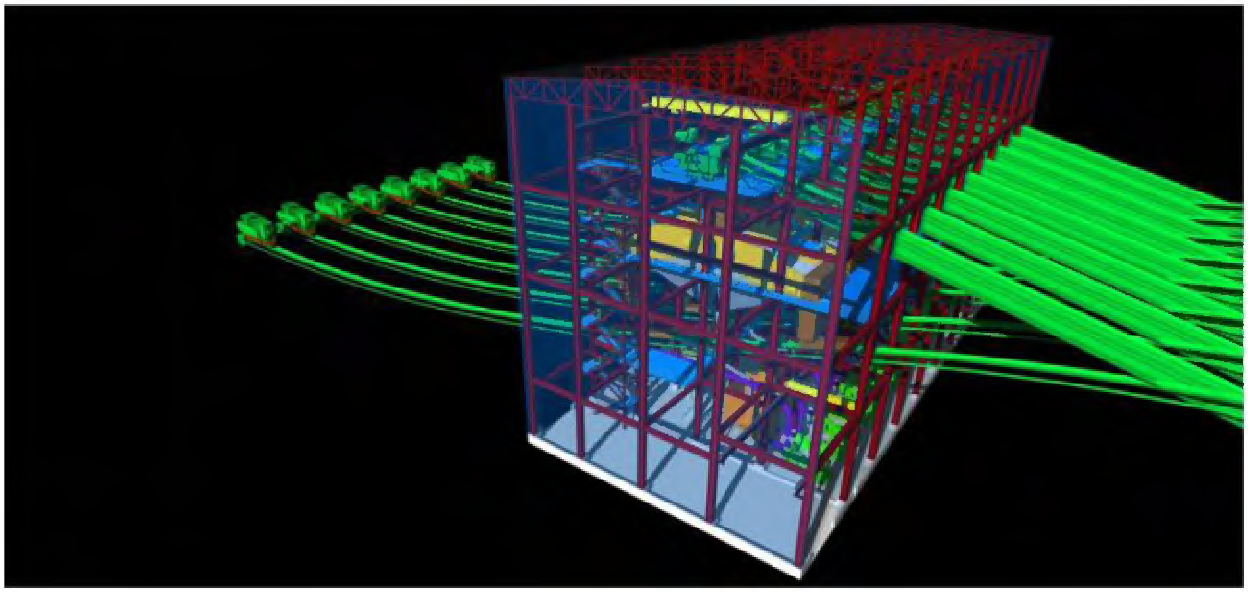


17.4.5 HPGR Screening

The eight (8) HPGR discharge will be sent to a screening building located midway between the fine ore bins and the HPGR building. The 16 wet vibrating screens aperture will be three (3) mm, the oversize (140 % of the fresh feed tonnage) will be recirculated back with fresh material on the HPGR feed conveyors exiting the fine ore bins and the screen undersize will be fed directly below the screen to pump boxes and pumped by eight (8) slurry pumps to the cobber magnetic separators.

Figure 17.11 shows the screening plant details.

Figure 17.11 – Screen House Plant Detail



17.4.6 Wet Plant Comminution

The objective of the wet plant is to continue the staged size reduction of the ore particles and liberate the iron ore mineral (magnetite) from the gangue minerals. The majority of the magnetite particles are liberated at a grind size of >45 microns, and the best results for iron grade and mass recovery occur in a range of 100 to 38 microns. In order to achieve maximum liberation of the magnetite, the wet plant (i.e. ball milling and classification circuit) reduces the size of the ore to a P_{80} of 50 microns.

In a recirculating stream, cyclones feed the rougher magnetic separation plant with the coarse underflow while the fines are carried to the final concentration stage in the overflow. The concentrate from the rougher circuit is then returned to the ball mill circuit. In this way, the gangue minerals, which are liberated at coarser sizes, can be discarded and the grinding power requirement is reduced prior to the final concentration stage.

The ball mill work index of approximately 16 kWh/t at the required grind size is comparable to other taconite iron ore deposits and is used for sizing of the comminution equipment.

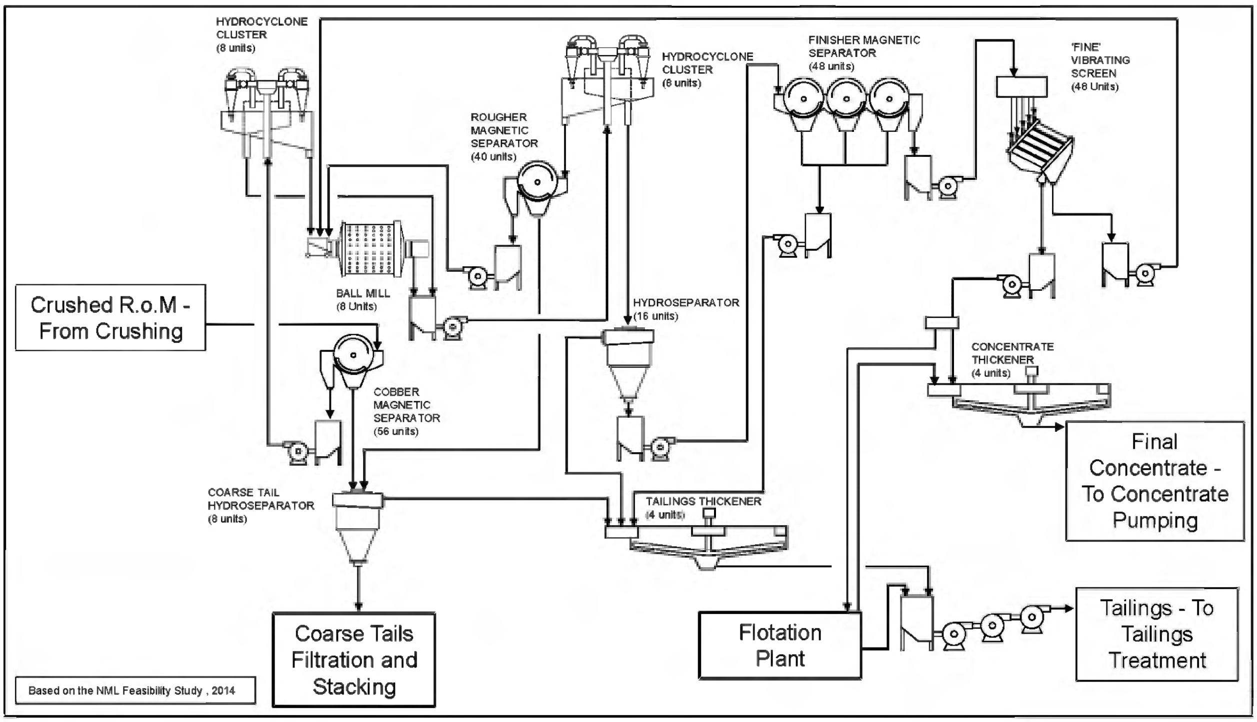
17.4.7 Low Intensity Magnetic Separation

The circuit incorporates three (3) stages of magnetic separation to take advantage of the progressive liberation of the ore and limit the amount of power needed to reach the final product size. Cobber separation and rougher separation each use single-drum LIMS units. The recovered concentrates are sent to the ball mill grinding stage for further liberation.

The cleaner separation is performed at the end of the process after desliming the ball mill product. The cleaning uses triple-drum LIMS units. The final recovered concentrate is screened after magnetic separation to remove the coarse fraction and to improve the final product grade.

Figure 17.12 below shows the flow sheet for the HPGR, the ball milling and magnetic concentration.

Figure 17.12 – Ball Milling and Magnetic Concentration Flow Sheet



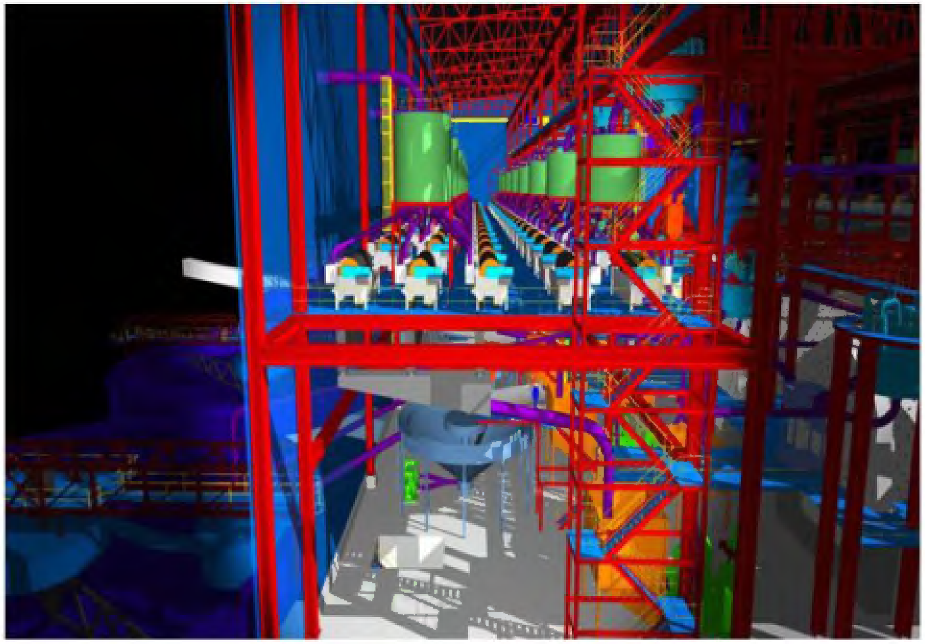
17.4.8 Cobber Magnetic Separation

The eight (8) HPGR screen undersize at 35 % solids are pumped directly to eight (8) lines of cobber magnetic separators where a weight recovery of 58 % is achieved.

The cobber tails are sent to hydroseparators to remove the coarse particles, which will be sent to a dewatering circuit for dry stack disposal. The hydroseparators overflow feeds the tailings thickener for recovery of fine particles for disposal in the tailings impoundment.

Concentrate from the cobber magnetic separators is pumped to dewatering hydrocyclones and the fine overflow sent to the ball mill discharge pump box, bypassing the mills. The coarser fraction is sent to ball mill feed. Figure 17.13 shows the cobber and rougher magnetic separation details.

Figure 17.13 – Cobber and Rougher – Magnetic Separation



17.4.9 Ball Mill Circuit

The cobber concentrate cyclone underflow, the rougher magnetic separators concentrate as well as the fine screen oversize enter the eight (8), parallel, 17.5 MW ball mills [measuring 13.4 m long (44 ft) with a diameter of 7.9 m (26 ft)] through eight (8) retractable chutes which decelerate the streams and provides for adjustment of slurry density. The chutes are also used to add grinding media and when moved back, they permit access to the mills with a liner handler. The feed is reground down to a P_{80} of 50 microns.

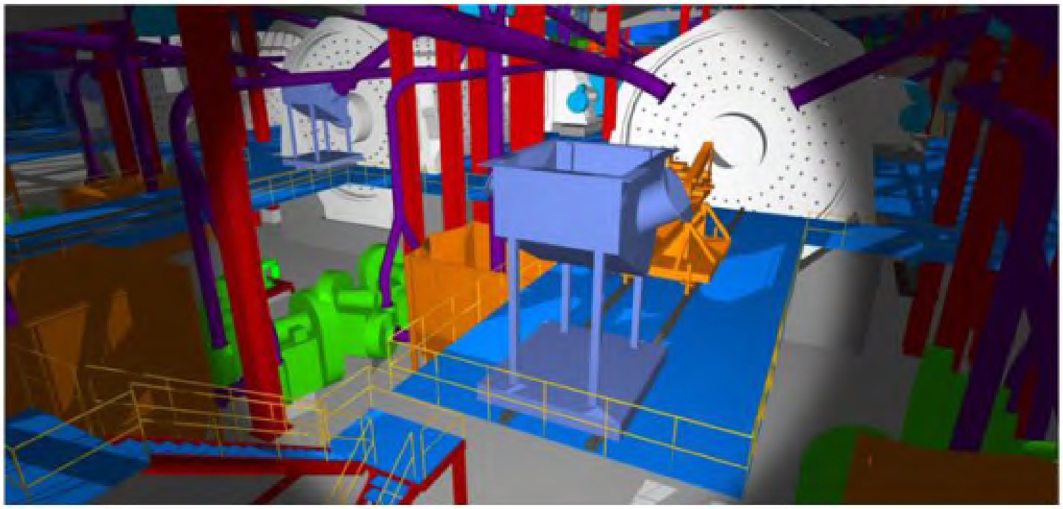
Maintenance of each ball mill motor, discharge pump and motor and other equipment will be aided by a pair of overhead bridge cranes running along the front and rear sides of the ball mill aisles.

a) Grinding Media

Ball usage is estimated at 120 tonnes per day. Grinding media (32 % chromium alloys) will be received in bulk at the outside wall of the ball mill building and dumped in a steel-lined concrete receptacle in batches of 60 tonnes. The balls will be reclaimed from the bottom of the receptacle by a bucket elevator. At the receiving end of the bucket elevator, an eight (8) tonne capacity clam shell bucket will receive the balls, handled by the overhead crane that services the loading end of all eight (8) ball mills.

Operators will direct the bucket over the ball loading through and open the clam shell bucket to recharge the balls. In normal operation, the operator will repeat this operation once per shift per ball mill. Figure 17.14 shows the ball mill feed end with the feed chute in retracted position and the liner handler in its working position.

Figure 17.14 – Ball Mill Feed End



b) Hydrocyclones Clusters and Deslimers

There are two (2) types of cyclone processes that feed the ball mills, the dewatering cyclones and the classifying cyclones. The first cluster of four (4) hydrocyclones per ball mill is a dewatering process. Its function is to lower the water content of the feed to the mills. By doing so, it also extracts five (5) % of the fines, which will report to the ball mill discharge. The second set of cyclones comprises eight (8) units per cluster for classifying particles. Like a screen it separates fines from coarses to feed either the rougher magnetic separators or the deslimers.

The deslimers, or density separators, will be used to remove the very fine particles of silica carried over with the concentrate. The slurry will enter the top of the deslimer where it is dispersed by a mixing box. Water is injected at mid level of the tank to create a rising current. Particles with a smaller size or a lesser density are carried upwards by the rising current and are collected in the tank overflow. Larger and heavier particles will settle at the bottom of the tank and will be extracted through the underflow valve when the settled bed reaches a pre-determined density or level, and will be sent to the finisher magnetic separators.

Although both SGA and SGS have been successful in using deslimers on bench scale devices, manufacturers have been hesitant to provide guarantees on a full scale basis for particle size separation under 100 microns. The two (2) Floatex units in the current design may not manage such a high feed without losing magnetite. Since the process requires separating particles with a P_{80} of ten (10) microns, larger desliming equipment should be considered in the next phase of engineering to guarantee the full scale application.

17.4.10 Rougher and Finisher Magnetic Separation

The last steps of beneficiation prior to flotation are rougher and finisher magnetic separation. They will receive pre-concentrate from the classifying cyclones underflow and deslimers underflow respectively. The final concentrate obtained with the finisher magnetic separators will have a Fe content of 69.6 % and approximately 3.0 % SiO_2.

Roughers are single-drum counter rotation (Figure 17.15), LIMS while finishers are triple-drum LIMS units with counter-current tank design (Figure 17.16).

Figure 17.15 – Schematic Illustration of a Wet Drum Magnetic Separator with a Counter-Rotation Tank Style (Rougher)

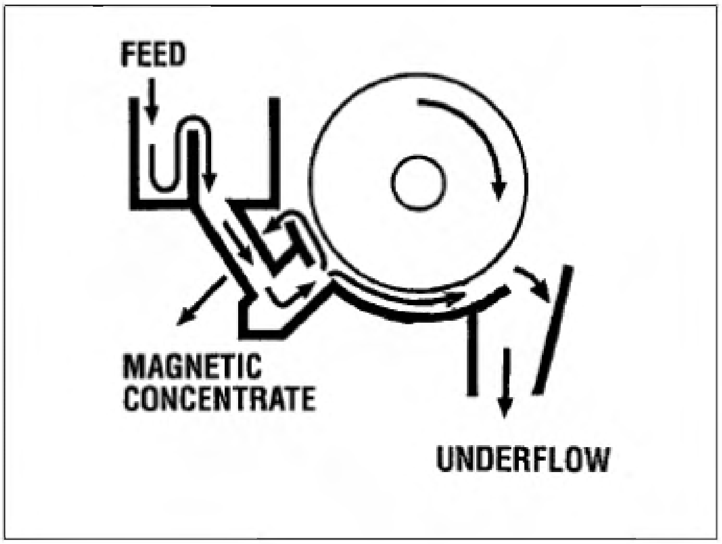
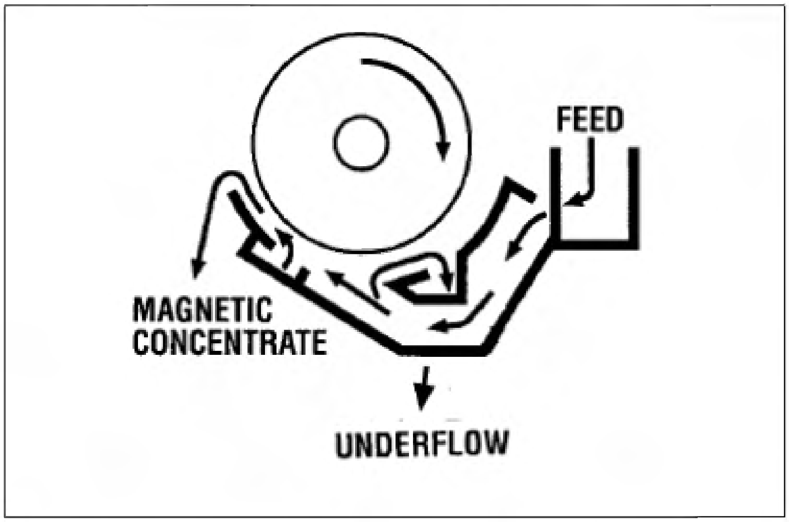


Figure 17.16 – Schematic Illustration of a Wet Drum Magnetic Separator with a Counter-Current Tank Style (Finisher)



a) Rougher Concentrate Stream

The slurry from the milling circuit classification cyclones underflow will be fed by gravity to the rougher feed distributors which in turn will feed the rougher magnetic separators by gravity. The rougher circuit comprises 40 single-drum counter rotation wet LIMS. The magnetite will be recovered by the rougher magnetic separators and delivered into the concentrate launder leading to the rougher concentrate pump box for further grinding.

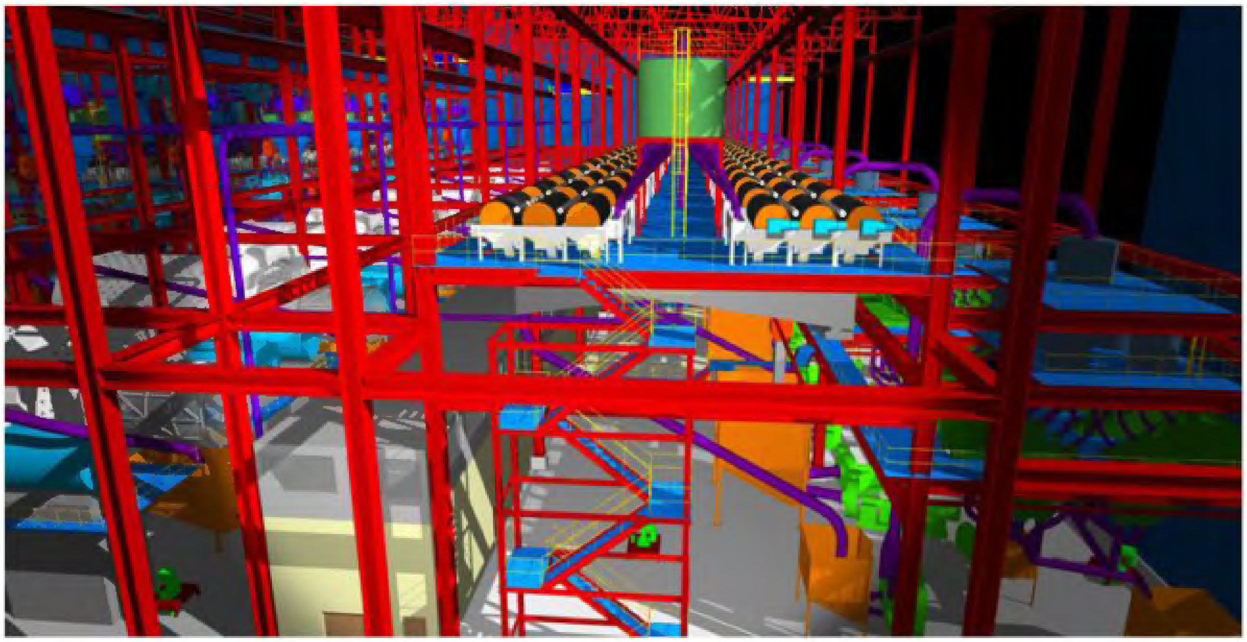
b) Rougher Tailings Stream

The non-magnetic tailings slurry will be directed to the coarse tailings hydroseparators via the rougher tailings launder. Before entering the hydroseparators, the rougher tailings will be mixed with the cobber tailings.

c) Finisher Concentrate Stream

The deslimer underflow will feed the finisher magnetic separators. The finisher circuit will have 48 triple-drums counter current wet LIMS. The finisher concentrate will be recovered into the concentrate pump box and pumped to the fine screens. Figure 17.17 shows the arrangement of the finisher LIMS.

Figure 17.17 – Finisher Magnetic Separators



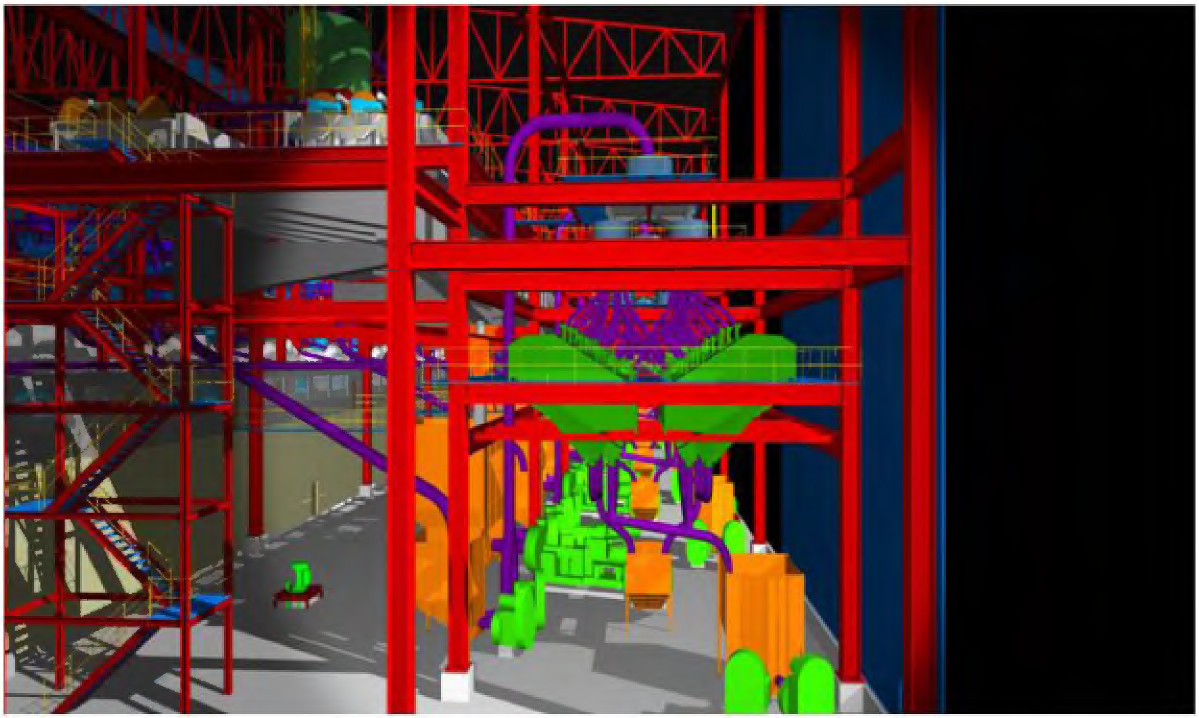
d) Finisher Tailings Stream

The non-magnetic tailings slurry will be directed to the tailings thickeners via the tailings launder and pump box.

e) Fine Screens

Fine screening of the finisher concentrate is a critical step in achieving the lowest possible silica level in the concentrate prior to flotation. Few suppliers can match the expertise of Derrick in being able to supply screens that work at such a small aperture. For this task, 48 units (5-deck Stacksizers) with a 0.053 mm aperture are proposed. Concentrate coming from the finishers will be evenly distributed in each Stacksizer. The oversize will be pumped back to the ball mill circuit, while the passing material will be sent to the concentrate thickeners or to the flotation plant for further silica reduction. It is expected that concentrate at this stage will contain 3.0 % silica. Figure 17.18 shows the arrangement of the fine screens.

Figure 17.18 – Fine Screens



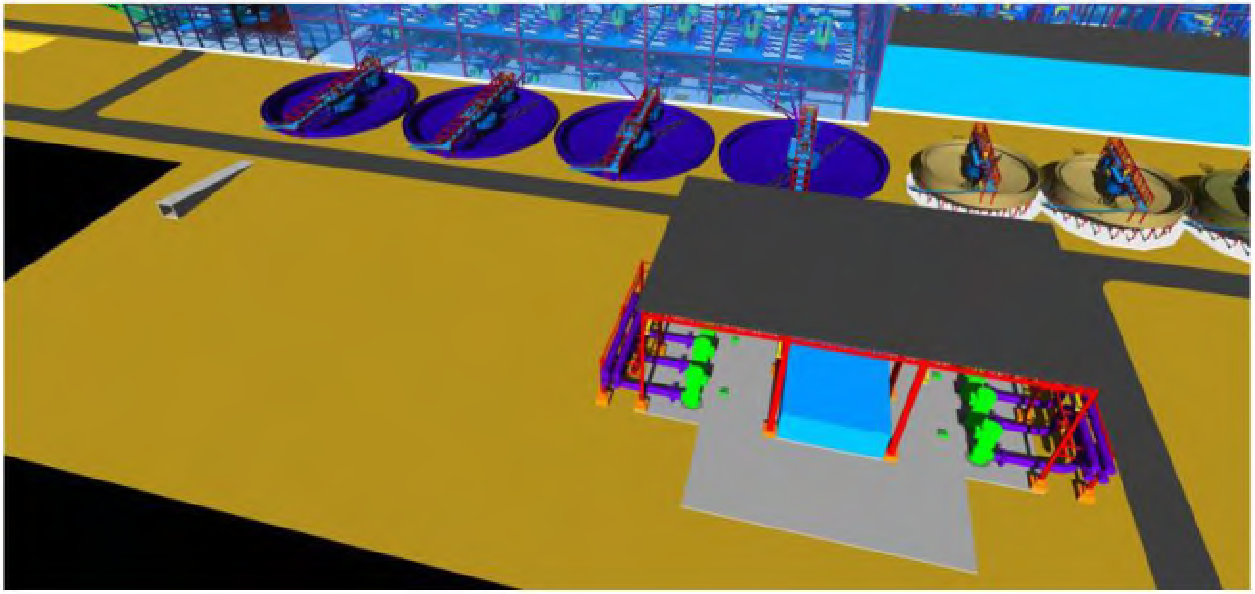
17.4.11 Tailings Processing

a) Tailings Thickening

The tailings thickening area will be used to partially dewater the tailings generated at the process plant, providing suitable material for discharge into the TSF in slurry form. The first stream of tailings is generated at the cobber and rougher magnetic separation and will be coarse tailings. Coarse tailings can be used as a dyke building material to extend the tailings storage facilities. The second stream will come from the deslimers and finisher magnetic separation and has a finer particle size. A third and much smaller stream will be generated at the flotation plant.

Figure 17.19 shows the tailings thickening area.

Figure 17.19 – Tailings Thickening



b) Coarse Tailings Hydroseparators

The tailings from the cobber and rougher circuits will pass through the tailings hydroseparators and exit in two (2) streams; the coarser tailings fraction at a cut point between +0.15 to +0.3 mm which will be determined during operations. The fines portion, along with most of the water, will flow to the tailings thickeners where it will be combined with the other tailings streams.

c) Tailings Thickeners

Coarse tailings hydroseparators overflow as well as finisher tailings and deslimer overflow will be combined in four (4) 60 m diameter thickeners.

The tailings solids will settle to the bottom of the thickener where they will be raked into the central discharge cone. The clarified process water will overflow to the process water storage basin via a channel.

The thickened tailings from the hydroseperator and thickener underflow will then be pumped to the tailings pumpbox where they will be combined with the flotation tailings. The tailings will then be pumped to the TAI. The heavier coarse fraction that settles first on the beaches reclaimed to raise the tailings impoundment area dykes.

Four (4) heavy duty concrete thickeners have been selected. The center drive steel rake mechanisms will be designed for high torque to provide an average of 60 % solids by weight underflow. The center column supports the drive and rake assembly and the access bridge.

Each thickener has dual slurry pumps [one (1) running – one (1) stand-by] located underneath the thickener in a service tunnel. This tunnel provides maintenance access to slurry pumps and piping and it is open at both ends to provide emergency egress from this tunnel.

d) Flocculants for Tailings Thickeners

Flocculants and coagulant will be used to assist the settling of the solids in the tailings thickeners. Coagulants use mainly the electro-chemical forces to form small agglomerates with the ultra fine particles. With the addition of the flocculant, larger flocs are formed by agglomeration of multiple smaller agglomerates from the coagulation as well as with the larger particles. These heavier flocculants can settle much faster than the single particles on their own.

e) Tailings Disposal

The KéMag site benefits from a natural depression in the terrain which allows for conventional disposal of fine tailings with few geographical restrictions. The initial dykes would be built with moderate quantities of earthwork prior to the start of production enabling three (3) years of tailings disposal. As tailings are pumped and discharged as slurry in the impoundment area, the coarse fraction settles first on the top of the beach and dewaters naturally. The tailings discharge point is then moved to another location and the coarse fraction is reclaimed to raise the dykes.

i) Description of Tailings Management Facilities

For KéMag, it is proposed to impound tailings in a valley (Figure 17.20) located in a different watershed basin west of the pit on claims recently acquired by NML. The proposed design uses favorable topography to store the first three (3) years of tailings production with the least amount of embankment fill material.

The total annual tailings production is 63 Mt, the tailings are pumped to the TMF at 60 % solids. The tailings pond shall be storing nearly 38 Mm^3 of tailings per year. The preliminary mass balance indicates that nearly 45 % of the tailings will be coarser than 0.100 mm and 55 % finer. Finally, preliminary tests indicate that the tailings are not likely to be acid generating (SENES 2011);

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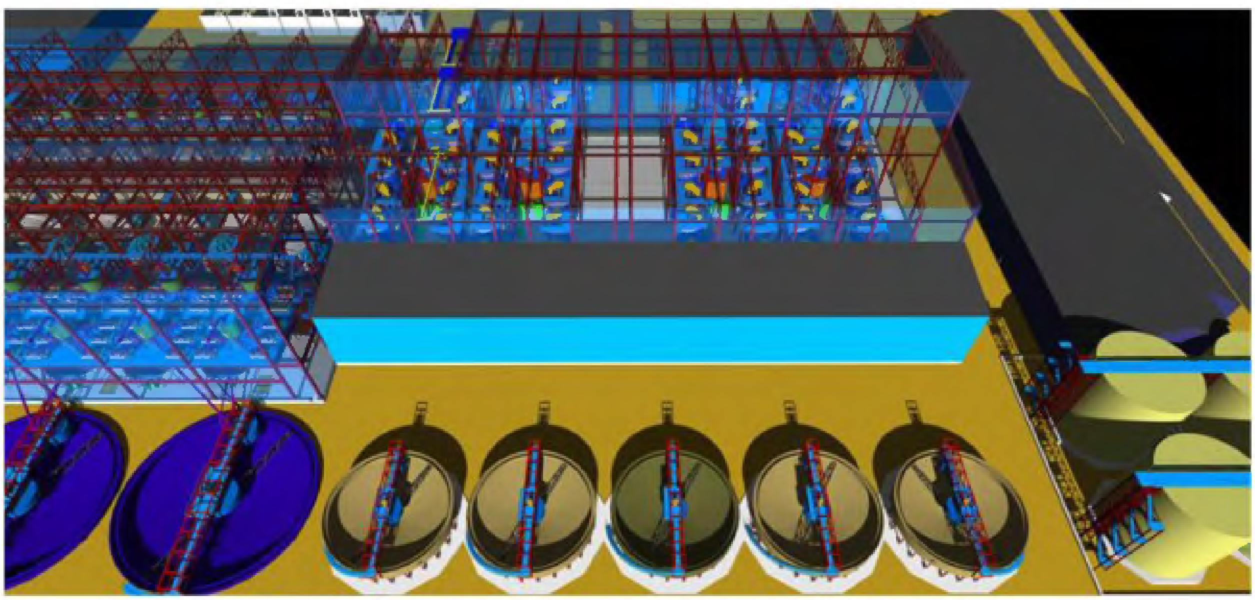
The following criteria apply for the closure of the different tailings impoundment areas:

- The fine tailings will be progressively covered with a layer of coarse tailings topped with a thick layer of non acid generating waste rock;
- The tailings water pond will have an adequate freeboard and flood storage capacity and will be equipped with a closure spillway to safely discharge the probable maximum flood without overtopping any of the embankments.

17.4.12 Concentrate Thickening and Handling

The concentrate from the flotation lines and the magnetic separation plant will be combined in four (4) concentrate thickeners operating without flocculant (see Figure 17.21). Each thickener will be 26 m in diameter with a capacity of 750 t/h. The underflow (at 70-75 % solids) will be pumped to the slurry storage tank distributor and distributed to the four (4) agitated storage tanks. The thickener overflow will be directed to the flotation process water basin by gravity.

Figure 17.21 – Concentrate Thickening and Handling



17.4.13 Flotation

To produce concentrate with less than 2.2 % silica, flotation is required.

Pilot plant testing of magnetic separation has shown that its limit is reached at a silica level between 2.6 % and 3.2 %. In order to produce low silica BF fluxed or DR grade pellets, the concentrate must contain 2.2 % and 1.5 % silica respectively so flotation has to be added to reduce the silica grade.

All flotation steps will use tank type flotation cells. Regrinding will be performed in fine ore grinding mills and classified with hydrocyclones.

The plant and equipment layout is based on the same principle as the rest of the process plant: four (4) distinct parallel lines, each with its three (3) circuits (sub-areas):

- The silica rougher flotation circuit, where 77 % of the flotation iron concentrates will be produced at 1.31 % SiO_2. The silica froth produced in this sub-area will be transferred at a rate of 601 t/h to the flotation regrind circuit;
- The flotation regrind circuit, where the silica froth will be ground to a P_{80} of <18 µm to liberate the iron particles. After magnetic separation, the non-magnetic tailing will be transferred to the flotation tailings thickener and the magnetic material will be floated in the cleaner flotation circuit;
- The cleaner flotation circuits, where magnetic material will be floated again to produce the second iron concentrate at 2.28 % SiO_2 and a second flotation tailings.

The flotation plant will operate at a nominal rate of 2,491 t/h of iron concentrate (i.e. the two (2) iron concentrates from rougher and cleaner flotation circuits together) at 1.50 % SiO_2. If the silica level for DR grade pellets feed is above 1.5 %, an organic binder will be utilized at the pelletizing stage in order to keep the SiO_2 level in the finished product at or below 1.8 %. The flow of final tailings (from the regrind and cleaner flotation circuits) will be 145 t/h at 53 % Fe and 23.96 % SiO_2.

Figure 17.22 below shows a simplified flow sheet of the flotation plant while Figure 17.23 shows the flotation area.

Figure 17.22 – Flotation Plant Flow Sheet

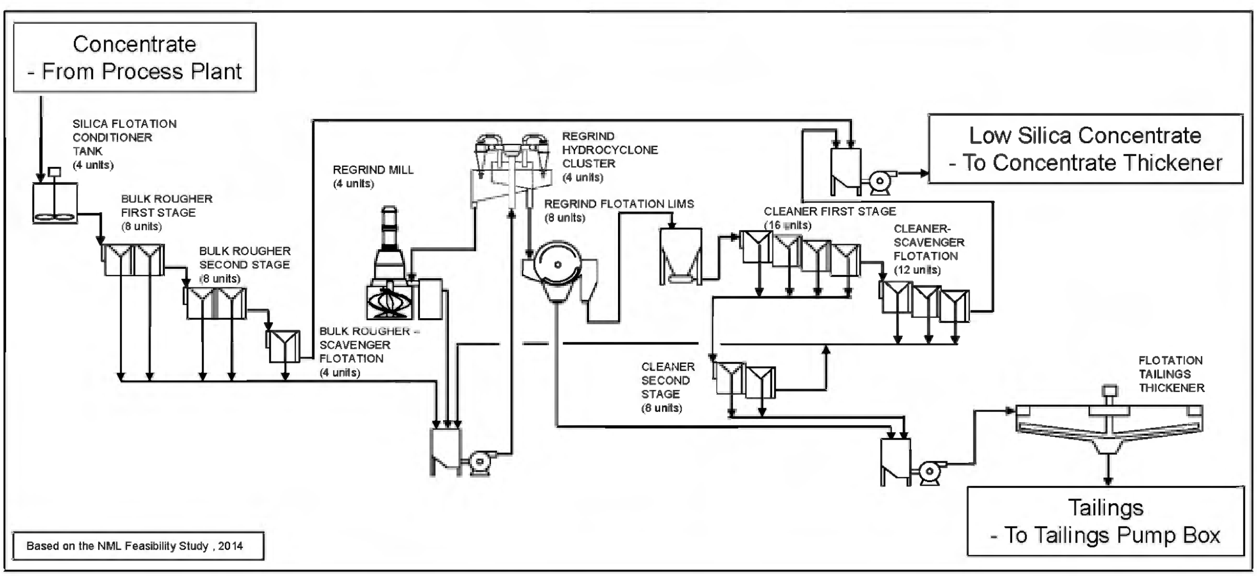
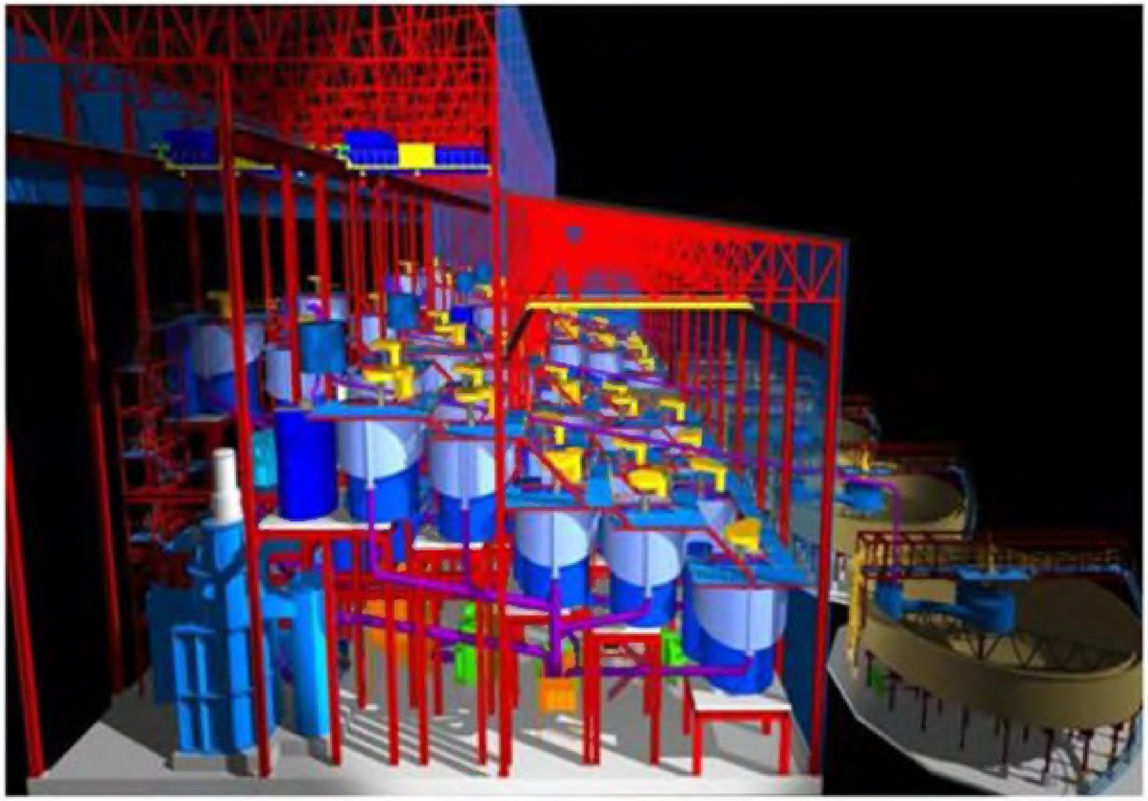


Figure 17.23 – Flotation



The flotation circuit may be bypassed at any time, thus increasing the silica level in the final concentrate. When all four (4) flotation lines are in operation, the lowest achievable silica content will be produced. When lines are bypassed, the silica content in the final concentrate will be the result of blending the concentrates from the magnetic separation with material which has been floated. Magnetite plant concentrate and flotation concentrate will be blended in the concentrate storage tanks after the concentrate thickeners and will be pumped to Pointe-Noire.

a) Silica Rougher Flotation Circuit

Rougher flotation will be performed in a single-pass open circuit. The feed from magnetic concentration will first be sent to conditioning tanks, where caustic starch is added as a depressant for magnetite. The amine collector will be added in various stages along the rougher flotation cells.

Each line will comprise of two (2) sets of two (2) flotation cells in series which will be followed by a fifth cell. Air will be injected into the bottom of the cells to generate froth. The silica froth will then be sent to the regrind circuit. The sink fraction containing over 70 % Fe and low levels of silica will be collected and sent to the concentrate thickeners.

b) Flotation Regrind Circuit

The froth from rougher flotation cells will be pumped to hydrocyclones for classification. One (1) cluster of cyclones will service each line. Each cluster will contain 20 cyclones. For each line, the coarse fraction will be reground in a fine grinding mill in closed circuit with the cyclones. The cyclone overflow at a D_{80} of <18 microns will be fed to two (2) single-drum magnetic separators in parallel for recovery of the magnetite and for density control. The magnetic concentrate will be treated in the cleaner flotation circuit. The non-magnetic material containing 54.4 % silica and 19.9 % Fe represents less than one (1) % of the flotation plant feed. This material will join the main tailings stream going to the flotation tailings thickener.

c) Cleaner Flotation Circuit

The cleaner flotation circuit composed of two (2) stages of cleaners followed by a cleaner-scavenger stage will increase the iron content of the magnetic concentrate from the regrind circuit. In this circuit, each line will be composed of four (4) cells as a first cleaner stage, two (2) cells for the second cleaner stage and three (3) cells for the cleaner-scavenger stage.

The silica froth of the first cleaner stage will be sent to the second cleaner stage and the iron-rich sink fraction will report to the cleaner-scavenger flotation cells.

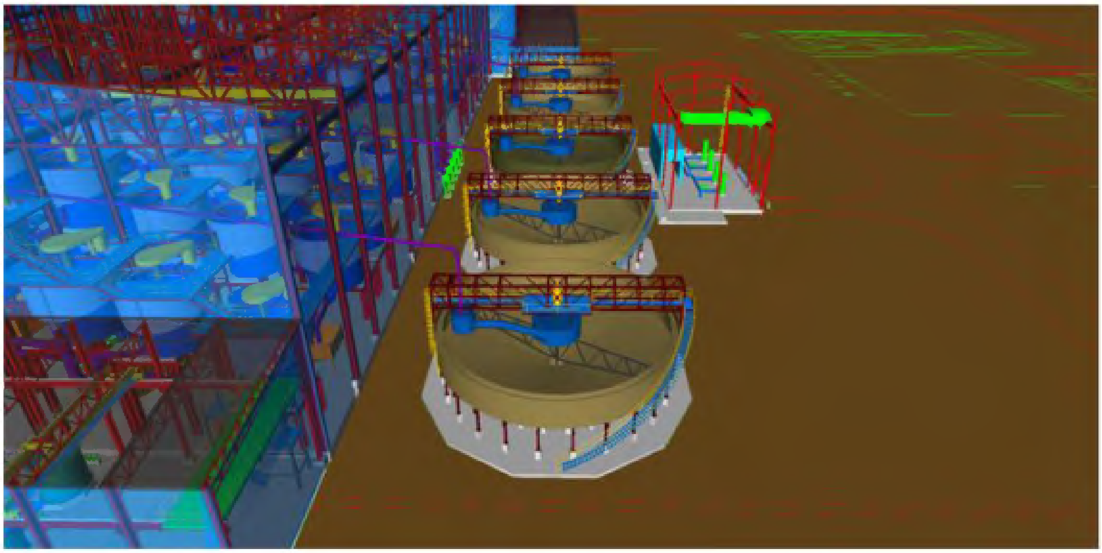
The second cleaner stage froth is the final tailings stream of the flotation plant and will be mixed with the regrind circuit magnetic separators tails and will then be sent to the flotation tailings thickener. The sink fraction, containing around 63.8 % Fe, will return to the regrind mill for further liberation.

The cleaner-scavenger froth will be directed to the regrind mill and the iron-rich sink fraction will be mixed with the rougher flotation concentrate and sent to the concentrate thickeners.

d) Flotation Tailings Thickeners and Flotation Process Water Facilities

Figure 17.24 shows five (5) thickeners of 26 m diameter, the third thickener (in the middle) is the flotation tailings thickener and the other four (4) are the concentrate thickeners.

Figure 17.24 – Flotation Tailings Thickener, Concentrate Thickeners and Flotation Water Facilities



The flotation tailings from the regrind circuit and the cleaner circuit of each line will be mixed in four (4) pump boxes and sent to the flotation tailings thickener. The thickener underflow at 60 % solids (145 t/h of solids) will be combined with the rest of the process plant fine tailings in the final fine tailings pump box and sent to the TSF.

The overflow from the flotation tailings and the four (4) concentrate thickeners will be directed to the flotation process water basin. All process water used in the flotation plant will originate from this basin. The excess water will be pumped to the tailings thickeners.

17.5 Plant Control System

The KéMag concentrator and associated slurry transportation system equipment will be controlled, operated, and monitored from a Central Control Room ("**CCR**") located in the concentrator building. Equipment status, process parameters and alarms will be transmitted to the pellet plant CCR via a fibre optic link.

The flotation plant and the KéMag crusher area equipment will be controlled, operated and monitored from dedicated Local Control Rooms ("**LCR**") located in their respective areas.

Control system processors will be able to exchange process data, alarms, interlocks and status of mechanical equipment in real time.

Plant Control System ("**PCS**") Input/Output cabinets will be located in area electrical rooms. All analog (process controls) and digital (on-off) signals associated with various on-site process units, equipment interlocks, alarm functions, and shutdown functions will be implemented in the PCS. Inputs and Outputs associated with skid-mounted equipment will be wired to the nearest PCS cabinet.

Local Control Panels ("**LCP**") will be provided for operation, monitoring, and start-up of major vendor supplied mechanical packages. Cables will enter through the floor.

The main component of the control system will be:

- Concentrator Central Control Room;
- Technical Room ("**TR**");
- Local Control Rooms:
  - Primary Crushers;
  - Secondary Crushers;
  - Flotation Plant;
- Local Control Panels.

17.6 Slurry Transportation System

The concentrate will be transported from the concentrator to the export terminal located at Pointe-Noire, where the slurry will be filtered and pelletized or sold as pellet feed. The description of the slurry transport system is presented in Section 18.1.

17.7 Filtering and Pelletizing

This Section summarizes the study prepared by Outotec for filtration and pelletizing plant including development of the design criteria and the design for these plants. Outotec's mandate was to engineer two (2) pellet lines each with 8.5 Mt/y of capacity with complete capital and operating cost estimate. The plants will be located in Pointe-Noire adjacent to the stockyard and near the shipping facility.

Pot grate tests were conducted in two (2) stages. The initial work was done in 2012 with Outotec's input at the SGA laboratory because of their familiarity with NML's material and processing. The collected data was used by Outotec to complete engineering and estimating for the Feasibility Study. The second round of tests were undertaken in 2013 at Outotec's facility to fine-tune the design and provide a process guarantee for pelletizing plant throughput and pellet quality.

The following Sections outline the test work, design criteria and the details of the facilities included in Outotec's scope of work.

17.7.1 Scope of Work

Once the concentrate slurry is received in the agitated tanks, all 22 Mt/y will be filtered using ceramic disc filters. The filter cake will then be transported to the pelletizing plant and/or to concentrate storage for direct sales as iron ore concentrate at an annual rate of six (6) Mt/y. Two (2) pelletizing lines of 8.5 Mt/y each can produce low silica fluxed

BF grade pellets or DR grade pellets. The design capacity is for 17 Mt/y of pellets (16 Mt/y of concentrate plus additives and gain on conversion).The production is divided as follows:

- Twelve (12) Mt/y of low silica BF pellets with 2.5 % SiO_2;
- Five (5) Mt/y of DR pellets with 1.8 % SiO_2; and,
- Six (6) Mt/y of concentrate for direct sales.

The pelletizing facilities include the following:

- Port slurry reception, handling and storage;
- Coarse additive handling, storage, reclaim and grinding for limestone, dolomite and bentonite;
- Organic binder handling, preparation and distribution;
- Feed preparation and filtration;
- Blending and mixing of filter cake and additives;
- Green balling;
- Pellet induration;
- Product screening and pellet coating for DRI product;
- Process gas and plant dedusting systems;
- Cooling water system;
- Heavy fuel oil ("**HFO**") system;
- Compressed air system;
- Intermediate pellet storage;
- Pellet storage and reclaiming facility;
- Concentrate for sale storage and reclaiming facility;
- Process automation control;
- Lifting equipment, passenger lifts, cranes, monorail trolleys with hoists;
- Firefighting system with required pumps and hydrants within the pelletizing plants.

Automatic process control is considered in most process areas. Modern plant design and automatic control elements help to improve plant availability and decrease the specific production costs.

Supporting facilities such as pellet and concentrate storage and handling, administration building, maintenance facilities, sewage treatment plant, water treatment plant, diesel tank farm, and flocculants storage are described in Section 18.0.

17.7.2 Pellet Test Work and Process Development

a) Pot Grate Test Campaign at SGA

The tests conducted at SGA with Outotec representatives sought to define and optimize the operating parameters which would deliver the desired metallurgical and physical quality of pellets at the required throughput. Between March and July 2012, a total of 24 pot grate tests were run with a focus on optimizing various parameters such as: temperature profile, bed height, zone configuration and basicity in order to meet Outotec's and NML/TS's requirements. Ten (10) tests used DR grade concentrate and 14 tests used BF grade from KéMag samples.

The SiO_2 content of the concentrate was adjusted to 1.5 % and 2.25 % for the DR grade and BF grade pellets respectively. This was done by mixing in appropriate proportions of four (4) different types of concentrates which were available from previous beneficiation work. The basicity levels, CaO/SiO_2, were adjusted between 0.4 % and 0.8 % for DR grade pellets and between 0.7 % and 1.0 % for BF grade pellets. Limestone was used as the fluxing medium in all tests except in Test #7 for BF grade pellets, which used dolomite. The level of bentonite was maintained constant at 0.48 % to 0.49 % throughout the tests.

After reviewing early test results, it was decided to limit the specific surface of the concentrate to 1,800 Blaine value to achieve optimum drying and firing during the pelletizing process.

b) Pot Grate Test Campaign at Outotec

The second round of tests were conducted in 2013 at Outotec to target 8.5 Mt/y under the conditions developed at SGA with some optimization. The main purpose of these tests was to improve the productivity in view of providing the throughput guarantee for the two (2) 816 m^2 machines. Outotec performed 12 pot grate tests using a new sample of LabMag Block B concentrate prepared at CMRL in Minnesota. In nine (9) of the tests, BF grade pellets were produced from a blend of floated and non-floated concentrate. The remainder was DR grade from floated concentrate.

The first test used three (3) wind boxes for Updraft Drying ("**UDD**") and three (3) for Downdraft Drying ("**DDD**") as per the SGA configuration. Subsequent tests settled on four (4) UDD and four (4) DDD as this showed the desired increase in productivity. The firing pattern was also varied (preheating, firing and after-firing zones) in order to obtain the desired physical and metallurgical qualities.

The chemical and metallurgical design criteria were determined by market study for high quality BF and DR grade pellets. The SiO_2 content of the concentrate was at 1.6 % and 2.3 % for DR and BF grade pellets respectively. This was achieved by flotation at pilot scale by SGA in Germany. The DR grade pellet feed results from 100 % floated material while the BF grade pellet feed was a mixture of 50 % magnetic separation concentrate (unfloated) and 50 % floated DR grade pellet feed material as will be applied in the full scale plant.

The basicity levels (CaO/SiO_2) were at 0.4 for DR grade pellets and 0.85 for BF grade pellets. Limestone was used as the fluxing medium in all tests. The level of bentonite was at 0.48 % for the first three (3) tests and increased to 0.54 % in the subsequent tests.

Based on the test results, Outotec issued a confirmation letter guaranteeing a nominal annual capacity of 8.5 Mt/y for each of two (2) 816 m^2 pellet machines. It was also stated that each pellet line was expected to ramp up to 9.0 Mt/y after a period of stable and routine operation.

i) Green Ball Properties

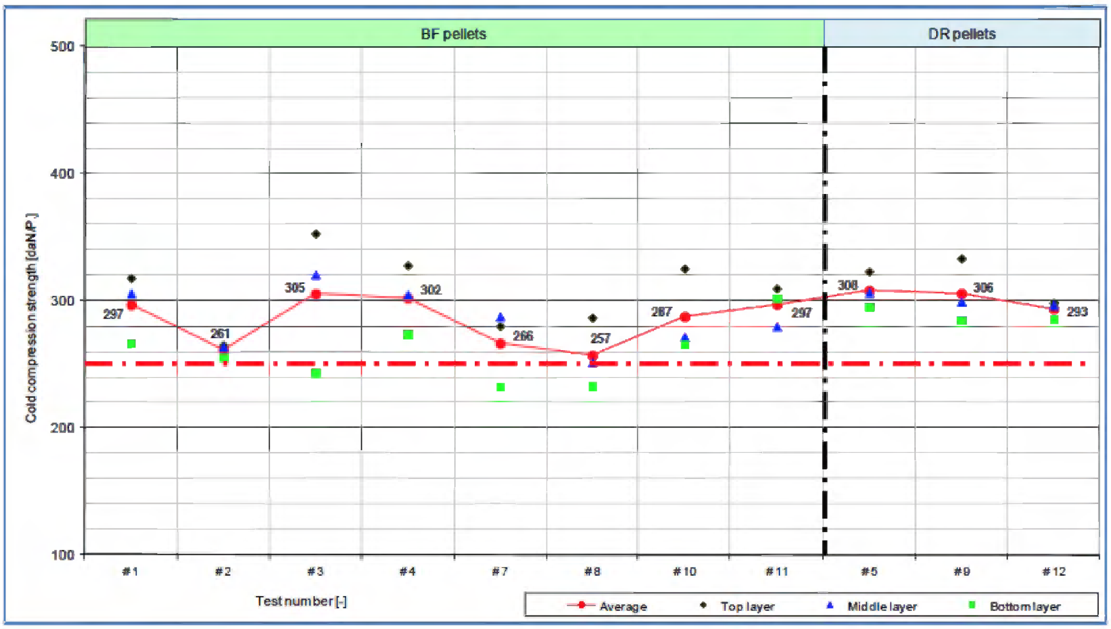
The Outotec green ball wet compression strength was acceptable throughout the test work, being always above one (1) kg/pellet. The dry compression strength was between 2.5 and 3.8 kg/pellet. The drop number varied between 2.7 and 3.8.

ii) Properties of Fired Pellets

Twelve (12) tests were undertaken [nine (9) BF and three (3) DR grade] to develop a design basis for the proposed equipment and to demonstrate that it will meet the throughput and quality requirements. While some parameters fluctuated during the testing period, the conditions of Tests #9 and #10, for DR and BF pellets respectively, were selected as the basis for the final equipment design and process guarantee.

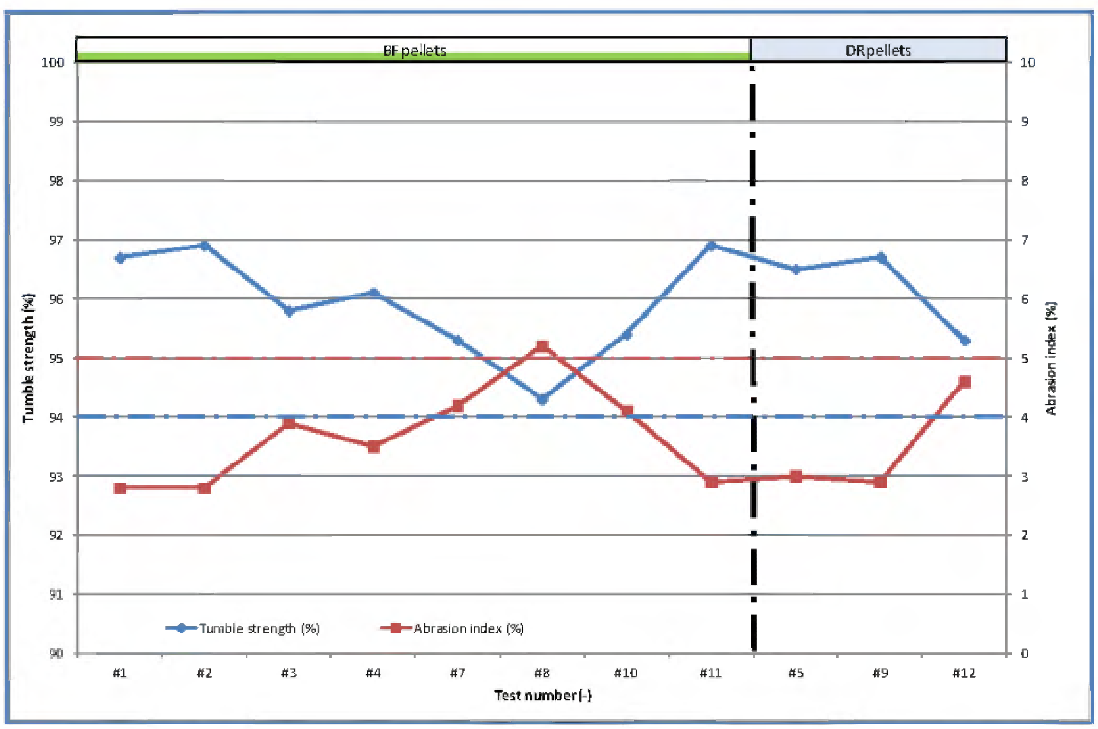
Cold compression strength values required are a minimum value of 280 daN/pellet for DR grade pellets and of 250 daN/pellet for BF grade pellets. This requirement could be reached for the average values in almost all tests, with results between 293 to 308 daN/pellet for DR pellets, 257 and 305 daN/pellet for BF pellets (average values over all layers). The cold compression strength values are shown in Figure 17.25.

Figure 17.25 – Cold Compression Strength (Outotec)



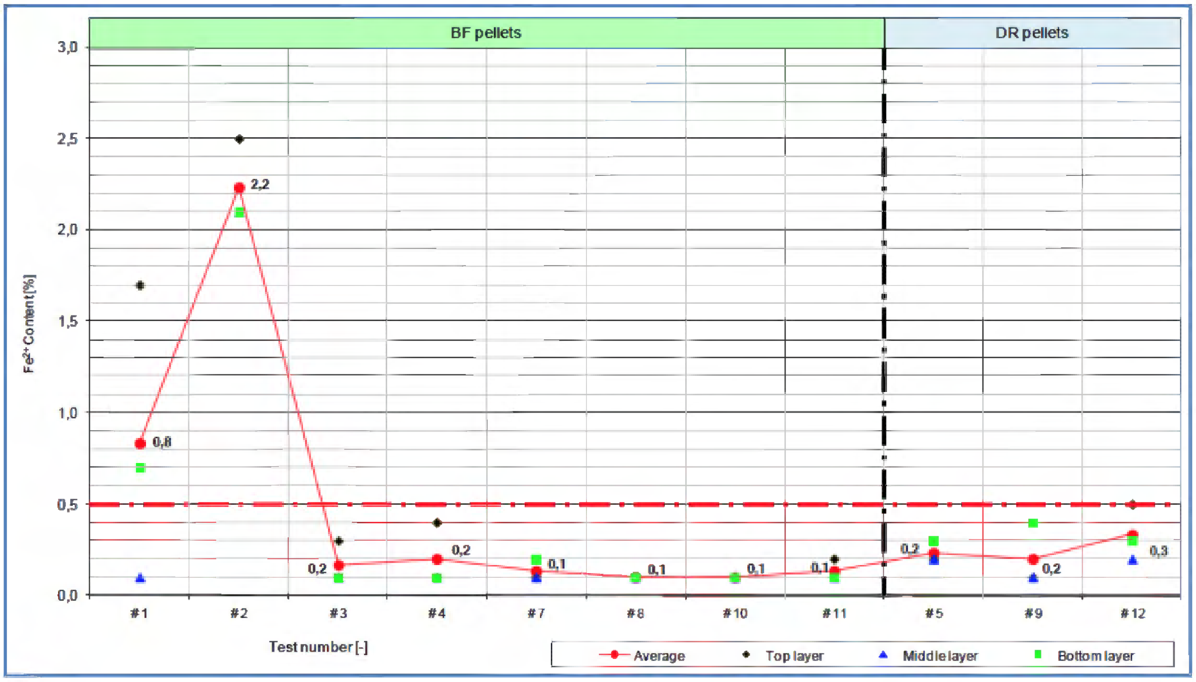
Tumble Index was very good in all tests (always above the lower limit of 94 %) as shown in Figure 17.26. The Abrasion Index values were lower than the desired five (5) % maximum.

Figure 17.26 – Tumble and Abrasion (Outotec)



After two (2) initial high bivalent iron readings, the hot gas temperature was decreased to 1,250°C from Test #3 onwards, and all subsequent tests were successful in that regard. The full results are shown in Figure 17.27.

Figure 17.27 – Bivalent Iron (Outotec)



iii) Grate Factor/Productivity

The targeted Grate Factor of 36.3 t/d/m² was achieved in almost all of the tests to reach a production of 8.5 Mt/y in 330 days per year with a safety margin of 15 %. The bed depth was maintained at 370 mm for all tests.

The net safety margins over the targeted capacity of 8.5 Mt/y varied between +11.4 % and +21.8 % for BF grade pellets and the DR pellets tests remained above 15 %.

iv) Metallurgical Properties

The BF pellets' metallurgical properties were analyzed in Tests #4 and #10 and the results are shown in Table 17.6. All the results met the required values except for the Free-Swelling Index of Test #10 at 22.4 % when the maximum should be 20 %. This target should be achieved in the operating plant by adding some additional dolomite in the flux.

Table 17.6 – BF Pellets Metallurgical Properties (Outotec)

Description	Units	Requirements		Test #4	Test #10
		Minimum	Maximum		
Swelling Test (ISO 4698)					
Free-Swelling Index	%		20.0	18.8	22.4
Static Test for Low-Temperature Reduction Disintegration (ISO 4696-1)					
Reduction Disintegration Index $RDI_{+6.3}$	%	90.0		93.2	97.0
Reduction Disintegration Index $RDI_{-3.15}$	%			4.1	2.1
Reduction Disintegration Index $RDI_{-0.5}$	%			3.2	2.0
Determination of Reducibility (ISO 4695)					
Degree of Reduction (t=240)	%			97.3	98.1
Minimum dR/dt at R=40%	%	1.0		1.1	1.2
Dynamic Test for Low-Temperature Reduction Disintegration (ISO 13930)					
Low Temperature Disintegration Index $LTD_{+6.3}$	%	80.0		81.1	96.6
Low Temperature Disintegration Index $LTD_{-3.15}$	%			11.9	2.4
Low Temperature Disintegration Index $LTD_{-0.5}$	%		11.0	7.7	2.2

The DR metallurgical properties are shown in Table 17.7. Pellets from Tests #5 and #9 were analyzed. The DR pellets met the target values except for the sticking test of Test #5 where the Clustering Index of 18.8 % surpassed the 15 % maximum. This test was performed without a dolomite coating however, which is common practice in industrial plants. Pellets from Test #9 had a very good result at 4.1 % Clustering Index using a bauxite coating.

Table 17.7 – DR Pellets Metallurgical Properties (Outotec)

Description		Requirements		Test #5		Test #9	
		Minimum	Maximum	5-1	5-2	9-1	9-2
Sticking Test (ISO 11256)							
Coating				n.a.		Bauxite	
Clustering Index	%		15.0	18.8		4.1	
Reduction Disintegration Test (ISO 11257)							
Degree of Metallization	%			92.2		86.7	94.7
Reduction Disintegration Index DR RD_{+10}	%			95.7		97.4	96.0
Reduction Disintegration Index DR $RDI_{3.15}$	%		1.5	1.2		0.8	0.9
Reduction Test (ISO 11258)							
Minimum Degree of Metallization (Calculated)	%	95.0		98.8	96.0	94.8	97.0
Degree of Reduction	%	95.0		98.6	97.2	96.4	97.9
dR/dt at R=40%	%			2.2	1.6	1.4	1.4
dR/dt at R=90% Estimated	%			0.7	0.4	0.5	0.7

c) Summary of Test Work

The two (2) testing campaigns at SGA and Outotec indicated that the required capacity of 8.5 Mt/y can be achieved with the pellet feed provided for testing. Outotec's letter of guarantee is in hand.

The green ball properties from the SGA campaign did not show any issues but during the Outotec testing program, the dry compression strength and drop number did not reach the desired levels. Although this does not affect the machine capacity, Outotec recommended corrective actions for future process development.

The physical properties of the indurated pellets such as cold compression strength, tumble strength and Abrasion Index met or exceeded the desired levels in most tests, especially in the tests which were used as basis for plant design. The metallurgical properties of the pellets were very good though some improvements may be made in selected parameters.

17.7.3 Mass Balance

The mass balance for a pellet production of 17.0 Mt/y of sized pellets with 0.76 Mt/y of undersized pellets for DR pellets is shown in Table 17.8 while Table 17.9 shows the mass balance for BF fluxed pellets.

Table 17.8 – Mass Balance for Production of DR Pellets

Feed Materials	t/y Wet	t/h Wet	t/t Pellets
Fine Iron Ore Feed (Slurry from Mine)	32,020,966	4,043.1	1.866
Coarse Bentonite	142,073	17.9	0.008
Coarse Limestone	279,688	35.3	0.016
Coarse Dolomite	79,688	10.1	0.005
Process Water to Mixer	197,243	24.9	0.011
Process Water to Additive Grinding	134,581	17.0	0.008
Process Water to Dust Slurry	605,116	76.4	0.036
Process Water to Coating Station	148,750	18.8	0.009
Bivalent Iron Content	534,599	67.5	0.031
Total Feed	**34,142,877**	**4,311.0**	**1.989**
Products	**t/y Wet**	**t/h Wet**	**% of Total**
Product Pellets @ 0.5% Moisture	17,162,312	2,167.0	50.3
Undersize Pellets	761,577	96.2	2.2
Filter Cake Concentrate for Sale	6,521,739	823.5	19.1
Loss on Ignition	165,977	21.0	0.5
Evaporation from Induration	1,812,611	228.9	5.3
Dust	909	0.1	0.0
Dust Slurry	672,351	84.9	2.0
Filtrate Water	7,045,401	889.6	20.6
Total Products	**34,142,877**	**4,311.0**	**100.0**

Table 17.9 – Mass Balance for Production of BF Fluxed Pellets

Feed Materials	t/y Wet	t/h Wet	t/t Pellets
Fine Iron Ore Feed (Slurry from Mine)	31,482,326	3,975.0	1.852
Coarse Bentonite	147,844	18.7	0.009
Coarse Limestone	896,535	113.2	0.053
Coarse Dolomite	0	0	0.000
Process Water to Mixer	200,137	25.3	0.012
Process Water to Additive Grinding	346,038	43.7	0.020
Process Water to Dust Slurry	605,116	76.4	0.036
Process Water to Coating Station	0	0.0	0.000
Bivalent Iron Content	529,602	66.9	0.031
Total Feed	**34,207,597**	**4,319.1**	**2.012**
Products	**t/y Wet**	**t/h Wet**	**% of Total**
Product Pellets @ 0.5% Moisture	17,000,000	2,146.5	49.7
Undersize Pellets	761,577	96.2	2.2
Filter Cake Concentrate for Sale	6,521,739	823.5	19.1
Loss on Ignition	393,685	49.1	1.2
Evaporation from Induration	1,743,228	220.1	5.1
Dust	909	0.1	0.0
Dust Slurry	672,351	84.9	2.0
Filtrate Water	7,114,108	898.2	20.8
Total Products	**34,207,597**	**4,319.1**	**100.0**

17.8 Pointe-Noire Process and Plant Description

The following description is based on block diagrams with water and mass balance, process flow diagrams, and plot plans for a pelletizing plant having two (2) dedicated pelletizing lines.

The two (2) pelletizing lines each have an annual capacity of 8.5 Mt of pellets based on 330 operating days per year (7,920 hours).

17.8.1 Port Slurry Reception, Handling and Storage

The port facilities and pelletizing plant have the required installations for all process areas from the reception of the iron concentrate as a slurry up to and including shiploading of product pellets and iron ore concentrate for sale. The iron ore concentrate is transported at a rate of 22 Mt/y from the concentrator to the port pelletizing facilities via a 645 km slurry transportation system. The slurry is received at the port facilities at a density of 65 % solids. Through a slurry distribution system, the slurry is fed to thickeners to raise the density to 72 % solids before being pumped to the four (4) slurry surge tanks.

Twenty-two (22) Mt/y of filter cake, with moisture of eight (8) %, will be produced. Of this quantity, 16 Mt/y of concentrate will be used to produce 17 Mt/y pellets and the remaining six (6) Mt/y will be stored and shipped as an iron ore concentrate for sale.

17.8.2 Additive Handling and Grinding

The following additives/binders are considered:

- Limestone;
- Dolomite;
- Bentonite;
- Organic Binder.

The coarse additives, limestone, dolomite and bentonite will be supplied as bulk material. A dry grinding system will be used for bentonite and a wet grinding system for limestone and dolomite. The additive grinding areas are designed for 22-hours-per-day operation.

The bentonite grinding plant must achieve two (2) targets:

- Bentonite fineness that is suitable for pelletizing (min 85 % < 45 μm);
- Drying the material with HFO to moisture level that is suitable for pneumatic transport.

Dust emissions at transfer points will be collected by a dedicated bag filter system. The collected dust will be discharged via a rotary valve into the coarse bentonite feed bin.

A wet grinding ball mill will be used to grind limestone and dolomite to a pelletizing fineness (min 85 % < 45 μm). The product will be in the form of slurry with a density of 65 % solids by weight. The mill slurry discharge will be pumped to the designated slurry tank in the filtration area.

Dust emissions at transfer points will be collected by a dedicated bag filter system. The collected dust will be discharged via a rotary valve into the coarse additive feed bins.

17.8.3 Feed Preparation and Filtration

a) Filter Cake for Pellet Production

The concentrate slurry from the four (4) slurry storage tanks is pumped as required to maintain the desired level of the two (2) filter feed preparation slurry tanks. To ensure the mechanical and metallurgical properties of the fired pellets are maintained, the basicity of fired pellets is adjusted by pumping limestone and dolomite slurry to the filter feed preparation slurry tanks as a ratio of the iron ore slurry received from the four (4) slurry storage tanks.

From the two (2) filter feed preparation tanks [one (1) for each pelletizing line], the slurry will be pumped to two (2) slurry distributors, each feeding the four (4) ceramic disc filters. The ceramic disc filters have 240 m^2 filtering area each and will produce a filter cake as a pellet feed with a moisture content of approximately eight (8) %.

b) Filter Cake for Sale as Concentrate

The excess concentrate slurry from the four (4) slurry storage tanks is pumped as required to maintain the desired level of the one (1) filter feed slurry tank part of an independent concentrate filtering system. From this tank, slurry will be pumped to a slurry distributor feeding the three (3) ceramic disc filters. The ceramic disc filters have 240 m^2 filtering area each and will produce a filter cake as a pellet feed with a moisture content of approximately eight (8) %.

c) Filter Cake Transport for Pellet Production and for Sale as Concentrate

The filter cake will be discharged and conveyed on a reversible belt conveyor systems either to feed the filter cake bins for pellet manufacturing or the concentrate storage shed as a concentrate for sale.

Part of the filtrate will be used for backwash cleaning of the filter ceramic disc. Make-up water will be added to backwash tanks and supplied through backwash pumps together with filtrate to disc filters.

The filtrate will then be transported by means of the filtrate pumps to the filtrate tanks and returned to the slurry receiving thickeners.

17.8.4 Mixing, Pre-wetting and Balling

The filter cake containing additives will be conveyed to the cake bins [two (2) bins per pelletizing line] each with a volume of 1,000 m^3. The filter cake will be extracted from the bins with vibrating cones and dosing belt scales will control the feed rate to the collecting belt conveyors feeding the mixers. "Loss-in-weight" feeders will dose the ground bentonite and/or organic binder at predetermined proportions onto the conveyor feeding the mixers. Water will be added at the mixers to obtain a moisture content of 8.5 - 8.9 % by weight for subsequent balling process.

Mixing of filter cake with bentonite and water will be done in intensive drum type mixers. The mixed material will be transported via belt conveyors to the balling feed bins. Dedusting of the mixing area will be done by means of bag filters.

The bin volumes for the ground bentonite and organic binder will be 100 m^3 and 40 m^3 respectively. Additives are pneumatically conveyed from the additive grinding area to the storage bins. All additive bins are equipped with bag filters for venting purposes.

The mixed material will be transferred by a belt conveyor system to feed the balling bins. Plows will be provided directly over the conveyor feeding the balling feed bins and will operate in an automated sequence to maintain the level of each bin within a predetermined range. The overall level of the 14 feed bins for each line is maintained by adjusting the feed rate to mixers.

In each pelletizing line, the balling area consists of two (2) identical rows, each having seven (7) balling feed bins, seven (7) dosing belt scales, seven (7) pelletizing discs and seven (7) single deck roller screens. There will be a total of 14 pelletizing discs for each pelletizing line.

Mixed material from the dosing belt scales will fall through fluffers for optimal dispersion of the material to the pelletizing discs. Each disc will have a diameter of 7.5 m and a rim height of 650 mm. The discs will be fitted with water sprays for optimal moisture adjustments and treated water will be utilized for this purpose. The discs will be mounted with an adjustable inclination between 44 and 53 degrees. Optionally, the discs may be fitted with a motorized inclination control. The discs will rotate at an adjustable speed for green balls with a target size of nine (9) - 16 mm.

In the balling area, the mixed material (or the balling feed) will be transformed into green balls with a preferred size of nine (9) to 16 mm in diameter by balling discs. These green balls must exhibit sufficient mechanical strength to withstand transportation and charging to the pallet cars of the indurating machine.

Green pellets, in the size range of zero to 18 mm diameter discharging from balling discs, will be screened on a single deck roller screen. Oversize >16 mm and undersize < nine (9) mm green pellets will be rejected and screened out by the roller screens and recycled back to the balling disc via a recycling conveyors system.

17.8.5 On-Size Green Balls

The on-size green balls from the single deck roller screen will be conveyed to the travelling grate by means of a series of belt conveyors to a reciprocating conveyor discharging on a wide belt. The reciprocating conveyor will spread the green balls onto a wide belt conveyor through the reciprocating movement of the head pulley. The green balls will then be delivered from the wide belt conveyor to a double deck roller screen feeder (“**DDRS**”).

The top deck of the DDRS will screen at 12.5 mm and the bottom deck will screen at nine (9) mm. The <nine (9) mm reject material is re-circulated back to the balling feed bins via the reject re-circulating conveyor system. The <12.5 mm and >nine (9) mm from the bottom deck is charged on top of the hearth layer on the travelling grate as the bottom layer of the green pellet bed. The >12.5 mm from the top deck is charged as top layer of green pellet on the travelling grate forming the total height of green pellets.

Disintegrators will be mounted on rejected pellet conveyors to disintegrate and the undersize pellets before they are returned via belt conveyors to the mixed material bins.

A reversible belt conveyor will allow the rejection of green balls from the process onto a dump pile outside of the building to prevent off-spec material from entering the induration machine and upsetting the process. The total flow of material to be recycled will be measured by a single idler belt scale mounted below the conveyor.

The green pellet bed level and depth across the width and length of the travelling grate is controlled by adjustments of the reciprocating conveyor, speed of the wide belt and by utilizing ultrasonic level sensors to automatically control the speed of the travelling grate to maintain predetermined bed depths.

17.8.6 Pellet Induration

The pellets will be completely heat hardened and cooled on the travelling grate. The travelling grate process ensures that during the preheating, firing and after firing stages, sufficient oxygen is always available in the process gases for proper magnetite oxidation.

The travelling grate consists of an endless chain of pallet cars on rails which continuously rotate. The travelling grate, on which the green balls will be indurated and cooled, has a useful area of 816 m² [four (4) m wide and 204 m long].

A layer of approximately eight (8) cm thick of >12.5 mm indurated pellets (hearth layer) will be provided to the bottom and side wall areas of the pallet cars as a protection from high temperatures. The first layer of green pellets in the size range of <12.5 mm and >nine (9) mm discharging the lower deck of the double deck roller screen will be charged on top of the hearth layer, followed by the second layer of >12.5 mm green pellets.

Hearth layer for bottom and side layer of travelling grate is fed by gravity from a storage bin installed above the feed-end of the travelling grate. The pallet car side layer is fixed at eight (8) cm and the bottom layer can be adjusted by a motorized gate, normally at eight (8) cm.

The three (3) components will be fed onto the pallet cars in the following order:

- Hearth layer;
- Side layer;
- First layer of <12.5 mm green balls; and,
- Second layer of >12.5 mm green balls.

The pelletizing cycle, based on the test work conducted at Outotec's laboratories in Germany, is summarized in Table 17.10.

Table 17.10 – Induration Cycle

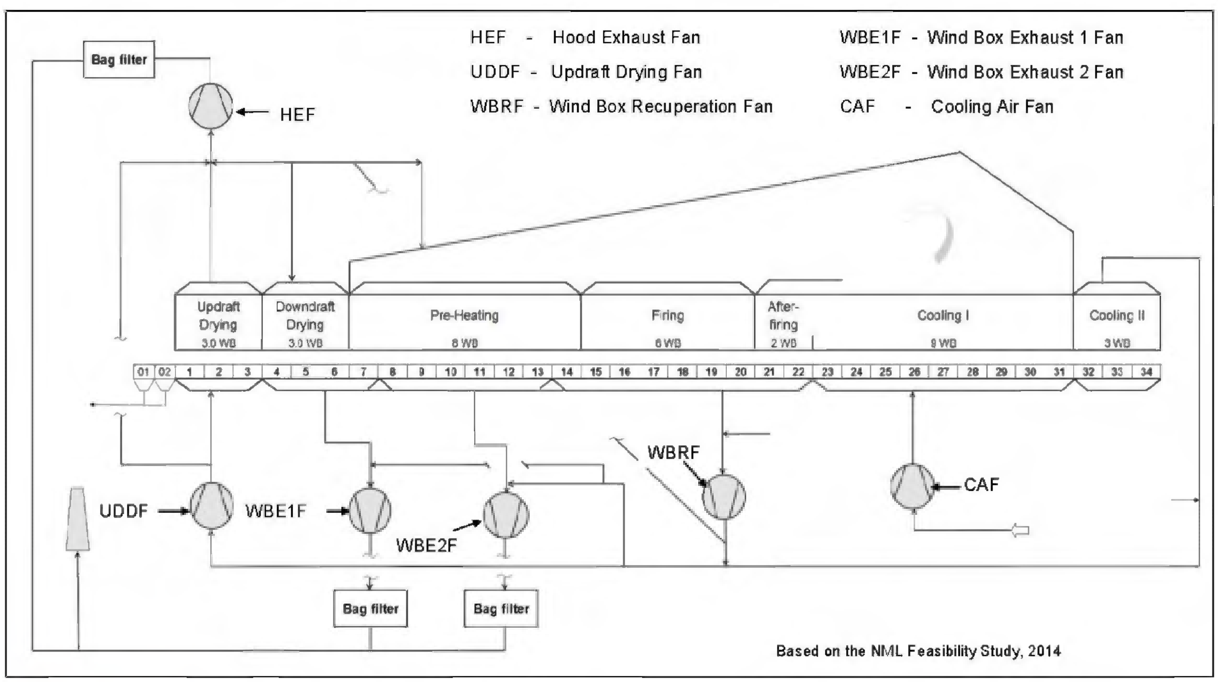
Process Zone	No. of Wind Boxes	Reaction Area (m^2)	Process Gas Temperature (°C)
Updraft Drying	4	96	330
Downdraft Drying	4	96	350
Preheating	5	120	350 – 1,250
Firing	4	96	1,250
After firing	5	120	950
Cooling 1	8	192	< 1,150
Cooling 2	4	96	< 550
TOTAL	34	816	0

17.8.7 Induration Process Gas Flow

a) Cooling Air

Ambient air from the cooling fan is forced through an arrangement of ducts and wind boxes through the bed of hot indurated pellets as a cooling agent for the first and second cooling zones of the indurating furnace. See Figure 17.28.

Figure 17.28 – Induration Gas Flow



b) Up-Draft Drying

Off-gas from the second cooling zone, being lower in temperature than the first cooling off-gas, is utilized for up-draft drying of the green pellets bed. Hot gas from the second cooling hood is extracted by the suction of the up-draft drying fan to maintain process gas flow and pressure in the windboxes of the up-draft drying zone. Ambient air bleed-in dampers control the inlet temperature of the updraft-drying fan. Windbox pressure in the updraft-drying zone is automatically controlled by a damper which bleeds off excessive air to the hood exhaust gas system.

c) Down-Draft Drying

Hot combustion gases from the last section of the firing and the after-firing zones are recuperated by the windbox recuperation fan and serve as drying gases in the downdraft-drying zone. Ambient air bleed-in dampers control the inlet temperature of the windbox recuperation fan. A bypass duct from the off-gas of windbox recuperation fan also serves to control the pressure of the downdraft drying zone and to raise the temperature of the humid low temperature of the hood exhaust fan.

d) Pre-Heat

The principal process gas for the preheat zone is sourced via the first cooling off-gas, direct recuperating header and down-comers. The temperature profile of the preheat zone is controlled bleeding-in lower process gas temperature for the off-gas of the windbox recuperation fan by adjusting bleeding-in dampers installed in the connecting ducts of the windbox recuperation off-gas and the direct recuperating header down-comers.

e) Firing – After Firing

The first cooling zone off-gas enters the direct recuperation header and down-comers, by the suction of the windbox recuperation and windbox exhaust fans to be further utilized as a process gas or waste gas. Burners are sized and arranged opposite to each other on the longitudinal sides of the preheating and firing zones to ensure a uniform hot gas distribution over the width and length of the pellet bed. Burners are self-aspirator type utilizing heavy fuel oil. Since the burners are divided into several control zones, temperature profiles can be adjusted, thus permitting an optimum heat treatment of the pellets.

f) Waste Gas

The temperature of the off-gases of the downdraft-drying, preheating and part of the firing zone is too low for further utilization as a process gas and is removed from the indurating process via two (2) windbox exhaust fans. The off-gas of two (2) windbox exhaust fans is cleaned by respective bag filters before being released via a stack to the atmosphere.

The hood exhaust fan pulls-off the humid and low temperature exhaust gas from the up-draft-drying hood. The hood exhaust gas is cleaned in the bag filter and directed to atmosphere together via the common waste gas stack.

17.8.8 Product Screening and Intermediate Pellet Storage

Cooled pellets from the indurating machine are discharged into a surge bin. The surge bins of each pelletizing line have two (2) dedicated outlets and screening systems. A predetermined level of the surge bin is ensured by adjustable gates and variable speed product conveyors for each system. During normal operation, the fired pellets from both product conveying systems will be distributed directly to the product screens via a common and a two-way chute.

The product conveyors also receive spillage collected over the length of the indurating machine via a spillage collecting conveyor and two-way chutes onto two (2) product conveying and screening systems for each pelletizing line.

Indurated pellets are screened to remove the undesired <5 mm pellet chips and fines from the final product and for the >12.5 mm larger pellets to be used as hearth and side layers. The >12.5 mm, larger pellets are required as hearth layer to avoid clogging of pallet car grate bars and maintaining the permeability of the travelling grate pellet bed, thus reducing the pressure drops and energy consumption. Screened >5 mm and <12.5 mm pellet and the > 12.5 mm pellet which is not used as hearth, form the final product which is transported to the storage yard via a conveyor and stacking system equipped with belt scale for production and inventory control.

In case of a product screen or materials handling failure, the unscreened pellets can be directed to an emergency stockpile conveying system and later recycled to the product screening system.

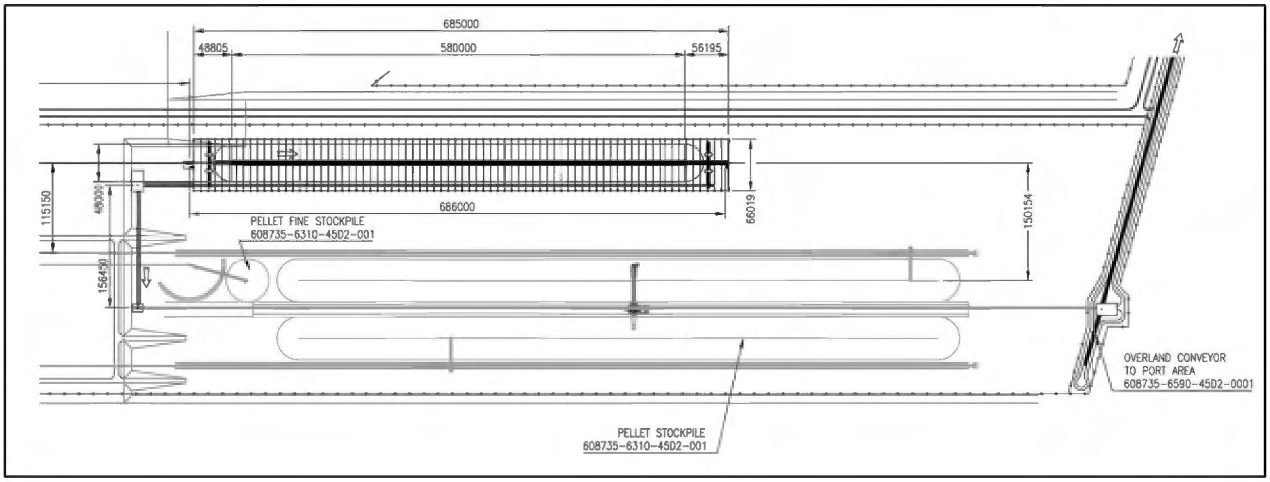
The pellet chips <5) mm produced by the product screens will be discharged onto a common pellet chip conveyor, equipped with belt scale and sent to the pellet chips stockpile. The pellet chips can be sold as pellet feed or sinter fines.

17.8.9 Pellet Stockpiling and Ship loading

Final pellet product, received from the screening area of each pelletizing line, is transported and stored via dedicated conveying and stacking systems in accordance to an established yard storage plan for the various pellet qualities. Each pellet storage yard has a capacity of 750,000 Mt. Each conveying and stacking system has a capacity of 3,200 t/h and can handle the total production of the two (2) pelletizing lines, this to ensure the production of both pelletizing lines in the event of a failure on one (1) of the conveying and stacking systems. The stackers are fitted with lofting and slewing features to provide the possibility of stacking the product in a chevron or windrow-row pattern for product blending.

Pellets for ship loading will utilize a common bucket wheel reclaimer having a capacity of 9,200 t/h for both pellet stockpiles. Both pellet and filter cake concentrate for sale will utilize a common overland conveyor on a time sharing basis for transportation to the Sept-Îles Port Authority ship loading facility. See Figure 17.29.

Figure 17.29 – Concentrate and Pellet Stockpiling and Reclaim



17.8.10 Filter Cake Concentrate Storage and Shipping

Filter cake concentrate received from the filtration plant is transported via a conveyor system to a covered storage shed, having a capacity of 600,000 Mt.

Filter cake concentrate for ship loading will utilize a 9,200 t/h capacity bridge reclaimer. Both pellet and filter cake concentrate for sale will utilize a common overland conveyor on a time sharing basis for transportation to the Sept-Îles Port Authority ship loading facility.

17.8.11 Storage and Ship Loading of Pellet Chips

The pellet chips and fines [< five (5) mm] produced by the product screens will be transported and stored in a dedicated pile at the end of the pellet storage yard. The pellet reclaimer will also be utilized for reclaiming and ship loading pellet chips and fines.

17.8.12 Pellet Coating

Part of the dolomite slurry coming from the additive grinding will be used as coating material for DR grade pellets. The slurry (15-20 % solids by weight) will be stored in agitated slurry tanks. From there, it will be pumped via slurry pumps to the coating spray system positioned on top of the product belt conveyors.

17.8.13 Gas Cleaning Systems

a) Process Gas Dedusting

The dust content of the windbox and hood exhaust gases will be collected by bag filters installed in front of the windbox exhaust and the hood exhaust fans.

The dust from the bag filters will be discharged through double pendulum valves positioned under every row of dust hoppers and transported via collecting chain conveyors to the dust slurry tank and mixed with process water prior to pumping to the thickeners.

b) Plant-Dedusting

The ambient atmosphere in the pellet plant will be kept at low dust levels in order to maintain clean working conditions as well as facilitating plant maintenance. All areas within the pellet plant where dust may arise will be covered with hoods and connected to the plant de-dusting system.

The feeding and the discharge stations of the indurating system will be de-dusted together with the screening section by a central de-dusting system. This system will be similar to the process gas system and consist of a de-dusting fan, a bag filter and the separate stack.

The dust from the bag filter will be discharged through double pendulum valves to a chain conveyor from where the dust is conveyed to the slurry tank. In the slurry tank, the dust will be mixed with process water prior to pumping to the thickener.

All dust recovered from the different gas cleaning systems will be returned back to the main process. By implementing this concept, no iron units will be lost throughout the process, with the exception of very limited quantities of dust in the clean gases of the stacks.

17.8.14 Fuel – Heavy Fuel Oil

The burners were designed to use heavy fuel oil (Bunker C) as the fuel source for the indurating machines. The characteristics of fuel oil are depicted in Table 17.11.

Table 17.11 – Characteristics of Fuel Oil

HFO Bunker C	Unit	Min	Max	Value
Density @ 15° C	kg/l	0	1.01	0.973
Pour Point	°C	0	0	6
Sulphur	wt %	0	1.5	1.31
Nitrogen	ppm	0	0	3.63
Flash Point	°C	65	0	104.0
Viscosity @ 50°C	cSt	150	650	54
Water & Sediment	% vol.	0	1.0	0.10
Heat Value	MJ/l	0	0	41.95
Vanadium	mg/kg	0	0	19
Silica	mg/kg	0	0	89
Sediment by Hot Filtration	wt %	0	0.1	0.02
Compatibility/ Stability		0	2	1
Ash	wt %	0	0.1	0.063
Carbon Residues	wt %	0	0	9.33

17.8.15 Cooling and Gland Seal Water

Cooling water will be supplied to various consumers and the return hot water from the same consumers will be collected in the cooling water tank and routed through the heat exchangers, where it is cooled. A small amount of make-up water will be added to replace evaporated water from the cooling system.

17.8.16 Process Water

The filtrate pumps and thickener overflow will supply the process water to a tank from where it will be distributed to the various users.

17.8.17 Glycol Cooling

Glycol will be supplied for motor and bearing cooling of the process fans. The return glycol will be returned to the glycol collecting tank. The heated glycol will be routed through the fin fan coolers to be cool down and re-used for cooling.

17.8.18 Compressed Air

Compressed air will be required for plant air (bin blasters, bag filters cleaning air, etc.) and instrument air. The system pressure will be designed for six (6) bar (g). Four (4) compressors will serve the compressed air system. Instrument air will be required for pneumatic valves, instruments, cleaning of bag filters and for instrument functions. The required air will be dried in the absorption dryers. The estimated consumption of compressed air is shown in Table 17.12.

Table 17.12 – Estimated Consumption of Compressed Air

Description	Consumption (m^3/h)	Pressure (bar)	Temperature (°C)	Purpose
Plant air	1,050	6	-2	Cleaning, Pneumatic Tools
Instrument air	2,080	6	-20	Cleaning of bag filters, field instruments

17.8.19 Plant Automation

The Plant Automation System will monitor and control all significant variables in accordance with the process requirements. It will provide all operating facilities with necessary sequencing, interlocking and safety functions, including alarms for abnormal conditions in the sections of the plant specified in the process and plant description.

The control system structure was based on a Distributed Control System ("**DCS**") network system with modular architecture. Monitoring, optimized control and regulation of the process, supervision of individual equipment, and documentation of the production process will be done by these levels:

- Level 2 – Operation and Monitoring/Human Machine Interfaces ("**HMI**"); and,
- Level 1 – Automation/Process Control Station.

Operation, engineering and the process control stations will be selected from the product range of one (1) supplier only. The proposed advanced control and safety system will have a distributed plant structure for automation and data acquisition functions.

The Levels 1 and 2 systems will be integrated into a network. The connection between the operator, engineering and process control stations will be an Ethernet TCP/IP structure.

The Level 2 equipment will be located in the CCR of the plant from where all the various process areas may be monitored. They will include start-up/shut-down support or software modifications and tests in the process plant.

A continuously working stand-alone gas analyzer for SO_2 and NOx monitoring will be provided for measurements at the process gas stacks. The analyzer will be complete with all necessary auxiliary instruments such as: power supply units, gas cooler, gas preparation units, filters, pressure reducers etc., installed in a local analyzer cabinet.

17.9 Utilities – Consumption Overview

Consumption figures will depend on the characteristics of raw materials and finished products but typical values are provided in Table 17.13 for electrical energy and in Table 17.14 for thermal energy which was average for both types of pellets.

Table 17.13 – Typical Consumption for Electrical Energy

Electrical Energy	Specific Consumption (Summer) kWh/t	Specific Consumption (Winter) kWh/t
Total	**32.3**	**32.5**
Raw material handling, additive grinding, dosing and mixing, green balling, induration, fans and dedusting, product handling and screening	30.7	30.7
Filtration	1.1	1.3
Other area (Process and cooling water, plant and instrument air, firefighting)	0.5	0.5

Table 17.14 – Average Typical Consumption for Thermal Energy for BF Fluxed and DR Pellets

Thermal Energy		Consumption (p/h)	Specific Consumption per $t_{pellets}$
Total	Mcal	2 x 70,198	65.4
	MJ	2 x 293,904	274
HFO 1C @ 9,600 kcal/kg	Mcal	2 x 70,198	65.4
	kg	2 x 7,312	6.81

18.0 PROJECT INFRASTRUCTURE

18.1 Slurry Transport System

BRASS Engineering International (**"BRASS"**) was contracted by ISLLP to prepare a study for the slurry transportation system to transport 22 million Mt/y of iron concentrate slurry, from the mine site to the terminal station at Pointe-Noire. The route selection process was performed by Johnston Vermette, a sub-consultant to NML providing expertise on slurry transport routing.

The slurry transportation system is designed to operate continuously with an operating range bounded by minimum safe limiting velocity, maximum throughput, and solids concentration by weight ("**% Cw**"). The slurry transportation system can only safely transport concentrate within a throughput window equivalent to 18 Mt/y minimum and 24 Mt/y at maximum flow rate.

The proposed slurry transport route (corridor) covered for the Feasibility Study, is summarized in this Section.

The KéMag route which transports the slurry a distance of 645 km from the KéMag deposit to the pellet plant at Pointe-Noire

18.1.1 Slurry Transportation System

The slurry transportation system will deliver the concentrated iron ore in the form of slurry from the concentrator to the pellet plant to be located at Pointe-Noire. The KéMag slurry transportation system will follow a corridor completely in Quebec. The slurry transportation system will be approximately 645 km long. The main pumping station at the head of the slurry transportation system will be located in a building adjacent to the concentrator.

The predominant design principle for this slurry transportation system will be to bury it underground and to avoid permafrost zones. The design will adopt a "No Freezing, No Plugging philosophy", which means that the slurry transportation system will either be buried below the frost depth (no freezing) or if it is buried above the frost line, it will have the necessary reliability and emergency back-up equipment to maintain no-freeze conditions during an emergency stoppage and the slurry transportation system slope will be restricted to avoid plugs even if the line is shut down full of slurry.

Where it is not possible to bury the slurry transportation system, an appropriate thickness of insulation and/or heat tracing will be provided, to guard against freezing of the slurry. A protective three-layer polyethylene external coating will mitigate external corrosion. Cathodic protection systems, either active or passive, will be included in the slurry transportation system design.

There will be many stream crossings between the concentrator and Pointe-Noire. Trenching will be the preferred method of construction of the crossings while horizontal directional drilling will be used to cross large streams when trenching is impractical or not permitted.

A fiber-optic "backbone" will carry all slurry transportation system communications, including SCADA system and video surveillance of the booster and valve stations, as well as office data and voice telephone channels between the concentrator and the pellet plant. The multi-fiber optical cable will be installed in a conduit in the slurry transportation system trench.

18.1.2 Slurry Characteristics

The particle size distribution is presented in Table 18.1.

Table 18.1 - Slurry Particle Size Distribution

Sieve Size (Tyler mesh)	Percentage Cumulative Passing
65	100.00
100	100.00
150	99.97
200	99.81
270	94.82
325	90.66
400	86.54

The dry solids specific gravity is 5.05.

18.1.3 Slurry Flow Rates

The design (or nominal) concentrator throughput would be 22 Mt/y of solids with a Cw of 65 % and process plant availability of 95 % which equates to 2,644 t/h of solids in the slurry. The total nominal slurry throughput would be 4,068 t/h or 1,955 m^3/h with a density of 2.08 t/m^3.

The minimum slurry throughput is calculated based by the limiting velocity under which solids will start sedimenting and laminar flow must be avoided as it is unstable and carries the risk of plugging the line as well as causes premature wear of the pipe. In Both KéMag and LabMag, this is found to be 2,200 t/h of concentrate and based on the 95 % availability of the slurry transportation system would equate to 18.3 Mt/y of concentrate.

The maximum throughput using six (6) of the seven (7) positive displacement pumps at their maximum rated output would be the equivalent of 23.5 Mt/y. Would more slurry being required, the stand-by pump could be used and the maximum throughput would climb to the equivalent of 25 Mt/y.

Table 18.2 lists the estimated flow rates of slurry under various conditions of concentration, Cw, and throughput.

Table 18.2 - Slurry Flow Rates

Design (22 Mt/y)			Maximum (24 Mt/y)		
Concentrate, Percentage Cw	**Density, t/m^3**	**Slurry, m^3/h**	**Concentrate, Percentage Cw**	**Density, t/m^3**	**Slurry, m^3/h**
63	2.02	2,084	63	2.02	2,311.0
65	2.08	1,955	65	2.09	2,167.3
67	2.15	1,833	67	2.16	2,032.2
69	2.23	1,719	69	2.24	1,904.8
70	2.27	1,664	70	2.28	1,843.9

Note: Density of slurry is calculated using 5.05 t/m^3 as the density of solids

18.1.4 Throughput and Availability

Historically, an operating slurry transportation system achieves an availability of more than 98 % when operated and maintained under design conditions. The design is based on 95 % availability.

The slurry transportation system has been designed to transport 22 Mt/y (dry basis) of concentrate slurry at 65 % solids by weight.

- Annual throughput (nominal): 22,000,000 tonnes;
- Daily throughput (nominal): 60,274 tonnes;
- Hourly throughput (nominal): 2,511.4 tonnes;
- Design throughput (@ 95 % availability): 2,643.6 t/h.

18.1.5 Design Factors

Pressure loss calculations were performed using BRASS's proprietary hydraulic computer models.

In addition, the following design criteria were used:

- Six (6) % flow safety factor is included for hydraulic calculations;
- Eighty (80) % of Specified Minimum Yield Stress ("**SMYS**") per ASME B31.11 is used to calculate the Maximum Allowable Operating Pressure ("**MAOP**");
- Thirty (30) m minimum clearance between the hydraulic gradient line ("**HGL**") and slurry transportation system line profile;
- Thirty (30) m minimum clearance between the static (shutdown) head and slurry transportation system profile;

- Fifteen (15) % safety factor in the limiting velocity calculation;
- Test pressure hoop stress is less than 90 % of SMYS;
- The ratio of outside diameter (“**OD**”) to weight (“**WT**”) is less than 80.

18.1.6 Corrosion Rate and Erosion Rate

Pending laboratory test results, the corrosion rate is assumed to be six (6) mil per year (m/y), equivalent to 0.1524 mm/y in the first ten (10) km of the slurry transportation system and four (4) mil per year or 0.1016 mm/y for the remaining length. No significant abrasion component is expected from the iron concentrate slurry if the system is operated within the operating range. This is also consistent with other similar systems in operation.

The slurry transportation system has been designed with extra conduit wall thickness to compensate for corrosion metal loss.

18.1.7 Design Life

The design life of the slurry transportation system is 22 years although a much longer life will be possible if operated within the design parameters.

18.1.8 Hydraulic Calculations

Pressure loss calculations were performed using BRASS’s proprietary hydraulic computer model for defining the hydraulic gradient line, based on the following criteria:

- The maximum internal velocity is less than 2.5 m/s;
- A flow safety factor of six (6) % is included in the system design. Pumps and slurry transportation system are designed to accommodate additional flow to cover possible variability in slurry properties in the future;
- A ten (10) % overpressure is allowed during transient conditions which are in compliance with ASME B31.11.

18.1.9 Minimum Safe Operating Velocity

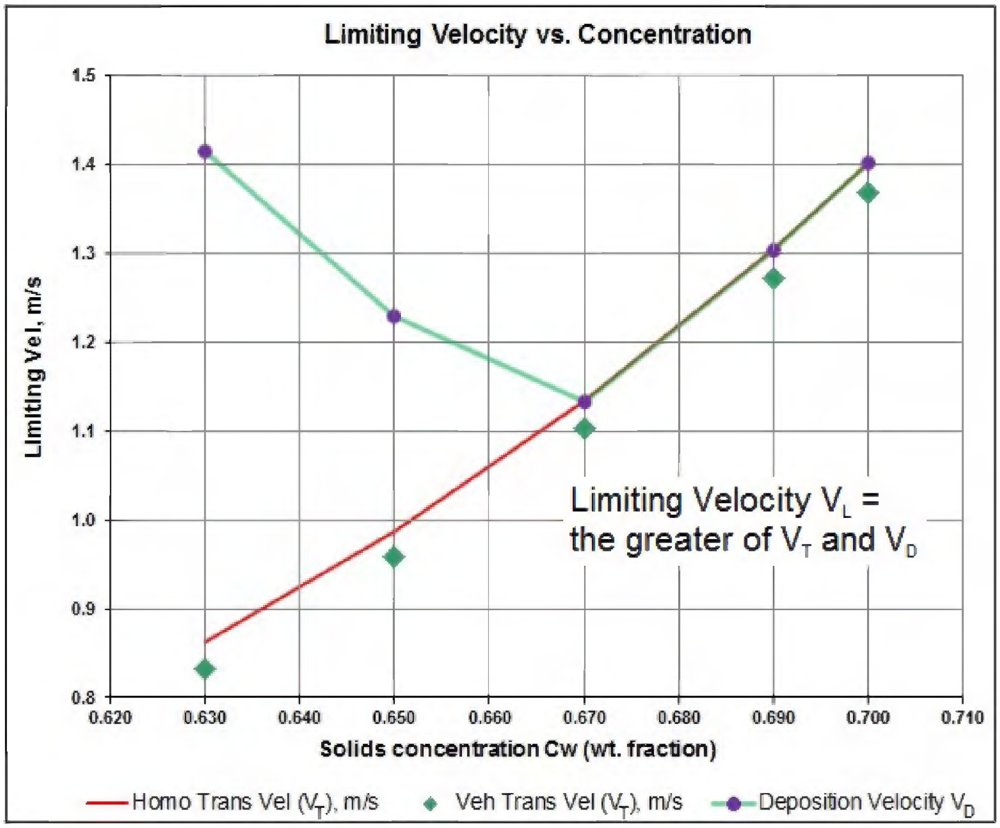
The minimum safe operating velocity is intended to maintain pseudo-homogeneous flow behavior (particles stay suspended) in the slurry transportation system. This minimizes wear along the bottom of the system which would otherwise result from the heavier particles dragging along the bottom of the system. In more extreme cases, accumulation of these particles may result in slurry transportation system plugging.

Minimum safe operating velocities for the slurry transportation system are governed by the deposition and transition velocities (Vd and Vt, respectively) for the solids content considered for the Project [from 63 to 70 % concentration by weight (Cw)]. The minimum (limiting) velocity is the greater of Vd and Vt. The actual solids concentration range will be tested and finalized during start-up and commissioning.

The minimum safe operating velocity determined for the slurry transportation system at 67 % Cw is 1.13 m/s. This corresponds to a flow rate of 1,537.1 m^3/h, which at a slurry density of 2.15 t/m^3 [density of slurry is calculated using 5.05 t/m^3 for solids and one (1) t/m^3 for water], will transport less solids than the nominal rate of 2,643.6 t/h.

The operating velocity would, therefore, have to be higher than this minimum safe operating velocity of 1.13 m/s in order to transport the nominal flow rate of solids (2,643.6 t/h) at 65 % Cw. The limiting velocity vs. concentration relationship is shown in Figure 18.1.

Figure 18.1 - Limiting Velocity vs. Concentration



18.1.10 Flow Velocity and Slurry Concentration

Slurry flow velocities are generally limited to a narrow range. As mentioned, low flow velocities for coarse materials may result in heterogeneous flow which increases both the wear rate along the bottom of the pipe and the risk of plugging the slurry transportation system. Higher flow velocities can result in uneconomical frictional pressure losses and pipe wall abrasion.

As a result, pipe diameters are selected to achieve a design velocity, which provides a reasonable margin above the minimum safe operating velocity while meeting the throughput and pressure requirements and avoiding abrasion.

Concentration parameters must also be considered. A minimum concentration is required to provide sufficient rheological support to maintain pseudo-homogeneous flow behavior which suspends the top size particles and in turn, avoids bottom wear.

18.1.11 Shutdown in Cold Weather

Over the course of each winter season, the frost depth will increase with time and can reach as far as three (3) metres in depth from the ground surface. It can also be less than one (1) metre if the snow cover is thick and the soil of organic nature.

Therefore, any flow stoppage that could exceed the freeze time of the water in the slurry transport system should be prevented or mitigation measures put in place to maintain a flow of water or drain the system.

Currently, the most conservative estimate for the time it would take to freeze stagnant water in the system is 36 hours for the one (1) metre soil cover over an uninsulated pipe. It must be noted however, that the slurry transport system operation and control philosophy for prolonged shutdown states that if the shutdown period exceeds 24 hours with the system full of slurry, (to be ascertained from slurry rheology), the system must be flushed with water to prevent settling of solids and possible plugging in the slurry transportation system. BRASS chose this 24 hour time period based on their experience with similar slurry systems.

Therefore, the 24 hour plugging time period is the design limiting condition. By mitigating the problem of protecting the slurry transportation system from plugging, then the design mitigates the problem of protecting the system from freezing, which is predicted to happen when the slurry stops moving, with no insulation on the slurry transportation system and a 1-m burial depth, in 36 hours. Adding insulation to the system or increasing the burial depth of the system will increase the 36-hour freezing time period, but won't affect the 24-hour plugging time period.

If the slurry flow is stopped for longer than 24 hours, then the slurry must be flushed from the slurry transportation system using water with no slurry. To ensure that the slurry transportation system flushing can commence within 24 hours, the following must be implemented during operation:

- A robust maintenance program to help ensure that slurry transportation system and equipment failures are rare;
- Repair teams with appropriate equipment and materials on stand-by to ensure that the slurry transportation system can be repaired and restarted within 24 hours.

The current pump station design is robust and has full power backup in the form of diesel emergency generators. Additional spares and redundancy could be designed to further reduce the risk of stoppage.

If the mitigations indicated above have been implemented, then only a single additional step is required to protect the slurry transportation system from freezing. In winter, the water flow must be maintained in order to prevent the water from freezing. If the water flow is stopped for longer than 36 hours, then the slurry/water must be flushed from the system and the system must be dried. This can be done using the same procedure used to dewater and dry the slurry transportation after hydrotesting with pig trains and a drying agent. This latter alternative has not been implemented since it is planned to keep a minimal amount of water flowing in the slurry transportation system to prevent freezing.

For planned shutdowns, the slurry will be pumped to the filtration plant at Pointe-Noire and the slurry transport shutdown full of water. If the shutdown is during the cold season and that it extends over 36 hours, the water will be allowed to flow directly to the water treatment facility and discharged in the Gulf.

If the shutdown is unplanned, or if it is caused by the inability of the filtration facility to receive slurry, two (2) discharge ponds have been planned at PS2 and at Pointe-Noire. They both have the capacity to handle their respective volumes contained in the segment of the slurry transportation system they serve and both pump stations feeding them are a capable of replacing the slurry with water from adjacent freshwater streams.

This being an emergency situation, reclaiming the concentrate from the ponds has not been engineered at this stage. Reclaiming with trucks and payloaders could be a solution at the export terminal while pumping with a suction dredge to the mixing tanks appears a more efficient method.

Discharge ponds should be cleared of snow after storms; this is easily achieved through local snow removals contracts.

18.2 High Voltage Electrical Distribution Network

It is anticipated that the utility companies will supply three (3) incoming lines at different voltage to each of the Project installation locations, being at mine and process plant area, slurry intermediate pumping station area and at Pointe-Noire and port area.

18.2.1 Mine and Process Plant

It is anticipated that the plant will be powered by a new 315 kV aerial power lines by the utility companies.

The site will be fed by two (2) High Voltage (“**HV**”) drops connected to one (1) single gas isolated switchgear (“**GIS**”) in a prefabricated building. This GIS will convert voltage down to 34.5 kV through six (6) stepdown transformers in N+2 redundancy configuration.

All local sub-stations will reduce voltage to 4.16 kV from 34.5 kV aerial lines. There are only two (2) sub-stations powered at 13.8 kV instead of 4.16 which are flotation and second stage grinding. The power cables from GIS to these two (2) sub-stations will be installed in buried duct banks.

The 34.5 kV switchgears will be installed in the main electrical room.

The power system studies aims to:

- Validate adequate transformer loading;
- Propose transformers' off circuit tap changer settings to obtain optimal nominal bus voltages;
- Ensure that service voltages are within design criteria limits;
- Estimate the capacitor bank necessary at the main bus to meet the Hydro-Québec's power factor requirements;
- Establish minimum electrical equipment rating for adequate operation under normal and contingent operation;
- Recommend equipment continuous and short-circuit ratings to provide adequate margins for project growth;
- Propose a method to start the large ball mill motors with an acceptable voltage drop in accordance with the design criteria.

The results obtained for the load flow study conclude that the voltage profile conforms to the design criteria and the power factor conforms to a utility demand of 95 % or better.

The transformer sizing is suitable for the intended duty. The main transformers are equipped with secondary on-load automatic tap changers. The tap position should control the secondary bus according to the voltage profile. The transformers which feed the mine loop should be set to +2.5 %.

Direct on line starting of 13.8 kV ball mills motors [two (2) x 8,750 kW] using a dual liquid rheostat starter ("**D-LRS**") is the worst case motor start-up. The calculation is based on typical preliminary and estimated motor data and load characteristics.

The feasibility of starting a set of ball mill motors [two (2) x 8,750 kW] simultaneously with a voltage drop of less than 15 % at the motor feeder bus was verified. The utility company might impose stringent constraints on the voltage drop at the Point of Client Connection ("**PCC**"). During large motor start-up, the voltage drop at the main 315 kV bus has also been considered. Starting a set of ball mill motors simultaneously causes a voltage drop of seven (7) % at the 315 kV bus. It should be validated with the utility during next phase study whether this starting method is acceptable or an alternate method must be employed. Values obtained are for reference only.

A complete study should be carried out when more data is available from the motor manufacturer at the detailed engineering phase.

The coincidence factor (1/diversity factor) used for the calculations is 0.87 and is based on mining industry practice, taking 1.0 of the grinding equipment and 0.8 average for the remaining equipment at their load level. Calculations are giving the following results: 383.4 MW at 0.95 power factor giving 403.6 MVA.

The sizing of the 315 kV to 34.5 kV main transformers 112/150/186 MVA, was based on the worst case scenario. However, with the load variations during the study, the sizing can be optimized (slightly on low side) at the detailed engineering phase.

The sizing of the 315 kV to 13.8 kV main transformers 54/72/90 MVA, can also be optimized (slightly on high side) at the detailed engineering phase. This is due to the late increase, close to the end of the Feasibility Study, of the 13.8 kV motors. The optimization should then apply to the whole plant design.

a) Infrastructure and Utilities – Overhead Lines

Two (2) 34.5 kV ACSR pole-mounted overhead lines will distribute power to the following areas:

- Tailings Water Pumping Station;
- Fresh Water Pond Pumping Station;
- Explosive Preparation Plant Building;
- Raw Water Supply Pumping Station;
- Acid Rock Drainage Treatment Plant;
- Workers accommodation complex (sub feeding Complex utility buildings and helipad;
- Domestic Waste Disposal Facility.

Two (2) 34.5 kV pole mounted isolated type MV105 cables with messenger cable will feed the mine area. The lines are redundant.

Two (2) 4.16 kV ACSR pole-mounted overhead lines will distribute power to the small pumping stations around the tailings storage basin.

b) Slurry Intermediate Pumping Station

Hydro-Québec will supply the plant with one (1) line of 34.5 kV from their Poste Normand, near route 389 part of the grid that supplies nearby Fermont.

There will be two (2) drops of the 34.5 kV line to feed the plant connected to the 34.5 kV main switchgear. This switchgear will feed the two (2) 34.5 kV to 4.16 kV main transformers 15/20 MVA capacity that will feed the 4.16 kV switchgear supplied by BRASS.

The plant electrical distribution design for the Slurry Intermediate Pumping Station area is based on the following documents:

- Mechanical motor list;
- Electrical load list.

The coincidence factor (1/diversity factor) used for the calculations is 0.87 and is based on mining industry practice.

Calculations are giving the following results 16.4 MW at 0.95 power factor giving 17.3 MVA. The selected sizing of the 34.5 kV to 4.16 kV main transformers is 15/20 MVA.

c) Pointe-Noire and Port

Hydro-Québec will feed the Pointe-Noire and port area at Sept-Îles from their Poste Arnaud with one (1) single line of 161 kV. This 161 kV GIS will feed and protect the two (2) main transformers 161 kV to 34.5 kV transformers 90/120/150 MVA capacity feeding the 34.5 kV arc-resistant metal-enclosed switchgears. The 34.5 kV switchgears will be installed in a stick-built main electrical room.

The plant electrical distribution design calculations for the Pointe-Noire and port area at Sept-Îles are based on the list of electrical consumers including Outotec, the mechanical motors, and the Electrical Load List. The coincidence factor (1/diversity factor) used for the calculations is 0.707 and was given to ISLLP by Outotec based on their data from existing pellet plant. Calculations are giving the results of 116.8 MW at 0.95 power factor giving 122.95 MVA.

The sizing and quantity of the three (3) main transformers 161 kV to 34.5 kV 90/120/150 MVA were due to the preliminary information from Outotec, assumptions about the large handling equipment and conveyors before receiving the bids and finally, before the decision of changing the heating power source. Outotec provided the clarifications regarding the loads calculations and information about coincidence factor in existing plants.

18.3 Emergency Power Plant

18.3.1 Mine and Process Plant

The mine and process plant will be equipped with one (1) set of three (3) 10,000 kW 4.16 kV diesel generators. This will cover the emergency load requirements at the process plant (including the slurry transport system at pump station PS1). The generators are "skid type" (as far as practicable) and will be installed in a stick-built building. A larger number of smaller units may be considered in the detailed engineering phase.

The preliminary estimated generators capacities were considered for the process buildings services, the non-process buildings and the facilities.

18.3.2 Intermediate Pumping Station

The slurry intermediate pumping station (PS2) will be equipped with four (4) 1,800 kW 4.16 kV diesel generators. The generators are "skid type" and will be installed in a stick-built building.

18.3.3 Pointe-Noire and Port

The Pointe-Noire and port area will be equipped with four (4) 1,800 kW 4.16 kV diesel generators to cover all the emergency loads requirements. All units will be supplied mounted in its own structural type walk-in enclosure to be installed on a concrete slab (s).

The process emergency loads are shown in the mechanical motor list.

18.4 Explosives Preparation and Storage

The explosives preparation and storage facilities are based on the requirements of an explosives supplier to be on site. The specialized manufacturer and supplier of explosives will be responsible for the following services:

- Transportation and storage of explosive products as well as blasting accessories on site;
- Manufacturing of bulk emulsion explosives;
- Loading and priming of blast holes.

All required turn-key infrastructure for the manufacturing and the delivery of bulk explosives as well as for a down-the-hole blasting service will be provided and executed by an explosives supplier who will be under contract.

Two (2) sites have been selected for the contractor to establish the explosives preparation operation. The site selection accounted for the required minimum distances as specified by the Canadian explosives regulations. Approvals and permits are required from the government regulating bodies prior to construction of:

- Explosives Plant – this site includes the storage facility for raw materials, the offices and garages as well as the emulsion plant and pump truck loading area;
- Explosive Magazines – this site includes the magazines to store the blasting caps, primers, detonation cord and packaged explosives.

For the fabrication of the bulk emulsion, the explosives supplier will transport ammonium nitrate prills by train to Schefferville. The explosives plant will consist of predesigned/prebuilt modules that are easily transported and assembled and will comprise, as a minimum, the following components:

- Emulsion plant;
- Storage area for ammonium nitrate, emulsion and explosives;
- Support installations.

The estimated amount of explosives required per year is approximately 45 million kg.

Storage of initiating explosives will be in separate explosives magazines. Magazine facilities include at least two (2) separate locked storage buildings, both surrounded by earthen bunds, equipped with lightning protection inside a fenced and fully guarded compound.

The explosives magazine building and the ammonium nitrate storage and explosives plant building will be located on the west side of the waste dumps. A minimal safe distance of one (1) km is provided from explosives plant.

In order to support the explosives supplier, the mine operator will build and maintain the access road to the two (2) sites and will supply electric power, communications and diesel fuel for the manufacturing of the emulsion as well as the operation of mobile equipment. The mine operator is also required to mobilize and house the contractor's workforce.

18.5 Main Access Road

The main access road from Schefferville to the mine site will be designed to permit heavy traffic to circulate at normal speed as regulated by provincial governments. The existing gravel road from Schefferville to the mine site will be upgraded.

Site roads to be used by off-highway mine trucks will be built to a stronger construction standard than the main access road.

18.5.1 Mine Haul Roads

Access for mine haul trucks and other large equipment to maintenance facilities, diesel fuel service and primary crushing areas are by mine roadways.

It has been decided that mine service vehicles will use the heavy vehicle haul roads and designing a light vehicle road network within the active mining areas is not required. Strict operating procedures will be in place to safeguard light vehicles and delivery of consumables.

18.5.2 Process Plant Site Access and Service Roads

Plant roads are used for normal operation traffic, which consists of light vehicles, small personnel buses and freight transport trucks. Site access roads are provided for the main entrance to the facility in order to allow delivery of materials, large personnel buses, and light vehicle traffic. Plant roads and access roads have provision for two-way traffic.

Service roads are typically one (1) lane unsealed roadways and are suitable for maintenance vehicles and operational access.

During the detailed execution phase, it is recommended that the following be reviewed:

- The impact of construction, and operations traffic within the plant to ensure separation of heavy vehicles from light vehicles and to assess if mitigation is required;
- Project driving standards and road rules;

- The impact of construction, and operations traffic on the local roads, and community, to assess if mitigation is required;
- Community awareness training, traffic movement protocols regarding times of movements through villages (e.g. avoid school transit times and worker start and finish times), traffic control and managed movements through towns; Delivery trucks access and traffic management to the site contractor's lay down area from the existing public road from Schefferville;
- Light vehicles reporting directly to the car parking area. Personnel access the site through the main security gate. New employees and visitors are directed to the training room for induction training that is compulsory before access to the site is allowed.

18.5.3 Slurry Transport Access and Service Roads

The KéMag slurry transportation system is in Québec for its complete length of 645 km. Roads along the slurry transport system route only exist around Sept-Îles. The KéMag, slurry transportation system will parallel the Arcelor Mittal railway at times and then follow route 389 between Fire Lake and Mont-Wright, totaling less than 100 km of shared right of way ("**ROW**"). It will occasionally benefit from forestry and wood logging routes before climbing to higher grounds beyond Mont-Wright where no infrastructure exists. This last 300 km stretch is entirely undeveloped territory.

Access routes to and along slurry transport system will be construction roads and require slow speed driving in four-wheel drive vehicles.

18.5.4 Export Terminal Access and Service Roads

Pointe-Noire is a well developed industrial and port sector and as such, is well served in all aspects of infrastructure. A provincial highway (HWY 138) runs alongside the sector and a two (2) way paved route serves the area (*Chemin de la Pointe-Noire*) leading all the way to the Alouette aluminum smelter. A municipal road runs along the south edge of the area (rue Alban Blanchard). These roads can be used in all weather conditions by the largest vehicles although some limitations in weight may apply in the spring.

Main access to the facilities will be from the south. Road construction will be needed to branch off rue Alban Blanchard and will run alongside the stockpiles up to the pellet plant. This configuration avoids all power line and railway crossings although it is a longer driving distance, it has the shortest length and construction costs.

The on-site roads will provide access to all facilities including the water treatment pond and access along the site security fence. These roads around the facilities will have two (2) lanes each 3.5 m wide for a total width of seven (7) m and will be paved with asphalt. The road base construction will be from CBR 60 % material and include 1.0 m shoulders.

The other on-site roads will be one (1) lane with passing areas, four (4) m wide and will be granular paved.

All on-site main roads will be provided with 11 m galvanized steel pole mounted Cobra type 250 watt HPS Roadway light fixtures, spaced at 50 m apart to meet the local regulations for roadway lighting. Traffic lights will be provided at cross-roads and restrictive entrances.

In general, traffic speeds will be restricted to 50 km/h and 30 km/h close to the facilities.

Adjacent to the auxiliary buildings will be asphalt paved areas for parking. And at the main gate will be an asphalt paved area for off-site car parking and a bus station with a covered drop-off and collection area.

The pavement structure will be in accordance with the recommendations provided by the Geotechnical Consultant.

18.6 Tailings Management Facilities

Water management for the Tailings Management Facilities (“**TMF**”) consists of determining the amount of water (and ice) stored in the TMF to:

- Ensure the process water demand is met according to the selected design criteria;
- Determine the TMF pond volume taking into account all the inflows and outflows;
- Ensure the TMF dykes and spillways can handle the design floods considering wind effects such as waves and wind setup.

18.6.1 Tailings Storage Basin

Fine tailings are cumulated in the tailings pump boxes installed in tailings pumping station. From there, the fine tailings are pumped to the tailings storage basin. There are two (2) tailings lines running from the pumping station to the tailings storage basin.

One (1) tailings line is in operation while the second one is on standby. According to geotechnical needs, the discharge point of the operating tailings line can be moved to form beaches along the tailings dams.

The pumps are sized for the longest run of tailings lines and are variable frequency driven. Since the pumping distance varies, the pumps can accommodate a lower discharge head. However, in the future, the pumps may be changed as the discharge head requirements change.

18.6.2 Process Water Pumping Station

A process water pumping station is installed inside the process plant with eight (8) vertical turbine pumps, fixed or variable frequency driven. Each pump delivers the water necessary for one (1) ore processing line.

The two (2) make-up water pipes discharge in the pumping station basin. Before discharging into the basin, a branch from the pipe provides water to the fire and potable water pumping station.

The main contributor to the process water basin is the overflow of tailings thickeners. The overflow from thickeners flows by gravity into the pumping station basin. Some additional water comes from the flotation water pump station.

18.7 Camps

18.7.1 Temporary Construction Camps

Lodging requirements are estimated based on the total construction hours for the project and the planned schedule. The construction camp for KéMag is presently sized to 1,700 rooms taking into consideration 400 beds of the permanent accommodation complex at the mine will be utilized during construction.

For the Sept-Îles area, it is estimated at 570 rooms excluding the camp for the Pellet plant. The slurry transportation camp is sized at 1,600 rooms. Camp population will include multidisciplinary construction workers, contractors, management staff, service people, owner's representatives, security and visitors. The design of camps have included all required sub-facilities as catering, recreational rooms and others amenities.

18.7.2 Permanent Residential Complex

The accommodation complex will be designed to meet the sleeping, hygiene, dining and recreational requirements for 400 workers and future employees. The surroundings will provide a level of comfort intended to optimize individual productivity and minimize the adverse effects of being separated from home and family for extended periods of time. Interior design, as well as the selection of furnishings, fittings and fixtures, will be considered. The concept could be standard or modular. Each room will be equipped with shower/water closets and all furniture and related equipment.

The residential complex proposes 25 units per sleeping quarter wing for a total of 400 individual rooms. Ninety-six (96) parking spaces are planned. The Complex allows flexibility with multiple shifts and fly in – fly out strategy.

Heating, air conditioning (optional), ventilation and make-up air will be provided by air-handling units. Ventilation rates will satisfy construction code requirements. The central kitchen is used to prepare all meals that are served in the central dining area for the operation personnel.

Fresh water will be provided by two (2) water wells located south to the complex. Single and double gates with galvanized steel fence will also limit access.

A dedicated water treatment plant, including a 6.5 m diameter fire water tank will provide potable and heated water for workers and complex utilities. A helipad of 250 m^2 is also annexed to the complex. Helicopter hangar is assumed to be located at Schefferville airport.

Offsite airstrip has not been developed as part of this study. Flight services will be provided from Schefferville's commercial airport.

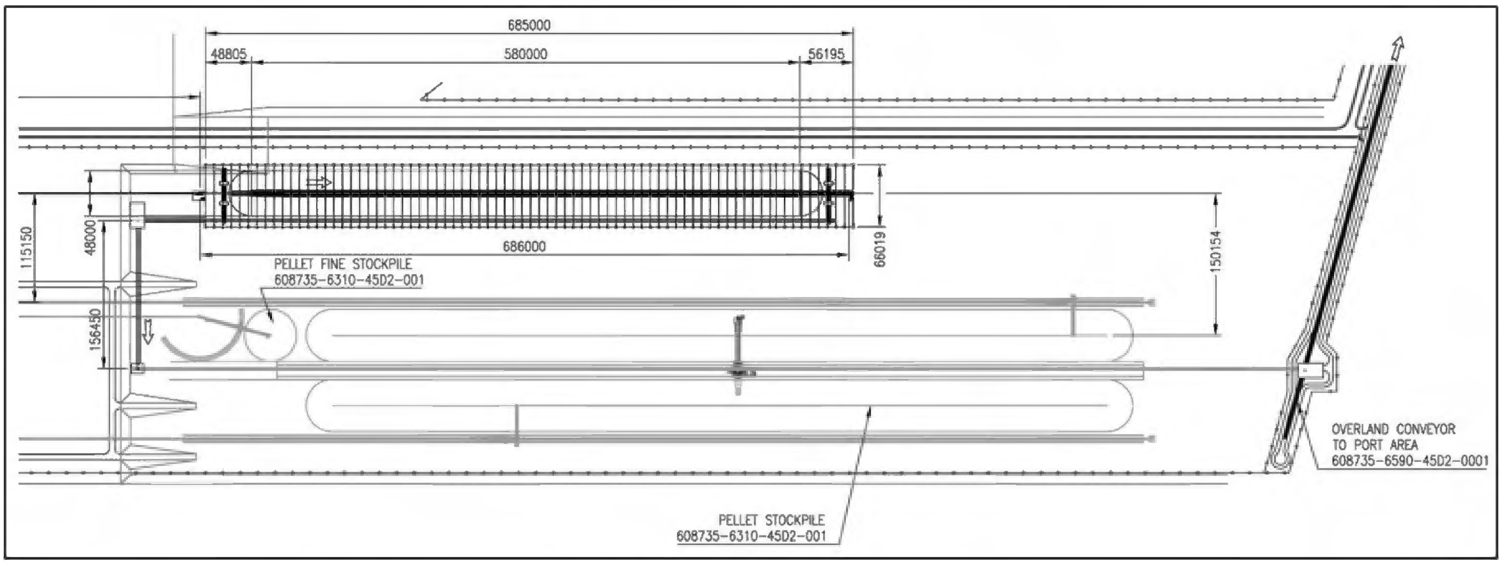
There will be no permanent residential complex for personnel in Pointe Noire.

18.8 Yard and Material Handling

18.8.1 Pellet Storage and Ship Loading

Final pellet product received from the pellet plants is transported and stored in accordance to an established yard storage plan for the various pellet qualities in two (2) dedicated yards each having a storage capacity of 750,000 Mt. The layout is depicted on Figure 18.2.

Figure 18.2 – Pellet and Concentrate Storage Area General Arrangement Plan View



Product from the two (2) pelletizing lines, each having a capacity of 1,073.2 t/h, is transported to dedicated storage yards and is stockpiled by two (2) stackers on rails. Each stacker is dedicated to a single stockpile. The transfer conveyors are fitted with belt scales to maintaining inventory of each pellet quality. The two (2) stackers have lofting and slewing capability to provide the possibility of stacking the product in a chevron or windrow pattern for product blending.

Depending on the pellet quality being shipped, pellets will be reclaimed by one (1) single bucket wheel reclaimer and transported to ship loaders via one (1) 7.6-km long overland conveyor to the transfer tower and ship loading system that will be operated by the Port Authority of Sept-Îles ("**SIPA**"). One (1) belt weighing scale has been installed on the reclaim conveyor for monitoring the tonnage being loaded.

One (1) product sampler is installed at the discharge end of the reclaim conveyor for sampling and analysis of the product that is being shipped. The capacity of the reclaiming system exceeds the 8,000 t/h shiploader rate to account for irregularities in the stockpile and reclaimer direction change.

a) Pellet Fines Storage and Ship Loading

Pellet clips received from the pellet plant are transported to a stacking area via one (1) transfer chute, one (1) pellet fines transfer conveyor and one (1) fines stacker. The stacker has the possibility of being slewed manually to maximize storage and has a capacity of 1,000 t/h.

Pellet fines will be reclaimed from the pellet fines stockpile with the pellet reclaiming system at the tail end of the pellets reclaim conveyor. All pellet ship loading equipment such as the bucket wheel reclaimer, conveyors, belt weighers, samplers etc. will be utilized.

b) Concentrate Storage and Ship Loading

The concentrate will be shipped via a slurry transportation system and filtered at Sept-Îles facilities. The concentrate filter cake will feed either the two (2) pellet plants (see Section 17.0) or be stocked in covered stockpile of 600,000 t. The concentrate will be reclaimed by a bridge reclaimer and will be transferred to ship loaders by the same overland conveyor used for pellets shipment.

One (1) belt weigh scale has been installed on the reclaim conveyor for monitoring the tonnage that is being loaded. One (1) product sampler has been installed at the discharge end of the reclaim conveyor for sampling and analysis of the product that is being shipped.

18.9 Maintenance Workshop and Main Warehouse

18.9.1 Mine Trucks and Light Vehicles Garage and Warehouse

The mine trucks and light vehicles garage and warehouse area is designed for the maintenance of heavy equipment like 363 t haul trucks and flexible for light vehicles.

The mining fleet availability for different machine classes does vary slightly and may in some cases be as high as 90 % but 85 % is deemed appropriate as an overall percentage for design purposes.

A further 15 % downtime has then been allocated according to machine type, i.e. "tracked" or "rubber-tired" and for routine/regular maintenance compared to major repairs and overhauls.

The assumption made for "tracked" equipment is that all routine service is done in the pit by means of mobile maintenance crews and maintenance service vehicles fully equipped with service and lubrication equipment. Occasional major repair work and major machine rebuilds will be performed in the workshops.

Similarly, the percentages applied to rubber-tired equipment assumes that these machines are normally driven back to the workshop for routine repairs and maintenance and major rebuilds, and will only be repaired in the field when they break down there, and it is easier to repair on site rather than move them back to the workshop for repairs.

Following this philosophy and assumptions, the 6,370 m^2 mine truck maintenance garage will include:

- Seven (7) bays with door opening of 12 m wide x eight (8) m high for truck box horizontal position;
- Two (2) bays with door opening of 12 m wide x 16 m high for truck box fully extended;
- Two (2) maintenance pits;
- Two (2) overhead crane of 60 t with service platforms;
- Bollards of 300 mm diameter x two (2) m long.

Provision has also been made for roller doors to be fitted to all openings on the bays.

There will be separate parking areas for all heavy vehicles and light vehicle traffic at all the facilities for shift change and any other idle time.

18.9.2 Mine Truck Tire Change and Washing Facility

The mine truck washing facility is designed on an area of 1,475 m^2. It takes into account two (2) opposite door opening of 12 m wide x eight (8) m high. This area includes three (3) distinct stages:

- Warming stage (air blowers);
- Washing stage;
- Drying stage.

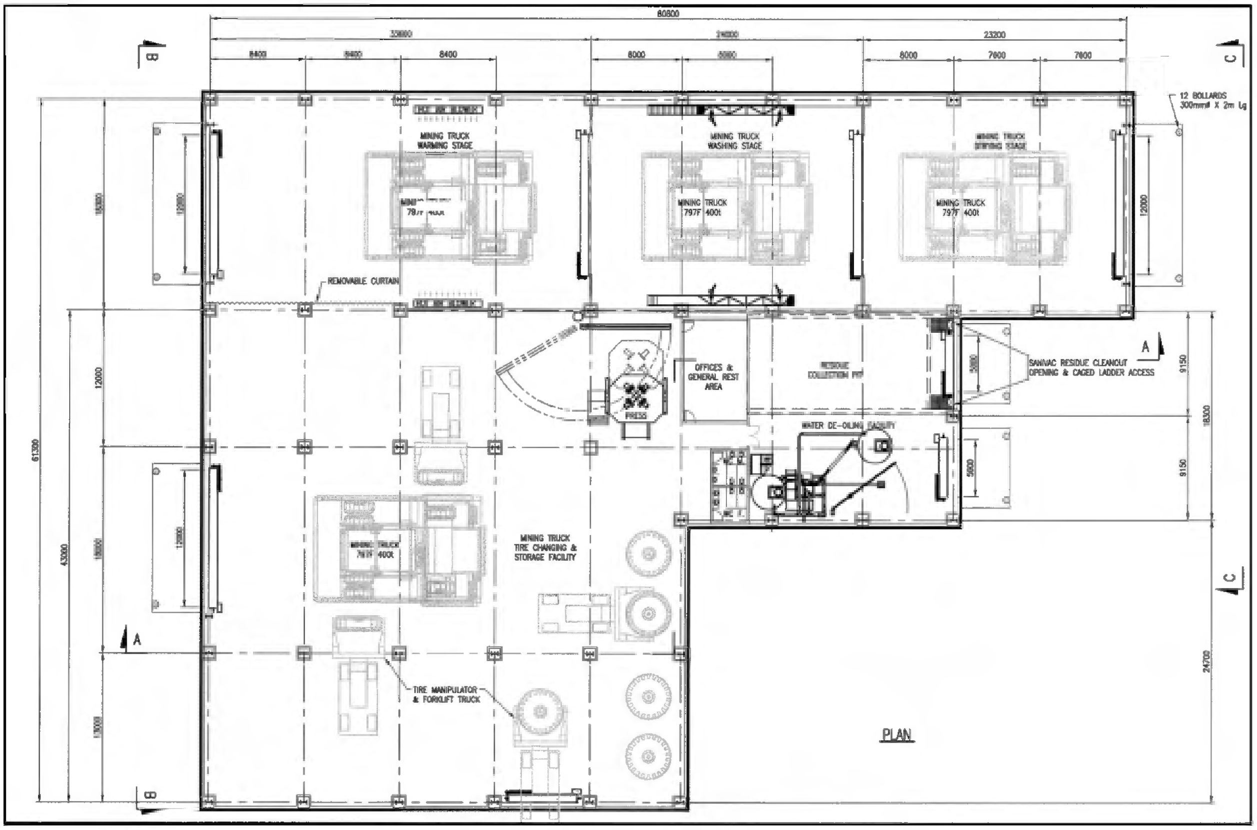
Drainage of washing residue is directed into a 4,65-m deep underground pit. This pit is provided with a cleanout opening, waste oil traps and sumps.

The mine truck tire shop area is 2,230 m^2 and will contain:

- Main door opening of 12 m wide x 8 m high;
- Area for inspection of mining equipment tires;
- Areas for tire storage, stripping, mounting and washing;
- Tire handlers;
- Administrative offices, changing rooms and parking areas.

The proposed mine truck tire change and washing facility is shown in Figure 18.3.

Figure 18.3 – Proposed Mine Trucks Tire Change and Washing Facility



18.9.3 Site Main Shop

The location of these facilities has been driven by the general location and proximity to the crushing, screening and processing areas. This 6,050 m² facility has been determined based on the philosophy of having day-to-day maintenance work done and major repairs carried out at a central point. The design has taken into account the number of equipment and size in conjunction with estimated maintenance requirements based on overall equipment availabilities. Sufficient clearance is provided for free maneuvering around the equipment. The warehouse is the storage place for all consumables and spare parts for the process plant.

The base case mine infrastructure requirements have been derived from current estimates of mine production and schedules.

18.9.4 Pointe-Noire

The maintenance workshop and main warehouse are sized to support the filtering, balling and pelletizing and stockage, reclaim and shiploading equipment. They also include stores and sanitary facilities

There will be no vehicles maintenance garage at Pointe-Noire facilities. The maintenance will be performed by others in town of Sept-Îles.

18.10 Ship Loading and Jetty Facilities

The ship loading and jetty facilities have been developed and will be operated by the Port Authority of Sept-Îles (SIPA).

18.11 Boiler Houses

18.11.1 Mine

There are two (2) boiler house facilities and both boiler facilities are identical.

One (1) boiler house is located adjacent to fire and potable water pumping station. It currently consists of six (6) steam water tube boilers, each with a 100,000 lb/h (45,455 kg/h) capacity. Of the six (6) units, four (4) are in operation, one (1) is stand-by and one (1) is considered future or capital spare. Diesel fuel is provided from two (2) nearby storage tanks. The boilers provide heating for all process buildings and shop areas.

The second boiler house is located adjacent to the main trucks maintenance garage building. It consists of three (3) steam water tube boilers, each with a 50,000 lb/h (22,730 kg/h) capacity. Of the three (3) units, two (2) are in operation and one (1) is stand-by. Diesel fuel is provided from nearby Diesel storage tank and two (2) used oil tanks.

The boiler provides heating for the shops, garages and storage buildings. They also provide steam for truck washing. The mine dispatch building is electric heated.

18.11.2 Diesel Fuel Supply, Storage and Distribution

a) Mine Equipment

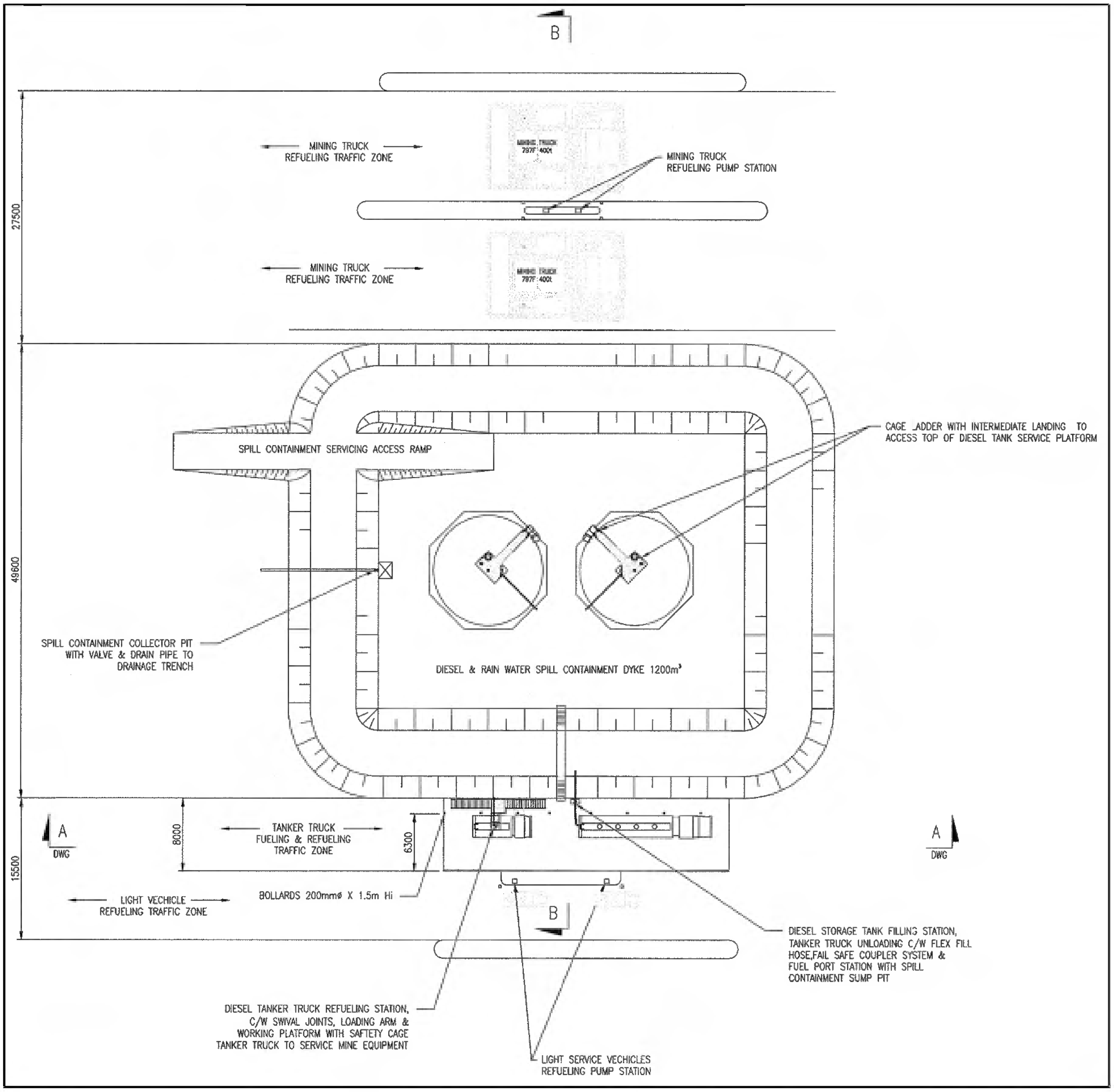
Two (2) diesel storage tanks, 1,000 m³ capacity each, provide sufficient capacity for mine and process plant operations for 15 days. Two (2) vehicle refuel stations are provided. One (1) is dedicated to mine trucks and one (1) for light vehicles.

A tank truck will supply fuel to Diesel powered mining equipment located in the pit. The proposed diesel fuel tank farm with the mine truck refueling station is shown in Figure 18.4.

An emergency power plant Diesel fuel tank farm provides fuel for Diesel generators. The diesel storage capacity is 1,000 m³. The containment area is about 1,100 m³. With Diesel Generators consumption between 250 to 300 tonnes/day, this gives three (3) days of autonomy.

At intermediate pumping station (PS2), the emergency diesel generators consumption will be approximately 45 tonnes/day while maintaining a minimum flow in the transportation system.

Figure 18.4 – Plan View of Diesel Fuel Tank farm with Mine Truck Refueling Station



b) Boiler Fuel Storage at Mine site

There are two (2) fuel storage facilities:

- One (1) boiler house is located adjacent to fire and potable water pumping station. Two (2) boiler fuel storage tanks, each with a 1,000 m³ capacity, provide sufficient fuel for eight (8) days of operation. A 50 m³ diesel storage tank provides the fuel required for starting the boilers. All tanks are installed in a dyke area; and

- The second boiler house is located adjacent to the main trucks maintenance garage building. One (1) boiler fuel storage tank with a 1,000 m³ storage capacity provides sufficient fuel for fifteen (15) days of operation. A 50 m³ diesel storage tank provides the fuel required for starting the boilers. A 100 m³ used oil storage tank provides an alternative fuel for boilers. All tanks are installed in a dyke area.

c) Pointe-Noire

Two (2) Diesel storage tanks, 1,000 m³ capacity each, provide sufficient capacity for Pointe-Noire installations operation for twenty five (25) days.

Two (2) vehicle refuel stations are provided: one (1) dedicated to tanker trucks and one (1) for light vehicles.

d) Boiler Fuel Storage at Pointe-Noire

There is no boiler at port. Heating will be by natural gas or electric.

18.12 Sewage Treatment Plant

The sewage water treatment plant is designed in respect of the environmental design rules and regulations. It will comply with water resources Act, the environmental control water and sewage regulations 20039 and the environmental quality act (Quebec). All equipment will operate 24 hours per day, 365 days per year.

18.12.1 Mine Plant, Residential Complex and Pointe-Noire Sewage Water Treatment Plants

The Bio-discs process is proposed for the respective sewage water treatment plants. The design will handle the following operating conditions:

- Average flow of 80 m³/d (based on a maximum operating requirement of 400 persons and an average water consumption of 200 litres/persons/day);
- A maximum flow of 220 m³/d;
- The operation of the plant will allow an incoming peak flow of domestic sewage water of 13.4 m³/h. The peak flow will be considered for one (1) hour, twice a day in the morning and in the evening.

18.12.2 Booster Pumps Station (PS2)

For the booster pumps area, a sanitary pit is proposed and will be cleaned periodically.

18.12.3 Sanitary Waste Disposal

a) Mine Site

There will be no domestic waste disposal site. Uncontaminated solid waste generated during construction and operations will be managed from Schefferville. Therefore, no sanitary landfill ("**SL**") has been considered.

b) Pointe-Noire

There will be no domestic waste disposal site. It will be managed from Sept-Îles.

18.13 Contaminated Waste

A number of oil spill containment areas are incorporated into the plant design where mineral oils and lubricant are frequently used. The containment areas are comprised of sumps with low capacity oil skimmers or high demand oil separators. The oil contaminated materials will be disposed of in an offsite contaminated waste treatment facility.

It is proposed to transport all contaminated waste products to an off-site recycling facility. Approaches could be recommended to an offsite contaminated waste disposal contractor who manages a facility near the city of Schefferville or Sept-Îles. This facility is able to destroy contaminated wastes by incineration and/or safely store contaminated wastes for future safe disposal.

18.14 Acid Rock Drainage Treatment Plant

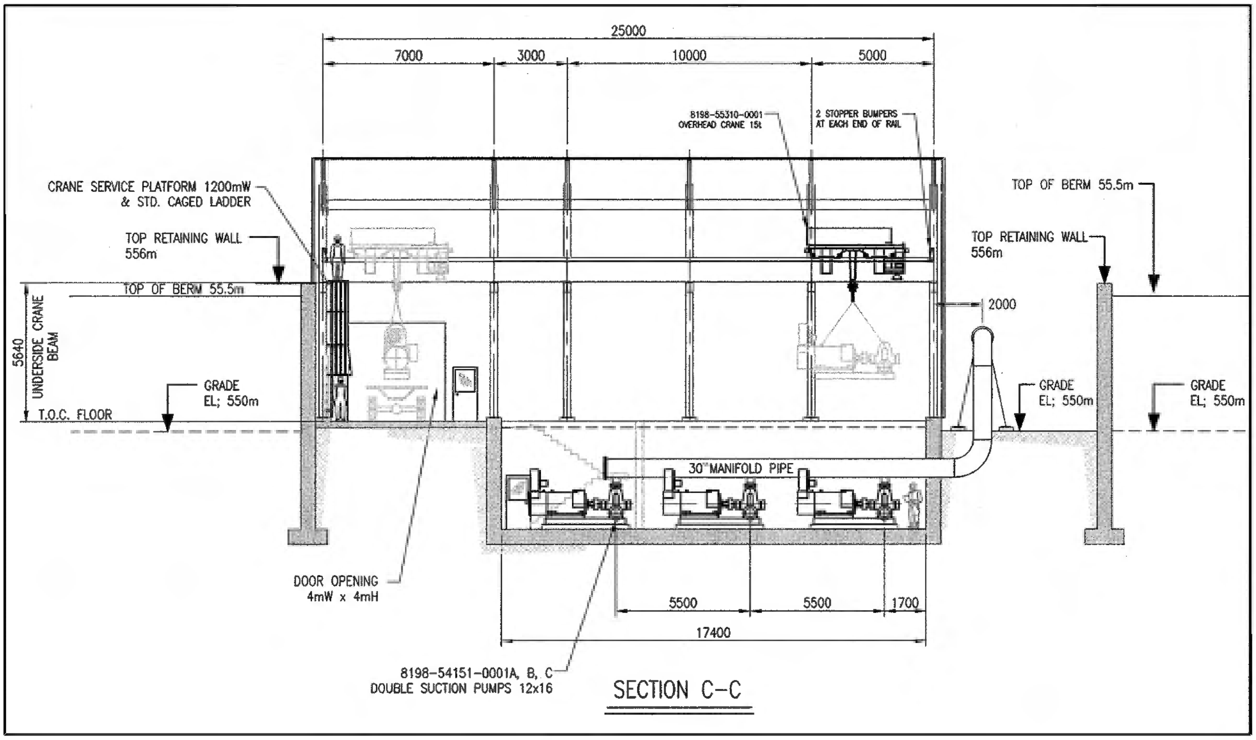
The waste dump runoff water is collected by gravity in a water pond 100 m by 100 m. From this pond, water flows by gravity in a polishing pond 40 metres by 60 metres. An underground pipe installed under the dams separating the two (2) ponds allows the water flow from one (1) pond to the other. The water flow is controlled by a sluce gate installed on the dam separating the two (2) ponds.

On the dam separating the two (2) ponds, a pH analyzer provides information regarding water treatment needs. According to the needs, a lime solution is injected into the pipe connecting the two (2) ponds.

The raw water from the river pumping station is discharged into the polishing pond and will eventually mix with the treated water.

At the end of polishing pond, a polished water pit and a pumping station are provided as show in Figure 18.5. The water is pumped to the fresh water pond.

Figure 18.5 – Acid Rock Drainage Pumping Station



18.15 Communications

18.15.1 Voice and Data Communications

Project network communication requirements are met via a Local Area Network ("**LAN**") connected to Wide Area Network ("**WAN**") via a service provider located at Pointe-Noire. The WAN network communication enables contact with global internet and telephone service whereas the LAN network communication provides all telephone and information technology needs inside the plant. In addition, a Voice Radio System is provided to enable communication among individuals working in different areas.

To ensure system reliability, redundant communications systems are installed at the process plant connected via a fiber optic cable along the slurry transportation system. The LAN network communication is based on Ethernet protocol. It consists of fiber optic and copper cables. Telephone service is provided in all offices, control rooms, electrical rooms, and accommodation complex.

Telephone service is based on Voice over Internet Protocol ("**VoIP**") technology. Communication servers are provided for call management and specific applications such as Auto-Attendant functions. Computer and internet facilities are provided in all offices, control rooms, electrical rooms, and accommodation complex.

A Public Address ("**PA**") system is installed in all buildings and accommodation complex. Close Circuit Television ("**CCTV**") and an Access Control System are provided in all buildings and accommodation complex including slurry pumping stations. Both systems use IP-based technology.

Satellite phones are used for communication along the slurry transportation system. The intermediate pumping station is provided with a LAN network communication connection for office network and telephone services.

18.15.2 Fiber-Optic Network

The telecommunication system design is based on a fiber optic "backbone," which carries all communications, including office data, process data, voice and video.

A ring topology fiber optic cable connection will be provided to each electrical, control and technical room.

18.15.3 Radio Networks

A digital multichannel two (2) way voice radio system is provided. The two (2) way radio communication system is designed for continuous operation, 24 hours a day, and 365 days a year.

Radio network covers the following areas:

- Mine;
- Primary crusher;
- Secondary crusher;
- Process plant building;
- Remote pump houses;
- Tailings area;
- Pellet plant and port facilities;
  - Dewatering plant;
  - Pellet plant building;
  - Product storage and ship loading.

18.15.4 Telephone System

Telephone system is IP-based technology. For security, plant site and port site are provided with dedicated IP Private Branch Exchange ("**PBX**") and voice message server. If for any reason, communication between Pointe-Noire and the plant site is lost, the plant site internal telephone system will stay active.

Telephones are provided at the following location:

- Offices;
- Electrical rooms;

- Control rooms;
- Accommodation complex.

Access to Public Switched Telephone Network ("**PSTN**") and internet is via a service provider located at Pointe-Noire.

18.16 Fire Alarm and Detection Systems

In the absence of any specific requirements from the Owner's insurance underwriters, this system will be designed to meet the National Building Code requirements.

The fire alarm system will be of the addressable type and comprise the following devices:

- Intelligent addressable detectors (smoke, heat and combined fire detectors);
- Manual pull stations;
- Audible horns, including strobes where required in noisy process areas.

The main fire alarm panel will be installed in the guard house and a remote annunciation panel will be provided in the plant Central Control Room. Any remote fire alarm panels will be installed locally, for interconnected distant areas, on a communication loop with the main fire alarm and the annunciation panels.

19.0 MARKET STUDIES AND CONTRACTS

The Project aims to be a significant, highly competitive producer in terms of cost and product quality, with the ability to supply pellets for the blast furnace (BF) and direct reduced iron (DRI) sectors, along with concentrate/pellet feed, to consumers in Europe, North America, the Middle East/North Africa ("**MENA**") and the Asia/Pacific region.

The marketing strategy and plan for the Project is being jointly developed and implemented by NML and TS, working in conjunction with World Steel Dynamics Inc. ("**WSD**") of Englewood Cliffs, New Jersey, USA, a leading steel information service whose team includes Dr. Joseph J. Poveromo, President, Raw Materials and Ironmaking Global Consulting.

To supplement WSD's work, UK based Papillon Mineral Services Ltd., was engaged to both model and provide detailed analysis of the pellet market. In addition, NML and TS have met with iron and steelmakers in each of the above market areas to discuss future iron ore requirements and sourcing strategies, with encouraging results. The collective views are incorporated into this Report.

19.1 Steel Trade and Production

The iron ore market is directly related to iron and steel production. WSD's crude steel production forecast is shown in Table 19.1. A return to steady global steel production growth is projected over the period 2014-2025, with a more cautious view of Chinese production than in positions taken by some other forecasters. As mentioned above, the marketing objective for the Project is to have a geographically diversified sales portfolio.

Against this background, there are good sales opportunities for the Project.

Firstly, one of the Project's most favorable attributes is the possibility of having TS as an equity investor as well as off-taker of a substantial portion of production, and with that, a core market in the integrated, Blast Furnace/Basic Oxygen Furnace ("**BF/BOF**") steelmaking sector.

In addition, WSD's Global Metallics Balance data point to other positive factors.

Table 19.1 – World Steel Dynamics' Crude Steel Forecast

Location	2012 (Mt)	2013e (Mt)	2014e (Mt)	2015e (Mt)	2025e (Mt)	CAGR 2014-2025 (%)
Advanced Countries	**464**	**448.7**	**460.8**	**470**	**536**	**1.4%**
Japan	107.2	108.8	110.2	111.0	121.0	0.9%
Western Europe	140.8	132.9	136.9	140.4	158.1	1.3%
United States	88.7	87.0	89.5	92.5	110.0	1.9%
Small Cap. Adv.	126.8	120.0	124.2	126.6	147.0	1.5%
China	717	770	790	802	850	0.7%
Developing World	**368**	**355.3**	**363**	**376**	**486**	**2.7%**
Africa	7.3	6.8	7.2	7.6	10.0	3.0%
Brazil	34.7	33.4	34.0	35.0	42.5	2.0%
CIS	111.5	106.6	108.0	109.0	122.5	1.2%
Eastern Europe	14.2	12.0	12.6	13.5	17.5	3.0%
Developing Asia	21.4	21.5	22.0	24.1	40.0	5.6%
India	76.8	78.5	78.0	82.0	113.0	3.4%
Latin America	31.7	30.0	31.6	34.0	45.0	3.3%
Turkey	35.9	33.5	34.0	34.5	33.9	0.0%
MENA	34.2	33.0	35.1	36.2	62.0	5.3%
World Total	**1,548**	**1,574**	**1,613**	**1,648**	**1,873**	**1.4%**
World Ex-China	831.2	804.0	823.3	846.4	1,022.6	2.0%

Source: WSD estimates

The growth of USA steel demand in the next decade is expected to boost investment in EAF capacity relying on DRI and, in turn, DR grade pellets. Lower natural gas prices in the USA are making the construction of DRI facilities an attractive option for a USA-based steel mill. Meanwhile, further expansion of EAF steelmaking in the MENA region will also increase the need for DR grade pellets.

Although an overall surplus of steel scrap appears to be developing, the future supply/demand balance for prime scrap is expected to be tight, creating the demand for more supply of clean iron units in the form of DRI, especially as EAF steelmakers move up market into sheet production.

19.2 Market Opportunity for Pellets

Table 19.2 shows the size of the market for pellets relative to overall iron ore demand, along with a projection of growth to 2022.

Table 19.2 – Iron Ore Demand Forecast Showing Pellets

Product	2012 (Mt)	2015e (Mt)	2018e (Mt)	2020e (Mt)	2022e (Mt)
Fines	1,330	1,485	1,589	1,660	1,725
Lump	230	252	265	273	280
Pellets*	314	363	408	441	474
Total	**1,874**	**2,100**	**2,262**	**2,374**	**2,479**

*excludes estimated Chinese demand that is supplied domestically
Source: Papillon Minerals Limited

Along with the previously mentioned growth in the market for DR grade pellets, the demand for pellets in the seaborne market generally is expected to increase among BF/BOF steelmakers as a result of the following structural changes:

- Carbon tax measures and stringent environmental regulations;
- Reduced sinter productivity resulting from increased share of ultra-fine materials in the sinter burden;
- Increase silica content in sinter fines creating demand for low-silica pellets;
- Declining availability of high quality lump ore.

The supply-side response appears limited, and WSD's analysis shows a tightening of the pellet supply-demand balance beginning in 2018 leading to a potential deficit if no new pellet plants are built other than those currently planned. A regional breakdown of the future pellet demand shown in Table 19.1 indicates that the Project would be well placed to compete with existing as well as prospective new suppliers. These areas again include Tata Steel Europe (“**TSE**”), which would be the core pellet customer with demand of approximately 5.5 million tonnes annually.

Tata Steel's operations in India are presently self-sufficient in iron ore and thus not considered as a candidate market for purposes of this review.

19.3 Products and Cost Position

Extensive test work at leading laboratories has been carried out on the Project's targeted range of concentrate/pellet feed, acid and fluxed pellets for BF/BOF steel making, and DR grade pellets. With its comprehensive iron ore database, WSD is able to benchmark the quality and cost of the Project's products against competitor parameters, and has concluded as follows:

- BF Pellets – Would be well accepted by any ironmaking operation as comparable to the best Canadian and Swedish pellets, better than Brazilian pellets, and clearly better than pellets from elsewhere; including the CIS and India. Can contribute to improved blast furnace performance at Tata Steel Europe;
- DR Grade Pellets – Quality will be comparable to the leading producers: Vale, Samarco, LKAB, IOCC and Arcelor Mittal Mines Canada;
- Global Cost Curve Position for Pellets – The Project would potentially be the lowest cost pellet producer in North America;
- Pellet Feed – The Chinese steel plants are the main consumers of the domestic magnetite pellet feed to produce pellets. Due to a projected growth in the Chinese pellet demand and at the same time forecasted closures of high cost domestic mines, the Chinese demand for seaborne magnetite pellet feed is expected increase.

19.4 Markets

The Project is expected to require a mix of captive and merchant market sales. Its planned range of pellets and concentrates and highly competitive cost structure, coupled with access to the new, deep-water ship loading facilities at Sept-Îles, Québec, creates an attractive opportunity for steelmakers worldwide seeking high quality products from a stable mining and processing jurisdiction with a long history of reliably supplying the seaborne and North American markets.

19.5 Tata Steel Europe – The Main Captive Market

Tata Steel's three (3) integrated steel works in Europe (Port Talbot – Wales, Scunthorpe – England and IJmuiden – The Netherlands) are shown in Figure 19.1. In January 2007, Tata Steel acquired the Anglo-Dutch steel maker Corus Group, thereby establishing a major presence in the European market to supplement its core production base in India. TSE iron ore consumption is approximately 20 million tonnes per year.

TSE is pursuing a strategy of greater self-sufficiency in raw materials, and currently the only captive supply is TS's DSO product, which had its first shipment in September 2013.

Since TSE's operations are both geographically proximate to and historically familiar with supply from Canada, they are also natural outlets for the Project and could consume 40 – 50 % of the Project's output. Accordingly, product specifications are the result of collaboration with TSE's supplies team.

As the only European steel maker with on-site pelletizing, the IJmuiden works in The Netherlands provides the Project with a unique captive market opportunity for pellet feed in what will be a highly competitive open market environment as discussed above.

Figure 19.1 – Natural Fit of the Project to Tata Steel Europe's Operations

19.6 Other Market Opportunities

In order to supplement the core off-take requirements of TSE and minimize the Project's market risk, NML and TS are working together to identify reputable and reliable steel or steel tie-in companies with an interest in Project investment and/or a long-term supply contract. The relationships begin with Project introduction and interest is confirmed via a confidentiality agreement.

Simply stated, the process continues through technical evaluation and other due diligence which, if successful, then leads to a letter of intent or memorandum of understanding, commercial negotiations and the eventual goal of a long-term and stable relationship with the Project.

To date, this marketing campaign has produced interest from companies in North America, Europe, the Middle East and the Asia/Pacific. The process is on-going and will accelerate upon finalization of the Project Feasibility Study.

19.7 Pricing

It is well known that the global pricing mechanism for iron ore has since the mid 2000's, transitioned from the historical, annually negotiated benchmark system to shorter term settlements mainly using the 62 % Fe CFR China price as a reference, with upward or downward adjustments for product quality as appropriate.

The period since mid-2012, has seen extreme price volatility, signaling the difficulty of predicting even a one-year price. With regard to the long-term price for iron ore, there are numerous views among industry analysts and forecasting services, resulting in a wide range of predictions.

Steel is a cyclical business, and by extension iron ore prices should reflect this cyclicality over time. The methodology used by WSD for arriving at a long-term price reflects this likelihood by assuming that the steel industry is subjected to the conditions shown in Table 19.3 below and applying a percentage to each for the share of the cycle in which the condition applies.

Using the 62 % Fe CFR China price as a baseline, with appropriate quality adjustments, the result is a weighted average price reflecting the overall impact of the five (5) conditions over the cycle being analyzed.

Table 19.3 – Global Steel Industry Scenarios

Scenario	Includes
Boom Times	Possibility of a steel (hot-rolled band) shortage
Good Times	Rising steel demand, "pricing power" returning to the hands of the steel mills to a substantial extent
Fair Times	Not much change in steel demand, wide swings for hot-rolled band and steel scrap on the world market
Bad Times	Slack steel demand, lower prices for steelmakers' raw materials, lower steel mill production costs and reduced "pricing power" in the hands of steel mills
Shake-Out Times	Weak apparent steel demand and the possibility of brief episodes in which the price of hot-rolled bands falls to the marginal cost of the median-cost mill

Source: World Steel Dynamics

Based on the above methodology, the long-term prices, Free on Board ("**FOB**") vessel at Sept-Îles, used in the Project are shown in Table 19.4. Given the obvious importance of pricing to project economics and uncertainties around long-term price projections, the outcomes below are subjected to sensitivity analysis using various market assumptions.

Table 19.4 – Long-Term Pricing

Product	Free on Board - Sept-Îles
Blast Furnace Pellets	US$ 116.61/dry-tonne
Direct Reduction Grade Pellets	US$ 126.86/dry-tonne
Pellet Feed	US$ 90.00/dry-tonne

19.8 Conclusion

The marketing conclusions to date are as follows:

- Assuming a positive investment decision in 2014, the Project would come on stream at a time of resurgence in the global steel industry;

- The Project's range of high quality pellets and pellet feed and favorable cost curve position give it a competitive and global market reach;
- With its year round shipping capability and ability to handle among the largest ore carriers, the Port of Sept-Îles provides a competitive advantage to the Project;
- The Project would have a core market for 40 – 50 % of its output at Tata Steel Europe's operations;
- Lower natural gas prices in the US are providing electric furnace as well as blast furnace based steelmakers an attractive option to produce cleaner iron units at competitive prices by constructing DRI facilities, which require high quality pellets. This is a natural market geographically for the Project;
- Other market opportunities exist in Western Europe, MENA and the Asia/Pacific areas.

Working with WSD, the Project has incorporated cyclical steel industry conditions into its analysis aiming to replicate market behavior that drives long-term pricing.

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Applicable Environmental Assessment Regimes and Permitting

20.1.1 Environmental Assessment

The Project is expected to trigger several regimes of environmental assessment ("**EA**").

The Project is expected to trigger the EA regimes of general application established by the Canadian Environmental Assessment ("**CEA**") Act and the Environment Quality Act ("**EQA**") (Chapter I, Division IV.1), in addition to the provincial regimes of section 23 of the James Bay and Northern Québec Agreement ("**JBNQA**") (EQA, Chapter II, Division III) and section 14 of the Northeastern Québec Agreement ("**NEQA**") (*Règlement sur l'évaluation et l'examen des impacts sur l'environnement dans une partie du Nord-Est québécois*).

The federal regimes of section 23 of the JBNQA and section 14 of the NEQA are not expected to be triggered by the Project, because the components of the Project in the areas in question come under provincial jurisdiction. Where impacts on matters of federal jurisdiction are anticipated in the areas in question, the CEA Act, the Fisheries Act, the Metal Mining Effluent Regulations ("**MMER**") or other federal regulatory instruments will apply.

The regulatory regime of the National Energy Board Act ("**NEBA**") is not expected to be triggered for the Project, since the slurry transportation system will not be inter-provincial.

Ideally, the implementation of all the applicable regimes should be harmonized to the extent possible. To that effect, a conference call with the CEA Agency, the Major Projects Management Office ("**MPMO**") and the National Energy Board ("**NEB**") was held on May 3, 2013 to discuss how the respective regimes of the CEA Agency and the NEB might apply, particularly in terms of their harmonization. In brief, the regimes are anticipated to be coordinated as follows:

- NML would file with the CEA Agency an Environmental Impact Statement ("**EIS**") for the whole of the Project;
- The NEB would delegate to the CEA Agency its status as a Responsible Authority for the purposes of the EA, but it would participate in the EA as a body with relevant expertise.

20.1.2 Permitting

Once the regulatory authorities have released the Project from further EA, the applications for the various permits that are required for site-preparation and construction can begin to be filed, followed by applications for permits for the start of operations. In order to expedite the start of construction, preparation of the permit applications can begin before the completion of the EA. The principal permits that will potentially be required are listed below.

An amendment to schedule 2 of the MMER will be required for draining water bodies or watercourses frequented by fish for purposes of creating tailings disposal areas. An amendment to schedule 2 of the MMER will not be granted unless it can be shown that there is no viable alternative to disposing of them in the water bodies or watercourses in question. A thorough evaluation of the alternatives will be conducted and will be included in the EIS.

Representatives of Environment Canada and Fisheries and Oceans Canada have advised that the process of obtaining an amendment to schedule 2 can take up to 18 months. Until it has been completed, no work affecting the water bodies/watercourses in question can be undertaken. Other construction could proceed, but NML/TS would have to assume the risk that the Project might not be allowed to go ahead if the required amendment is not obtained.

In order to minimize the delay between the federal EA release and the approval of an amendment to MMER schedule 2, NML/TS may invest early in collecting and compiling the required information, in particular the preparation of a complete fish habitat compensation plan. The risk involved in that course of action is that time and money might be spent without the assurance that the Project will be authorized by the GoC.

As far as the fish habitat compensation plan to be included in the EIS is concerned, the CEA Agency and Fisheries and Oceans Canada have confirmed that a conceptual plan that provides a clear understanding of the effectiveness of the proposed compensation would be adequate.

a) Government of Canada

- Approval under Section 52 of the CEA Act 2012;
- Authorization under Section 35 of the Fisheries Act;
- Amendment of schedule 2 of MMER pursuant to the Fisheries Act;
- Permit under Section 7 of Explosives Act;
- Certificate of Public Convenience and Necessity under the National Energy Board Act;
- Permits under Section 7 of the Transportation of Dangerous Goods Act;
- License under Section 5 of the Radiocommunication Act.

b) Government of Québec

- *Loi sur la qualité de l'environnement* (LQE), Chapter I, Division IV.I;
- LQE, Chapter II, Division III;

- Authorization under *Loi sur les terres du Domaine de l'État*, art. 55 and permit under *Loi sur les forêts*, art. 31;
- Authorization under LQE, art. 22 and 32
- Authorization under LQE, art. 22 and *Règlement sur les carrières et sablières*, art. 2
- Approval under Loi sur le régime des eaux, art. 71
- Authorization under Loi sur la sécurité des barrages, art. 5 à 13 (*Règlement sur la sécurité des barrages*, art. 57 à 63);
- Authorization under LQE, art. 46 s) (*Règlement sur le captage des eaux souterraines*, art. 31);
- Lease under *Loi modifiant la loi sur les Mines*, art. 100;
- Authorization under *Loi modifiant la loi sur les Mines*, art. 241;
- Authorization under *Loi modifiant la loi sur les Mines*, art. 232.2;
- Lease of land under *Règlement sur la vente, la location et l'octroi de droits immobiliers sur les terres du domaine de l'État*, art. 39;
- Depollution attestation under Loi sur la qualité de l'environnement, art. 31.11 (see *Règlement sur les attestations d'assainissement en milieu industriel*);
- Wildlife Management Permit under *Loi sur la conservation et la mise en valeur de la faune*, art. 26;
- Authorization to alter a wildlife habitat under *Loi sur la conservation et la mise en valeur de la faune*, art. 128.7;
- Authorization under *Loi sur les explosifs*, art. 2, 3 and 11;
- Authorization under *Loi sur le Bâtiment (Code de Construction).*

c) Kativik Regional Government

- Certificate under *Règlement relatif à l'application de la Loi sur la qualité de l'environnement*, art. 8.

d) MRC de Caniapiscau

- Certificate under *Règlement relatif à l'application de la Loi sur la qualité de l'environnement*, art. 8.

e) MRC de Sept-Rivières [*Territoire non organisé ("**TNO**") du Lac Walker et de la rivière Nipissis*]

- Permit under the *Règlement relatif aux permis et certificats, aux conditions préalables à l'émission de permis de construction, ainsi qu'à* *l'administration des règlements de zonage, de lotissement et de construction numéro 06-92.*

20.2 Baseline Data Collection and EIS Status

20.2.1 Biophysical Aspects

Intensive field seasons to collect biophysical baseline data were conducted in fall 2011 and spring-fall 2012. The data are currently being analyzed, and most of the field reports have already been reviewed and approved. It is worth mentioning that the characterization of the watercourses crossings along the slurry transportation system has been done using an accepted practice at the time of the survey which requires characterizing only a representative sample for the purpose of the EIS. However, in recent months the guidelines issued by the CEA Agency have required the characterization of each watercourse crossing by linear infrastructure.

NML and SLE met with representatives of Fisheries and Oceans Canada, Environment Canada and the CEA Agency on August 30, 2013 to explain that it is more practical to characterize the ≈400 water crossings that each slurry transportation system scenario involves based on a representative sample for the purposes of the EIS (which has been done), and to study each crossing of the chosen route during the permitting phase. Based on that meeting and on subsequent discussions in September 2013, there is a high level of confidence that the existing baseline data are sufficient.

Therefore the collection of biophysical baseline data is essentially complete for the project except for the location of the future the discharge point for the excess treated water from the slurry transportation system which is still under discussion. The location of the discharge point for the excess treated water from the slurry transportation system is being refined through the analysis of alternative sites and discussions with the Ville de Sept-Îles. The exercise is being conducted in collaboration with SLE and the *Institut nordique de recherche en environnement et en santé au travail* (**"INREST"**).

Also, there is an area for which the completeness of the baseline data must be confirmed: archaeological data in the portion of the slurry transportation system south of Emeril. Desktop archaeological potential studies have been conducted in the full corridor of slurry transportation system. The issue is whether, as may be required pursuant to the 2012 *Guide pour l'initiateur de projet* issued by the *Ministère de la Culture et des Communications du Québec*, field surveys will have to be conducted prior to the tabling of the EIS, or whether they can be carried out in the selected corridor during the permitting phase once it is known which slurry transportation system will be built. The matter was discussed with representatives of the *Ministère de la Culture et des Communications* on September 24, 2013 and it was agreed to work towards finding a mutually acceptable solution.

A representative of NML was appointed as a member of the Corporation de protection de l'environnement de Sept-Îles (**"CPESI"**) in 2013. NML is also a member of the

Table de concertation sur la qualité de l'air à Sept-Îles, on which government, industry, environmental groups and civil society work together towards the production of a global portrait of atmospheric quality at Sept-Îles.

Finally, analyses of the metal leaching/acid rock drainage potential of samples of waste rock and ore from the deposit are being conducted and a program to conduct mineralogical examinations of composite ore samples in order to determine whether it contained asbestiform fibres that may affect human health using optical and transmission electron microscopy revealed the presence of no asbestiform amphibole fibres in the breathable fraction of the sample.

20.2.2 Social Aspects

On the socio-economic side, lists of stakeholders have been prepared, and all of the research instruments (e.g., questionnaires, interview guides) are virtually complete. The collection of data from secondary sources is very advanced. The field campaign to collect socio-economic data cannot begin until the Project Description has been filed and NML has initiated its public information and consultation program, since the public dissemination of the Project description is a prerequisite for that campaign.

The preliminary visual impact assessment for mine site and the Pointe-Noire infrastructure is nearly complete.

20.2.3 Project Description

The Project Description is the document that activates the various EA regimes. The GoC and the GoQ have agreed that a single Project Description covering all the components of the Project should be submitted. A draft of the Project Description was submitted to the CEA Agency, the MPMO and the NEB on March 18, 2013. Relatively minor comments were received from the CEA Agency and the NEB; the MPMO confirmed that it would not submit comments.

Subsequent to the receipt of the preceding comments, design changes to several components of the Project were adopted as a result of the review and value engineering exercises undertaken in mid 2013. The Project Description will be reviewed in the light of those changes and, if necessary, the revised draft of the Project Description will be submitted to the federal regulators for review.

20.2.4 Sustainability Assessment

NML/TS have prepared a tool to assess the sustainability of the Project.

The tool will be applied to the Project at various stages of its design, starting with the pre-feasibility study stage, followed by the feasibility study stage and then following the second and third rounds of the public information and consultation program. That approach will be one way of demonstrating to the concerned governments and the public how their input was taken into account. The results of the sustainability assessment will be included in the EIS. The sustainability tool will thereafter be applied to the Project periodically throughout the construction, operation and decommissioning/rehabilitation phases.

20.2.5 Environmental Impact Statement

The drafting of the EIS is underway. At this time, the biophysical data are nearly completed except for the future location of the discharge point for the excess treated water from the slurry transportation system. At a minimum, marine survey should be carried out when the location will be known.

The data collection (desktop study) is nearly completed. Data collection during the public consultations, once the project description will be tabled, will allow the completion of this part of the EIS.

Other sections of the EIS will be finalized once the EA guidelines will be issued by the governmental authorities.

20.2.6 Timetable

The timelines for issuing EA guidelines vary according to the regulatory agency. Given the 365-day timeline for completion of EAs under the CEA Act, and based on recent EAs, the CEA Agency's guidelines should be received within 60 days of the decision to require an EA (which would follow a regulatory 45-day period for the review of the Project Description) following receipt of a satisfactory Project Description.

The Government of Quebec ("**GoQ**") is expected to issue generic guidelines for the components south of the 55th parallel of latitude assessed within one (1) month of the tabling of the Project Description. Receipt of the guidelines for the components assessed under the JBNQA regime could take several months after the tabling of the Project Description.

Delays in receipt of the guidelines should not materially delay the completion of the EIS, since analysis of other recent guidelines for mining and pipeline projects permits confident predictions of the contents of the guidelines for the Project. The Project may require additional delays (8 to 12 months) in case the MMER is triggered.

20.2.7 Post-EIS Regulatory Timetable

The CEA Act provides a "government" timeline (i.e., the government "clock" is stopped when the government is awaiting information from the proponent) of 365 days for EAs (including the Minister's decision), unless there is a panel review, in which case the timeline is 24 months (also including the Minister's decision). Based on discussions with the CEA Agency in late 2011, it is not anticipated that the Project will be subject to a panel review, but the use of a panel review may prove to be the best solution to the challenge of harmonizing at least some of the several EA regimes. That should have the effect of shortening the NEB process from approximately 15 months to perhaps nine months or less.

The timeline for EAs under the GoQ regime of general application can be estimated to be roughly 15 months of "government" time following the tabling of an acceptable EIS. The timeline under the JBNQA regime is expected to be approximately 25 months of "government" time.

The following timetable for the EA releases assumes that the EIS is filed in August 2015:

- Receipt of questions from regulators (CEA Agency, MDDEFP/KEQC) on the EIS – Q4 2015;
- Submission of responses to questions on EIS – Q1 2016;
- Acceptance of EIS by regulators for purposes of public hearings – Q2 2016;
- Public hearings (BAPE, KEQC) – Q3-Q4 2016;
- Submission of responses to questions filed at the hearings – Q1 2017;
- Issuing of EA releases and conditions of release – Q2-Q3 2017 (taking into account the expected long delay for the KEQC);
- Completion of MMER process – Q3 2019.

However, this timeline does not consider the request for the MMER which should require an additional 12 to 18 months if not made in parallel.

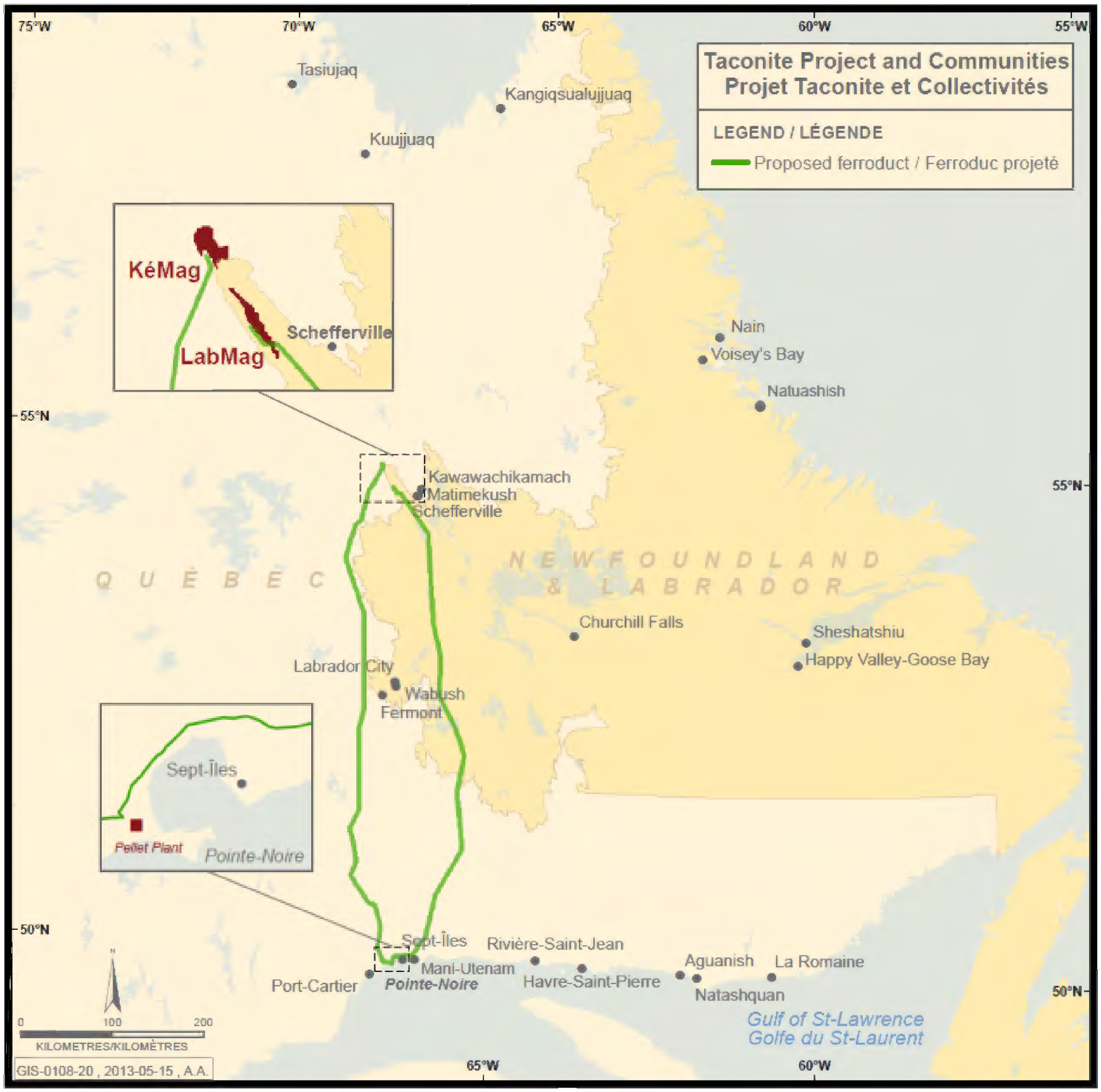
20.3 Social and Community

20.3.1 Consultation Process

Information/consultation tours are planned for each study area during the Project description and EIS preparation stage. The target communities (shown on Figure 20.1) have been grouped by local and regional study areas according to the components of the Project, as follows:

- Local Study Area 1 (the Schefferville-Menihek area, including the mine/concentrator and related facilities): Naskapi Nation of Kawawachikamach ("**NNK**"), Nation Innu Matimekush-Lake John ("**NIMLJ**"), Ville de Schefferville;
- Local Study Area 2 (the area between Schefferville and Sept-Îles, encompassing the potential slurry transportation system route): Innu Takuaikan Uashat mak Mani-Utenam ("**ITUM**"), Ville de Fermont, Labrador City/Wabush;
- Local Study Area 3 (the Sept-Îles area, including the pellet plant and related infrastructure): ITUM, Ville de Sept-Îles, Ville de Port-Cartier;
- Regional Study Areas (indirectly affected communities): Kuujjuaq, Churchill Falls, Innu Nation ("**IN**"), NunatuKavut Community Council ("**NCC**"), Happy Valley-Goose Bay/North West River.

Figure 20.1 – Project Target Communities



BCP was mandated to prepare a draft communications plan for the non-Native population of Sept-Îles. It presented its plan to NML/TS on January 10, 2013. A simplified version thereof for the first phase of the public information/consultation program has been prepared, that can be adapted for all of the concerned Aboriginal groups and non-Native populations. Currently, internal and public questions and answers are being prepared, the required visual materials are being identified and the text that will be used for the micro website for the public information/consultation program is being drafted.

In order to avoid misconceptions about the Project in the months preceding the tabling of the Project Description, meetings have been held with key individuals and organizations, mainly in the Sept-Îles area, to brief them about the Project and to reassure them that a full public information/consultation program will be conducted. That activity was designed to reduce the danger of public opposition to the Project and to expedite the attainment of social acceptability.

20.3.2 Social Acceptability

Following controversy with certain new development projects (excluding thus far the Project), the Ville de Sept-Îles commissioned the development of a tool to measure the degree of sustainability/social acceptability of new projects on its territory. The tool, developed by a Québec-based university research chair, was tested by the Ville de Sept-Îles in December 2012, but the results seem to indicate that it must be fine-tuned.

The social acceptability of the Project (at least in the Sept-Îles area) will be gauged by means of a specific tool to be developed by NML/TS taking into account the work described in the preceding paragraph and other relevant work. Open-mindedness, transparency, respect and trust are keys to gaining social acceptability. The public information and consultation program, a series of informal meetings with influential persons and organizations and the initiatives described below are some of the measures designed to enhance the efficient attainment of social acceptability.

a) Social Initiatives

From the outset, NML/TS have striven to create and maintain a harmonious relationship with all the Aboriginal and other groups that may be affected by their activities.

The most important achievements to date are:

- Signing four (4) IBAs and a CA for the DSOP;
- Preferential hiring and contracting on the DSOP for the members and corporations of the Aboriginal groups;
- A scholarship program for secondary students in seven (7) Aboriginal communities;
- Summer employment of Native students and training in geological exploration techniques;
- Numerous donations to support cultural and sporting activities, with a focus on youth;
- Specific assistance in upgrading sports facilities;
- Sponsoring a Native student training to become a nurse practitioner;
- Popular-language summaries of technical reports;

- Information and consultation sessions with the local governments and populations;
- An adapted workplace for the DSOP;
- A toll-free telephone number for questions and complaints;
- A quarterly newsletter in English, French and several Aboriginal languages;
- Advance notice of field activities;
- Visits to schools and participation in career fairs and similar events;
- Participation in and financial support for the Inuit Youth Mining Education Strategy;
- The creation of an Elders' Committee for the DSOP in collaboration with the NNK;
- Periodic flights to allow the Elders of the NNK and the NIMLJ to observe for themselves whether caribou are being disturbed by NML/TS's activities;
- Site-restoration pilot project with *Université Laval*;
- Support for a joint research project between the NNK and the Canadian Business Ethics Research Network.

Through initiatives such as the foregoing, NML/TS are committed to ensuring that their mining activities will drive the socio-economic regeneration of the entire area and to structuring the Project in such a way as to provide the concerned Aboriginal groups with the opportunity to continue to live sustainably in their traditional lands for the next four (4) or five (5) generations and to prepare themselves for the decades that will follow in ways that they determine for themselves.

Also the sustainability assessment exercise to be included in the EIS should demonstrate the quality of the Project to the stakeholders.

20.3.3 Memoranda of Understanding and Collaboration Agreements

Based on land claims and currently available land use data, Table 20.1 lists the potentially directly impacted Aboriginal groups for each component of the Project. Table 20.2 summarizes the status of the relevant land-claims settlements or negotiations.

The Aboriginal groups potentially affected by the Project are: Kuujjuaq Inuit, NIMLJ, NNK and ITUM.

Table 20.1 – Potentially Impacted[1] Aboriginal Groups

Component	Aboriginal Groups
Mine/concentrator complex and access road	Naskapi Nation of Kawawachikamach Nation Innu Matimekush-Lake John Innu Takuaikan Uashat mak Mani-Utenam, Kuujjuaq Inuit
Slurry transportation system	Naskapi Nation of Kawawachikamach Nation Innu Matimekush-Lake John Innu Takuaikan Uashat mak Mani-Utenam
Pellet plant	Innu Takuaikan Uashat mak Mani-Utenam

1 Refers to treaty or asserted rights, primarily those related to lands and resources that might be directly affected.

Table 20.2 – Status of Relevant Land-Claims Settlements/Negotiations

Aboriginal Group	Status of Land-Claims Settlements/Negotiations
Naskapi Nation of Kawawachikamach	Comprehensive land claim agreement – Northeastern Québec Agreement – signed in 1978 resolved claims in and to Québec. Naskapis assert rights in and to parts of Labrador, but claim not yet accepted by GoC. In 2009, the NNK and the GoQ signed the Naskapi-Québec Partnership Agreement. It is contractually binding, but it does not have the status of a treaty. The agreement provides that the GoQ will encourage the proponents of mining projects to enter into agreements with the Naskapis.
Nation Innu Matimekush-Lake John and Innu Takuaikan Uashat mak Mani-Utenam	Matimekush-Lake John and Uashat mak Mani-Utenam Innu assert rights in and to parts of Québec, including parts of JBNQA "Territory". Claim accepted by GoC in 1979 and by GoQ in 1980. A Framework Agreement was signed in 1988. Negotiations were halted in 1994, after the dissolution of the Atikamekw-Montagnais Council. In 2005, the Innu of Uashat mak Mani-Utenam and Matimekush-Lake John created Ashuanipi Corporation to resume land claim negotiations. Formal negotiations between Ashuanipi Corp. and the governments have been on hold since 2008. Matimekush-Lake John and Uashat mak Mani-Utenam Innu are part of the Innu Strategic Alliance (a political grouping that also includes Ekuanitshit, Pessamit and Unamen Shipu).
Inuit of Nunavik [2]	Comprehensive land claim agreement – James Bay and Northern Québec Agreement – signed in 1975 resolved Québec claim. In 2002, the Nunavik Inuit and the GoQ signed the Sanarrutik Agreement. It is contractually binding, but it does not have the status of a treaty. The Sanarrutik Agreement provides that the GoQ will encourage the proponents of mining projects to enter into agreements with the Inuit.

2 Not deemed to be directly affected as defined in Table 20.1, but will be consulted because of potential for indirect effects, role in JBNQA Section 23 environmental and social impact assessment regime, administrative role of the Kativik Regional Government and requirements of the Sanarrutik Agreement.

NML/TS wants to avoid a situation in which any of the Aboriginal groups reacts negatively when the Project is officially announced in order to trigger the EA processes. It is proposed, therefore, to enter into a Memorandum of Understanding ("**MOU**") with each of the Aboriginal groups that may be affected by the Project.

The MOUs would serve to reassure the Aboriginal groups that NML would try to negotiate a Collaboration Agreement ("**CA**") with the groups that will actually be affected, the identity of which will be known only once the investment decision has been taken.

Ideally, the MOUs should be signed before the Project Description is made public, since they are intended to eliminate the danger of complaints from the Aboriginal groups that they are not being informed/consulted/accommodated. If that is not possible, the process should at least be initiated by then.

20.4 Anticipated Impacts

20.4.1 Positive Impacts

The major foreseeable positive impacts of the Project include the following:

- Positive contribution to the labour market through the hiring of employees and the awarding of contracts for construction and operations;
- Positive contribution to the GDP of the various governments;
- Positive contribution to government revenues through royalties and taxes;
- Indirect and induced economic impacts locally, regionally and nationally;
- Providing contracts, training and jobs to the local and regional Aboriginal groups;
- Numerous, varied and important benefits through the CAs to their Aboriginal signatories.

20.4.2 Negative Impacts

The major foreseeable negative impacts of the Project include the following:

- The draining of several lakes and the infilling of others for tailings impoundment areas and waste rock piles at the mine site, which will result in the loss of fish and fish habitat
- The potential for metal leaching and acid rock drainage from some of the waste rock;
- The transformation of sizeable areas of land, which entails impacts on the natural, social and visual environments;
- Reduced enjoyment of hunting, fishing, trapping and gathering;
- The crossing of numerous watercourses and water bodies for the construction of the slurry transportation system. The vast majority of the watercourses and water

bodies that will be crossed can, however, be considered to be minor from a fisheries perspective;

- The intrusion of the slurry transportation system into certain protected areas (either projected or existing);
- The potential disturbance of protected species (e.g., Rusty blackbird);
- The public's perception of the inter-basin transfer of a large volume of water through the slurry transportation system and the annual discharge of $\approx$12 Mm^3 of water treated to the required norms into the Saint Lawrence River;
- The perception of the Project's contribution to cumulative impacts, particularly in the Sept-Îles area (e.g., water quality; atmospheric quality; visual environment; increased cost of living) and in the Schefferville-Menihek area (interference with traditional activities; increased cost of living);
- The presence of a large work force in both the Schefferville and Sept-Îles regions that is predominantly male and from the outside;
- Increased pressure on the infrastructure and services in the Schefferville and Sept-Îles regions;
- The emission of almost 900,000 tCO_2 eq/y from the pellet plant.

It is anticipated that all the afore-mentioned negative impacts can be mitigated to a satisfactory degree. Doing so will require the implementation of large-scale and long-term mitigation, compensation, monitoring, follow-up and adaptive management measures. They will also be offset to varying degrees by the positive impacts listed in previous section.

20.5 Closure Plan

NOTE: A closure plan will be developed in the next phase of the KéMag Project. It will be similar to LabMag Closure Plan described below.

Vegetation and topsoil will be cleared prior to the start of mining by a mining contractor using a fleet of dozers, small excavators and articulated haul trucks ahead of the mining operation. Suitable organic material will be stockpiled for future reclamation use. Overburden will then be stripped using a fleet of excavators and hauled to the waste dump by trucks.

The objective of the rehabilitation work is to restore the site to a satisfactory condition by:

- Eliminating unacceptable health hazards and ensuring public safety;
- Limiting the production and emission of substances harmful to the receiving environment, and in the long term, aiming at eliminating the need for maintenance and monitoring;

- Restoring the site to a condition in which it is visually acceptable to the community; and
- Reclaiming the areas where infrastructures are located (excluding waste dumps, stockpile, and tailings impoundment areas) for future use.

All areas affected by mining activities (building sites, tailings pond, waste rock pile) must be revegetated to control erosion and restore the site to its natural condition.

Closure planning should be integrated into the early stage of the engineering of the Project. And then, as much as possible, closure and rehabilitation measures should be carried out at every phase of the mine life so as to reduce the mine's footprint and allow for rapid re-establishment of biodiversity.

20.5.1 Revegetation

The principal concerns expressed by the Aboriginal participants included the need to restore and rehabilitate the mining sites, the loss of habitats for traditional activities, the impact of mining activities on caribou and caribou hunting, the loss of fish habitats, and the general access to the territory to carry out traditional activities. Populations living around the Project area or using the land for traditional purposes may be affected positively or negatively. For example, mining activities may affect the presence and quality of natural resources, which may impose changes in land-use and resource-use patterns.

Progressive rehabilitation measures should be implemented during construction and operation phases by:

- Re-vegetation of affected lands no longer required for future operation; and
- Rehabilitation of potentially contaminated lands in areas no longer required for a production or an active support function.

A specific issue for progressive rehabilitation was identified for the mine site:

- The tailings storage areas will be divided in several cells during the exploitation. At the time one of the cells reaches full capacity, the reclamation works should be carried out on the cell as described below. The progressive implementation of the cover on the tailings surface will limit the erosion of the fine tailings at the surface of the storage areas.

The Project is at the feasibility stage; however, all works will be designed to meet the geotechnical stability requirements from the Canadian Dam Association (2007). The waste rock pile will be designed so as to promote re-vegetation at closure, i.e. with berms and a gentler slope than the natural slope on a rock pile. For the first seven (7) years of the mine plan, all the waste rock is planned to be placed outside of the mined out pit. It was assumed that 50% of the Menihek Shale will be non acid generating and therefore will be placed in the mined out pit from Year-8 and onwards. The rehabilitation of the waste dump will include the unloading of topsoil/overburden from the top of the slopes. The material should distribute itself by gravity and the surface will be seeded.

The low grade ore is expected to be processed at the end of the mining activities. It is assumed that no material will remain at closure. A layer of top soil or overburden will be placed on the leveled surface that will then be vegetated.

20.5.2 Soil Decontamination

Mining activities can potentially cause contamination of underlying soils and adjacent to the mining site. Upon termination of mining activities, the soil surrounding risk areas (storage areas for petroleum products, spills any residue along the line residuals, any ore concentrate along the pipeline, spill, etc.) will be characterized. If this characterization reveals the presence of contaminants in concentrations exceeding regulatory values, NML will take the necessary measures to manage the contaminated soils in accordance with the regulation.

At closure, soils will be characterized at each site and should contaminants be found, the soils will be managed as per applicable regulations. This characterization concerns soils but also groundwater and surface waters in areas that are likely to have been contaminated by human activities, especially the handling of petroleum products. If results of the study show that parameters exceed the criteria set in the regulation, a rehabilitation plan specifying the environmental protection measures to be undertaken will be submitted to the Authorities.

20.5.3 Dismantling of Buildings and Infrastructures

A significant amount of waste material will be generated by the demolition of the existing buildings on the site. These materials must be managed responsibly in order to minimize their impact on the environment and on workers' health with regards to the provincial requirements.

Progressive rehabilitation measures should be implemented during construction and operation phases by dismantling of buildings and infrastructures no longer in use.

All buildings (at mine site as well as Pointe-Noire) should be cleaned and demolished or dismantled to ground level. Whenever possible, demolition materials will be sold. The walls will be razed or dismantled to their foundations. Floor slabs and foundations that are contaminated by hydrocarbons will initially be decontaminated before being broken. The area of the plant will be leveled in order to reestablish the natural drainage. All the above ground concrete slabs should be removed. The concrete foundations should be broken and removed to a depth of one (1) metre and the remaining foundations should be covered with overburden material and revegetated.

All process vessels, pipelines and equipment should be drained at the end of operations. The equipment will be cleaned and dismantled to be sold or recycled off-site. Chemicals and reagents should be picked up and adequately handled/stored in accordance with regulation.

The main access road from Schefferville to site and main access road at Pointe-Noire will remain. Most of the haul roads and services roads at Pointe-Noire built by NML/TS will no longer be required and will be scarified to facilitate revegetation. Some portions of the haul roads will be required for maintenance works and post-closure monitoring program. Service roads, helipad and main access roads will be scarified and revegetated. Culverts and unnecessary pipes will be removed in order to reestablish the natural flow of water.

The electrical infrastructures (e.g. high voltage transmission line), electrical substation and transformers, and emergency power plant) will remain on site as long as required for the water treatment plant supply. However, most of the mine site and Pointe-Noire site transmission lines should be dismantled if it is of no use to other parties.

The support infrastructure (e.g. potable water supply system, sewage collection system and treatment plant, compressed air plant and distribution, boiler houses, diesel fuel tank farm, additive storage stockpiles, concentrate stockpile, pellet stockpile, double conveyor to port and product silos) should be dismantled if it is of no use to other parties. The solid waste disposal facility will be covered with 0.5 m of overburden and revegetated.

The demolition of buildings and infrastructure will generate a large amount of waste materials. In order to minimize the quantity of non-hazardous demolition waste that will be disposed of into a landfill. A Waste Material Management Plan will be prepared prior to undertaking demolition works. The Plan will strongly emphasize the application of 3RV-E principles (reducing, reusing, recycling, valorizing and eliminating) whenever feasible and applicable.

Despite a large amount of materials to be recovered or reutilized, the dismantling of buildings and infrastructures will require the elimination of a large amount of non hazardous debris of all kinds. The most responsible solution in the context of sustainable development is to have these materials landfilled locally since the transportation of a large amount of materials will produce an important quantity of greenhouse gas emissions that will largely cancel all the benefits that would otherwise be obtained by the recuperation/recycling of the waste materials. Given the isolated geographic setting of the site, local landfilling is by far the best alternative from the sustainable development standpoint.

All hazardous waste should be managed off-site in accordance with applicable laws and regulations.

20.5.4 Waste Rock and Tailings Pile

A preliminary waste rock characterization program has been undertaken between 2008 and 2012 in order to study the metal leaching and acid generation potentials of these materials that will be produced and stored during the mining operations. The program included, amongst others, static modified acid base accounting tests, shake flask extraction and kinetic humidity cell tests[1]. Some of the waste rock samples showed a potential for acid generation and metal leaching. However the representativeness of these samples is questioned since the average sulfur content for the different geological units is significantly higher compared to the value measured during a previous geochemical campaign. In this stage of the Project, for design purpose, it is considered that half of the volume of waste rock that will be produced will be acid generating and metal leaching.

One (1) sample of process tailings produced from metallurgical tests was tested in a humidity cell. The results showed that it was non acid generating and the potential for metal leaching was uncertain. In this stage of the Project, for design purpose, it is considered that process tailings will be non acid generating and non metal leaching.

A thorough environmental characterization of the tailings and the waste rock should be carried out to determine whether or not the accumulation areas are likely to produce acid rock drainage ("**ARD**") or leach metals.

All pipelines related to the TIAs and water pump stations will be demolished or dismantled and the materials should be sold, whenever possible, or managed off-site. Run-off water will be managed by the implementation of a network of ditches at the surface of the TIAs and of a spillway located on the retention dam.

For the purpose of this closure and rehabilitation plan, all tailings are considered to be non acid generating and non leachable. However, a detailed characterization of the tailings that will be deposited in the tailings impoundment areas and the waste rock that will be extracted from the pits, is necessary to adequately plan the rehabilitation measures for the mine waste management facilities. This characterization should include physical (grain size, permeability, etc.) and geochemical properties (acid generation potential, metal leaching potential, etc.). In the next engineering phases, a tailings impoundment management strategy should be developed. Moreover, the water treatment requirements at closure should be re-evaluated once additional data on mine waters and mine waste geochemistry become available.

At the surface of the TIAs, as previously mentioned, a network of ditches will be built to drain water downstream of the infrastructure and to limit the cover erosion. A spillway will be built on the retention dam for each TIA.

The run-off from the waste dump will be treated as long as necessary. The collection pond, the ARD water treatment plant, and the polishing pond will be kept in place as

[1] 2 SENES Consultants Inc., 2012. ML/ARD Assessment for LabMag Project, July 2012.

long as the water quality does not meet the applicable regulatory criteria for direct discharge to the environment. The pipeline from ARD treatment plant to process plant will be demolished or dismantled. After closure, treated water will be directed to the pit.

20.5.5 Open Pit

At the moment, no specific future land use has been identified by NML. The local community has not expressed any special demands. Closure and rehabilitation measures should allow the local community to access freely the mine site as it was before beginning of mining operations. It is planned that at closure all fences will be removed once the pit filled with water. Future potential uses of the newly formed lake may be identified in collaboration with authorities, environmental groups and research centers. This will, amongst other parameters, depend on the water quality.

Once mining activities have ceased, dewatering operations at the open pits are expected to stop. Consequently, the pit will gradually fill up with water until equilibrium has been reached. The overburden slope around the pit should be designed safely from the beginning in order to avoid any risks to workers and to promote the integration of the pit in the landscape.

To secure the access to the pit during the filling phase, the mine pit perimeter dyke will remain in place to limit the access to the pit during this phase.

Warning signboards should be placed at regular intervals. Once pit filling is completed and if the water quality meets the required criteria, the perimeter dyke will be breached at three (3) or four (4) locations to reestablish the original water flow pattern.

20.5.6 Mining Effluent

Regarding the mine dewatering, the in-pit sump pump will be removed and the pipelines will be dismantled. The pit perimeter ditch will be backfilled. The mine pit perimeter dyke will remain in place as long as the pit water level and quality are not stabilized as previously mentioned. Subsequently, the dam will be breached in several places so as to reestablish the original water flow pattern.

20.5.7 Other Installations

Mobile and fixed equipment as well as heavy machinery will be sold or recycled (trucks of ± 100 tonnes, front loader, small vehicles, compressors, fans, generators, pumps, etc.).

The raw water supply pumping station and pipeline from Elross Lake will be dismantled at closure. The potable water supply system will also be dismantled.

To minimize and reduce non-hazardous demolition waste to be landfilled, waste should be managed according to 3RV-E principles (reducing, reusing, recycling, valorizing and eliminating). All hazardous waste should be managed off-site and accordance with existing laws and regulations.

Despite a large amount of materials to be recovered, the dismantling of buildings and infrastructures will require the elimination of a large amount of debris of all kinds. The most responsible solution in the context of sustainable development is to have these materials disposed of in authorized sites or sorting centers that will maximize their revalorization. To this end, at closure, a quote for the demolition including a materials management plan for decommissioning should be written. This will guide the contractor to perform the work.

At closure, petroleum wastes still remaining on site will be comprised of used engine and hydraulic oil plus rear end/differential gear lubricants from fleet equipment, oil from various site generators, used degreasing solvents, and contaminated or expired diesel. Petroleum waste will be managed as per applicable regulations.

The diesel fuel tanks and their surface and groundwater pipes will be removed in accordance with applicable regulations. These tanks will be sold, stored for future use or disposal, while ensuring compliance with applicable regulations. The geomembrane protection, piping and non-refillable containers will be disposed of in accordance with applicable regulations.

Most of the concentrate slurry transportation system is planned to be underground and therefore, will be left in place at closure. Aboveground sections of the system will be sectioned and removed. All the system openings will be sealed with concrete. The intermediate pumping station and the monitoring stations will be dismantled and the equipment and materials will be sold, whenever possible, or managed off-site. The high voltage transmission line will be dismantled if it is of no use to other parties and the service roads and helipad will be scarified and revegetated.

20.5.8 Emergency Plan and Monitoring Program

During the mine site operation, a monitoring program will be implemented with some instrumentation (e.g. groundwater monitoring wells, surface water monitoring stations, etc.). This system will be used to continue the environmental monitoring of the site after its closure and rehabilitation.

After closure, the components that will remain on site are the two (2) tailings storage areas, the waste rock pile and the pit. The stability of the tailings dams, the waste rock pile and of the pit walls will be monitored and signs of erosion, settlement and/or land slide will be noted. Corrective measures will be implemented if required. This surveillance will be conducted once a year for a minimum of five (5) years following the closure of the mine.

The environmental monitoring program will be implemented to ensure the efficiency of the rehabilitation measures. The quality of surface water and groundwater will be monitored in areas where infrastructures will remain after site rehabilitation. The duration of the post closure monitoring will depend on the geochemical properties of the accumulated materials and the results of the water quality monitoring during the operation.

An agronomic monitoring will be performed to assess the efficiency of the re-vegetation done as part of the rehabilitation works during operations and after mining activities have ceased. Monitoring will be conducted annually for three (3) years following the implementation of vegetation as part of progressive rehabilitation. At closure, the rehabilitation plan will be implemented and a significant part of the site will be re-vegetated. Re-vegetation success will be monitored for five (5) years at the end of the rehabilitation works. Re-seeding will be carried out, when necessary, in areas where vegetation growth is deemed unsatisfactory.

21.0 CAPITAL AND OPERATING COSTS

21.1 Capital Costs

The capital cost estimate for the Feasibility Study was assembled by ISLLP. Parts of the capital costs were developed by ISLLP and other areas were provided by consultants under the supervision of ISLLP (i.e. mining costs by Met-Chem and slurry transportation system by BRASS). The filtration and pelletizing plants prepared by Outotec were provided to NML/TS as a full package and added as such to the estimate. NML/TS provided ISLLP with the Owner's cost estimates. Details of the various costs are shown further in this Section.

The Project is designed to produce 23 Mt/y of product, namely 12 Mt/y of BF fluxed pellets with 2.5 % SiO_2, five (5) Mt/y of DR pellets with 1.5 % SiO_2 and six (6) Mt/y of concentrate with 2.1 % SiO_2.

The Project includes the mine, process plant, a slurry transportation system and a pellet plant and associated storage and material handling system at a common port in Pointe-Noire. All are located in Quebec.

The estimate has been developed using approved design criteria, flow sheets, engineering drawings and equipment lists. Enquiries were sent to suppliers for pricing of major equipment.

The capital cost estimate has been developed on the basis of one (1) overall EPCM contractor who will provide the design, procurement and construction activities for the Project. All sub-contracts would be managed by the EPCM contractor.

This capital cost estimate qualifies as an American Association of Cost Engineers ("**AACE**") Class III estimate. The intended accuracy of this Feasibility Study estimate is ± 15 %. Although some individual elements of the estimate may not achieve the target level of accuracy, the sum of all estimate elements falls within the parameters of intended accuracy.

Table 21.1 shows the organization and division of responsibilities at the major area level with respect to Engineering, Material Take-offs ("**MTO**") and Estimating.

Table 21.2 shows a high level summary of the estimated direct and indirect capital costs.

Table 21.3 shown the third party infrastructure excluded from the capital cost and accounted by an annuity in the financial analysis.

Table 21.1 - Responsibility Matrix at the Major Area Level

Major Area	Responsibility Entity		
	Engineering	MTO's	Estimating
Direct Costs			
Mining Area	Met-Chem	Met-Chem	Met-Chem
Process Plant	ISLLP	ISLLP	ISLLP
Tailings Disposal	ISLLP	ISLLP	ISLLP
Slurry Transportation System	BRASS/ ISLLP	ISLLP	ISLLP
Pelletizing Plant	Outotec	Outotec	Outotec
Material Handling	ISLLP	ISLLP	ISLLP
Infrastructure	ISLLP	ISLLP	ISLLP
Indirect Costs			
EPCM Services	ISLLP	N/A	ISLLP
Construction Field Indirects	ISLLP	N/A	ISLLP
Owner's Costs	N/A	N/A	NML/TS
Contingency	N/A	N/A	ISLLP
Escalation	Excluded	Excluded	Excluded
Risks	NML/TS/ISLLP	NML/TS/ISLLP	NML/TS/ISLLP

Table 21.2 – Capital Costs Estimate Summary

Area	Title	Total Costs ($'000)
1000	Mine Area	106,128
2000	ROM Crushing, Storage and Reclaim	382,938
3000	Concentration, Grinding, Separation & Upgrade	1,006,560
4000	Tailings Disposal and Water Management	23,022
5000	Concentrate Slurry Transport	237,845
5210	Slurry Transport	1,539,965
6000	Port Area (Sept-Îles)	332,376
6200	Pellet Plant (Outotec LSTK)	1,226,084
8100	Infrastructure and Utilities (Schefferville)	531,508
8200	Infrastructure and Utilities (Sept-Îles)	144,359
	Total Direct Cost	**5,740,784**
9000	Indirects Costs	1,421,000
9000	Contingency	867,000
	Total Indirect Cost	**2,288,000**
	Total Direct + Indirect	**8,028,784**
	Owners Cost (by NML)	210,000
	Total Project Capex	**8,239,000**
	EXCLUSIONS FROM Capex	
	Mine Mobile Equipment in Opex	323,000
	Hydro-Quebec Transmission Line in Opex	661,000
	Risk	334,300
	Escalation	Excluded

The above total sums are rounded up to $1,000.

Table 21.3 – Capital Cost vs Leasing

Area	Title	Total Costs ($'000)
	Capex – Mine and Process[1]	5,227,000
	Capex – Infrastructure[2]	3,012,000
	Total Project Capex	**8,239,000**

Costs of major mining equipment and power transmission line are not included in Capex, but servicing costs are in cash cost.
2 Consists of slurry transportation system, product storage and reclaiming system are to be financed on the basis of long term debt.

21.1.1 Basis of Estimate

The capital cost estimate covers the facilities included in the scope of work described in the previous Sections.

The estimated capital cost is based on the following key assumptions:

- The estimate is expressed in February 2013 Canadian dollars since the majority of the bids were received during that period;
- The proposed construction work week is on the basis of a 60-hour with rotations of 21 days in followed by seven (7) days out [travel during the seven (7) days out];
- Fluctuations to nominated currency exchange rates are excluded;
- Allowance for industrial dispute or lost time arising from industrial actions is excluded;
- Environmental permitting is excluded;
- All taxes and duties are excluded;
- Escalation is excluded; (some included see paragraph below);
- Project financing and interest during construction is excluded;
- No allowance is made for acceleration or deceleration of the Project schedule;
- Plant operating costs are excluded from the capital cost;
- Project insurance is included in Owner's cost.

The Project schedule is presented in Section 24.1 and the Capex associated with the schedule is based on an advance period whereby the design concepts are frozen and basic engineering commences. The timing for this start is in the late summer following approval of the EIS and obtaining initial permits. The work would then continue through its life-cycle until the end of construction and commissioning.

21.1.2 Currency

The estimate base currency is Canadian dollars. The capital cost estimate consists of items quoted in various foreign currencies which have been converted into Canadian dollars using exchange rates as of February 5, 2013. The vast majority of pricing for equipment and bulk materials are based on Canadian dollars. Table 21.4 shows the currency exchange rates and the percentages content in different currencies and the percent content of costs for each of the listed currencies, excluding the pellet plant (by Outotec).

Table 21.4 – Currency Exchange Rates and Percent Content

Currency Code	Currency Name	Canada	Percent Content (%)
CAD	Canadian Dollar	1.00	85.6
USD	US Dollar	1.00	8.1
EUR	Euro	1.30	6.3

21.1.3 Cost Basis and Methodology

a) Labour Rates

The labour crew mix wage rates was developed on the basis of the Quebec Labour collective agreement for April 2013. Two (2) sets of labour crew rates have been developed to account for work performed north and south of the 55^{th} parallel.

Labour hourly wage rates for construction work was developed as labour crew mix wage rates and include base salaries, casual overtime, fringe benefits, equipment rental rates (with operations and maintenance), tools, consumables, mobilization/demobilization, temporary services, temporary installations, all applicable premiums (including those for safety and security), union delegates, PPE, site supervision and administration as well as sub-contractor's overhead and profit.

Table 21.5 outlines typical hourly rates for construction labour crews working north of the 55^{th} parallel for KéMag. Labour rates were also established for working in the slurry transportation system and port areas which are south of the 55^{th} parallel.

Transportation cost for workers from their point of origin to site was evaluated and the costs carried under construction field indirect. The same applies to room & board (camp and catering) as well as to all other necessary accommodations.

Table 21.5 – Construction Labour Crews

Crew Code	Description	Crew Rates North of 55th Parallel ($/h)
41A	Civil, light	$169
41B	Civil, local/structural	$121
41C	Rock excavation	$247
41E	Civil, heavy	$237
42A	Concrete	$130
43A	Structural Steel	$166
44A	Architectural, siding and roofing	$159
44B	Architectural, interior and finish	$122
45A	Mechanical, heavy	$147
45C	Field erected tanks	$157
46A	Piping	$143
46B	Piping, light	$130
46I	Insulation	$121
47A	Electrical work	$129
48A	Automation work	$124

b) Labour Manhours and Productivity

The estimate base man-hours are based on typical craft performance for the northeast and Quebec North industrial productivity from other similar projects. Direct field labour is the skilled and unskilled labour required to install the permanent plant equipment and bulk materials at the Project site. Direct field installation man-hours have been developed using estimated unit man-hours for each commodity multiplied by the quantity.

Installation man-hours are based on ISLLP historical data adjusted for specific site conditions at Schefferville and Sept-Îles, namely extended overtime, weather and schedule.

The Productivity Loss Factors were established from the Global Construction Cost and Reference Yearbook and the AACE.

The Gulf-Coast factor of one (1) is based on working an eight-hour per day (8 h/d) in daylight hours in moderate temperatures based on the average productivity from 30 major cities located in North America (Gulf-Coast projects are used as model). Location factors typically reflect disparities in construction hourly rate productivity. The productivity data used were developed over an extended period so as not to be influenced by abnormal variations and reflects a typical average. Labour productivity and site considerations for the Project are listed below in Table 21.6.

Table 21.6 – Labour Productivity Factors

Labour Productivity	Factors
Civil Earthwork	1,2
Concrete	1,5
Structural Steel	1,5
Architectural	1,5
Mechanical (Average)	1,4
Piping	1,7
Electrical	1,7
Instrumentation / Automation	1,6
Average	1,5

c) Unit of Measure

The International System of Units ("**SI**") has been used as the unit of measure for this Project, except for the nominal pipe diameter which is defined in inches.

d) Pricing and Quantity Development

All quantities generated for the estimate are mainly based on MTO and deliverables which exclude contingencies of any kind.

e) Civil Work

Quantities for mass earthwork including excavation, backfill, rock excavation, structural backfill, site development, U/G drainage, dams and dykes, and permanent roads are provided by engineering and are based on the latest detailed layouts. No piling is needed at the mine since the topography of site is rock and also at the port as per the latest study.

The aggregate price at the mine is calculated based on crushing of the overburden and the rock excavated during the site and buildings preparation. The quantities of till are assumed to be available on site, no screening or treatment is required.

f) Concrete

The concrete quantities are provided by engineering; for the concentrator and flotation buildings are based on the mechanical 3D model design, and for non-process buildings the quantities are based on engineering calculations, layout and sketches. MTO templates covering all potential concrete activities consist of a table listing a series of CRC's (construction site activities) as a function of the WBS (physical location).

Prices for the supply and installation of concrete are based on similar projects within the same region.

g) Structural Steel

The structural steel quantities are provided by engineering; for the concentrator and flotation buildings they are based on the mechanical 3D model design, and for non-process buildings the quantities are based on engineering calculations, layout and sketches. MTO templates covering light, medium and heavy steel are used to issue quantities.

Historical data and information from the most recent projects are used for the pricing of structural steel for non-process buildings and quotes received by email from Chinese suppliers were used elsewhere. Chinese steel is being utilized for the major process buildings at the mine site and for the concentrate storage building at the port.

h) Architectural and Building Services

Architectural quantities for roofing, cladding, equipment doors, etc. are provided by engineering based on the latest layouts and elevation drawings. MTO templates covering all potential architectural activities are used to issue quantities.

Historical data from most recent projects are used for the pricing for cladding, roofing and all other architectural items.

i) Mechanical

The mechanical equipment are identified as per the latest mechanical equipment list which provides capacities and dimensions for all major and most minor equipment. All equipment is tagged according to the latest Process Flow Diagrams ("**PFD**"). The estimating group verified that the majority of equipment in the equipment list matches the PFD. The quantities of bins, hoppers and chutes are provided by an engineering MTO. The pump boxes capacities and sizes are provided by engineering and verified by estimating group according to the latest mass flow sheet.

Quantities for HVAC, fire protection and building utilities are estimated on square metre basis for buildings.

Budgetary quotes were received for most of the major and minor equipment packages. Technical and commercial evaluations for all the packages were completed and provided by engineering and procurement. Equipment pricing for the pellet plant is based on the quote provided by Outotec to NML/TS.

j) Piping

The piping quantities for the process area are provided by engineering based on a 3D model; moreover, most of the piping quantities are based on P&ID and on preliminary engineering criteria and basic pipe routing sketches.

ISLLP engineering has developed the quantities for the civil works for the main slurry transportation system, by assuming two (2) cross sections (rock and no rock assumption) based on quantities issued from a Terrestrial Ecosystem Model ("**TEM**"). The results were compiled based on Groupe Hemispheres' identification of ground types that pose particular engineering constraints and opportunities. This information was validated from similar studies for a slurry transportation system within the same area. However, the hydraulic design of the system is based on technical recommendations from BRASS.

Historical data and information from most recent projects were used for the pricing of relevant piping works.

k) Electrical

The Capex is based on the estimated cost the system at one (1) m depth below surface which, with the proposed back-up arrangements, provides a low risk option. The additional Capex for the system with an average burial depth at three (3) m (no risk option) is provided as a component of the overall Risk Assessment – Electrical.

Electrical equipment and wiring quantities are based on engineering deliverables such as equipment list, single-line diagrams, motor and cable lists.

Electrical duct-banks, cable trays, conduit and wire and cable for 34.5 kV, 4.16 kV and 600 V distribution systems are estimated based on historical data. Lighting and receptacles are calculated based on quantity requirements for the major buildings. Generators, GIS, and transformers pricing are based on latest budgetary bids received.

The capital costs for the HV transmission lines from HQ to the mine, the intermediate pumping station and port are assumed to be paid through the per kilowatt hour used and therefore included as an operating cost. Should this assumption fail to materialize, the Capex would increase by $ 661 M. The main incoming sub-station is included in the Capex.

l) Instrumentation and Controls

Instrumentation quantity and pricing are based on a percentage of mechanical equipment cost by area.

Only the cost for the Process Control System is based on budgetary quotes. The cost of telecommunication systems are estimated using historical data.

21.1.4 Indirect Costs

a) Construction Field Indirect Costs

The construction field indirect costs include all temporary roads (only on the Project sites), fencing and facilities, lay down areas, material handling and warehousing, temporary power, construction services (surveying, security, medical, scaffolding, janitorial, concrete testing, craft training, etc), construction vehicles, consumables, on-site toilets, medical check-up, passes, IDs, security and allowances for construction and safety signs, plant sheds, barricades, guard rails, etc.

b) Construction Camp

Based on the total hours estimated for the Project and the planned schedule, the size of construction camp is presently sized to 1,700 rooms (taking into consideration that the extra 400 beds will be part of the permanent accommodation complex at mine) including all required sub-facilities as catering, recreational rooms and others amenities.

At Sept-Îles, the camp requirement is estimated at 570 rooms excluding the camp for the Pellet plant which is estimated to require 1,000 direct workers at the peak of construction.

For the slurry transport system, three to four (3-4) camps would be required to support the construction activities totaling about 1,600 rooms.

Budgetary pricing for Camp supply and catering has been received for both mine and Sept-Îles and has been incorporated in Capex.

c) Freight

Freight costs are included in the indirect cost accounts. In general terms, all bulk material pricing is inclusive of freight to the site. Equipment freight cost from origin to Sept-Îles port has been evaluated at five (5) % of total equipment cost, excluding the pellet plant which is delivered only to Sept-Îles.

In-land transportation to bring equipment and material from Sept-Îles to the mine site using rail and trucks is calculated from budgetary pricing received for rail and truck transportation.

d) Vendor Representatives

Vendor support for equipment erection and pre-commissioning is estimated as two (2) % of total equipment cost excluding the pellet plant part, which is included in the total cost of the pellet plant.

e) Spare Parts

Commissioning and operational spare parts for two (2) years is estimated as three (3) % of total equipment cost excluding the pellet plant, which is included in the total cost of the pellet plant.

f) First Fills

An allowance of two (2) % of total equipment costs has been included for first fills excluding the pellet plant part, which is included in the total cost of the pellet plant.

g) EPCM Services

EPCM services have been estimated based on engineering deliverables and a breakdown of manpower per area as per the construction schedule to supervise and control works up-to and including pre-operational verification and assistance during commissioning.

h) Contingency

Contingency is an integral part of the estimate and can be best described as an allowance for undefined items or costs elements that will be incurred and will be spent, within the defined Project scope, but that cannot be explicitly foreseen due to a lack of detailed or accurate information.

Contingency has been evaluated by using both deterministic and probabilistic approaches.

ISLLP's recommendation for contingency at the feasibility study level is P_{80} which yields contingency levels of 12.1 %. The Monte Carlo simulation results assign a suggested contingency amount against the desired probability of underrun at P_{80}.

i) Owner's Costs

Owner's Costs was provided by NML/TS for inclusion in the estimate. ISLLP has incorporated these costs directly into the Capex. The costs include the following:

- Cost of royalty or license fees;
- Owner's team;
- Environmental monitoring, water analysis etc.;
- Public relations;
- Labour relations (Union);
- Owner's travel, legal and other corporate office charges to the Project;
- Environmental permits/government approvals;
- Owner's consultants (legal, environmental etc.);

- Commissioning and start-up (Note: cost of construction workers have not been considered);
- Operations training;
- Project insurance including comprehensive general liability and insurance for construction equipment and tools, builder's all risk insurance;
- Allowance for upgrade of any offsite facilities;
- Removal and disposal of hazardous materials.

j) Risk

Three (3) risk sessions were conducted to identify potential Project risks (threats and opportunities) and captured in a risk register using the risk software Stature. Each risk was assigned a level of consequence, probability of occurrence and manageability for which a probable outcome value was derived. Mitigation actions were assigned to each risk item.

A Monte Carlo simulation was performed and the simulation results @P_{80} result in a risk amount of $334.3 M.

21.2 Operating Costs

Year-wise unit cost-variation vis-à-vis the concentrate production is shown in Figure 21.1. The Project achieves the steady-state capacity in Year-3 after the initial ramp-up in Year-1 and Year-2, when the capacity utilization is 60 % and 85 % respectively. This level of production "creep" represents a reasonable increase in output and is adjudged well within the design parameters underpinning the Project.

Operating cost per tonne produced is higher during the early years when ramp-up to nameplate capacity is being achieved.

The area-wise summary of estimated annual operating costs are outlined in Table 21.7, by major area for the LOM, as well as the average production cost per tonne of final products.

Summary analyses of operating costs are presented in the following Sections.

Figure 21.1 – Summary of Estimated Opex by Year

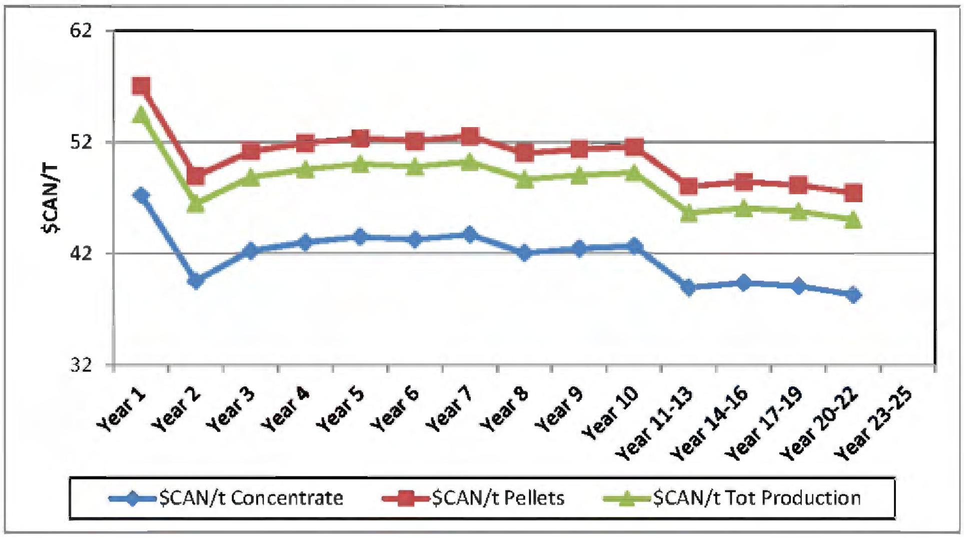


Table 21.7 – Summary of Estimated Opex by Area (Average)

Areas	Description	Costs for LOM ($'000)	Unit Costs ($)	Units
1000	Mine	8,380,119	17.76	per tonne of concentrate
2000, 3000, 4000	Concentrator	7,614,038	16.13	per tonne of concentrate
5000	Slurry Transport	421,237	0.89	per tonne of concentrate
6100 - 6200	Pelletizing Plant	4,043,786	11.09	per tonne of pellets
6300 - 6600	Port Operation	1,279,841	2.59	per tonne of total production
8000	G&A and Infrastructure	1,615,287	3.27	per tonne of total production
		Unit Cost for Concentrate	**40.65**	**per tonne of concentrate**
		Unit Cost for Pellets	**49.68**	**per tonne of pellets**
Note: Benefit and other payments are not included in unit costs				

21.2.1 Accuracy of the Estimate

The Opex presented is based on the estimated consumptions of consumables, reagents and power. Maintenance supplies cost factors were used for buildings. Vendor data was used where possible.

ISLLP's in-house database information and supplier queries for wear parts were used for major items of equipment e.g. crushers, HPGRs, ball mills and tower mills.

The accuracy level is in the range of ± 15 %, as required by this estimate class and is deemed appropriate a feasibility study level.

21.2.2 Basis of Estimate

The Opex was developed based on the following key assumptions:

- A mine plan based on a waste to ore ratio of no more than 0.54, and typically 0.33 over the life of mine;
- An assumed ramp up to 13 Mt/y concentrate production within four (4) quarters in Year-1 followed by an increase in production to 19 Mt/y in Year-2 to finally reach the full capacity for 22 Mt/y of product to be produced in the third year of operation;
- Product capacity of 23 Mt/y of product, from which 12 Mt/y will be BF pellets, five (5) Mt/y will be DR pellets and the rest will be kept as concentrate. The ratio of pellets/concentrate is 17/16 whereby 16.0 Mt/y of concentrate will produce 17 Mt/y of pellets and the balance of the 22 Mt/y, i.e., six (6) Mt/y, would be sold as concentrate;
- 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 22 years of operation, utilizing best practices 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 0.044 per kWh for Quebec consumption (L rate from Hydro-Québec);
- It is considered in the Feasibility Study that the power line will be built by a public utility (Hydro-Québec at their cost);
- The power line cost is estimated at $661 M (estimated amount given by Hydro-Québec);
- The power line cost was converted into an operational cost absorbed in the kWh price, assuming an interest rate of 7 % over a ten (10) years period (an increase of $0.021 per kWh over the first ten (10) years of operation). The power consumption was based on the electrical load list developed for the various areas.

The following criteria were used for the operating cost estimate:

- Salaries are based on average in the same geographic area for similar projects. The costs represent the full cost of employment to the Project and include local taxes and all benefits;
- Prices for internationally sourced material and consumables were based on enquires to international suppliers or obtained from ISLLP's in-house database;
- The pellet plant operating costs have been prepared by Outotec and provided to ISLLP for inclusion in the overall operating cost estimate;

- The operating costs for slurry transportation were prepared by BRASS;
- All major mobile mining equipment is on a leasing basis;
- Consumption data were obtained from equipment suppliers and ISLLP's in-house estimates;
- Annual working hours include vacation and sick leave are estimated at 1,957 hours for each employee with a working schedule of two (2) weeks in and two (2) weeks out for the mine site and process plant based employees;
- Accommodation (camps) and aircraft transportation costs for the workforce were calculated using ISLLP extensive internal database;
- The working capital was estimated to be equal to a three-month period of the operating cost of Year-1.

21.2.3 Escalation, Currency Exchange and Contingency

The base date of the operations cost estimate is February 2013. All escalation beyond that date is excluded. The capital cost used to factorize operating supplies, maintenance of buildings and of fixed equipment was updated in October 2013. The Bunker C fuel oil is dated August 2013 as well as the fuel price.

No escalation has been applied either to the operating costs or to the financial model for the Project.

The currency exchange rates applied are the same rates used for the Capex presented in this Section.

No contingency has been applied to the operating costs reported here.

21.2.4 Summary of Operating Costs by Area and Cost Item

a) Mining Operating Costs Estimate

Mining operating costs were estimated on the assumption that mining operations will be carried out by NML/TS. Met-Chem has developed a "bottoms-up" operating cost estimate for the mining operation.

Mining operating costs include:

- General and safety consumables for all the employees of the mine;
- Administration and operating supplies;
- Buildings maintenance;
- Leasing of the mobile equipment;
- Other similar costs.
- Mining fleet and mobile equipment operating and maintenance costs;

- Electricity cost that is not included in fleet cost and explosives costs, for all mining activities;
- Mobile equipment amortization and replacement;
- Mobile equipment operating labour;
- Consumption of fuel and lubricants, where not already included in hourly operating costs for equipment.

Mining includes the following sub-areas:

- Mine pit;
- Mine waste disposal;
- Mine installations and explosives handling.

The mine operating cost was estimated for each period of the mine cost. This cost is based on leasing and operating the mining equipment, the manpower associated with operating the mine, the cost for explosives as well as dewatering, road maintenance and other activities.

Operation and maintenance costs of services and buildings such as central workshops, fuel handling and other facilities are included in the G&A and Infrastructure.

In order to determine the operating cost, the following assumptions were used;

- Diesel Fuel Price – $1.16 /litre;
- Electricity – Variable by year $0.052 to $0.083) /kWh;
- Explosives Cost - $0.29 /t plus a fixed annual cost of $7 M (based on supplier pricing).

The mine operating cost was estimated to average $17.76 /t of concentrate or $2.69 /t mined for the life of the mine.

b) Mine Operating Cost Breakdown by Major Components

Mining operating cost estimates based on unit operations is as provided in Table 21.8.

Table 21.8 – Mining Operating Costs Breakdown – Activities and Manpower

Category	Costs ($/t Mined)	Costs ($/t of Concentrate)	Percent of Total Costs (%)
Loading	0.28	1.83	10
Hauling	0.89	5.84	33
Drilling and Blasting	0.42	2.80	16
Support & Service	0.15	1.02	6
Manpower	0.40	2.67	15
Leasing	0.50	3.30	19
Other	0.05	0.30	2
Total	**2.69**	**17.76**	**100**

c) Beneficiation Plant Operating Costs Estimate

Beneficiation plant includes the following sub-areas:

- ROM, crushing, storage and reclaim;
- Concentration (grinding/separation/upgrading);
- Tailings disposal and water management.

The average operating costs for the Beneficiation Plant are outlined in Table 21.9.

- The power cost is based on the annual operating time and average loading of the equipment;
- The labour cost includes operating labour and maintenance labour and is shown in Table 21.16;
- The grinding rolls wear parts replacement is based on grinding roll replacements after 6,000 hours for the eight (8) HPGR units;
- The grinding media consumption is based on a calculation assuming typical consumption properties for type of ore, typically referenced to an Abrasiveness Index;
- The ball mill liner replacement is based on calculations. It corresponds to one (1) set of lining replacement per ten (10) months for each of the eight (8) ball mills;
- The operating spares for the filters and baghouses are based on typical wear and change out rates;

- The operating spares for the HPGR screens decks are based on typical wear and change out rates;
- Other operating supplies include safety supplies, operator supplies and miscellaneous general items;
- Spare parts consumption is factored from mechanical equipment and building costs.

The laboratory supply costs are covered in this area although the analytical services will be provided across all operating areas, including geology, water treatment etc.

Table 21.9 – Summary of Average Crushing, Screening and Concentration Area Opex

Items	Costs for LOM ($)	Unit Costs ($/t of concentrate)
Power	2,954,008,918	6.259
Flocculants	99,825,106	0.212
Frother	3,687,157	0.008
Amine Collector	329,068,862	0.697
Starch	48,708,988	0.103
Lime	1,434,838	0.003
Gyratory Liner Concaves	88,570,872	0.188
Gyratory Mantle	87,890,000	0.186
Cone Liner	38,536,960	0.082
Cone Mantle	37,668,260	0.080
HPGR Tyres	429,444,000	0.910
Ball Mill Grinding Media Consumption	1,955,799,540	4.144
Ball Mill Liners Requirements	230,436,956	0.488
Tower Mill Grinding Media Consumption	140,761,868	0.298
Tower Mill Liners Requirements	51,600,000	0.109
Primary Screen	17,062,920	0.036
Secondary Screen	17,055,150	0.036
HPGR Screen	80,548,800	0.171
Fine Screen	31,106,880	0.066
Cobber Drums	11,271,600	0.024

Items	Costs for LOM ($)	Unit Costs ($/t of concentrate)
Rougher Drums	6,890,000	0.015
Finisher Drums	15,520,000	0.033
Fixed Equipment Maintenance	224,339,764	0.475
Building Maintenance	79,006,955	0.167
General and Safety Consumables	4,950,000	0.010
Labour	604,656,976	1.281
Administration Operating Supplies	24,186,279	0.051
Total	**7,614,037,650**	**16.13**

The summary of the manpower cost for the beneficiation plant can be found under Table 21.16 at the end of this Section.

d) Slurry Transport Opex

The slurry transport system includes the following areas:

- Slurry storage and pumping (Schefferville);
- Slurry transport;
- Slurry transport intermediate pumping station;
- Slurry transport monitoring station;
- Slurry reception area (Sept-Îles).

The operating costs for the slurry transport system area are outlined in Table 21.10.

Table 21.10 – Summary of Average Slurry Transport System Opex Estimate

Cost Items	Costs for LOM ($)	Unit Costs ($/t of concentrate)
Power	235,173,654	0.498
Equipment maintenance and Spare Parts	46,486,000	0.098
Building Maintenance	13,233,646	0.028
General and Safety Consumables	630,000	0.001
Labour	120,878,296	0.256
Administration Operating Supplies	4,835,132	0.010
Total	**421,236,728**	**0.892**

Data from sub-contractor BRASS

Maintenance cost is factored from equipment and building costs. The power cost is based on the annual operating time and average loading of the electrical equipment.

The summary of operation salaries for the slurry transport system for Year-4 can be found in Table 21.16.

e) Pelletizing and Port Opex

Pelletizing Plant and Port facilities include the following areas:

- Slurry thickening and feed preparation;
- Pelletization plant;
- Pellet and concentrate storage area;
- Water treatment area;
- Materials handling and storage at Jetty.

The operating costs for the Pellet Plant developed by Outotec are outlined in Table 21.11.

Table 21.11 – Summary of Average Pellet Plant Opex

Cost Items	Costs for LOM ($)	Unit Costs ($/t of Pellets)
Power	571,702,174	1.568
Bunker Type C Fuel Oil	2,065,390,678	5.664
Limestone	181,993,632	0.499
Bentonite	186,872,400	0.726
Organic Binder	203,775,000	1.900
Dolomite	28,885,563	0.079
Sulfuric Acid	19,308,488	0.053
Oxalic Acid	7,723,207	0.021
Maintenance Parts	239,744,151	0.657
Building Maintenance	44,396,000	0.122
General and Safety Consumables	2,366,000	0.006
Labour	472,720,248	1.296
Administration Operating Supplies	18,908,810	0.052
Total	**4,043,786,351**	**11.09**

The summary of operation salaries for the pellet plant for Year-4 can be found in Table 21.16 at the end of this Section.

The summary of operating costs for the port operation area is outline in Table 21.12. The port fees were provided by NML/TS and they are $0.37 /t passed through port for overhead royalty and $0.80 /t for loading cost from the common transfer tower.

- The power cost is based on the annual operating time and average loading of the electrical equipment;
- The labour cost includes operating labour and regular maintenance labour;
- Other operating supplies include allowances for safety supplies, operator supplies and miscellaneous items;
- Maintenance costs are factored from mechanical equipment costs and building costs;
- Other operating supplies include allowances for safety supplies, operator supplies and miscellaneous general items.

Table 21.12 – Summary of Average Port Opex Estimate

Cost Items	Costs for LOM ($)	Unit Costs ($/t of Total Production)
Power	404,210,414	0.819
Water Treatment Chemicals Costs	4,909,586	0.010
Fixed Equipment Maintenance	81,322,684	0.165
Building Maintenance	16,618,934	0.034
General and Safety Consumables	854,000	0.002
Labour	187,119,856	0.379
Port Fees	577,320,493	1.170
Administration Operating Supplies	7,484,794	0.015
Total	**1,279,840,761**	**2.59**

f) General and Administration and Site Infrastructure Opex Estimate

Site Infrastructure includes the following areas:

- Infrastructure and utilities (Schefferville);
- Infrastructure and utilities (Sept-Îles).

G&A – Sales, engineering and administration include:

- Management labour;
- Administration cost;
- Office supplies.

The operating costs for the site infrastructure area are outlined in Table 21.13.

- The main electrical loads in this area are the service buildings;
- The labour cost includes operating management, engineering, warehouse, and similar services;
- Maintenance includes a buildup of site mobile equipment maintenance costs covering spare and replacement parts, lubricants and tires;
- Other operating supplies include allowances for safety supplies, operator supplies and miscellaneous general items.

Table 21.13 – Summary of Average General and Administration Cost Estimate

Cost Items	Costs for LOM ($)	Unit Costs ($/t of Total Production)
Power	544,058,731	1.103
Fixed Equipment Maintenance	39,894,812	0.081
Building Maintenance	44,758,062	0.091
General & Safety Consumables	3,256,000	0.007
Labour	385,799,929	0.782
Administration Operating Supplies	15,431,997	0.031
Fly In Fly Out	151,543,517	0.307
Catering	228,836,437	0.464
Other Allowances	201,707,422	0.409
Total	**1,615,286,908**	**3.27**

g) Power Operating Costs Estimate

The distribution of the power by area is outlined in Table 21.14.

Table 21.14 – Power Distribution by Area – Year-4

Area	Consumption (MWh)	Costs ($)
Mine	159,409	8,670,003
Concentrator	2,318,043	134,273,133
Slurry Transport	182,295	10,689,712
Pelletizing Plant	590,601	25,986,462
Port	417,573	18,373,201
G&A and Infrastructure	421,727	24,729,942
Total	**4,089,647**	**222,722,452**

21.2.5 Operations and Manpower

NML/TS propose to establish an integrated iron ore mine and process plant North of Schefferville, Quebec. An export terminal facility will be located in Sept-Îles on the Gulf of St-Lawrence coast and a buried slurry transport system that will transport the concentrate slurry from the plant to the pelletizing facilities at the export terminal. The four (4) primary business units (mine, plant, slurry transportation system, export facilities) will be fully supported onsite by maintenance, engineering, technical, Health, Safety, Environment and Community ("**HSEC**"), Human Resources ("**HR**"), and finance/commercial groups of the organization.

The mine and process plant operations are based on 24 hours a day, 358 days a year to produce 22 million dry tonnes of magnetite concentrate. To allow for process disruptions, planned maintenance and other unforeseen events, an overall availability of 95 % has been applied to the design of the plant and to estimate steady state flow rates. During outage periods, production on the remaining operating streams will be optimized to minimize the production losses.

The export facility will manage the slurry receiving, filtering, pelletizing and stockpiling. The marine facility is a multi user export terminal managed by others.

The operation phase will be staffed using a combination of North American and local and maybe expatriates. All non-local mine and process plant employees will be accommodated for the duration of their on-site rotation at a permanent village on the outskirts of the plant and will fly out from Schefferville for their return home. Local employees will reside in their residences and drive to work. If practical, a bus service can be established from their village to the mine site.

The slurry transportation system, pellet plant and executive management staff will reside in Sept-Îles as well as the corporate members of the NML/TS team.

Crews will man the head pump station and intermediate pumping station under the same conditions as the people operating the process plant.

For start-up, there will be expatriates representing the NML/TS's team and some commissioning specialists. The majority of these will be integrated into the organization as part of the operations phase workforce. Over a period of three (3) years, it is intended to reduce this number, replacing the expatriate workforce with local or regional personnel.

An operational readiness program will be implemented to ensure the operating organization is ready to successfully manage the facility from start-up, and that the workforce is well-trained and capable.

a) Operating Philosophy

A primary Project objective is to provide a safe working environment that creates value for NML/TS shareholders, the local communities and the people of the Schefferville and Sept-Îles areas. It is critical to the success of the Project to perform effectively in the commissioning and operation of ramp-up phases, and then to further improve through lessons learned in the operations phase.

Useable, controlled and accessible documentation will be developed during the Project Execution Phase ("**PEP**"), including equipment documents, design documentation, operating procedures, maintenance procedures and work practices. This documentation will be instrumental in maintaining efficient, safe, consistent practices throughout the operations phase.

i) Language of the Operation

French and English will both be used as the official languages of the operation. In Quebec, availability and use of French in the working environment is governed by provincial law and policies. The legislation requires that all signage and documentation be available in French with few exceptions.

ii) Expatriate / Employee Transition

It is envisaged that the Project will have about 50 expatriates for start-up to fill specialized roles across the Project, including the mine, process plant and export terminal. These expatriates will provide the leadership and experience necessary to ensure successful commissioning and ramp-up to full production.

An additional key responsibility for all expatriates is the training and mentoring of selected local employees in order to achieve a timely and effective transfer of responsibilities to a largely local workforce.

b) Organization

The organizational structure for the operations phase is designed to support the successful operation of an iron ore mine and beneficiation plant in a remote location, a slurry transportation system, a pelletizing facility and associated material handling facilities while producing cost-effective, quality iron ore concentrate and pellets. An organization chart is provided in Figure 21.2 to depict the main operation areas.

A dedicated commissioning manager will lead the commissioning phase activities. The commissioning team will be drawn from the existing organizational structure and, the construction crew, the equipment supplier representative and the EPCM area specialists.

c) Resources and Sources of Personnel

It is essential to the success of the Project to have high-quality people with iron ore experience involved as soon as possible. The area is well developed with knowledgeable personnel since it has been under exploitation since the 1960's. Experienced staff is available in mining, concentration and pelletizing as well as in developing and supporting infrastructures and living quarter in northerly latitudes.

Figure 21.2 – Operation Organizational Structure by Facility and Services – Overall

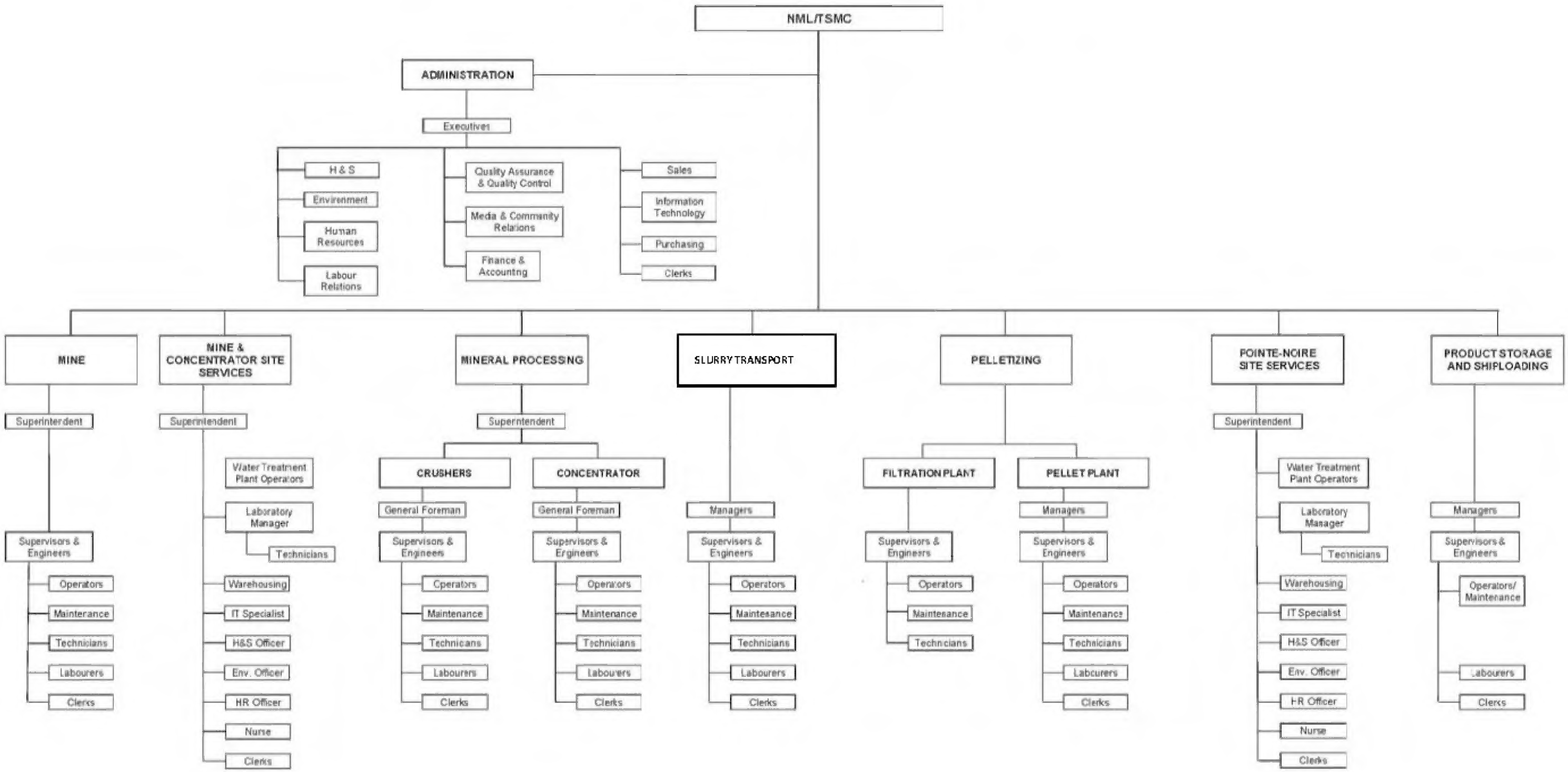


Table 21.15 – Labour Force Year-4

Area	Senior Executives	Superintendents, Managers, and General Foremen	Supervisors and Engineers	Operators	Maintenance Personnel	Technicians	Others	Total
Mine	1	2	10	344	123	39	4	523
Crushing	0	1	6	33	14	0	0	54
Concentrator	1	7	19	55	55	16	18	171
Slurry Transport	1	4	4	7	27	0	2	45
Pellet Plant	0	6	21	42	73	9	18	169
General & Administration and Infrastructure	3	11	0	0	0	72	62	148
Port Facilities	7	7	8	17	3	14	5	61
Total	**13**	**38**	**68**	**498**	**295**	**150**	**109**	**1,171**

d) Accommodation

Accommodations at the export terminal will be borne by the employees. Sept-Îles is a major centre and it is unusual to offer housing in an urban environment.

The situation at the mine is entirely different and housing and catering will be provided for the fly-in/fly-out staff.

e) Summary of Operating Labour Costs Estimate

The estimated overall operating labour cost for Year-4 is outlined in Table 21.16.

Table 21.16 – Estimated Operating Labour Costs – Year-4

Area	Number of Employees	Costs ($)
Mine	523	57,857,694
Concentrator	225	27,484,408
Slurry Transport	45	5,494,468
Pelletizing Plant	169	21,487,284
Port	61	8,505,448
G&A and Infrastructure	148	17,536,360
Total	**1,171**	**138,365,662**

22.0 ECONOMIC ANALYSIS

The economic/financial assessment of the KéMag Project is based on first-quarter 2013 price projections and cost estimates in Canadian currency.

No provision is made for the effects of inflation. The evaluation is carried out on a 100 % equity basis. Current Canadian tax regulations are applied to assess the corporate tax liabilities while the recently proposed regulations in Quebec (Bill 55, December 2013) are applied to assess the mining tax liabilities.

The assessment of the Project assumes that certain infrastructure elements, namely, the slurry transport infrastructure between the mine site and Pointe-Noire, the mine site accommodations, and the stock yard and materials handling facilities for ship loading are leased. The financial indicators under base case conditions are presented in Table 22.1.

Table 22.1 – Base Case Financial Results (100 % Equity)

Financial Indicators	Results	Units
Before-tax NPV @ 8%	5,261.8	$M
After-tax NPV @ 8%	2,241.3	$M
Before-tax IRR	17.5	%
After-tax IRR	13.2	%
Before-tax Payback Period	4.9	Years
After-tax Payback Period	5.7	Years

A sensitivity analysis reveals that the Project's viability is not significantly vulnerable to variations in capital and operating costs, within the margins of error associated with feasibility study estimates. However, the Project's viability remains vulnerable to the larger uncertainty in future sale prices.

22.1 Methodology

The financial assessment is based on estimates of capital and operating costs, a Project execution schedule as well as a production schedule, which combined with price forecasts, enables the projection of revenues over the operating life of the Project. The operating life is expected to run over a 22-year period.

Cash flows are developed on an annual basis in a Microsoft Excel model. Financial results are derived on both a before-tax and after-tax basis. Taxation is based on current federal and Quebec corporate tax rules as well as on the recently proposed Quebec mining tax legislation (Bill 55, December 2013).

The base case assumes 100 % equity financing. However, results based on 30 % equity/ 70 % long-term debt financing, showing the effect of financial leverage, are presented as well. No provision is made for the effects of inflation.

22.2 Summary of Input Data

22.2.1 Base Case Assumptions

The base case assumptions are listed below:

- All cost estimates are expressed in constant first quarter 2013 Canadian currency;
- An exchange rate of USD 0.90 to CAD 1.00 is assumed to convert the USD price projections into CAD. As well, this rate is used to adjust Opex estimates and 7.0 % of the original Project Capex initially priced in USD at an exchange rate of 1 to 1. A similar adjustment is made to 5.5 % of the original Project Capex initially priced in Euros at an exchange rate of EUR 0.77 to CAD 1.00 to account for a revised exchange rate of EUR 0.71 to CAD 1.00
- The analysis is performed on the basis of 100 % equity financing;
- The FOB Pointe Noire price forecasts for the three (3) products are:
  - Type BF pellets – US$ 116.61/t;
  - Type DR pellets – US$ 126.86/t;
  - Concentrate – US$ 90/t
- The assessment assumes that the assets associated with slurry transport of the concentrate from the mine site to Pointe-Noire, the mine site accommodations, and the stock yard and materials handling facilities for ship loading are leased. As the original Capex includes the cost of the leased facilities along with their construction schedule, the cost estimates have been modified for the purpose of the leasing assumption. Thus, the capital expenses associated with these assets are removed from the original Capex (a total of $ 3,011.8 M) and replaced by a time-adjusted annuity representing an additional operating expense over the 22-year operating life of the Project. The time adjustment considers the construction schedule of the facilities in relation to the start of production. The annual lease rate is assumed to be seven (7) %.
- Project NPVs are determined at a discount rate of eight percent (8 %), the base case, and at two (2) additional rates of ten percent (10 %) and 12 %, meant to represent typical costs of equity capital.
- Working capital requirements are based on 45 days of receivables and 30 days of payables. Benefit Agreement payments as described in Section 22.3 of this Report are not considered as payables. Working capital levels fluctuate over the operating life as sales and operating expenses increase or decrease. The remaining working capital is recovered at the end of operations.

- Closure costs are set aside annually during the operating life of the Project and thus, are included in the Opex. Consequently, no closure bonds have been modeled.

22.2.2 Revenues

Annual revenue projections are based on the pellet and concentrate production schedules documented in Table 22.2 below and the price forecasts given above. Expected annual production rates are 12 million, five (5) million and six (6) million tonnes for type BF pellets, type DR pellets and concentrate, respectively. The production schedule provides for a ramp-up over the first two (2) years of production. Full production is reached in the third year of operation and maintained until the final year of life. Table 22.3 provides life-of-mine revenues associated with the three (3) mine products.

Table 22.2 – Iron Ore Shipments per Year

Year	BF Pellets (t)	DR Pellets (t)	Concentrate (t)
1	7,200,000	3,000,000	3,609,646
2	10,200,000	4,250,000	5,085,884
3	12,000,000	5,000,000	6,005,277
4	12,000,000	5,000,000	6,033,557
5	12,000,000	5,000,000	6,000,248
6	12,000,000	5,000,000	5,989,747
7	12,000,000	5,000,000	5,978,974
8	12,000,000	5,000,000	6,024,438
9	12,000,000	5,000,000	5,993,018
10	12,000,000	5,000,000	6,011,626
11	12,000,000	5,000,000	6,014,480
12	12,000,000	5,000,000	6,014,480
13	12,000,000	5,000,000	6,014,480
14	12,000,000	5,000,000	6,001,222
15	12,000,000	5,000,000	6,001,222
16	12,000,000	5,000,000	6,001,222
17	12,000,000	5,000,000	5,987,496
18	12,000,000	5,000,000	5,987,496

Year	BF Pellets (t)	DR Pellets (t)	Concentrate (t)
19	12,000,000	5,000,000	5,987,496
20	12,000,000	5,000,000	6,014,770
21	12,000,000	5,000,000	6,014,770
22	12,000,000	5,000,000	6,014,770

Source: ISLLP Feasibility Study, 2013

Table 22.3 – Life-of-Mine Revenues

Products	LOM Revenues ($M)
Type BF Pellets	33,350.5
Type DR Pellets	15,117.5
Concentrate	12,878.6
Total	**61,346.6**

22.2.3 Operating Costs

Annual operating cost estimates are based on the mine production schedule in Table 22.2 above and the operating cost estimates documented in Section 21.2 of this Report. These costs have been adjusted for foreign content with the assumed USD exchange rate. Table 22.4 provides a summary of operating costs associated with the Project. The production costs include the Benefit Agreement payments as described in Section 22.3 of this Report.

Table 22.4 – Summary of Production Costs

Operating Costs	Unit Costs ($)	Units
Average Production Cost	42.56	$/t concentrate
Average Production Cost	51.60	$/t pellets
Average Leasing Cost	14.17	$/t product
Total LOM Production Costs	**31,288.9**	**$M**

The production costs include the Benefit Agreement payments.

22.2.4 Capital Costs

Initial Capex and sustaining capital costs are documented in Section 21.1 of this Report. These costs have been adjusted for foreign content with the assumed USD and Euro exchange rates. Table 22.5 summarizes the Capex of the Project. The information related to initial Capex represents the initial investment without the cost of the leased assets as listed in Section 22.2.1 of this Report.

Table 22.5 – Capex Summary

Capital Items	Costs ($M)
Initial Capital Cost	5,291.9
Initial Working Capital	120.6
LOM Sustaining Capital Costs	1,096.9

22.3 Royalties, Benefit Agreement and Taxation

22.3.1 Royalties

No royalty agreement applies to the Project.

22.3.2 Benefit Agreement

Benefit Agreement payments apply to the concentrate production from the KéMag Property. These are accounted for in the financial model but the terms of the agreement remain confidential at this time.

22.3.3 Taxation

A tax model incorporating corporate and mining taxes was developed by a fiscal expert.

Corporate taxes are levied both by the federal and provincial/territory governments in Canada. Generally, provincial corporate income taxes are levied on federal taxable income pro-rated to the particular province or territory. However, Alberta and Quebec have particular rules for determining taxable income that differ slightly from the federal rules. Thus, these provinces collect their own corporate taxes.

In the present case however, those rules that differ in Quebec have no impact on the determination of provincial taxable income. Therefore, Quebec corporate taxes are based on federal taxable income. Mining taxes are levied by each province/territory in Canada. Thus, Quebec mining taxes are levied on taxable income associated with mineral products extracted in Quebec.

The fiscal conditions listed below are assumed to apply.

For federal and Québec provincial corporate income taxes:

- Federal income tax rate of 15 % throughout the Project's life;
- Quebec provincial income tax rate of 11.9 % throughout the Project's life;
- Depreciation at a rate of 25 % per year on a declining-balance basis for all depreciable assets, including mining assets, concentrator, pellet plant, power supply assets and port installation assets (as per the 2013 Federal Budget);
- Canadian development expenditures depreciated at a rate of 30 % per year on a declining-balance basis;
- Canadian exploration expenditures allowable at a rate of 100 %;
- Deductibility of mining taxes.

For Quebec mining taxes (based on Bill 55, December 2013):

- A minimum annual royalty payment based on the value of output at the pit's mouth; the rate is one (1) % for output valued at less than $ 80 M plus four (4) % of any excess; this minimum payment is considered a pre-payment of the "regular" mining tax;
- The "regular" mining tax includes a basic tax of 16 % of taxable income, plus two (2) possible additional levies: an additional tax of six (6) % to the extent the "profit margin" in a given year exceeds 35 %, and an additional tax of six (6) % to the extent the "profit margin" in a given year exceeds 50 %; these apply incrementally to the excess over each threshold margin;
- Annual processing allowance of ten (10) % (20 % if ore is processed beyond the beneficiation stage) of the original cost of processing assets located in Quebec, not to exceed 75 % of the profit for the year, as calculated immediately before this deduction, and not to be less than 30 % of the value of output at the pit's mouth determined before the deduction of the processing allowance;
- Exploration and development expenditures allowable at a rate of 100 %;
- Depreciation at a rate of 30 % per year on a declining-balance basis for all depreciable assets, including mining assets, concentrator, pellet plant, power supply assets and port installation assets.

22.4 Residual Value

For the purpose of this financial evaluation, it is assumed that any proceeds realized from the sale of fixed assets at the end of the Project's life are offset by the cost of site rehabilitation. Therefore, the net residual value consists only of the recovery of working capital.

22.5 Financing

Although the base case assessment assumes 100 % equity financing, the financial model was created to address the case in which debt financing is considered. Here, it is assumed that 70 % of the initial capital cost will be financed through long-term debt sources. To the extent possible, the disbursement of borrowed funds is delayed in order to reduce the amount of interest payable.

As financing arrangements are not yet in place, it is assumed that the annual interest rate on borrowed funds is seven (7) % (current U.S. Prime rate of 3.25 % + 3.75 %).

The assumed terms of repayment are single payments at the end of each year starting at the end of the first full year of production over a period of ten (10) years. It is important to note that no additional work was done to confirm these assumptions with potential lenders. The financing terms are used only to demonstrate the effect of leverage.

22.6 Financial Model and Results

Table 22.6 presents the financial analysis results on a before-tax basis and Table 22.7 presents the cash flow statement of the Project.

Table 22.6 – Before-tax Financial Indicators

Before-tax Financial Indicators	Results ($M)
Total LOM Revenue	61,346.6
Total LOM Operating Costs*	24,298.3
Total LOM Lease Payments	6,990.6
Initial Capital Cost	5,291.9
Total LOM Sustaining Capital	1.096.9
Total Cash Flow	23,668.9
Net Present Value @ 8%	5,261.8
Net Present Value @ 10%	3,455.0
Net Present Value @ 12%	2,135.8
Internal Rate of Return	17.5
Payback Period	4.9

* Includes Benefit Agreement payments

Table 22.8 summarizes the financial analysis results on an after-tax basis. The first five (5) lines of Table 22.6 (i.e., total revenue to total sustaining capital) are not reproduced in Table 22.8 because they are identical.

Table 22.8 – After-tax Financial Indicators

After-tax Financial Indicators	Results ($M)
Total LOM Corporate Taxes	7,271.4
Total LOM Mining Taxes	3,175.9
Total Cash Flow	13,221.6
Net Present Value @ 8%	2,241.3
Net Present Value @ 10%	1,153.8
Net Present Value @ 12%	359.0
Internal Rate of Return	13.2 %
Payback Period	5.7 Years

Note: The NPVs given in the tables above are based on the mid-year convention rather than the more common end-of-year convention. Some financial experts favour the mid-period convention for projects in which cash flows occur more or less continuously through time, as is the case for industrial projects. Using this convention, cash flows are assumed to occur in the middle of each year and are discounted as such. A net present value determined in this manner is greater than that obtained using the conventional method. The ratio of the larger value over the lesser value is simply $(1+i)^{0.5}$*, in which "i" represents the annual discount rate. This relationship is exact as long as all cash flow components follow the same convention. The internal rate of return is not affected by this issue.*

For the purpose of illustrating the effect of debt financing, Table 22.9 compares the financial outcomes with 100 % equity financing versus those with 30 % equity/ 70 % debt financing (i.e. levered). The impact of leverage is greater on an after-tax basis than on a before-tax basis because interest payments are tax deductible. For this reason, the results are shown only for the after-tax assessment.

Table 22.9 – Effect of Leverage

After-tax Financial Indicators	Results ($M)
Net Present Value @ 8%	2,241.3
Levered Net Present Value @ 8%	2,697.1
Internal Rate of Return	13.2 %
Return on Equity ("**ROE**")	21.9 %

Note: According to financial theory, the cost of "levered" equity is higher than the cost of "unlevered" equity. The difference between these two (2) costs increases with the debt/equity ratio. Consequently, if the cost of unlevered equity is eight (8) % for instance, the cost of levered equity is higher. Thus, discounting the levered cash flows at this higher rate may in fact reduce the NPV to a level lower than that of the unlevered value observed in Table 22.9 above. Likewise, the ROE (i.e. the levered IRR) must be compared to the higher cost of levered equity and not to the cost of unlevered equity, which is assumed to be eight (8) % in Table 22.9.

22.7 Sensitivity Analysis

A sensitivity analysis is carried out on the base case scenario described above to assess the impact of changes in market prices (across-the-board variations in all three (3) product prices), total pre-production capital cost and operating costs on the Project's NPV @ eight (8) % and IRR. Each variable is examined one-at-a-time. An interval of ± 30 % with increments of ten (10) % was used for all three (3) variables. It is to be noted that the margin of error for cost estimates at the feasibility study level is typically ± 15 %. However, the uncertainty in price forecasts usually remains significantly higher, and is a function of price volatility.

The before-tax results of the sensitivity analysis are shown in Figure 22.1 and Figure 22.2. Figure 22.1 showing variations in NPV indicate that the Project's before-tax viability is not significantly vulnerable to the under-estimation of capital and operating costs, taken one at-a-time. The vertical dashed lines represent the typical 15 % margin of error associated with the Capex and Opex estimates. The NPV is more sensitive to variations in operating expenses than it is to capital expenditure, as shown by the steeper curve. As expected however, the NPV is most sensitive to variations in prices. Across-the-board reductions of about 25 % in prices result in break-even conditions, i.e., a net present value of zero.

Figure 22.1 – Sensitivity of the Before-tax Net Present Value

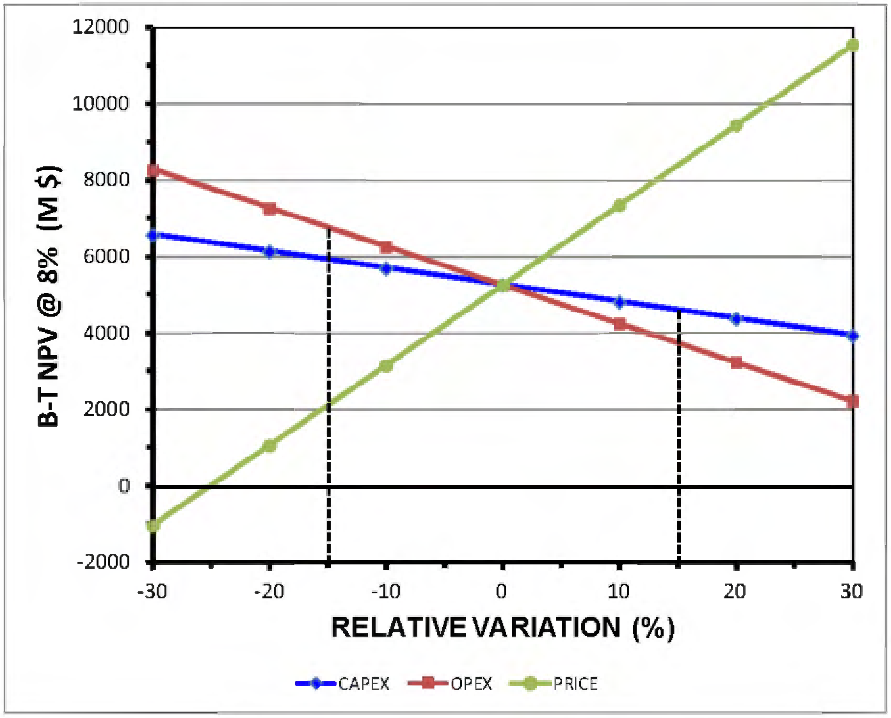


Figure 22.2 – Sensitivity of the Before-tax Internal Rate of Return

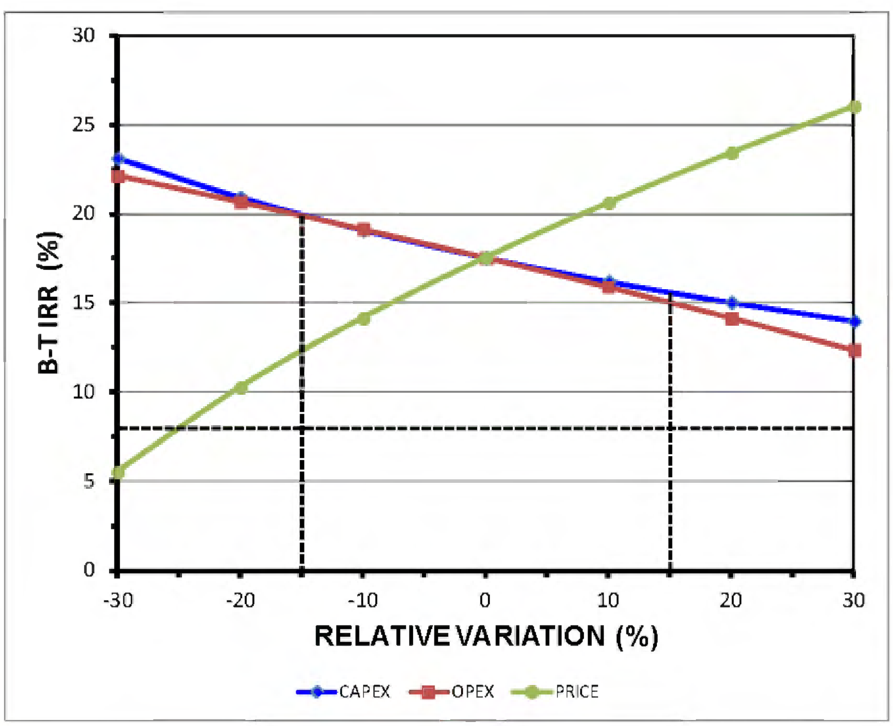


Figure 22.2, showing variations in internal rate of return provides similar conclusions. In contrast with Figure 22.1 which shows linear variations in NPV for the three (3) variables studied, variations associated with internal rate of return are not linear. Due to the different timing of pre-production capital expenditure versus operating costs over the mine life, the internal rate of return is more sensitive to variations in capital expenditure, especially negative variations. The rate of return is reduced to eight (8) %, i.e., break-even conditions (shown by the horizontal dashed line), for the same across-the-board reductions in prices as noted above in the case of the net present value.

The after-tax results of the sensitivity analysis are shown in Figure 22.3 and Figure 22.4. The same conclusions about the sensitivity of the project's viability to variations in capital expenditure, operating costs and prices can be drawn here. On an after-tax basis, however, break-even conditions are reached at across-the-board reductions in prices of 19 %.

Figure 22.3 – Sensitivity of the After-tax Net Present Value

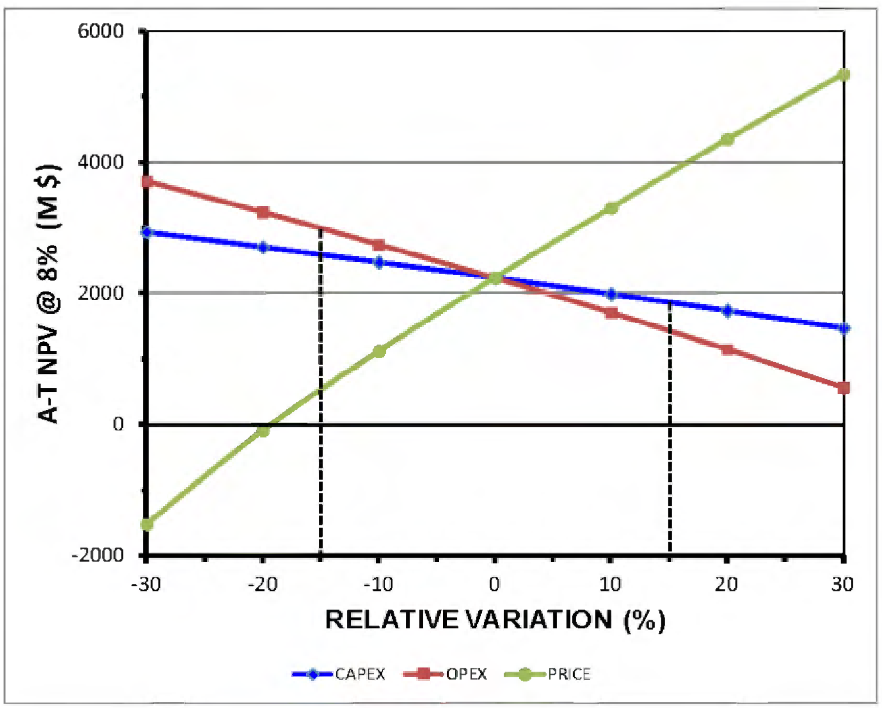
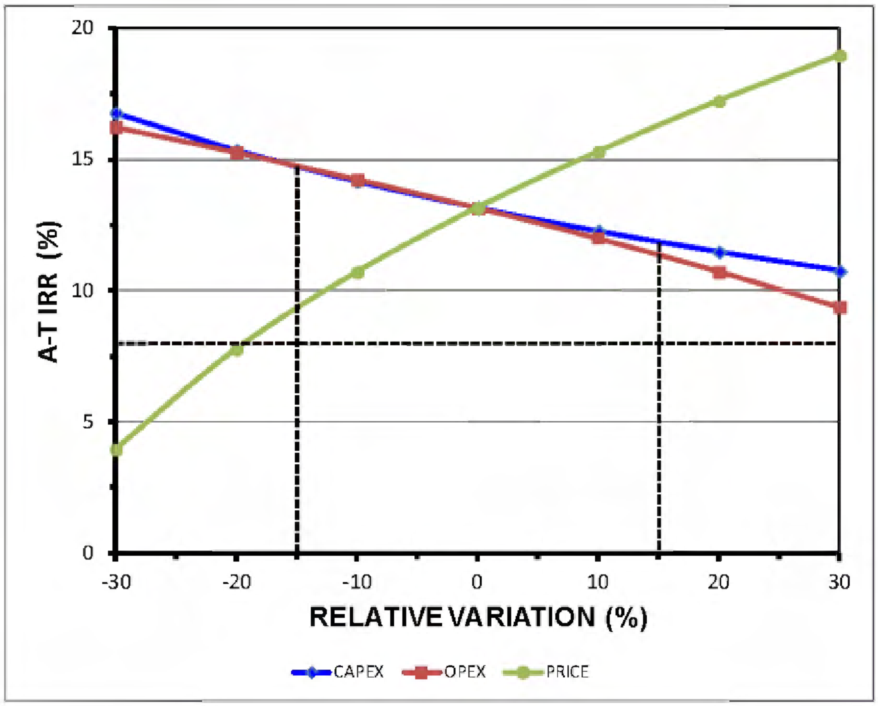


Figure 22.4 – Sensitivity of the After-tax Internal Rate of Return

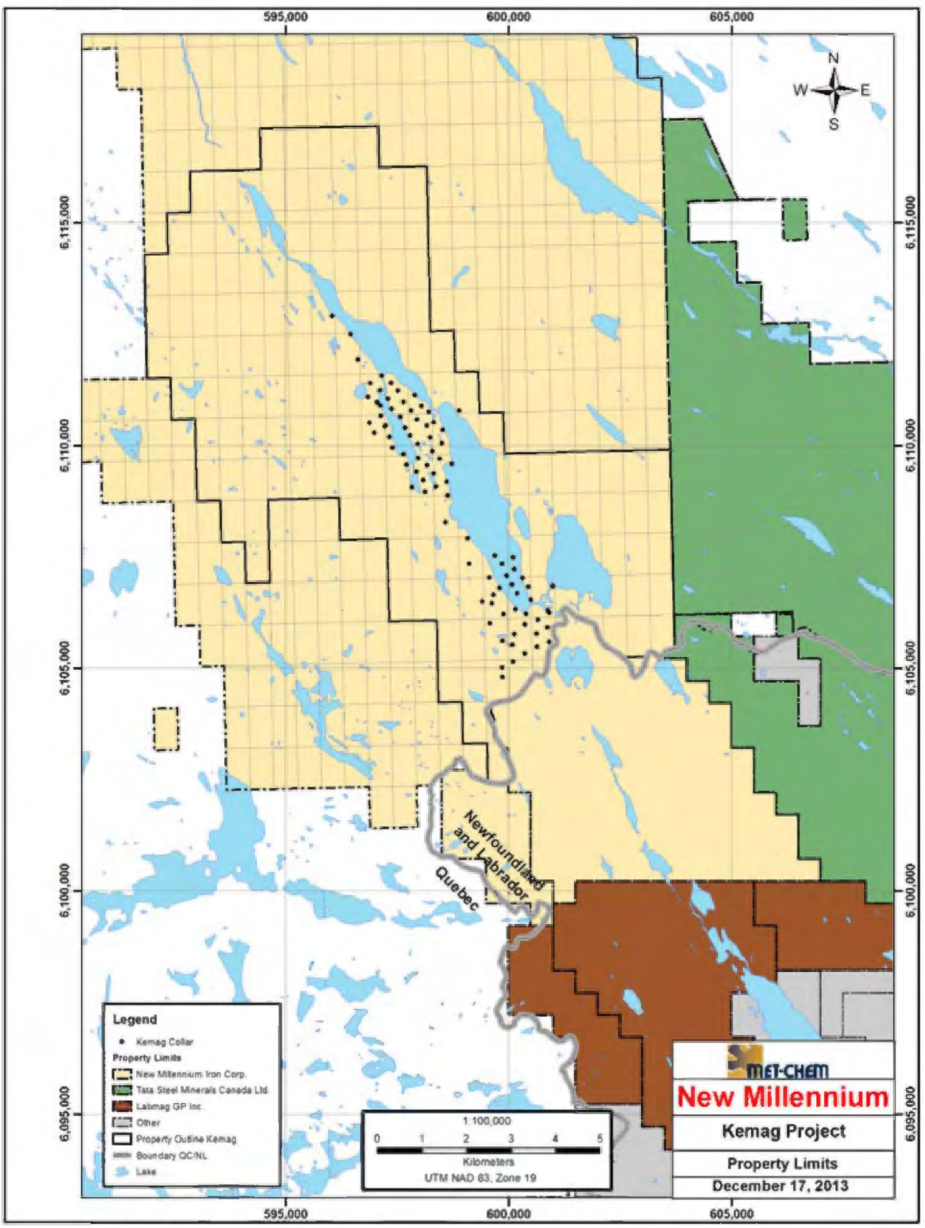


23.0 ADJACENT PROPERTIES

The KéMag Property is essentially surrounded on all sides by claims held by New Millennium Iron Corp., except for a block of adjacent claims at the southeastern boundary registered under Tata Steel Minerals Canada Ltd., as illustrated in Figure 23.1.

The southeast limit of the KéMag Property follows the Quebec-Labrador boundary and abuts against the Licenses held by New Millennium Iron Corp., Tata Steel Minerals Canada Ltd. and LabMag GP Inc.

Figure 23.1 – Location of the Claims Adjacent to the KéMag Property



The reader is advised that the information provided in this Section was publicly disclosed, derives from an Internet search and is mostly drawn from the Registry of *Ministère des Ressources Naturelles* and various published maps and reports. The Qualified Person has not attempted to verify the data and results. The presence of iron formation in adjacent properties is not necessarily indicative of the mineralization on the Property that is subject of the present Technical Report.

24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 Schedule

A Project master schedule for the Feasibility Study has been developed to cover the major Project milestones. The Project master schedule contains engineering, procurement, construction and pre-operational testing and commissioning activities at a level of detail commensurate with the progress of scope definition. It encompasses the WBS structure and master scope document developed in the Feasibility Study and is consistent with the contracting plan.

24.1.1 Schedule Assumptions

The Project schedule is based on the following assumptions:

- Design criteria, process and scope of work if frozen prior to start of basic engineering;
- Geotechnical and hydraulic surveys are completed prior to start of basic engineering;
- Major equipment package bid/award duration is ten (10) weeks;
- Resources are available for engineering and construction management personnel;
- Constraints for construction due to the lack of construction workers in the region, especially welders;
- Schedule activities are based on the Project WBS.

24.1.2 Project Schedule / Construction Sequence

The construction flow is the following:

- Mobilization of EPCM contractor to site;
- Start construction of temporary camp;
- Mobilization of site preparation sub-contractor;
- Site preparation;
- Civil works;
- Concrete construction;
- Structure erection;
- Architectural installations and finishes;
- Mechanical completion;
- Electrical and instrumentation installation;
- Pre-operational verification;

- Commissioning assistance.

24.1.3 Critical Path

The critical paths are through the procurement of steel, the construction of the slurry transport system and the construction of the pellet plant.

24.1.4 Duration of the Project Schedule

- 32 months detailed engineering: months 19 to 52;
- 44 months construction: months 22 to 65;
- 10 months pre-commissioning: months 57 to 67;
- 9 months commissioning: months 59 to 68;
- Total: 57 months from EPCM contract award to the 1st product feed.

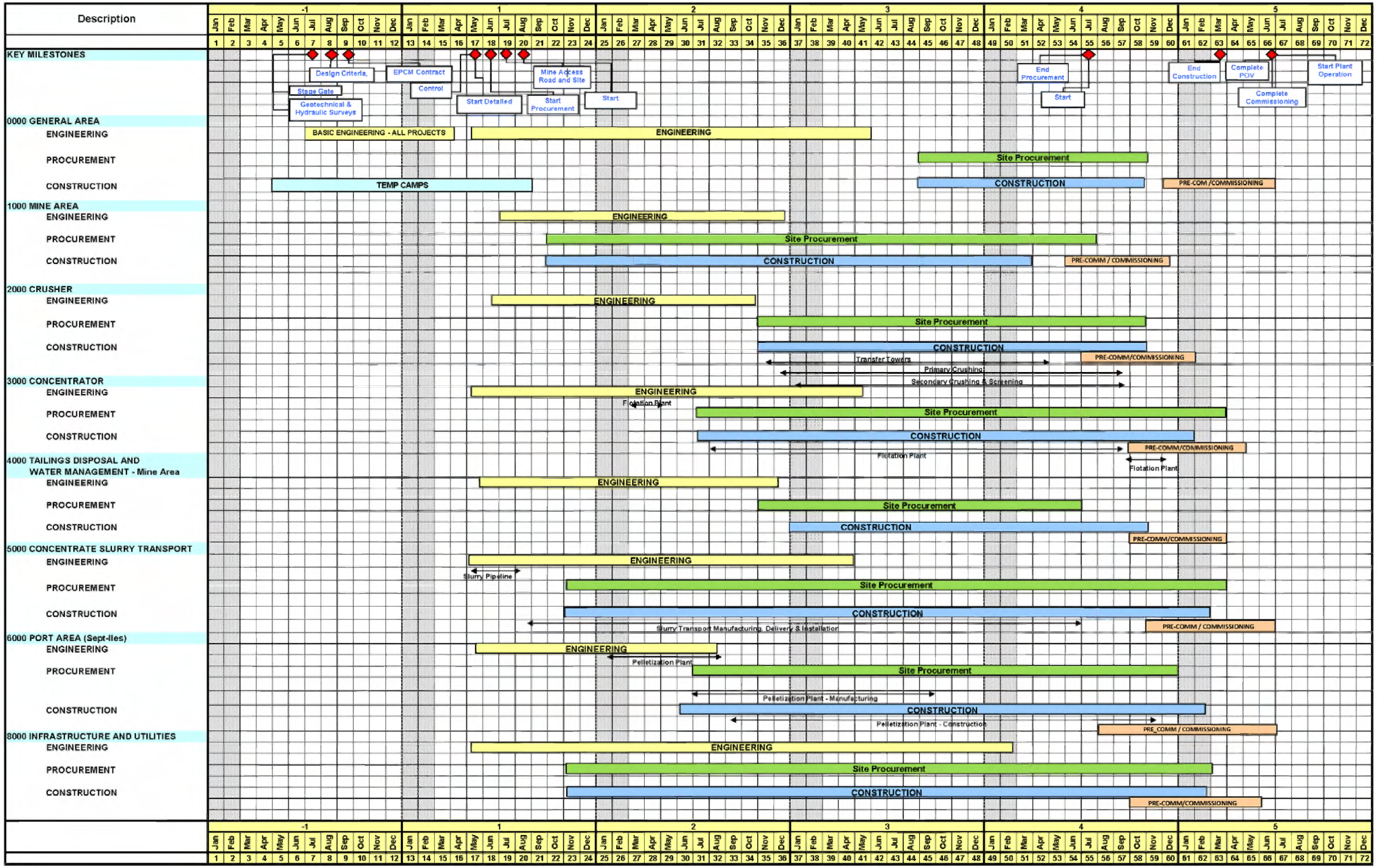
24.1.5 Major Milestones

The major milestones have been tabulated in the following Table 24.1 showing the Project activities through its cycle and Figure 24.1 represents the Project road map.

Table 24.1 – Major Milestones

Description	Month
General	
Final FS Study Submitted to NML/TS	0
Stage Gate Approval By NML/TS	4
Geotechnical and Hydraulic Survey Completed	9
Start Basic Engineering	9
EPCM Contract Awarded	11
Start Detailed Engineering	19
Procurement Start	20
Start Construction	21
Design Criteria, Process and Layouts Frozen	22
Establish Control Budget	23
Start Pre-Operational Verification	55
End of Construction	60
End of Pre-Operational Verification	64
Commissioning Completed	68
Plant Operation Start	68
Detailed Engineering	
Start Basic Engineering	9
Start Detailed Engineering	19
Detailed Engineering of Slurry Transport System	22
Detailed Engineering of Structural Steel	35
Procurement of Pellet Plant	35
Construction / Pre-Commissioning / Commissioning	
Start Temporary Camp Installation on Site	18
Start Civil Works Mine Pit	19
Start Slurry Transport System Construction	24
Start Flotation Plant Concrete Construction	35
Start Primary Crushing Concrete Construction	42
Start Structural Steel Installation	42
Start Mechanical Installation	43
Start Electrical Installation	45
Pelletization Plant Commissioning	64
Pre-commissioning Complete	67

Figure 24.1 – Master Schedule Road Map



24.2 Project Execution Plan

24.2.1 Project Execution Plan Document Structure

The Project Execution Plan ("**PEP**") has been developed based on the Feasibility Study work done to date on the Project. The PEP outlines the plans, resources, mechanisms and procedures that NML/TS expects from the EPCM contractor and covers all aspects of the Project from engineering design through pre-operational verification ("**POV**") and assistance during commissioning.

This PEP will form the foundation for execution of the Project, and will be used in conjunction with the Feasibility Study documentation by the EPCM contractor in creating their detailed Project execution plan.

24.2.2 Execution Strategy

The execution strategy of the Project is based on a construction–driven methodology for large capital project delivery.

The strategy implies that NML/TS will appoint an EPCM contractor to supply basic and detailed engineering, execute the complete procurement and contracting cycle, appoint and manage contractors and control schedules and budgets.

The Project schedule favours an improved level of Project definition prior to awarding packages, to minimize the financial risks to NML/TS. The EPCM contract can be converted to Lump Sum once over 75% of the detailed engineering has been executed and prior to contracting the major construction packages.

There are other strategies that could be adopted by NML/TS such as a strong Owner's management team that would direct the engineering, procurement and construction packages.

Detailed engineering for the mine, concentrator, slurry transport construction plans and port material handling and overall infrastructure will be performed in a single location to ease coordination, control scope and minimize duplication and rework.

Specialized engineering such as the pelletizing and slurry transport hydraulics for example will be sub-contracted to technology providers supervised by the EPCM contractor. Labour intensive engineering such as structural steel detailed drawings can be performed in low-cost centres overseas as long as the communication systems and supervision is mastered by the EPCM contractor.

Procurement and contracting strategy will be based on the list of commitment packages. The bidders list will be expanded to all approved bidders for a specific commodity and award will be based on the lowest bidders who meet the technical requirements. If applicable, preference shall be given to bidders who have First Nations affiliations or have offices or facilities located in Quebec. Specialized lots will be awarded on a turn-key basis while construction lots will be awarded on a unit rate basis. Since the Project definition will be advanced, quantities should not vary significantly from the engineering estimates.

Major equipment, specialised equipment, expensive, or long lead items will be purchased by the EPCM Contractor and will be supplied to the installation subcontractors.

Piping bulk material will be purchased by the EPCM Contractor and will be supplied to the fabrication shops. Piping spools will be pre-fabricated in shop and delivered at site prior to the planned start of their installation.

24.2.3 Site Conditions and Constraints

The weather conditions on site, as well as the working calendars and holidays have an impact on the Project schedule.

The working calendar is based on a six (6) days/week - 10 hours/day calendar. There will be two (2) breaks of two (2) weeks one for summer vacations and another during the holidays season (Christmas) as well as allowance for non-working periods on statutory holidays. The manpower rotation will be three (3) weeks in, one week out and for technical positions; an overlap will be worked for transiting from one person to another.

A seasonal work calendar is considered for some specific type of work, with non-working period from mid-October to mid-April for major concrete works. Temporary power will be used during the construction period.

A camp shall be built before starting early works.

24.3 Construction Execution Plan

The Project will be managed and constructed with different construction management teams which will all be supported and directed from the Project construction head office. There will be a construction management team with all technical and management forces that will be assigned to fulfill all the responsibilities to assure the proper execution of the Project.

Construction managers for each location will be assisted by a site engineering team, discipline technicians, area superintendents, HSE officers, labour relation coordinator, construction planners and progress monitors, cost controllers, estimators, contract administrators, material controllers, warehouse administrator and secretarial and clerical workers.

Each camp and construction management offices will be autonomous and will have all the required services to support the workforce in line with the planned and estimated size at their peak manpower requirement.

Piping should either be integrated into the equipment skids or be part of pipe-racks modules that will be transported to site.

Electrical and instrumentation will be integrated into the different parts of the process buildings using prefabricated electrical and control rooms and set in place once delivered to site. Building structural grounding loops will be executed using smaller contracts.

The objective of this approach is the reduction of the manpower requirements on site, thereby significantly reducing the site indirect costs, improving site safety records and maintaining the Project schedule.

25.0 INTERPRETATION AND CONCLUSIONS

The Project is technically feasible; ore can be mined, treated and delivered into export vessels by incorporating proven processes and technologies.

Based on 22-year cash-flow, the IRR before taxes is 17.5 % and 13.2 % after taxes assuming 100 % equity and standard mining income taxes. Therefore, the Project has financial viability.

The life of mine can be further enhanced by exploiting the lower grade resources currently being stockpiled.

25.1 Project Highlights

The geology and mineral reserve are well known and can supply a consistent quality ROM to the concentrator for 22 years of production.

The process has been developed after extensive test work and by experienced process engineers, although refinements and confirmation of certain processing steps are possible. There is enough confidence in the flow sheets so that refinements and optimization can be tackled during the basic engineering phase.

Conventional disposal of tailings can be engineered to meet the environmental requirements while remaining economically viable.

The slurry transport system is an economical and reliable mean of transporting iron ore slurry, especially fine taconite concentrate. Frost protection is of paramount importance and can be achieved by a combination of methods according to the geographical location and soil conditions. Such trade-off studies should be assessed early in the basic engineering phase of the Project along with field surveys.

Pelletizing testing has confirmed the high probability of achieving high plant throughput.

A new multi-user dock with a capacity of 50 Mt/y and capability of handling 350,000 DWT vessels is being built by SIPA and is planned to be commissioned by fall 2014. The product handling facilities will connect the pelletizing plant to this deep-sea port terminal. NML has invested $ 38 M in the new dock to obtain the rights to ship 15 Mt/y of product.

The market study indicates a future pellet demand and both the market trends and forecasts indicate higher pellet premium.

The Project is expected to trigger several regimes of EA. Discussions with the applicable governments have been held and it is concluded that a single EIS covering all the components of the Project is proposed to be submitted by August 2015.

Constructability strategy of using pre-engineered structures, skid mounted equipment and modularization to the maximum extent allowed by the transportation channels will lower the risk of cost and schedule overruns.

During the construction phase, the Project will create on average 2,500 construction jobs over 42 months and 300 management and engineering jobs spread over 60 months. The construction labour will peak at 4,600 labourers and 400 EPCM employees in the summer of the last full year of construction at all sites. During operation, 1,100 direct employees will be necessary and numerous indirect jobs will be created through the development of secondary and tertiary industries that will be supporting the operation and maintenance of this Project. This is a major component of the economic development of the Labrador Trough and Sept-Îles.

25.2 Risk Evaluation

ISLLP conducted risk reviews workshop with ISLLP experts in their various fields and with NML/TS Project team members. The objectives of the risk sessions were to identify risks, mitigations and action plans in the Feasibility Study phase to pursue up to the execution.

The scope is comprised of both risks and opportunities associated with but not limited to the following categories:

- Environment;
- Geotechnical and Hydrology;
- Engineering;
- Infrastructure;
- Mining;
- Public Perception;
- Health, Safety and Environment;
- Financial;
- Construction;
- Operation.

The outcome of these workshops was compiled in the risk register and an action log formulated for the Project.

25.2.1 Key Project Concerns

- Mining below Lake Harris, however the lakes are shallow and flow can be diverted;
- Lake with fish-habitat is an environmental concern, but likely, to be permitted. The cost has not been establish;

- Long-distance slurry transportation systems in freezing climates, depth of burial review;
- Environmental Assessment, EIS submission delayed to August, 2015;
- Lack of detailed geotechnical assessment could result in surprises and have a significant impact in the construction Capex and hence must be completed before start of basic engineering and the finalization of the Project budget.

The risk level as well as risk qualitative analysis was determined by the participants who selected the level of consequences, mitigation plans including action plans for each identified and evaluated risk.

On the positive side, during the next phase, there is good possibility to improve the flow sheet resulting in reduction of the Capex and Opex.

25.3 Conclusion and Next Phase

Early identification of constructability techniques to maximize pre-assembly of concrete, steel, equipment and any other way of reducing the installation time on site, is paramount for a successful project.

Early in the project's lifecycle, an experienced constructability engineering team should be reviewing the design layouts, specifications, objectives, and site conditions to work with the design engineers to establish possible guidelines to standardize buildings, foundations and any other criteria to minimize labour costs and impact on schedules.

The constructability team should also work very closely with the logistics group to determine the ideal transportation and construction philosophy for the engineering team.

The risk workshops were composed of experts, both internally to ISLLP and NML/TS. The discussions were open and transparent allowing for any member to raise concerns or comments. The P_{80} value of the probable consequence value of risks after mitigation was $334 M as the result of the Monte Carlo simulation. One major Opex risk was considered a very high risk because of the depth of the slurry transport system at one (1) m of cover considered a which could be prone to freezing in that Northern part of Quebec. To counter this risk, 100% back-up power through gensets was provided and this risk has also been mitigated by adding discharge ponds and fresh water supply in two (2) locations.

In summary, during the next phases, the following constructability activities should be considered:

- Incorporate an analysis of operations and maintenance into the facility design;
- Review site conditions and access;
- Design considering construction details and strategy;
- Design quality control;

- Standardize, modularize, coordinate and involve suppliers, consultants and subcontractors;
- Complete the 3D model of the plant and facilities, and
- Refine the scheduling strategy.

26.0 RECOMMENDATIONS

Based on the Feasibility Study schedule, it is recommended to proceed shortly with partial funding for basic engineering for lesser defined areas of the Project. It is also recommended to execute test work and field surveys that are required to support basic and detailed engineering.

Topographical surveys would be best executed prior to the basic engineering activities by one (1) year.

At this stage, there are areas of lesser definition that will be carried on before or during basic engineering. The main areas of lesser definition are:

- Mine dewatering (hydrogeology);
- Waste rock dumps and run-off water management (ARD);
- Tailings pumping, dewatering and material handling;
- Slurry transport hydraulic design due to lack of representative rheology testing;
- Slurry transport heat transfer study;
- Frost depth and earthwork for slurry transport;
- Export terminal earthwork and foundation design;
- Water treatment facility at Pointe-Noire due to lack of filtrate sample analysis.

These elements were engineered based on similar projects or processes but will require confirmation from test work carried on representative samples by laboratories and/or suppliers.

Following the completion of the Feasibility Study, a number of key activities should be undertaken to ensure there is a smooth transition from the study phase to the execution phase of the project development life cycle. These key activities can be summarized into the following:

- Establishment of Client Project Management Team;
- Commencement of field surveys, advanced test work and basic engineering activities;
- Continuation of environmental approval and permitting processes;
- No long lead procurement items are deemed to threaten the schedule at this stage. The EPCM approach was selected where engineering is prioritized in the first phase and the EPCM contractor integrates known technologies into a plant as opposed to integrating technology packages. This presents lesser risks to the schedule. There are however three (3) construction packages that by their nature will need to be tendered early as their duration extends for the full construction period:

- Slurry transport system construction;
- Pelletizing plant;
- Power transmission lines.

Environmental permitting is also key to two (2) of these contracts.

To accelerate the Project an estimated $ 6 M will be spent for completion of the EIS and miscallenous items mostly related with alternative tailing disposal option and confirmation of the depth of frost penetration along the slurry transport route. The expected cost is shown in Table 26.1 below.

Table 26.1 – Next Phase Estimated Costs

Activity	Estimated Costs ($)
Completion of the EIS Report	4,000,000
Study of an Alternative option on Dry Stacking of Tailings	1,200,000
Frost Dept Confirmation along the Slurry Transportation Route	600,000
Miscellaneous	200,000
Estimated Total Costs	**6,000,000**

›urce: NML

26.1 Mining and Geology

- Further geological work is recommended to better define the Iron Formation limit. Additional drilling or trenching in this area will either increase the resource base or allow for the surface infrastructure to be closer to the pit crest;
- It is recommended that measurements be conducted to better estimate the density of the overburden and Menihek Shale as well as the swell factor and moisture content of all material types to allow an optimization of the mine equipment requirement;
- Due to the limited scope of the 2006 hydrogeological study, the mine dewatering calculations need to be verified and updated as required accordingly with the results from the 2012 study;
- A change in the pit slope will either increase or decrease the waste stripping within the pit but not significantly affect the Mineral Reserves. A technical analysis is recommended to determine the final pit wall configuration before mining begins in this area;

- ARD potential of the overburden needs further sampling to support the design of waste rock dumps and to design mine waste rock runoff containment and treatment.

26.2 Process

26.2.1 Concentrator

The primary trade-off study is the validation and feasibility level engineering of an optional process flow sheet that is meant to provide a low silica concentrate at a low Blaine index allowing higher throughput in the pelletizing process. The flow sheet needs to be tested and the plant estimated and compared with the current flow sheet that also requires a validation run. The better of the two (2) can then be used for the plant design. As engineering (and time) progresses, the ability to influence costs, both in capital and operating, diminishes. It is therefore important to carry a structured value engineering exercise early to guide and maintain scope during the detailed engineering phase.

- Topographical survey at mine site to locate plant and infrastructures;
- Geotechnical survey to establish ground bearing capacities, borrow pits location and grounding requirements;
- Optimize the location and refine the construction estimates in view of the geotechnical recommendations following a detailed survey of the site selected and a field topographical survey;
- Update flow sheets and mass balance taking into account the data acquired through additional pilot scale testing;
- Pilot scale confirmation test run to freeze flow sheet and mass balance and to produce test samples for rheology testing (concentrate and tailings), pelletizing tests and supplier testing in all areas of processing.

26.2.2 Slurry Transport System:

- Perform calculations for the settling and plugging analysis to calculate the time that the slurry transport system can be shutdown before it must be purged or re-started in view of the rheology testing;
- Slurry transport heat transfer study to allow for frost mitigation design;
- Define compressed air requirements and heaters at both pump stations to drive a pig train to flush water from the slurry transport system;
- Slurry transport corridor topographic LIDAR survey to allow for optimization of route selection (can only be performed in summer);
- Slurry transport corridor geotechnical survey to assess the ground conditions for earthwork;

- Refine the inventory of crossings to design crossing methods according to soil conditions and environmental constraints;
- Geotechnical survey to determine location, extent and depth of bedrock and the location and feasibility of horizontal directional drilling for stream crossings;
- Permafrost and muskeg survey to determine location, extent and depth of permafrost and muskegs and to assess the thermal conductivities of the various soils along the route.

26.2.3 Pelletizing

a) Topographical and geotechnical surveys at Pointe-Noire

To allow for locating the filtration and pellet plant and the stockpiles over the better soils and to perform civil and concrete design at export terminal.

b) Boiler House

The original study scope of Outotec included Natural Gas. During the course of the Study NML/TS was notified that natural gas would not be available and NML would have to use HFO instead. Steam is the preferred method for heating HFO and for purging/cleaning burner. Therefore, the sizing and installation of a boiler is recommended for the following services:

- HFO tank farm, day tank, furnace HFO pumping and burner purging;
- Steam injection in filter feed preparation tanks to raise slurry temperature;
- Using steam instead of electricity for HVAC. (Outotec HVAC is for an electrical system in their scope);
- HVAC for other buildings such as slurry reception, storage tanks, thickeners, warehouse, shops, admin building, etc.

c) Additive Dosing Flexibility

Consider adding flexibility in additive dosing i.e. possibility of adding additives to the concentrate, on the occasions that this filter cake is necessary to supplement the requirement of the pelletizing capacity. This may mean the installation of additional pump (s) to the two (2) filter feed preparation tanks for pelletizing lines #1 and #2, thus being able to quickly switch slurries (no additives for with additives) to the concentrate for sale filtering system and reversing the filter cake from shed direction to pelletizing lines #1 or #2.

d) Process Gas and Plant Dedusting (Bag Filter vs ESP's)

Bag houses are a challenge to operate and maintain in installations for cleaning of dry gases. The temperature and moisture of the waste gases and plant dedusting will vary considerable, especially during shutdowns/start-ups. Investigate if there is any possibility of reverting to electrostatic precipitators.

26.2.4 Export Terminal

- Refine the design of transfer station to multi-user common discharge point to the Port-of-Sept-Iles shiploading facilities;
- Refine water treatment plant design and adapt point of discharge design to the result of the environmental and social assessments;
- Consultations with future port shipping infrastructure users to refine the access route to the Port of Sept-Iles transfer tower to the shiploader.

26.2.5 Market Study

- Refine the pellet market study with additional information on the sea-borne trade component and Chinese pellet demand.

26.3 Environment

26.3.1 Tailings and Water Management

- Study to select optimal tailings disposal strategy;
- Rheology testing of tailings to allow for tailings pumping and dewatering design;
- Bathymetry of surrounding lakes and streams;
- Conduct environmental assessment and public hearings on the treatment and discharge of treated water in the river stream to confirm treatment requirements and discharge point at Pointe-Noire.

26.4 Opportunities

26.4.1 Pelletizing

a) Product Screening

Consider re-designing the proposed layout, chute arrangement and multi stage [six (6) decks] product screening. It may well be worth the while to look at designing a product screening system, where the bigger pellets can be segregated naturally and/or plowed-off the product conveyor. The remaining portion (after hearth layer has been removed) of the product can possibly be segregated naturally and only the portion containing fines would be screened at five (5) mm. This would reduce the screening area size considerably.

NML should also investigate the possibility of selling their pellet without any screening as it is done by Arcelor Mittal ("**AM**") in Port-Cartier since the start of the plant in 1978. NML double screening of green balls, at the discs and at the feed end of the machine, results likely in better size distribution and fewer fines in the product than AM. This coupled with a segregation bin for the hearth layer would result in a substantial saving in Capex by doing away with the screening houses. It would also do away with the chip handling, stockpiling, reclaiming to the shiploading system.

26.4.2 Export Terminal

- Optimize location of facilities using the results from the detailed topographic and geotechnical surveys.

26.4.3 Financial Analysis

- Refine cash-flow and financial analysis to capture the residual value of the scenarios.

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