March 27, 2013

VIA SEDAR

Autorité des marchés financiers
Alberta Securities Commission
British Columbia Securities Commission
Manitoba Securities Commission
New Brunswick Securities Commission
Nova Scotia Securities Commission
Ontario Securities Commission
Saskatchewan Securities Commission
Securities Commission of Newfoundland and Labrador
Securities Office of Prince Edward Island

Dear Sirs/Mesdames:

Re: Voluntary Filing of Technical Report - Stornoway Diamond Corporation

Stornoway Diamond Corporation (the "**Company**") has filed this day a technical report on SEDAR titled "The Renard Diamond Project, Quebec Canada – Feasibility Study Update" dated February 28, 2013 in which a number of changes to the Renard Project are presented. Throughout 2012, the company undertook a review of the project that culminated in a report titled "The Feasibility Study Optimization Report for the Renard Project". The results of the optimization work are documented in the Feasibility Study Update report.

On January 28, 2013, the Company issued a press release summarizing the results of The Feasibility Study Optimization Report for the Renard Project. The updated study did not trigger the requirement to file a technical report; however the Company has prepared this report to conform to National Instrument 43-101 Standards of Disclosure for Mineral Projects ("**NI 43-101**") to provide the public with updated information on the project. The technical report also includes the results of a bulk sample collected from the R65 kimberlite that were released subsequent to the press release summarizing the Feasibility Study Optimization Report. The results of the R65 bulk sample were disseminated in a press release dated February 26, 2013.

This technical report is accordingly being filed voluntarily by the Company today. Upon the Company subsequently filing a disclosure document that would trigger the filing of a technical report under Section 4.2 (1) of NI 43-101, the Company will comply with the requirements thereof, including the filing of separate consents of qualified persons.

Please contact the undersigned at (604) 983-7762 should you have any question regarding the foregoing or the filing being made.

Yours truly,

(Signed) "Dave Skelton"

_____

Dave Skelton
Vice President, Project Development

The Renard Diamond Project
Québec, Canada

Feasibility Study Update

NI 43-101 Technical Report
February 28, 2013

Qualified Persons
Mr William Bagnell, P. Eng. AMEC Americas Limited
Mr Paul Bedell, P.Eng., Golder Associates Ltd.
Ms Valérie J. Bertrand, Géo., Golder Associates Ltd.
Dr Richard Brummer, P.Eng., Itasca Consulting Canada Inc.
Mr David Farrow, P.Geo., GeoStrat Consulting Services Inc.
Mr Charles Gagnon, Eng., Roscoe Postle Associates
Mr Louis-Pierre Gignac, Eng.,G Mining Services Inc.
Dr Lynton Gormely, P.Eng., AMEC Americas Limited
Mr Martin Magnan, Eng., Roche ltée, Groupe-conseil
Mr Jean-François St-Onge, Eng., SNC-Lavalin Inc.

Effective Date
February 28, 2013

IMPORTANT NOTICE

This report was prepared as a National Instrument 43-101 Technical Report for Stornoway Diamond Corporation (Stornoway). Portions of the Technical Report were authored by Qualified Persons employed by AMEC Americas Limited (AMEC) as identified in those Qualified Persons' Certificates. The quality of information, conclusions, and estimates contained in those sections prepared by AMEC employed Qualified Persons is consistent with the level of effort involved in AMEC' 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 is intended for use subject to the terms and conditions of Stornoway's contract with AMEC. Except for the purposes legislated under provincial securities laws, any other use of this report by any third party is at that party's sole risk.

CERTIFICATE OF QUALIFIED PERSON

Paul Micheau Bedell

To accompany the report entitled "The Renard Diamond Project, Québec, Canada, Feasibility Study Update, NI 43-101 Technical Report" (the "Technical Report"), dated February 28, 2013, I, Paul M. Bedell, P.Eng., do hereby certify that:

1. I am an Associate and Senior Geotechnical Engineer employed at Golder Associates Ltd. located at 500-4260 Still Creek Drive, Burnaby, British Columbia, Canada.
2. Graduated with Bachelor of Engineering Science degree in Civil Engineering and a Master of Engineering Science from The University of Western Ontario in London, Ontario in 1994 and 1997, respectively.
3. I am a registered Professional Engineer in British Columbia (membership number 28511), Ontario (membership number 100026653), Saskatchewan (membership number 11859), and the Northwest Territories and Nunavut (membership number L1746).
4. I have worked as an engineer since my graduation from The University of Western Ontario. For the past 15 years I have been employed with Golder Associates Ltd. During this period I have fulfilled the role of engineer on mining projects directing and completing geotechnical engineering for mine waste management and soils. I currently hold the position of Associate, Senior Geotechnical Engineer.
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 education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purpose of this NI 43-101.
6. I am responsible for Sections 10.1 (2010 and 2011 geotechnical and 2012 hydrogeological), 10.4 (2010 and 2011 geotechnical and 2012 hydrogeological), 16.1, 18.1 (pre-2012, 2012 ditches & monitoring wells)), 18.4, 20.3, 25 (PKC facility, water management and treatment), and 26.3 of the Technical Report.
7. I visited the Renard Diamond Project property from September 29 to October 1, 2010, inclusive.
8. I am independent of Stornoway Diamond Corporation applying the test set out in Section 1.5 of the NI 43-101.
9. I have had no prior involvement with the property that is subject of the Technical Report.
10. As of the date of this certificate, to the best of my knowledge, information, and belief, that Sections 10.1 (2010 and 2011 geotechnical and 2012 hydrogeological), 10.4 (2010 and 2011 geotechnical and 2012 hydrogeological), 16.1, 18.1 (pre-2012, 2012 ditches & monitoring wells), 18.4, 20.3, 25 (PKC facility, water management and treatment), and 26.3 for which I am responsible in this technical report, contain all the scientific and technical information that is required to be disclosed to make this technical report not misleading.

"Signed and sealed", Dated this 28th day of February 2013.

Paul M. Bedell, M.E.Sc., P.Eng.
Associate, Senior Geotechnical Engineer
Golder Associates Ltd.

CERTIFICATE OF QUALIFIED PERSON

Valérie Johanne Bertrand

To accompany the report entitled "The Renard Diamond Project, Québec, Canada, Feasibility Study Update, NI 43-101 Technical Report" (the "Technical Report"), dated February 28, 2013, I, Valérie J. Bertrand, géo., do hereby certify that:

1. I am an Associate and Senior Geochemist employed at Golder Associates Ltd. located at 32 Steacie Drive, Kanata Ontario. K2K 2A9, Canada.
2. I graduated with a Bachelor of Science degree in Geology from the University of Ottawa in Ottawa Ontario in 1991 and have a Master of Applied Science degree in Mining Engineering at the University of British Columbia in Vancouver BC which I obtained in 1999.
3. I am a registered Professional Geoscientist in Ontario (membership number 1458) and a member in good standing of l'Ordre des Géologues du Québec (membership number 1221).
4. I have worked as a geoscientist since my graduation from the University of Ottawa. For the past 13 years I have been employed with Golder Associates Limited. During this period I have fulfilled the role of geochemist on mining projects directing and completing environmental geochemistry investigations on mine wastes, soils and water. I currently hold the position of Associate, Senior Geochemist.
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 education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purpose of this NI 43-101.
6. I am responsible for Section 20.2 of the Technical Report.
7. I visited the Renard Diamond Project property from September 29 to October 1, 2010, inclusive.
8. I am independent of Stornoway Diamond Corporation applying the test set out in Section 1.5 of the NI 43-101.
9. I have had no prior involvement with the property that is subject of the Technical Report.
10. As of the date of this certificate, to the best of my knowledge, information and belief, Sections 20.2, for which I am responsible for in this technical report, contains all the scientific and technical information that is required to be disclosed to make this technical report not misleading.

"Signed and sealed", Dated this 28th day of February, 2013.

Valérie J. Bertrand, géo. M.A.Sc.
Associate, Senior Geochemist
Golder Associates Ltd.

ITASCA
Consulting Canada Inc.

CERTIFICATE OF QUALIFIED PERSON

Richard K. Brummer, PhD, Professional Engineer (Ontario)
Itasca Consulting Canada Incorporated,
166 Douglas Street
Sudbury, Ontario, P3E 1G1
rbrummer@itasca.ca

I, Richard K. Brummer, PhD, Professional Engineer (Ontario), am employed as Principal Geomechanics Engineer with Itasca Consulting Canada Incorporated. This certificate applies to the technical report entitled "The Renard Diamond Project, Québec, Canada, Feasibility Study Update, NI 43-101 Technical Report" dated February 28, 2013.

I am a member and "Designated Consulting Engineer" of Professional Engineers Ontario. I graduated with the degrees of Bachelor of Science in Engineering (cum laude) (1977), Master of Science in Engineering (1980), Bachelor of Commerce (Honours) (1990), all from the University of the Witwatersrand, Johannesburg, South Africa, and Doctor of Engineering (PhD) from the University of Johannesburg (formerly RAU) in 1988.

I have practiced my profession for 35 years. I have been directly involved in the design and evaluation of mining methods for over 100 underground and open pit mines in Africa, North and South America, Europe, Asia and Australia.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101).

I personally visited the Stornoway's Renard site on August. 21, 2012 and engineers under my direct supervision have visited the site at various times.

I am responsible for Section 10.1 (pre-2010, 2010 and 2012 geomechanical, pre-2012 hydrogeology), 10.4 (pre-2010, 2010 and 2012 geomechanical, pre-2012 hydrogeology), Section 16.2, Section 16.3, Section 25 (rock mechanics) and Section 26.1 (rock mechanics) of the technical report titled "The Renard Diamond Project, Québec, Canada, Feasibility Study Update, NI 43-101 Technical Report" and dated February 28, 2013. the ("Technical Report").

I am independent of Les Diamants Stornoway (Canada) Inc. as independence is described by Section 1.5 of NI 43–101.

I have been involved with the Renard Project on behalf of Les Diamants Stornoway (Canada) Inc. since October 2009.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

"Signed and sealed"

Richard K. Brummer, PhD, P.Eng. Dated: February 28, 2013

Certificate of Qualified Person

Certificate of **David Farrow**

I, David Farrow, of Vancouver, British Columbia, Canada, do hereby certify that as an author of this "The Renard Diamond Project, Québec, Canada, Feasibility Study Update, NI 43-101 Technical Report", dated, February 28, 2013, make the following statements:

1) I am a Geologist with GeoStrat Consulting Services Inc. of 40-4055 Indian River Drive, North Vancouver, British Columbia, V7G 2R7, Canada

2) I am a graduate of the University of the Witwatersrand, Johannesburg, South Africa (GDE (Geostatistics) 1998) and the University of Cape Town, Cape Town, South Africa (B.Sc.(Hons) 1982).

3) I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (License # 33860). I am also a member in good standing of The South African Council for Natural Science Professions (License # 400074/87).

4) I have practiced my profession continuously since graduation.

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

6) My relevant experience with respect to Renard Deposits includes over 26 years in exploration, mining geology and grade estimation in Canada and southern Africa.

7) I am responsible for the preparation of sections: 1.2, 1.3, 1.4, 1.7, 4.1, 4.2, 4.3, 4.4, 4.5, 6, 7, 8, 9, 10.1 (exploration), 10.2, 10.3, 10.5, 10.6, 12, 14, 15.4.4 (Indicated Resources), 23 and 25 (Resources) of this technical report titled "The Renard Diamond Project, Québec, Canada, Feasibility Study Update, NI 43-101 Technical Report", dated February 28, 2013.

8) I visited the Property March 5 to March 9, 2009 and September 13, 2012.

9) I was co-author of a previous NI 43-101 Preliminary Assessment report on the property, dated May 5, 2010 and primary author of previous NI 43-101 compliant Mineral Resources for the property, dated January 2010, and December 2010.

10) As of the date of this Certificate, to my knowledge, information and belief, the sections of this Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

11) I am independent of the Issuer as defined by Section 1.5 of the Instrument. I have read National Instrument 43-101 and the sections for which I am responsible in this Technical Report have been prepared in compliance with National Instrument 43-101 and Form 43-101F1.

"Signed and sealed", dated this 28th day of February 2013, at North Vancouver, British Columbia, Canada.

David Farrow, Pr.Sci.Nat. P.Geo.

CERTIFICATE OF QUALIFIED PERSON

I, Louis-Pierre Gignac, Eng., do hereby certify that :

1. I am a consulting mining engineer for G Mining Services Inc., 7900 Boul. Taschereau, Edifice D, Suite 200, Brossard, Québec J4X 1C2;
2. I have graduated from McGill University, Canada with a B.Sc. In Mining Engineering in 1999, and from École Polytechnique de Montréal, Canada with a M.Sc.A. in Industrial Engineering in 2002.
3. I am in good standing as a member of the Order of Engineers of Québec (#132995) and I am a member of the Canadian Institute of Mining.
4. I have worked in the mining industry continuously since my graduation from university.
5. I have read the definition of "qualified person" set out in the National Instrument 43-101 ("NI 43-101") and certify that as a result of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I am the author of sections 1.8 (OP), 1.9, 3.2, 15.1 (OP), 15.2, 15.4.1, 15.4.3 (OP), 15.4.4 (OP), 15.5.1, 16.0 (OP), 16.4, 16.6 (OP), 18.5, 19.1, 22, 25 (OP/financial analysis), and 26.1 (OP) of this Technical Report entitled "The Renard Diamond Project, Québec, Canada, Feasibility Study Update, NI 43-101 Technical Report" dated February 28, 2013;
7. I have visited the site on June 15, 2010;
8. I have no personal knowledge as of the date of this certificate of any material fact or change, which is not reflected in this 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 Stornoway Diamond 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 Stornoway Diamond 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 years from Stornoway Diamond Corporation., or any associated or affiliated companies
12. 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.

"Signed and sealed", dated at Brossard this 28th day of February 2013.

_____

Louis-Pierre Gignac, Eng.
Mining Engineer, G Mining Services Inc.
Brossard, Qc, Canada

CERTIFICATE OF QUALIFIED PERSON

Lynton Gormely, Ph.D., P.Eng.
AMEC Americas Limited
111 Dunsmuir Street, Suite 400
Vancouver, B.C. V6B 5W3
Tel (604) 664-3312 Fax (604) 669-9516
lynton.gormely@amec.com

I, Lynton Gormely, Ph.D., P.Eng., am employed as a Principal Process Engineer with AMEC Americas Limited.

This certificate applies to the technical report entitled "The Renard Diamond Project, Québec, Canada, Feasibility Study Update, NI 43-101 Technical Report" dated February 28, 2013.

I am a member of the Association of Professional Engineers and Geoscientists of British Columbia, registration number 10005. I graduated from the University of British Columbia with a Bachelor of Applied Science degree in 1968 and from the University of British Columbia with a Ph.D. in Chemical Engineering in 1973.

I have practiced my profession for 39 years. I have been directly involved in process engineering design and construction projects for the mining industry for the recovery of base and precious metals. I have experience with the principles of the design of the metallurgical testwork, the design of the process flow sheets, and the selection of the mineral processing equipment, which is relevant to diamond processing.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).

I have not visited the Renard property in northern Québec.

I am responsible for supervising the preparation of Sections 1.5, 11, 13, 17, 25 (processing) and 26.2 of "The Renard Diamond Project, Québec, Canada, Feasibility Study Update, NI 43-101 Technical Report".

I am independent of Stornoway Diamond Corporation as independence is described by Section 1.5 of NI 43–101.

I have had no previous involvement with the Renard Project.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

"Signed and sealed"

Lynton Gormely, Ph.D., P.Eng. Dated: February 28, 2013.

CERTIFICATE OF QUALIFIED PERSON

William Bagnell,P. Eng
AMEC Americas Limited
301 – 121 Research Drive
Saskatoon, Saskatchewan S7N 1K2
william.bagnell@amec.com

I, William Bagnell P. Eng., am employed as Manager, Mining with AMEC Americas Limited.

This certificate applies to the technical report entitled “The Renard Diamond Project, Québec, Canada, Feasibility Study Update, NI 43-101 Technical Report” dated February 28, 2013.

I am a member of The Association of Professional Engineers of Saskatchewan. I graduated from the Technical University of Nova Scotia with a B.Eng., Mining Engineering, in 1996.

I have practiced my profession continuously for sixteen years. I have been directly involved in mine design for Prefeasibility and Feasibility Studies for underground projects. These studies encompass diamond, gold, uranium and potash underground projects located in Canada.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).

I have not visited the Renard Project site.

I am responsible for Sections 1.1, 1.6, 1.8(UG), 1.10, 2.0 to 2.3, 3.0, 3.1, 3.3, 4.6, 5.6, 15.1(UG), 15.3, 15.4.2, 15.4.3(UG), 15.4.4(UG), 15.5.2, 16.0(UG), 16.5, 16.6(UG), 24.2, 25(UG), 26.1, and 27, of “The Renard Diamond Project, Québec, Canada, Updated Feasibility Study, NI 43-101 Technical Report”.

I am independent of Stornoway Diamond Corporation as independence is described by Section 1.5 of NI 43–101.

I have had no previous involvement with the Renard Project.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

“Signed and sealed”

William Bagnell, P. Eng., Dated February 28, 2013.

SNC·LAVALIN

Certificate of Qualified Person

I, Jean-Francois St-Onge eng., do hereby certify that:

1. I am employed as Chief Mining Engineer with SNC-Lavalin Inc Mining & Metallurgy division 1140 Blvd De Maisonneuve W., Montréal Qc., Canada H3A 1M8.

2. I graduated from the University Laval, Québec, Qc, Canada with a B.Sc.A. (Mining) Degree in 1993. I have practiced my profession continuously since then.

3. I am a member in good standing of the OIQ (Ordre des Ingénieurs du Québec - 111717).

4. I have worked on mining operations for over 15 years and in consulting for the last 5 years. I have been involved in various aspects of the design, assessment and development of mine projects over this 20 year period.

5. I am a “qualified person” as that term is defined in National Instrument 43-101 (Standards of Disclosure for Mineral Projects) (the “Instrument”).

6. I have not personally visited the property of The Renard Diamond Project, Québec.

7. I am responsible for Sections 10.1 (2012 R65 cement aggregate and geotechnical infrastructures), 10.4 (2012 R65 cement aggregate and geotechnical infrastructures), Section 18.1 (2012 geotechnical infrastructure) 18.2, 21.0, 24.1, 24.3, 24.4 and related parts of Section 25.0 (Interpretation and Conclusions of the Technical Report) of “The Renard Diamond Project, Québec, Canada, Feasibility Study Update, NI 43-101 Technical Report” dated February 28, 2013 (the “Report”).

8. As at the effective date of the Report, to the best of my knowledge, information and belief, the Sections of the Report stated above in paragraph 7 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 the Instrument and the Sections of the Report stated above in paragraph 7 have been prepared in accordance with the Instrument.

10. I have read and understand the Instrument and I am considered independent of the issuer as defined in Section 1.5.

Dated at Montréal, Qc, Canada, on 28th February 2013.

[signed&sealed]

Jean-François St-Onge eng.
Chief Mining Engineer
SNC-Lavalin Inc.

CERTIFICATE OF QUALIFIED PERSON

I, Martin Magnan, Eng., do hereby certify that:

1. I am currently employed as Project Manager – Environment in the consulting firm:

   Roche ltée, Groupe-Conseil
   3075, ch. des Quatre-Bourgeois
   Bureau 300
   Québec (Québec) G1W 4Y4
   CANADA

2. I graduated from Laval University of Québec, Canada with a B. Sc. A. in Geological Engineering in 1990 and from *Université du Québec à Chicoutimi* of Québec, Canada with a M. Sc. A in Geology in 1994.

3. I am in good standing as a member of the *Ordre des Ingénieurs du Québec* (#126033).

4. I have practiced my profession continuously since my graduation.

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

6. I visited the *Foxtrot Property* on the 16th of November 2011.

7. I am responsible for sections 4.7, 4.8, 4.9, 5.1 to 5.5, 18.3, 19.2, 20.1, 20.4, 20.5, 20.6 sections 25 (environment/permits) and 26 (environment/permits) of this Technical Report entitled "*The Renard Diamond Project, Québec, Canada, Feasibility Study Update, NI 43-101 Technical Report*" dated *February 28, 2013*.

8. As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

9. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

10. I have read national Instrument 43-101 and Form 43-101F1, and aforementioned sections of the Technical Report has been prepared in compliance with that instrument and form.

Prepared in Shawinigan, Québec, February 28, 2013.

"Signed and sealed"

_____

Martin Magnan Eng.

CHARLES GAGNON

I, Charles Gagnon, ing., M.Sc., as an author of this report entitled "The Renard Diamond Project, Québec, Canada, Feasibility Study, NI 43-101 Technical Report" (the "Technical Report"), dated February 28, 2013, do hereby certify that:

1. I am a Mining Engineer with RPA Inc. of Suite 302, 1305 Boulevard Lebourgneuf, Québec, QC G2K 2E4.

2. I am a graduate of Université Laval, Québec, Quebec, in 2002 with a B.Sc. degree in Mining Engineering and Mineral Processing (Cooperative) and in 2005 with a M.Sc. degree in Mining Engineering and Mineral Processing, Mine Ventilation.

3. I am registered as an Engineer in the Province of Quebec (Reg. #130730). I have worked as a mining engineer for a total of 7 years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - Mining ventilation consultant
     - Review and report on mining operations and projects for due diligence and regulatory requirements.
     - Underground ventilation design.
   - Mining Engineer, GENIVAR.

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

5. I have not visited the Renard property in northern Québec.

6. I am responsible for the preparation of Section No. 16.5.9 entitled: "*Ventilation and heating system*" of the Technical Report.

7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.

8. I have had no prior involvement with the property that is the subject of the Technical Report.

9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Ventilation section of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 28th day of February, 2013

(Signed & Sealed) "*Charles Gagnon*"

Charles Gagnon, M.Sc., ing.

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1.0 SUMMARY

1.1 Introduction

On December 29, 2011, Stornoway Diamond Corporation filed a National Instrument 43-101 (NI 43-101) Technical Report presenting a Feasibility Study for the Renard Diamond Project. This NI 43-101 Technical Report presents a summary of the Updated Feasibilty Study for the Renard Diamond Project that was completed in 2012.

SNC-Lavalin Inc., AMEC Americas Limited, Golder Associates Limited , G Mining Services Inc., GeoStrat Consulting Inc., Itasca Consulting Canada Inc., Roscoe Postle Associates Inc.and Roche ltée Groupe-Conseil were commissioned by Stornoway Diamond Corporation (Stornoway) to prepare a Canadian National Instrument 43-101 compliant independent Qualified Person's Review and Technical Report of the Renard Diamond Project, located in Central Québec, Canada.

The Renard Diamond Project (Renard Project or Project) is solely owned and operated by Stornoway Diamonds (Canada) Inc./Les Diamants Stornoway (Canada) Inc. (SDCI), a wholly-owned subsidiary of Stornoway.

This Technical Report describes the Mineral Reserve estimate for the Renard Diamond Project and details the results of the Updated Feasibility Study on the Project.

The Feasibility Study Report includes Mineral Reserve estimates and represents an economically sound, technically robust and environmentally compliant execution plan for the Project.

1.2 Property Location and Site Description

The Foxtrot Property is located approximately 820 km north of the city of Montréal, 120 km south of the all-weather Trans-Taïga highway and 150 km south-southeast of Hydro-Québec's LG-4 electricity generating station in the Otish Mountains area of north-central Québec. Current access to the property is by air. The closest settlement is Témiscamie, on the shore of Lac Albanel. Stornoway currently sources material from the regional centres of Chibougamau and Mistissini, both of which act as staging posts for the trans-shipment of samples and personnel. Power is currently provided to the exploration camp via diesel generators. Hydroelectric station LG-4, operated by provincial power producer Hydro-Québec, could potentially supply electrical power to the site in the future. Power transmission from LG-4 would require installation of a high-voltage transmission line connecting LG-4 to the Project site. In 2011, The Ministère des Transports du Québec (MTQ) initiated a project to extend road 167 from Témiscamie to the Project site. The road construction project is currently a joint effort between the MTQ and Storonway. Construction of the road extension is ongoing.

1.3 Property Ownership

The Renard Diamond Project, referred to as the Renard Project in this document, is situated on the Foxtrot Property in a block of claims known as Foxtrot 2.

The Foxtrot Property comprises three blocks of separate, but essentially contiguous landholdings, known as the Foxtrot 1, Foxtrot 2 and Foxtrot 3 blocks. Foxtrot 1 and Foxtrot 2 consist of 617 individual claims, one mining lease and one surface lease, and Foxtrot 3 contains 303 individual claims. Fifty-eight claims in five blocks belonging to Foxtrot 3, and included in the totals above, lie north of the main block and are non-contiguous. Mining lease BM 1 comprises 144 ha and surface lease 130310000 comprises 199 ha. Both leases are located within Foxtrot 1 and Foxtrot 2.

The claims are registered in the name of SDCI as a 100% interest. At the Effective Date of this report, all claims are reported by Stornoway to be in good standing.

Tenements in the Foxtrot Property area were initially staked in 1996 by a 50:50 joint venture between Ashton Mining of Canada Inc. (Ashton) and SOQUEM Inc. (SOQUEM), with Ashton as operator. Subsequently, the Joint Venture partnership interests were reassigned to Ashton Diamonds (Canada) Inc. and DIAQUEM Inc. (DIAQUEM), a wholly-owned subsidiary of SOQUEM. Stornoway acquired Ashton and Ashton's 50% property interest in January 2007. Stornoway acquired DIAQUEM's 50% interest in April 2011. SDCI currently holds a 100% property interest, subject to a direct royalty on future diamond production of 2% in favour of DIAQUEM. The Renard Project is operated by Stornoway through its wholly-owned subsidiary SDCI.

1.4 Geology and Mineralization

The Project area is located within the eastern portion of the Superior Craton. The Craton is considered to be an amalgamation of small continental fragments of Meso-Archean age and Neo-Archean oceanic plates, with a complex history of aggregation.

There are five known episodes of kimberlitic volcanism in Québec; from south to north, the kimberlite fields are Témiscamingue, Desmaraisville, Otish, Wemindji and Torngat. The Renard Cluster is considered to be part of the Otish kimberlitic volcanic event.

To date, nine kimberlite pipes have been identified over a 2 km^2 area in the Renard Cluster (Renard 1 to Renard 10; Renard 5 and Renard 6 being one kimberlite body, known as Renard 65). The kimberlites are typically spaced between 50 m and 500 m from each other. Whole-rock trace element compositions suggest a closer affinity to Group I kimberlite (olivine-rich, monticellite, serpentine, calcite kimberlites), with derivation from a garnet-bearing mantle. The Lynx and Hibou kimberlite dyke systems are situated to the west of the Renard pipes.

In this report, reference is made to the Renard Core Area, which is defined as a 37 ha area that contains the Renard 2, Renard 3, Renard 4, Renard 65 and Renard 9 kimberlite pipes as well as the Lynx and Hibou dyke systems. Kimberlites are discussed individually in Section 7 of this report.

Geophysical data and drill information from delineation and bulk sampling programs indicate that the Renard pipes are irregular and elliptical in plan view. Surface areas of the kimberlite portion of the pipes range from 0.1 ha to 2.0 ha, and they are situated within a larger halo of fractured country rock.

The pipes comprise root zone to diatreme facies rocks characterized by complex internal geology, with the dominant phase composed of massive volcaniclastic kimberlite classified as “tuffisitic” kimberlitic breccia (TKB). The TKB contains 15% - 90% by volume, fresh to moderately altered granitoid, country rock clasts in a matrix that is generally dominated by serpentinized olivine macrocrysts, carbonates and serpentine. A minor amount of coherent “hypabyssal” kimberlite (HK) with a small proportion of highly altered and digested country rock xenoliths is also present. These are considered to be later stage intrusions that occur throughout the pipe-like bodies, on the periphery or as standalone dyke systems. This material is characterized by abundant, disseminated calcite, olivine macrocrysts and less than 15% by volume crustal xenoliths. Extensive sampling programs conducted between 2001 and 2010 have demonstrated that the pipes and dykes are diamondiferous.

1.5 Mineral Processing and Metallurgical Testing

AMEC Americas Limited (AMEC) developed the process flow sheet and designed the process plant. The nameplate capacity of the process plant will be 2,160,000 tpa (dry solids basis) and the plant operating time will be 6,848 hours per year, which results in a head feed of 315 tph or 6,000 tpd.

The plant is designed to liberate, concentrate and recover diamonds in the size range 30 mm to 1 mm. Diamond liberation is accomplished through a three stage crushing process utilizing a primary jaw crusher, a secondary cone crusher and a tertiary high pressure grinding roll (HPGR). After crushing, the ore is washed and sized in preparation for the concentration process.

The concentration process utilizes Dense Medium Separation (DMS) to separate heavy minerals, including diamonds, from light minerals. The heavy minerals are processed in a secure recovery plant while the light minerals are sized to facilitate recirculation of the coarser fraction through the HPGR to release locked diamonds.

The recovery plant utilizes x-ray and grease processes to achieve high recovery efficiency. Recovered diamonds will be prepared for valuation in a purpose built facility within the recovery plant.

Plant rejects are dewatered and trucked to the Processed Kimberlite Containment (PKC) facility for disposal.

Process plant design and equipment selections were based on bulk sample plant operational data, test work, experience from other operations and proven technology.

1.6 Status of Project Development

The Project has advanced to the stage where NI 43-101 compliant Mineral Reserves have been prepared, founded on the mining, processing and infrastructure designs as detailed in this report.

The Project Execution Plan proposes concepts and practices that are consistent with those in the Canadian diamond mining industry. The Project design includes open pit mining of the Renard 2 and Renard 3 kimberlite pipes, followed by underground mining of the Renard 2, Renard 3 and Renard 4 kimberlite pipes by the blast hole shrinkage method. The mine plans propose the extraction of 17.0 million tonnes of waste and 23.8 million tonnes of ore over an 11.5-year period utilizing standard equipment and mine designs. Ore will be fed to a 2.16 Mt/a processing plant with three stages of crushing, DMS, and X-ray/grease diamond recovery circuits. Process plant designs and equipment selections utilize proven suppliers and are based on experience from other operations. Security designs are industry standard practices.

Supporting infrastructure includes a 18 MW packaged diesel power plant, 1,490 m gravel airstrip, five-bay truckshop, mine offices, emulsion plant, fuel tank farm, 250 bed accommodation complex and effluent treatment plant.

Cost estimates were built from first principles and benchmarked against relevant operations. Appropriate contingencies have been applied. Risk management and peer reviews were held during the Feasibility Update process.

The Project is environmentally sound and has one of the smallest land footprints of any diamond mine in Canada. Dry stacking of processed kimberlite is planned to dispose of plant rejects and management of all surface runoff water will be facilitated through a system of peripheral drainage ditches designed to collect all site runoff water. The ditches will direct runoff water to an excavated catch basin for treatment, if required, before release to the environment. Suspended solids were determined to be the main concern for water treatment. Storage capacity of the catch basin will accommodate spring runoff and a 100 year return storm event. Mine operations incorporate a program of progressive reclamation that minimizes costs and allows timely monitoring of performance. The waste rock generated by the mine will be re-introduced underground for support.

Risks for the Project have been identified and are considered reasonable for the Project's development status. The greatest economic risk/opportunity is related to the timely availability of road access.

1.7 Mineral Resources

Stornoway retained GeoStrat Consulting Services Inc., (GeoStrat) to provide an independent Mineral Resource Estimate for the Renard Project. Following the final summer/fall 2010 drilling program, an updated Mineral Resource Estimate was completed in accordance with the CIM Mineral Resource and Mineral Reserve definitions referred to in NI 43-101, Standards of Disclosure for Mineral Projects and reported on January 24, 2011 for the Renard 2, Renard 3, Renard 4, Renard 9 and Renard 65 kimberlite pipes as well as the Lynx and Hibou kimberlite dyke systems. The full technical report, entitled "*2010 Mineral Resource Update for the Renard Diamond Project*" is available on SEDAR at www.sedar.com.

The 2010 Mineral Resource Estimate comprised the integration of kimberlite volumes, density, petrology and diamond content-data obtained from 88,585 m of diamond drilling, 6,151 m of reverse circulation (RC) drilling, 16.4 t of samples submitted for microdiamond analysis, 600.8 cts of diamonds (6,457 stones) recovered from reverse circulation drilling and 8,611.6 cts of

diamonds (84,381 stones) recovered from surface trenching and underground bulk sampling. Results are tabulated in Table 1.1 and Table 1.2. The Mineral Resource Estimate is based on the continuity of geology between kimberlite at depth and kimberlite nearer surface, and the generally low variation in sample results for the different kimberlite phases with depth.

Table 1.1: January 2011 Indicated Mineral Resources Renard Diamond Project

Deposit	Total Tonnes	Total Carats	Average cpht	Average Dilution %
Renard 2	17,632,000	18,092,000	102.6	37.4
Renard 3	1,751,000	1,849,000	105.6	36.8
Renard 4	7,246,000	3,813,000	52.6	49.1
Renard 9	0	0	0	0
Lynx	0	0	0	na
Hibou	0	0	0	na
Total	26,629,000	23,755,000	89.2	na

Note: Totals may not equal the sum of the individuals due to rounding.

Table 1.2: January 2011 Inferred Mineral Resources Renard Diamond Project

Inferred				
Deposit	**Total Tonnes**	**Total Carats**	**Average cpht**	**Average Dilution %**
Renard 2	5,208,000	6,142,000	117.9	29.1
Renard 3	542,000	637,000	117.6	39.2
Renard 4	4,756,000	2,087,000	43.9	55.5
Renard 9	5,700,000	2,686,000	47.1	65.4
Renard 65	12,935,000	3,722,000	28.8	48.7
Lynx	1,798,000	1,924,000	107	na
Hibou	178,000	256,000	144	na
Total	31,116,000	17,454,000	56.1	na

Note: Totals may not equal the sum of the individuals due to rounding.

There is additional potential for the Project, as the geological models for Renard 2, Renard 3, Renard 4, Renard 65, and Renard 9 are based on conservative geometries for the kimberlites at depth, and the evaluation models do not incorporate areas of limited drilling at depth. The "target for further exploration" (TFFE) - previously known as "potential mineral deposit" (PMD) before the June 30, 2011 revisions to NI 43-101 - are detailed in Table 1.3. Total TFFE was identified as representing between 55 and 76 million tonnes, containing between 24 and 49 million carats of diamonds, at an average grade of 43 to 64 cpht. These were defined on a basis of geological-modelling, outcrop mapping, limited delineation drilling and surface sampling. The

potential quantity and grade of any TFFE is conceptual in nature, there is insufficient exploration to define a Mineral Resource and it is uncertain if further exploration will result in the target being delineated as a Mineral Resource.

Table 1.3: January 2011 Target For Further Exploration - Renard Diamond Project

Low Range				High Range		
Deposit	**Total Tonnes**	**Total Carats**	**Average cpht**	**Total Tonnes**	**Total Carats**	**Average cpht**
Renard 2	4,012,000	4,132,000	103	4,586,000	8,621,000	188
Renard 3	771,000	825,000	107	1,644,000	2,762,000	168
Renard 4	11,077,000	4,209,000	38	15,300,000	12,087,000	79
Renard 9	3,858,000	1,736,000	45	6,327,000	3,163,000	50
Renard 65	29,517,000	6,789,000	23	41,591,000	13,725,000	33
Lynx	3,089,000	2,966,000	96	3,199,000	3,839,000	120
Hibou	2,745,000	2,855,000	104	2,853,000	4,309,000	151
Total	55,069,000	23,512,000	43	75,500,000	48,506,000	64

Note: Totals may not equal the sum of the individuals due to rounding. Previously known as Potential Mineral Deposit prior to June 30, 2011 changes to NI43-101 legislation.

1.8 Mineral Reserves

A mine plan has been developed to extract the Indicated Mineral Resources of the Renard Project. The Renard 2 and Renard 3 kimberlite pipes will be mined through a combination of open pit and underground mining methods while the Renard 4 pipe will be mined by underground methods only. The mine design and estimate of the Mineral Reserves for the open pit was completed by G Mining Services Inc. (GMining) and for the underground by AMEC. Open pit and underground Mineral Reserves were estimated independently based on criteria specific to each method. For both underground and open pit methods, a detailed mine plan was developed and dilution and recovery modifying factors applied. All Mineral Reserves are classified as Probable Mineral Reserves.

The combined Mineral Reserves for the Project, summarized by mining method and pipe, are shown in Table 1.4.

Table 1.4: Mineral Reserve Summary – Open Pit and Underground

Pipe	Mineral Reserve (t x 1000)	Grade (cpht)	Carats (c x 1000)	% of Total Tonnes	% of Total Carats
Open Pit					
R2	1,313	94.6	1,242	5.5%	6.9%
R3	723	92.8	671	3.0%	3.7%
Sub Total	2,035	94.0	1,913	8.6%	10.7%
Underground					
R2	1617,029	80.0	13,616	71.6%	75.9%
R3	1,003	84.1	843	4.2%	4.7%
R4	3,718	42.4	1,575	15.6%	8.8%
Sub Total	21,750	73.7	16,034	91.4%	89.3%
Total	23,785	75.5	17,947	100.0%	100.0%

1.9 Financial Analysis

GMining completed a financial analysis for the Renard Project. The Post-Tax Net Present Value discounted at 7% (NPV 7%) is 392 million and $684 million Pre-Tax based on de-escalated nominal dollars. The internal rate of return is 16.3% on an after-tax basis and 20.3% on a pre-tax basis based on de-escalated nominal dollar cash flows. The payback period from the end of the pre-production period in December 2015 is 5.0 years on an after-tax basis and 4.9 years on a pre-tax basis.

The Project returns are more sensitive to the diamond price assumption, followed by capital costs and finally by operating costs.

1.10 Conclusions and Recommendations

Feasibility Study engineering designs are supported by scientific and technical data of sufficient quality and detail to support detailed engineering work, should Stornoway make the decision to advance the Project to production. The authors of the report recommend to Stornoway that financial, equity and commodity markets be observed to establish when, and under what conditions such a decision would be appropriate.

2.0 INTRODUCTION

In 2011, Stornoway Diamond Corporation (Stornoway) engaged SNC-Lavalin Inc. (SLI) as the Lead Consultant to complete an independent Feasibility Study on the 100% owned Renard Project in Québec. SLI was selected on the basis of its demonstrated expertise in large, remote infrastructure projects, cost estimation, scheduling, risk management and project management skills. The results of the Feasibility Study were disclosed in a press release dated November 16, 2011. A NI-43-1011 technical report in support of the November 16, 2011 disclosure was filed on January 3, 2012.

SLI retained AMEC Americas Limited (AMEC) for their diamond process and mass mining expertise based on having successfully designed and executed several recent diamond projects in northern Canada. .

A number of scope elements were prepared by additional consultants including:

- Resource estimation - GeoStrat Consulting Inc.;
- Open pit design and financial analysis – G Mining Services Inc.;
- Rock mechanics and hydrogeology with Itasca Consulting Canada Inc.;
- Geochemical characterization, geotechnical investigations, geotechnical recommendations for infrastructure, open pit overburden stability analysis, processed kimberlite containment facility design, water management, and water treatment - Golder Associates Ltd.; and
- Environmental and social considerations and permitting – Roche Groupe-Conseil Ltée.

SLI integrated conceptual design elements from all the authors in order to prepare the capital cost estimate, the operational cost estimate, the schedule and the Feasibility Study report. An important component of the study, which involved all study consultants, was the Risk Assessment managed by SLI.

Genivar Inc. was not retained by Stornoway, however Genivar Inc. agreed to provide independent commentary on the Route 167 Extension Project which is under the responsibility of the Ministère des Transports du Quèbec.

In 2012, Stornoway completed a review of the Renard Project to optimize the mine plan. The resulting report was titled “Renard Diamond Project - Feasibility Study Optimization Report”. In addition to the list of consultants involved in the 2011 Feasibility Study, Roscoe Postle Associates Inc was retained to prepare the ventilation requirements for the updated mine plan. With respect to the Route 167 Extension Project, Genivar’s commentary in the previous report is now the responsibility of Roche Goupe-Conseil Ltée.

On January 28, 2013, a press release was issued summarising the results of the Feasibility Study Optimization Report for the Renard Project. The updated study did not trigger the requirement to file a technical report, however the report herein was prepared to conform to the National Instrument 43-101 Standards of Disclosure for Mineral Projects to provide the public with updated information on the project. The technical report also includes the results of a bulk sample collected from the R65 kimberlite that were released subsequent to the press release

summarizing the Feasibility Study Optimization Report. The results of the R65 bulk sample were disseminated in a press release dated February 26, 2013.

This report is being filed voluntarily effective Februay 28, 2013. Upon Stornoway subsequently filing a disclosure document that would trigger the obligation to file a technical report, the Company will file separate consents of qualified persons in compliance with NI 43-101.

The general list of abbreviations is found in Table 2.1. Lithological abbreviations are found in Table 7.2.

Table 2.1: List of Abbreviations

Abbreviation	Description
$	Canadian Dollars
US$	United States Dollars
AMEC	AMEC Americas Limited
Ashton	Ashton Mining Canada Inc.
BHS	Blasthole shrinkage
BHSP	Blasthole shrinkage with pillars
ca	Circa
Canadian Environmental Assessment Act or CEAA	*Canadian Environmental Assessment Act* (S.C., c.37)
Capex	Capital Cost estimate
CCR	Cracked country rock
CIM	Canadian Institute of Mining and Metallurgy
CK	Coherent Kimberlite
cm	Centimetre
COMEV or Evaluating Committee	Evaluating Committee of the Ministère du Développement durable, de l'Environnement et des Parcs
Comex or Provincial Review Committee	Review Committee of the Ministère du Développement durable, de l'Environnement et des Parcs
cpht	Carats per Hundred Tonnes
cpt	Carats per Tonne
CR	Country rock
CRB	Country rock breccia
cts	Carats
DFO or Department of Fisheries and Oceans	Department of Fisheries and Oceans (Canada)
DIAQUEM	DIAQUEM Inc.
DICAN	Diamonds International Canada
DMS	Dense media separation
Engineering Procurement or CEAA	Engineering Procurement Construction and Management
Environment Quality Act or EQA	*Environment Quality Act* (RSQ c. Q-2)
ESIA	*Environmental and Social Impact Assessment*

Explosives Act	*Explosives Act* (R.S.C., 1985 c. E-17)
Fisheries Act	*Fisheries Act* (R.S.C., 1985 c. F-14)
Foxtrot Property	Foxtrot 1, Foxtrot 2 and Foxtrot 3 blocks
g	Gram
Ga	Giga Annun (Billion Years)
GCC or GCC-EI or Grand Council of the Crees	Grand Council of the Cree of Québec (Eeyou Istchee)
GDP	Gross Domestic Product
GeoStrat	GeoStrat Consulting Inc.
GMining	G Mining Services Inc.
GN	Fresh, competent, fine grained, banded, quartz-feldspar-biotite gneiss
Golder	Golder Associates Ltd.
GR	Fresh, competent, equigranular granite
ha	Hectare
HADD	Harmful alteration, disruption or destruction of fish habitat
HK	Hypabyssal Kimberlite
HKt	Transitional Hypabyssal Kimberlite
HPGR	High pressure grinding roll
Hz	Hertz
ISRM	International Society for Rock Mechanics
Itasca	Itasca Consulting Canada Inc.
ITH	In-the-hole
JBNQA	James Bay and Northern Québec Agreement 1975
kg	Kilogram
km	Kilometre
km^2	Square Kilometre
l	Litre
kV	Kilovolt
Lakefield	SGS Lakefield Research Ltd
LOM	Life of mine
m	Metre
m^2	Square Metre
m^3	Cubic Metre
Ma	Mega Annun (Million Years)

Majescor	Majescor Resources Inc.
masl	Metres Above Sea Level
MBJ	Municipality of James Bay
MDDEP	Ministère du Développement durable, de l'Environnement et des Parcs
mm	Millimetre
MRNF	Ministère des Ressources naturelles et de la Faune
Mt	Million Tonnes
Mt/a	Million Tonnes per Annum
MTQ	Ministère des Transports du Québec
MVK	Massive volcaniclastic Kimberlite
MW	Megawatts
NI	National Instrument
Norton Rose	Norton Rose OR LLP
NPV	Net Present Value
NRCAN	National Resources Canada
NTS	National Topographic Sheet
OK	Ordinary Kriging
Opex	Operating Cost estimate
PDA	Pre-Development Agreement
PEA	Preliminary Economic Assessment
PEG	Fresh, competent, equigranular negratite
PEM	Mining Exploration Licences
PK	Processed Kimberlite
PKC	Processed Kimberlite Containment
PMD	Potential Mineral Deposit
QC	Quality Control
QPs	Qualified Persons
Québec Explosives Act	*An Act respecting explosives (Québec)* (RSQ, c. E-22)
Québec Land use Act	*An Act respecting Land use planning and development* (RSQ, c. A-19.1)
Québec Mining Act or Mining Act	*Mining Act (Québec)* (RSQ, c. M-13.1)
RC	Reverse Circulation

Regulation respecting mineral substances	*Regulation respecting mineral substances other than petroleum, natural gas and brine* (R.Q., c. M-13.1, r.2)
Regulation respecting the application of the Environment Quality Act	*Regulation respecting the application of the Environment Quality Act* (c. Q-2, r.3)
Renard Project or Project	Renard Diamond Project
Roche	Roche ltée, Groupe-Conseil
ROM	Run-of-mine
Saskatchewan Research Council	Saskatchewan Research Council Geoanalytical Laboratories
SDCI	Stornoway Diamonds (Canada) Inc./Les Diamants Stornoway (Canada) Inc.
SNC-Lavalin or SLI	SNC-Lavalin Inc.
SOQUEM	SOQUEM Inc.
spt	Stones per Tonne
Stornoway	Stornoway Diamond Corporation
SWRPA	Scott Wilson Roscoe Postle Associates Inc.
t	Metric Tonne
TFFE	Target for further exploration
Thunder Bay Laboratory	Thunder Bay Mineral Processing Laboratory
TK	Volcaniclastic Kimberlite
TKB	Tuffisitic Kimberlitic breccia
TKt	Transitional Volcaniclastic Kimberlite
tpa	Metric Tonne per Annum
tpd	Metric Tonne per Day
tph	Metric Tonne per Hour
WWW	WWW International Diamond Consultants
V	Volts
yr	Year

2.1 Qualified Persons and Site Visits

Qualified Persons contributing to this report are listed in Table 2.2.

Table 2.2: Areas of Responsibility

Name of QP	Company	Site Visit	QP Sections
William Bagnell, P.Eng.	AMEC Americas Limited	No site visit	Section 1.1, 1.6, 1.8 (UG), 1.10 Section 2.0 to 2.3 Section 3.0, 3.1, 3.3 Section 4.6, 5.6 Section 15.1(UG),15.3, 15.4.2, 15.4.3 (UG) 15.4.4(UG) and 15.5.2 Section 16.0 (UG), 16.5, 16.6 (UG) Section 24.2 Section 25 (UG), 26.1 Section 27
Lynton Gormely, P.Eng.	AMEC Americas Limited	No site visit	Section 1.5 Section 11, 13, 17 Section 25 (processing) Section 26.2
David Farrow, P.Geo	GeoStrat Consulting Services Inc.	March 5-9, 2009 and Sept 13, 2012	Section 1.2, 1.3, 1.4, 1.7 Section 4.1 to 4.5 Section 6, 7, 8 and 9 Section 10.1 (exploration), 10.2, 10.3, 10.5, 10.6 Section 12, 14 Section 15.4.4 (Indicated Resources) Section 23 Section 25 (Resources)
Louis-Pierre Gignac, Eng.	G Mining Services Inc.	June 15, 2010	Section 1.8 (OP), 1.9 Section 3.2 Section 15.1(OP),15.2, 15.4.1, 15.4.3 (OP), 15.4.4 (OP) and 15.5.1 Section 16.0 (OP),16.4, 16.6 (OP) Section 18.5 Section 19.1 Section 22 Section 25 (OP/financial analysis) Section 26.1 (OP)
Charles Gagnon, Eng.	Roscoe Postle Associates	No site visit	Section.16.5.9

Name of QP	Company	Site Visit	QP Sections
Paul Bedell, P.Eng.	Golder Associates Ltd.	Sept. 29-Oct. 1, 2010	Section 10.1 (2010-2011 geotechnical and 2012 hydrogeological) Section 10.4 (2010-2011 geotechnical and 2012 hydrogeological) Section 16.1 Section 18.1 (pre-2012, 2012 ditches and monitoring wells) Section 18.4 Section 20.3 Section 25 (PKC facility, water management & treatment) Section 26.3
Valérie Bertrand, Géo.	Golder Associates Ltd.	Sept. 29-Oct. 1, 2010	Section 20.2
Richard Brummer, P.Eng.	Itasca Consulting Canada Inc.	Aug 21, 2012	Section 10.1 (pre-2010, 2010 and 2012 geomechanical, pre-2012 hydrogeology) Section 10.4 (pre-2010, 2010 and 2012 geomechanical, pre-2012 hydrogeology) Section 16.2, 16.3 Section 25 (rock mechanics) Section 26.1 (rock mechanics)
Martin Magnan, Eng.	Roche Ltée, Groupe-conseil	Nov. 16, 2011	Section 4.7 to 4.9 Section 5.1 to 5.5 Section 18.3 Section 19.2 Section 20.1, 20.4, 20.5, 20.6 Section 25 (environment/permits) Section 26.4 (environment/permits)
Jean-François St-Onge, P.Eng.	SNC-Lavalin Inc.	No site visit	Section 10.1 (2012 R65 cement aggregate and geotechnical infrastructure) Section 10.4 (2012 R65 cement aggregate and geotechnical infrastructure) Section 18.1 (2012 geotechnical infrastructure) Sections18.2 and 21 Section 24.1, 24.3, 24.4 Section 25

The information in Sections 1.5, 11, 13, 17 and 25 pertaining to metallurgy and processing was prepared by AMEC process specialist Harry Ryans. Lynton Gormely is an AMEC Principal Process Engineer with relevant experience in the design of metallurgical testwork, the design of the process flow sheets, the selection of the mineral processing equipment, etc., and is the Qualified Person taking responsibility for these Sections of this report. As such, Mr Gormely is relying on the experience and knowledge of diamond processing provided by Mr. Ryans, and on Mr. Ryans' visits to the site.

2.2 Previous Technical Reports and References

Bedell, P., Bertrand, V., Brummer, R., Farrow, D.F., Gignac, LP., Gormely L., Kroon, A., Magnan, M, Taylor, G.,Therrien, P.(2011):The Renard Diamond Project, Québec, Canada, Feasibility Study, NI 43-101 Technical Report December 29, 2011.

Farrow, D.F. (2010): 2009 Mineral Resource Update, Renard Diamond Project, Northern Québec, Canada. Prepared for Stornoway Diamond Corporation by Golder Associates. Available at www.Sedar.com.

Farrow, D.F. and Farrow, D.G. (2011): 2010 Mineral Resource Update for the Renard Diamond Project. Prepared for Stornoway Diamond Corporation by GeoStrat Consulting Services Inc. Available at www.Sedar.com.

Lecuyer, N.L., Roscoe, W.E., Cullen, R., Kozak, A. And Wiatzka, G. 2009. Technical report on the Preliminary Assessment of the Renard Project, Québec, Canada. NI-43101 Report. Published by Scott Wilson Roscoe Postle Associates Inc. Effective Date: December 12, 2008). Available at www.Sedar.com

Lecuyer, N.L., Roscoe, W.E., Farrow, D., L'Ecuyer, M., Kozak, A., 2010. Updated Technical Report on the Preliminary Assessment of the Renard Project, Québec, Canada. Published by Scott Wilson Roscoe Postle Associates Inc. Available at www.Sedar.com

2.3 Effective Date

The Effective Date of the report is February 28, 2013.

3.0 RELIANCE ON OTHER EXPERTS

The Qualified Persons (QPs) involved in this report are QPs for the sections identified in the certificates of QPs. The QPs have relied on expert opinions and information pertaining to mineral tenure, surface rights, taxation and diamond pricing.

3.1 Mineral Tenure and Surface Rights

The QPs have not reviewed nor independently verified, and therefore disclaim responsibility for information on the legal status or ownership of the Foxtrot Property. In support of Section 4.4 (Mineral Exploration Licenses and Claims), GeoStrat Consulting Inc. (Geostrat) and Amec Americas Limited (AMEC) have fully relied on Stornoway Diamond Corporation (Stornoway)-appointed law firm Norton Rose OR LLP (Norton Rose) to provide a legal opinion for information on the claims within the Foxtrot 1, Foxtrot 2 and Foxtrot 3 properties. The Foxtrot 2 claim block hosts the Mineral Resources and Mineral Reserves referred to in this report. On the basis of Norton Rose's legal opinion dated February 13, 2013, subject to the qualifications thereunder, Stornoway Diamonds (Canada) Inc./Les Diamants Stornoway (Canada) Inc. (SDCI), a wholly-owned subsidiary of Stornoway, is, according to the Public Register (as defined in such opinion), the registered titleholder of a 100% interest in the claims listed in such opinion and identified in Section 4.4 of this report.

Morever, AMEC has fully relied on, and disclaims responsibility for the legal opinion provided by Norton Rose dated February 13, 2013 with respect to Sections 4.6 and 5.6 (Surface Rights and Sufficiency of Surface Rights).

3.2 Taxation

In support of Section 22 (Economic Analysis), G Mining Services Inc. (GMining) has relied on Stornoway and its external advisors to provide guidance on taxes, royalties, and other government levies or interests, applicable to revenue or income from the Renard Project.

3.3 Diamond Valuation

Geostrat, AMEC and GMining have fully relied on WWW International Diamond Consultants (WWW) and disclaim responsibility for diamond price information presented in Sections 14.6 (Diamond Price Estimates), 19.1(Diamond Pricing and Market Studies) and 22 (Economic Analysis). It is reasonable for the Qualified Persons to rely on WWW for this information because WWW is an internationally recognized independent diamond valuation and advisory service to diamond mining and exploration companies, governments of diamond producing countries and private diamond companies. WWW, through Diamonds International Canada (DICAN) Ltd, serves as the valuator for the Federal Government of Canada and the Ontario Government.

In May 2011, WWW conducted a full diamond valuation of the Renard Project diamonds documented in a report entitled “Valuation and Modelling of the Average Price of Diamonds from the Renard Kimberlite Cluster - May 2011”. In October 2011, WWW provided a marketing report titled “Rough Diamond Supply and Demand Forecast 2011 to 2025 Plus Cost

Comparison". In the 2011 Feasibility Study, the diamond price information was used in the Mineral Resource/Mineral Reserve estimates and the financial analysis, and their diamond supply and demand forecasts were referenced in the Market Studies section. The diamond price assumptions used in the 2011 Feasibility Study were maintained in the Updated Feasibility Study.

The diamond price information could not be verified by the QPs due to the proprietary nature of the diamond price book used for the valuation. Risk exists that the diamond price values obtained from WWW may differ from those achieved during commercial production since the valuation was based on an exploration sized sample.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Foxtrot Property is situated in the Monts Otish (Otish Mountains) region of the province of Québec, Canada, 820 km north of the city of Montréal and 360 km north-northeast of the mining town of Chibougamau (Figure 4.1). The property lies 120 km south of the all-weather Trans-Taïga highway and 150 km south-southeast of Hydro-Québec's LG-4 electricity generating station.

The closest settlement is Témiscamie, on Lac Albanel approximately 210 km to the south, which is connected by all-weather provincial road #167 to the Cree Community of Mistissini.

Much of the property is encompassed within National Topographic Sheet (NTS) 033A16, although there are also smaller portions in NTS 033A09 and 033H01 (Figure 4.2). Project centroids are at approximately 52°49'N and 72°12'W.

There are currently no active mining operations in the immediate vicinity of the Foxtrot Property, but there are numerous active exploration projects primarily to the south and west.

4.2 Tenure History

Stornoway Diamond Corporation (Storonway or SDCI) was created in 2003 as a result of a plan of arrangement between Northern Empire Minerals Ltd. and Stornoway Ventures Ltd. Exploration leading to the acquisition of the Foxtrot Property commenced in 1996 under a 50:50 joint venture agreement dated March 14, 1996 (and subsequent amendments) between Ashton Mining of Canada Inc. (Ashton) and SOQUEM Inc. (SOQUEM), with Ashton as operator. The purpose of the initial agreement was to explore for diamonds within an area of mutual interest (AMI) in the Ungava Region of Québec.

Subsequently, the joint venture partnership was reassigned to Ashton Diamonds (Canada) Inc. and DIAQUEM Inc. (DIAQUEM), a wholly-owned subsidiary of SOQUEM. Stornoway acquired Ashton and Ashton's 50% property interest in January 2007. On December 14, 2010, Stornoway reported that it had entered into an agreement with DIAQUEM for the acquisition of DIAQUEM's 50% interest in the Renard Project. This acquisition was completed on April 01, 2011. Stornoway Diamonds (Canada) Inc., a wholly-owned subsidiary of Stornoway, now owns 100% of the Renard Project, subject to a 2% direct royalty interest on future diamond production in favour of DIAQUEM.

When initially granted through online staking activities in 2000 and 2001, the Property consisted of two Mining Exploration Licences, PEM 1555 and PEM 1556, and 2,121 claims. Since that time, claims were allowed to lapse in 2007 (500 claims), 2009 (1,161 claims) and 2011 (157 claims).

In June of 2010, in preparation for the expiration of the Mining Exploration Licenses at the end of each of their respective final terms on August 28, 2010, 744 mineral claims were staked via map designation over PEM 1555 and the northern portion of PEM 1556.

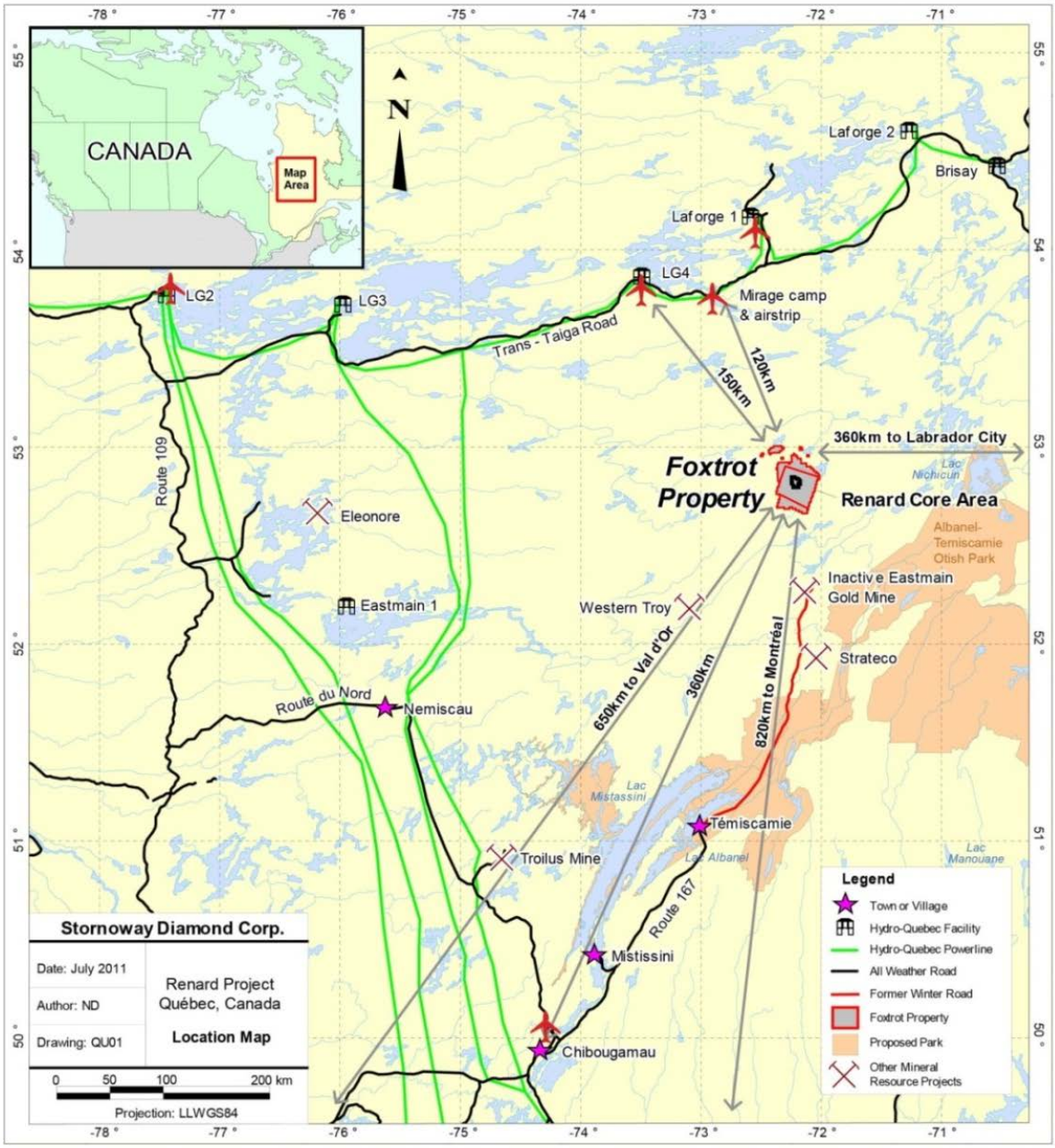

Figure 4-1: Location Map

Figure 4.1: Location Map

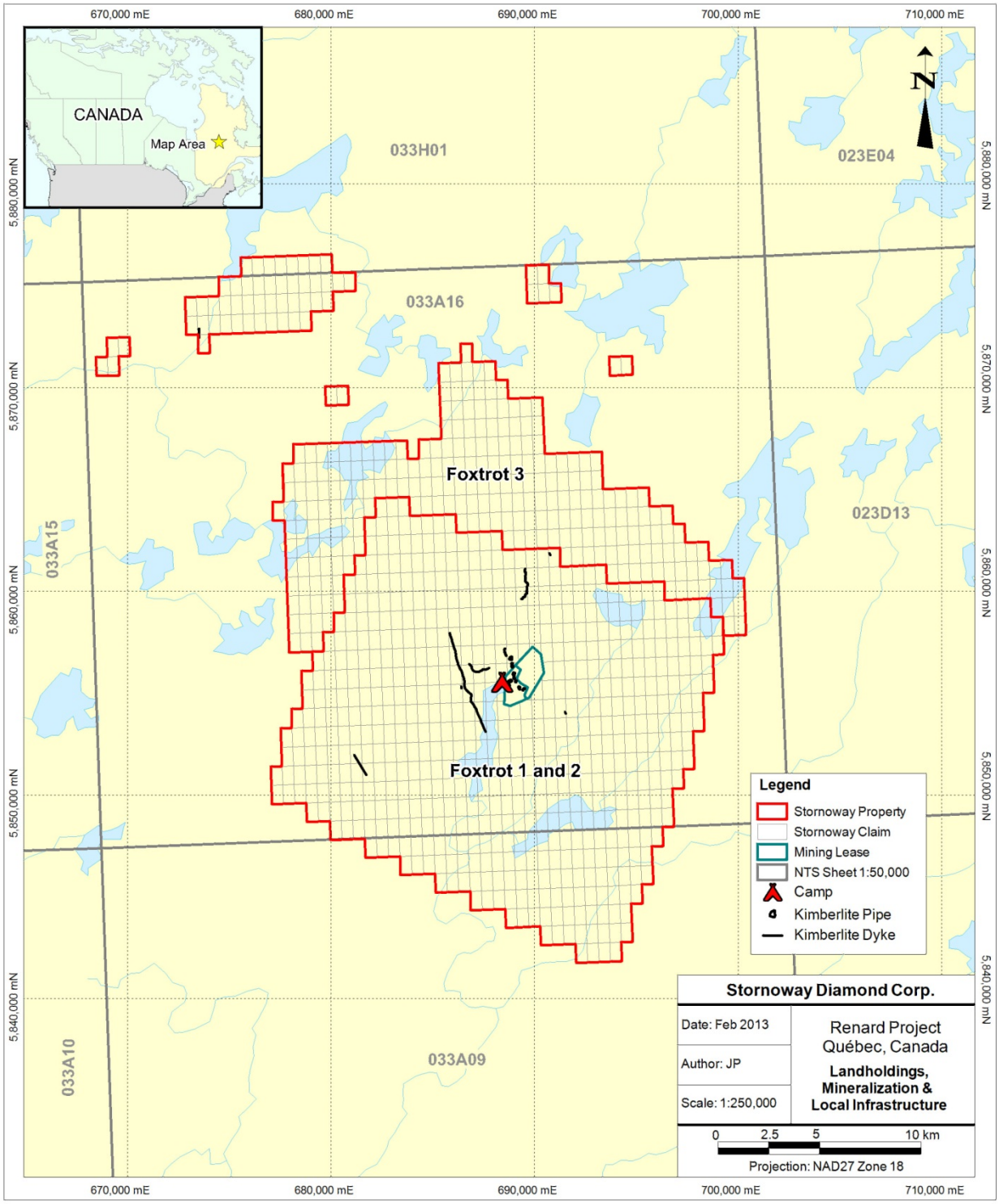

Figure 4.2: Landholdings, Mineralization and Local Infrastructure

During the map designation process 51 already existing partial cells from the Foxtrot 3 block which bordered PEM 1555 were extended to full sized cells.

The 744 claims staked over PEM 1555 and part of PEM 1556 in June of 2010 all came to the end of their first, two year terms in August of 2012. At that time 127 claims were allowed to lapse on their respective anniversary dates and 617 were renewed for an additional 2 year term, keeping them in good standing until August of 2014.

In Ovtober of 2012, mining lease BM 1021 and surface lease 1303 10 000 were granted to Stornoway.

The current landholdings (Figure 4.2) are registered in the name of Stornoway as to a 100% interest. The Renard kimberlite pipes and the Lynx and Hibou dykes, are all located well within the boundaries of the contiguous block of 744 mineral claims that comprise the Foxtrot 1 and Foxtrot 2 claim blocks.

4.3 Mineral Tenure in Québec

Mining Exploration Licences are issued in Québec for a period of five years and may be renewed for one additional five-year period provided the conditions of renewal described in the *Mining Act* (*Québec)*(R.S.Q., c. M-13.1) (Mining Act) are met. Prior to expiry of the Mining Exploration Licence (PEM), the entire area may be converted to mining claims in an online process, and the excess exploration expenditures (or credits) carried over from the PEM to the new claims.

In Québec, claims are issued based on a two-year assessment period with the work requirements increasing for each period. Renewal fees paid earlier than 60 days prior to the anniversary date are fixed by regulation and are currently set at $123.00 per claim. If a renewal application is submitted after that date but prior to the claim's date of expiry, the current prescribed fee of $126.00 per claim is doubled.

The claim holder has the exclusive right to search for all mineral substances in the public domain, with the exception of petroleum, natural gas, brine and loose surface materials. In accordance with the Mining Act a claim holder must comply with the obligation to carry out work on the claims in an amount based on the surface area of the claim and its location in relation to the 52nd parallel. When the work carried out is insufficient, or if work was not carried out, the titleholder may pay an amount equivalent to the required amount in lieu of work. Alternately, excess exploration credits from other claims within a 4.5 km radius may be transferred to meet any shortfall.

A Mining Lease can be issued to a claim holder in Québec who demonstrates that a deposit is mineable. Mining Leases are surveyed, valid for 20 years (renewable for 10 year periods) and have annual payments of $44/ha. In Québec, surface leases are granted for the purpose of storing mine residues. Fees for surface leases that are used to store mine residues are $94/ha.

4.4 Claims and Leases

The Foxtrot Property comprises three groups of separate but essentially contiguous landholdings known as the Foxtrot 1, Foxtrot 2 and Foxtrot 3 blocks, covering a total area of 47,949.44 ha (Figure 4.2). Foxtrot 1 and Foxtrot 2 consist of 617 individual claims (32,052.27) and one mining lease (143.71 ha), and Foxtrot 3 contains 303 individual claims (15,753.46ha). Fifty-eight claims in five groups belonging to Foxtrot 3, and included in the totals above, lie north of the main block and are non-contiguous (Figure 4.2).

The claims are registered in the names of SDCI as a 100% interest. At the effective date of this report, all claims are reported by Stornoway to be in good standing.

All but nine of the 617 claims comprising the Foxtrot 1 and 2 properties currently have anniversary dates of August 15, 2014 or August 16, 2014 and can be identified by the following claim numbers in the public register of real and immovable mining rights kept at the Ministère des Ressources naturelles et de la Faune (MRNF): CDC2245719 to CDC2245734 (inclusive), CDC2245737 to CDC2245968 (inclusive), CDC2245873 to CDC2246148 (inclusive), CDC2246150, CDC2246152, CDC2246154, CDC2246156, CDC2246158, CDC2246160, CDC2246162, CDC2246164, CDC2246166, CDC2246168, CDC2246170, CDC2246295 to CDC2246314 (inclusive), CDC2246321 to CDC2246473 (inclusive), CDC 2246493 to CDC2246499, and CDC2246572 to CDC2246573 (inclusive). CDC 22446378 to CDC2246380 (inclusive), CDC2246390 to CDC2246392 (inclusive) and CDC2246402 to CDC2246404 (inclusive) all have anniversary dates of January 16, 2015 as they were temporarily suspended during the time that the mining lease application was being processed (all partially covered the area which is now defined as BM 1021). Shortly after the issuance of BM 1021 these nine cells were reactivated, their cell size was reduced to reflect the area which is now covered by the mining lease and their anniversary dates were extended to reflect the amount of time that the cells were temporarily suspended.

Mining Lease BM 1021, totaling 143.71 ha in size, was granted to Stornoway on October 16, 2012. BM 1021 encompases the mine site and surface operations (excluding the Processed Kimberlite Containment area). A surface lease encompassing the Process Kimberlite Containment area was also granted on Oct 16, 2012. The surface lease, number 1303 10 000, is 199.85 ha in size.

All known kimberlite mineralization, including the pipes and the dyke systems, as well as the existing infrastructure, including Camp Lagopède, diesel generators, underground infrastructure, the on-site dense media separation (DMS) facility, and processed kimberlite containment facility, that is associated with the Foxtrot Property, are situated within the boundaries of the Foxtrot 2 claim block. A single, narrow (approximately 0.1 m thickness), north-trending kimberlite dyke with an estimated strike length of 450 m straddles mining claims 1009366 and 1009385. No other kimberlites are known within the Foxtrot 3 claims.

4.5 Agreements

Stornoway holds a 100% interest in the Property, subject to a 2% direct royalty on future diamond production in favour of DIAQUEM, effective April 01, 2011 and as documented in an Acquisition and Exchange Agreement between DIAQUEM and Stornoway dated December 14, 2010 and Amended on February 08, 2011.

4.6 Surface Rights

The Foxtrot Property, including the Renard kimberlite pipes and the Lynx and Hibou kimberlite dykes, is situated within the region of northern Québec governed by the *James Bay and Northern Québec Agreement 1975*, as amended (JBNQA), a land claims agreement executed by the Government of Québec, the Government of Canada, the Grand Council of the Cree of Québec (GCC) and the Northern Québec Inuit Association, amongst others.

The JBNQA provides for three categories of land, Categories I to III, each with specifically defined rights. The Foxtrot Property lies within Category III lands in an area in proximity to the community of Mistissini (the Cree Nation of Mistissini, or CNM). Category III lands are public lands where Cree communities have certain rights, particularly in regard to trapping, hunting, fishing and the development of outfitter operations. Surface and mineral rights on Category III lands reside with the Government of Québec and are governed by the applicable land use laws and regulations, implemented by the relevant regulatory authorities. Members of the CNM undertake hunting, fishing and trapping activities within the Foxtrot Property, with the Renard kimberlites occurring in an area known to them as “yuus-kanchiisu-saakahiikan” (mild rock ptarmigan lake). More specifically, the Renard kimberlites lie within the CNM trapline area designated as M-11, used by Sydney and Emerson (known as the ‘tallyman’) and their respective families.

Under the terms of the JBNQA, governments have the right to develop Category III lands. However, mining projects among others are subject to an environmental and social protection regime, which provides for the protection of the hunting, fishing and trapping rights of the Cree.

SDCI, as holder of a claim, has the exclusive right to explore for mineral substances on the parcel of land subject to the claim, with the exception of: (i) petroleum, natural gas and brine; (ii) sand other than silica sand used for industrial purposes, gravel, common clay used in the manufacture of clay products and every other mineral substance found in its natural state as a loose deposit, as well as inert mine tailings used for construction purposes; and (iii) on any part of the parcel of land that is also subject to an exploration license for surface mineral substances or an exclusive lease to mine surface mineral substances, every other surface mineral substance.

Moreover, the claim provides access rights to the parcel of land subject to the claims on which exploration work may be performed. A claim holder cannot access land that has been granted, alienated or leased by the State for non-mining purposes, or land that is the subject of an exclusive lease to mine surface mineral substances, without first having obtained the permission of the holder of these rights.

SDCI is required to carry out assessment work prior to the 60th day preceding the expiry date of a claim. The nature and value of the work are determined by regulation. The amount of the assessment work to be carried out during each term of a claim varies according to the number of terms since its issuance, the surface and the locations.

To mine mineral substances (except surface mineral substances, petroleum, natural gas and brine) in the Province of Québec, SDCI will be required to obtain a mining lease in accordance with the Mining Act. To obtain a mining lease in respect of all or a part of a parcel of land that is subject to one or several claims, SDCI will be required to establish the indicators of the

presence of a workable deposit, provided it also meets prescribed requirements and pays the applicable annual rent.

SDCI will, have as lessee the rights and obligations of an owner on the land that is subject to such mining lease, including surface and usage rights. On lands in the public domain, access and the right to use the surface of the land are limited to mining purposes only.

4.7 Permits

4.7.1 Exploration

Currently, work on the Project is conducted in accordance with the terms and conditions contained within the surface exploration permit and other permits that are required for the camp, fuel storage and exploration activities. Stornoway has received the permits and approvals it needed to operate and conduct the associated advanced exploration activities completed to date.

4.7.2 Future Work

The Renard Project must obtain global environmental approvals under various legislation, including the Mining Act, the *Environment Quality Act* (R.S.Q. c. Q-2) (EQA), the *Canadian Environmental Assessment Act* (S.C., c. 37) (CEAA) and the JBNQA. Table 4.1 lists the potentially necessary authorizations, permits, approvals, attestations and leases for the Renard Project. To date, Stornoway has received several important permits and authorizations.

In December 2012, Stornoway received the global Certificate of Authorization for the Renard Diamond Project from the Québec Ministère du Développement Durable, de l'Environnement, de la Faune et des Parcs (MDDEFP). The Certificate of Authorization represents the principal regulatory approval required to initiate mine construction. In October 2012, the Québec Ministère des Ressources naturelles issued the Mining Lease for the Renard Diamond Project. The Mining Lease represents the formal transfer of mineral title from the Québec Government to Stornoway, and it is valid for a period of 20 years. In December 2012, Stornoway received from the Québec Ministère des Ressources naturelles, the approval for the proposed Closure and Rehabilitation plan for Renard.

The JBNQA was implemented in 1975 by the Québec government, the Government of Canada, the Grand Council of the Crees (Eeyou Istchee) (GCC-EI) and the Association des Inuits du Nouveau-Québec. The JBNQA defines the social and environmental protection regimes for the regions of James Bay and Nunavik. The Renard Project is subject to the provincial process for the assessment and review of social and environmental impacts under the JBNQA.

Three land categories were established in the JBNQA: category I, II and III. Specific rights are defined for each one. The Foxtrot Property is located on category III land of the community of the CNM.

Chapter II of the EQA provides specific provisions for environmental assessments applicable to the James Bay territory, in accordance with the provisions of the JBNQA. The environmental assessment procedure specific to this region differs from the provincial procedure, as it involves

the active participation of the Aboriginal people who live there. The Ministère du Développement durable, de l'Environnement et des Parcs (MDDEP) is responsible for coordinating the review of the impact assessment study as a representative of the Québec government. However, other departments, such as the MRNF and the Ministère des Transports (MTQ) may also be involved in the review as experts.

The Québec government is responsible for mining activities in the province. This activity is subject to the Mining Act which defines ownership of the right to mineral substances (claims, mining exploration licenses, mining leases, mining concessions, etc.) and the rights and obligations of the claim holder or other mining right granted by the State.

The CEAA is applicable to projects for which the federal government has decision-making powers under Section 5 of the CEAA, as a proponent, administrator of federal lands, funding source or regulatory body. For the Renard Project, the Fisheries and Oceans Canada (DFO) must grant authorization to modify the fish habitat under Subsection 35(2) of the *Fisheries Act* R.S.C. 1985, C. F-14). The Canadian Environmental Assessment Agency is considered the responsible authority for the Renard Project and DFO and TC are considered experts. Under federal regulations, the Renard Project is subject to a comprehensive federal environmental assessment, in accordance with section 21 of the CEAA.

At the end of the impact assessment, after receiving global environmental approval from the MDDEP and from the federal government under the CEAA, the proponent may begin preparing the final plans and specifications. In addition to the mitigation measures set out in the environmental and social impact assessment, the final Project design must comply with the applicable standards for projected infrastructures and equipment. The preparation of plans and specifications must follow the legal framework of the governments of Québec and Canada.

Following the submission of the Project Notice in February 2010, the MDDEP and Comité d'évaluation (COMEV) issued guidelines to Stornoway on the scope of the impact assessment to be carried out for the Renard Diamond Mine Project (June 2010). A modification to the Project Notice was submitted to the MDDEP, COMEV and the Canadian Environmental Assessment Agency in February 2011, informing them of some changes made to the Project since February 2010. The Environmental and Social Impact Assessment (ESIA) was conducted in 2010-2011 by Roche ltée Group-Conseil. The ESIA report was submitted to the Administrator on December 28, 2011.

The ESIA process includes several consultation activities carried out by Stornoway and others managed by federal and provincial authorities. Public hearings on the ESIA, held separately by the federal and Québec regulators, are important milestones in the mine permitting process, and are designed to gauge the overall social acceptability of the proposed development. To this effect, the final round of provincial public hearings was held in Chibougamau and Mistissini in August 2012 and federal consultations were held in the same communities in June 2012.

4.8 Socio-Economics

The Foxtrot Property is located within the region of Northern Québec governed by the JBNQA, as amended, a land claims agreement executed by the Government of Québec, the Government of Canada, the GCC and the Northern Québec Inuit Association, among others. In

February 2002, Québec and the Cree Nation signed a fifty year political and economic agreement. This agreement is the La Paix des Braves. The GCC is the political body that represents approximately 14,500 Crees of Eastern James Bay and Southern Hudson Bay in Northern Québec. The GCC has twenty members: a Grand Chief and Deputy-Grand Chief elected at large, the chiefs elected by each of the nine Cree communities of the territory, and one other representative from each community.

The closest community to the Renard Project is Mistissini, located southeast of the largest natural lake in Québec, Lac Mistassini, approximately 90 km northeast of Chibougamau. The site on which the community is built was once used as a basic summer encampment where the Hudson Bay Company settled in the early 1800s. In terms of population (3,500 residents), Mistissini is one of the largest Cree communities in Québec. Its economic activities are mainly concentrated in the business and service sectors (health and education), forestry, trapping, tourism, outfitting, construction and transportation.

The limits of the Jamesian Territory essentially follow the boundaries of the Municipality of James Bay (MBJ), a vast regional administrative entity that includes the towns of Chibougamau, Chapais, Lebel sur-Quévillon and Matagami. The MBJ communities most affected by the Renard Project are Chibougamau and Chapais.

Mining exploration has played a leading role in the economic development of Chibougamau and Chapais. The completion of a cart road linking Saint-Félicien to Lac-Saint-Jean and Chibougamau led to the establishment of the village of Chibougamau, which received its status as a municipal corporation on September 1, 1954. After the Campbell mine became operational in 1955, Chibougamau saw rapid expansion with the development of rich polymetallic deposits (gold, copper and silver) and the population grew year over year until the late 1970s, when it reached its highest levels. The economy of Chibougamau and Chapais became progressively diversified with forestry and the development of the public and private service sector.

4.9 Risks

The ongoing ESIA process for the Renard Project is based on the most current and detailed information available on the physical, biological and human environment. The information was provided by the Feasibility Study team through best environmental assessment practices that are recognized by government institutions. These inputs, in combination with information from the local communities and consultation meetings held with local stakeholders, form the basis of the environmental assessment process.

The environmental assessment process includes environmental design measures which prevent and minimise environmental and social impacts and risks, and limit the extent and consequences of these potential risks and impacts. The environmental assessment process also allows for the development of specific mitigation and compensation measures that are consistent with government guidelines to minimize or compensate these impacts or risks to acceptable levels. Any residual impacts will be managed through an environmental management plan that includes environmental monitoring and surveillance. Environmental and social residual risks to the Project schedule or implementation were identified and addressed as part of the Feasibility Study in a risk register. They are discussed in Section 24.4 of the present report.

Table 4.1: List of Potentially Necessary Authorizations, Permits, Approvals, Attestations and Leases

Description	Department	Regulations/Act/Code
Provincial government		
Authorizations		
Certificate of authorization to carry out mining project *Construction of airstrip* *Installation of water intake in Lakes Poisson and Lake Lagopède* *Construction of dyke in Lake Lagopède and drainage of bay* *Dredged sediment storage* *Water diversion* *Construction and operation of power station* *Mining operations (pits, dumps and tailings sites)* *Operation of ore processing plant* *EDO - treated industrial water (mining operations)* *Explosives magazine and service building* *Installation and operation of oil separator - Garage - Power station* *Trench landfill* *Hazardous material management and storage* *Incineration of waste oil as fuel* *Incinerator - Waste management* *Lagopède mobile concrete plant* *Sand and gravel operation (borrow pits)*	MDDEP	Section 22 of the *Environment Quality Act* (R.S.Q., c. Q-2)
Authorization to conduct an activity likely to alter a wildlife habitat *Activities affecting fish habitat (water diversion, water removal and withdrawals, etc.)* *Does not apply to docks, trails, roads, culverts or bridges built in accordance with the conditions set out in Division VI of the* Regulation respecting wildlife habitats	MRNF - Wildlife	Section 128.7 of the *Act respecting the conservation and development of wildlife* (R.S.Q., c. C-61.1) Division VI of the *Regulation respecting wildlife habitats*
Bulk sampling authorization *Bulk sampling of more than 50 metric tonnes*	MRNF - Mines	Section 59 of the *Mining Act* (R.S.Q., c. M-13.1)
Authorization to install apparatus or equipment to prevent, reduce or cause the cessation of the issuance of contaminants into the atmosphere	MDDEP	Section 48 of the *Environment Quality Act* (R.S.Q., c. Q-2)
Authorization to establish water intakes, water purification appliances, and devices for the treatment of wastewater *Camp - permanent (wastewater)* *Camp - permanent (potable water)* *EDO - treated sanitary wastewater (permanent camp)*	MDDEP	Section 32 of the *Environment Quality Act* (R.S.Q., c. Q-2)
Authorization to build high-capacity dam *Lake Lagopède dam*	Centre d'expertise hydrique du Québec - MDDEP	Section 5 of the *Dam Safety Act* (R.S.Q., c. S-3.1.01) Division VI of the *Dam Safety Regulation* (c. S-3.1.01, r. 1)
Approval		
Approval for the localization of mine tailings concentration and disposal plant	MRNF - Mines	Sections 240 and 241 of the *Mining Act* (R.S.Q., c. M-13.1) Sections 124 and 125 of the *Regulation respecting mineral substances other than petroleum, natural gas and brine* (c. M-13.1, r. 2)

Attestation		
Depollution attestation (Industrial waste reduction program)	MDDEP	Section 31.10 of the *Environment Quality Act* (R.S.Q., c. Q-2) Order in Council 515-2002 of May 1, 2002, respecting the application of subdivision 1 of Division IV.2 of Chapter I of the Environment Quality Act to the mineral industry and primary metal manufacturing sector
Permits		
Forest management permit for mining activities *All deforestation activities*	MRNF - Forests	Chapter I, Divisions I and II of the *Forest Act* (R.S.Q., c. F-4.1) *Regulation respecting standards of forest management for forests in the domain of the State* (c. F-4.1, r. 7)
Occupation licence for work involving the collection or discharge of water *Water intake in Lake Poisson* *Water intake in Lake Lagopède*	Centre d'expertise hydrique du Québec - MDDEP	Section 10 of the *Regulation respecting the water property in the domain of the State* (c. R-13, r. 1.1)
Explosives permit (general, magazine and transport)	Sûreté du Québec	*An Act respecting explosives* (R.S.Q., c. E-22, a. 22) Division II of *Regulation under the Act respecting explosives* (c. E-22, r. 1)
High-risk petroleum equipment operation permit *Mine site* *Airstrip*	Régie du bâtiment du Québec	Section 120 of the *Safety Code* (c. B-1.1, r. 0.01.01.1) Section 8.12 of the *Construction Code* (c. B-1.1, r. 0.01.01)
Leases		
Mining leases	MRNF - Mines	Section 100, Division V of the *Mining Act* (R.S.Q., c. M-13.1)
Non-exclusive lease to mine surface mineral substances *Borrow pits, quarries and sand pits*	MRNF - Mines	Section 140, Division VII of the *Mining Act* (R.S.Q., c. M-13.1)
Lease for the occupation of the State's hydric domain *Lake Lagopède dyke*	Centre d'expertise hydrique du Québec - MDDEP	Section 19 of the *Regulation respecting the water property in the domain of the State* (c. R-13, r. 1.1)
Lease to occupy the domain of the State *Dumps and tailings sites* *Camp* *Airstrip* *Explosives magazine* *Trench landfill*	MRNF - Territory	Section 239 of the *Mining Act* (R.S.Q., c. M-13.1) Section 47, Division II of the *Act respecting the lands in the domain of the State* (R.S.Q., c. T-8.1) Section 35 of the *Regulation respecting the sale, lease and granting of immovable rights on lands in the domain of the State* (c. T-8.1, r. 7)
Notice to Minister		
Notice to Minister to establish an industrial camp	MDDEP	Section 2 of the *Regulation respecting sanitary conditions in industrial or other camps* (c. Q-2, r. 11)
Federal government		
Authorization		
Authorization to modify fish habitat *All activities affecting fish habitat*	Fisheries and Oceans Canada	Section 35 of the *Fisheries Act* (R.S.C., 1985, c. F-14)

Approval		
Approval of works in, over, under, through, or across any navigable waterway *Lake Lagopède dyke*	Transport Canada	Section 5 of the *Navigable Waters Protection Act* (R.S.C., 1985, c. N-22)
Licence		
Licence for the manufacturing of explosives	Natural Resources Canada	Paragraph 7(1)(a) of the *Explosives Act* (R.S.C., 1985, c. E-17) Section 25 of the *Explosives Regulations* (C.R.C., c. 599)
Registration		
Registration of an aerodrome[1]	Transport Canada	Section 301.03 of the *Canadian Aviation Regulations*, Part III, Subpart 1 - Aerodromes
Municipality of James Bay (municipality outside RCM)		
Certificates		
Certificate of compliance with municipal bylaws - Service infrastructure	–	*An Act to amend the James Bay Region Development Act and other legislative provisions* (Bill n°40)
Certificate of compliance with municipal bylaws - Road	–	
Certificate of compliance with municipal bylaws - Camp	–	
Certificate of compliance with municipal bylaws - Airstrip infrastructure	–	
Permits		
Permit for individual work on groundwater catchment (Airstrip)	–	*An Act to amend the James Bay Region Development Act and other legislative provisions* (Bill n°40)
Permit for septic system installation (Airstrip)		

[1] *Aerodrome: Any area of land, water (including the frozen surface thereof) or other supporting surface used, designed, prepared, equipped or set apart for use either in whole or in part for the arrival, departure, movement or servicing of aircraft and includes any buildings, installations and equipment situated thereon or associated therewith.*

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

5.1.1 Air

Historically, during summer months, float-equipped, fixed-wing aircraft and, during the winter months, ski-equipped aircraft were leased from various hunting and fishing outfitters in the area or from small air service companies in Chibougamau, Mistissini, Témiscamie or Wemindji. Rotary wing aircraft were also chartered. The Chibougamau–Chapais airport has daily scheduled flights to Montréal. Flight time from Chibougamau to the Property varies from one to two hours depending on the type of aircraft used. In 2012, a temporary 1,500 foot long land airstrip was constructed on site to accommodate wheeled aircraft year round.

An ice airstrip measuring up to 1,250 m in length has previously been constructed near Camp Lagopède to allow operation of large aircraft for mobilization of equipment and supplies. In the summer, fixed-wing access directly to the Property requires the use of float planes, which can land at a dock at Camp Lagopède.

5.1.2 Road

Highway 109, an all-weather paved road, originates in Matagami, Québec, and terminates approximately 500 km north at the LG-2 hydroelectric generating station in Radisson. Two major, all-weather, gravel roads lead east from this highway into the interior of Québec (Figure 4.1). The North Road connects Chibougamau with the James Bay Highway via Nemiscau. The Trans-Taïga Highway runs east from Radisson (LG-2) to Caniapiscau and provides gravel road access to the LG-3, LG-4, Laforge 1, Laforge 2 and Centrale Brisay hydro sites as well as to several caribou hunting lodges. LG-4 is approximately 150 km from the Renard site. Pourvoirie Mirage (Mirage), a hunting/fishing lodge located approximately 50 km to the east of LG-4 along the Trans-Taïga Highway, owns and operates an all-weather airstrip. Mirage is approximately 120 km from the Project, and provides a staging point for the transfer of trucked material to either fixed-wing or rotary-wing aircraft.

Provincial road 167 affords paved access to the city of Chibougamau and the village of Misstissini, and links these communities to the provincial road network. A 150 km, all-weather gravel road proceeds north from the village of Mistissini to Témiscamie.

Between 2004 and 2008, a system of unpaved local roads and trails was constructed between Camp Lagopède, the Renard Core Area, the Hibou and the Lynx dyke areas and various exploration drill sites. There is no road linking the Property to any outside roads.

On August 1, 2011 Stornoway Diamond Corpoation (Stornoway) executed two separate agreements with the Ministère des Ressources naturelles et de la Faune and the Ministère des Transports (MTQ) to provide financial assistance for construction of the Route 167 extension. Stornoway agreed to contribute $44 million to the development of the new road over a ten year period starting in July 2015.

Road construction started in early 2012, and first road access to the Renard site for construction mobilization is anticipated by the fourth quarter of 2013 through the utilization of an upgraded seasonal road and construction easements.

5.2 Climate / Operating Seasons

Climate at the Renard site is characterized by long winters and short summers. Temperatures range from summer maximums of +35°C to winter minimums of -45°C. Lakes freeze over in late October, thawing in late April–May. Abundant precipitation falls in the form of rain and snow. Total annual precipitation averages around 80 - 100 cm. The Otish Mountains form a local topographic high that affects both precipitation and fog formation and distribution in the area. Fog and low-lying clouds can cause delays to aircraft operating in the area.

Forest fires are common in the area during the spring and summer months, but to date have not adversely affected the Project.

5.3 Local Resources and Infrastructure

5.3.1 Local Resources

The nearest communities to the Foxtrot Property are Chibougamau (population about 8,000), Mistissini (pop 3,500), and Wemindji (pop 1,300). Chibougamau serves as the major supply centre for regional industries.

Two field camps were constructed on the Foxtrot Property: Emmanuel and Lagopède. During the winter, a 15 km snowmobile trail connected the two camps. Camp Emmanuel originally had the capacity to lodge up to 35 persons, but is currently decommissioned. Camp Lagopède, which can host 75 persons, is used to stage all project related activities. The site has a satellite communications system, a secure logging facility, a fenced storage area for holding kimberlitic samples, a garage equipped with a welding shop, and a 10 tph Dense Media Separation (DMS) facility.

5.3.2 Power

Power is supplied to the site by diesel generators. Currently, all fuel is flown to the site. The hydroelectric generating station at LG-4, located approximately 150 km to the north–northwest of the Foxtrot Property (Figure 4.1), could potentially supply power to the Property. Stornoway is financing an independent feasibility study and environmental assessment for a power line. The power line studies are being conducted by Hydro-Québec separately from the feasibility study.

5.3.3 Transport

The airport facilities at LG-4 and Chibougamau, and the fixed-wing float bases in Mistissini, Témiscamie and Mirage are used as staging areas for mobilization of equipment and personnel to the Project site.

5.3.4 Water

In the past, water was required to support camp operations and the process plant. Depending on the immediate requirements and the relevant usage permits, process water was sourced from local lakes or recirculated from surface facilities such as the processed kimberlite containment area or the underground exploration workings. During the 2007 underground bulk sampling program, three different types of water were managed on the Foxtrot Property:

- Used water from the Lagopède camp operation: A water treatment system was set up in 2006 and approved by the Ministère du Développement durable, de l'Environnement et des Parcs (MDDEP). It consists of four series of Enviro–Septic system units connected to a cleaning field established behind the campsite.
- Used water from the underground exploration program: A water treatment system approved by the MDDEP was set up underground, together with an outside water-settling pond that was monitored by Stornoway and reported monthly to the appropriate government agency. This pond was decommissioned in August 2008 following MDDEP regulations.
- Used water for the DMS operation: This water was completely recirculated from the DMS facility into the adjacent settling pond as approved by the MDDEP. DMS process water did not require any specific water treatment.

The potable water system used for the Lagopède camp operation is a portable ozone water treatment system, Expio, which is manufactured by Aquagenex in Clermont, Québec. There are two units available on site. The system can produce up to 100 gallons (378 L) per day, a sufficient quantity for the operations to date.

5.4 Physiography

Topographic relief within the Foxtrot Property consists of steep-sided hills with rounded tops separated by muskeg-covered valleys. Elevations range between 400 masl and 800 masl. Lakes, ponds and small rivers are common.

5.5 Flora and Fauna

The Foxtrot Property is located in the Taïga belt of Northern Québec. Vegetation consists largely of immature to mature black spruce, poplar, alders and muskeg, with increasing proportions of muskeg and black spruce toward the north. Animals such as bear, fox, moose, marten and caribou, as well as various species of birds, are present in the vicinity of the Foxtrot Property.

5.6 Sufficiency of Surface Rights for Mining Operations

The sufficiency of surface rights for mining operations is discussed in Sections 4.5 and 4.6.

6.0 HISTORY

Diamond exploration commenced in the Foxtrot Property area in 1996, with the formation of a 50:50 joint venture between Ashton Mining Canada Inc. (Ashton) and SOQUEM (SOQUEM). Prior to that time, regional exploration had been undertaken for gold and base metals by a number of parties using prospecting and geochemical techniques. These activities were limited in scope due to the location of the current property between two Archaean volcanic belts, an area that was considered to be non-prospective for traditional gold and base metal targets. Both BHP Billiton and De Beers Canada Exploration have conducted regional diamond exploration programs in the general area of the current Foxtrot Property, but results are not publicly available. Although there had been scattered small-scale claim blocks throughout the area explored by the Ashton-SOQUEM joint venture, no large-scale land acquisition had occurred prior to the August 2000 grant of Mineral Exploration Licences PEM 1555 and 1556.

Initial stage exploration by the joint venture comprised heavy mineral sampling that defined a number of areas with anomalous indicator mineral counts. Geophysical surveys over the geochemical anomalies were completed, using fixed-wing airborne, magnetic surveys, and ground geophysical surveys.

Core drill testing of these targets led to the discovery of the Renard 1 and Renard 2 kimberlites in September 2001, the Renard 3, Renard 4, Renard 5 and Renard 6 kimberlites in March and April 2002, Renard 7 in August 2002, Renard 8 in September 2002, Renard 9, Renard 10 and the 4.2 km long Lynx kimberlitic dyke occurrences in 2003, and the 850 m long Hibou kimberlite dyke in 2005. The Renard 5 and Renard 6 kimberlites were subsequently determined to be one kimberlitic body, and the occurrence was renamed Renard 65.

Mini-bulk sampling using trenching and core drilling commenced in 2002 on the Renard 2, Renard 3 and Renard 4 kimberlites. Additional mini-bulk sampling since that date has been undertaken on the Renard 1, Renard 65, Renard 7, Renard 9, Renard 10, Lynx, Lynx South and Hibou kimberlites. Mini-bulk samples of kimberlite boulders collected down-ice of the Lynx and Hibou kimberlites have also been completed.

During 2004, a conceptual, order of magnitude, tonnage and grade estimate was prepared for the combined Renard 2, Renard 3, Renard 4 and Renard 65 kimberlites, by Wardrop Engineering Inc. of Vancouver, BC (Maunala, 2004). This estimate included valuation of Renard 2, Renard 3, Renard 4 and Renard 65 diamonds recovered through RC sampling.

Additional exploration resulted in the discovery of the North Anomaly kimberlite dyke in 2005 and the Southeast Anomaly and G04-296 kimberlite dykes in 2006. An occurrence of kimberlitic float material was also discovered north of the Renard pipe cluster.

Bulk sampling commenced in 2006, testing the Renard 2, Renard 3, Renard 4, Renard 65 and Renard 9 kimberlites. Samples were sourced from trench, drill core, RC drill chips and underground. The underground exploration workings comprised a portal and the excavation of an inclined ramp to a depth of approximately 55 m below surface. Horizontal drifts were driven to access the Renard 2 and Renard 3 kimberlites. A modular 10-tonne-per-hour dense media separation (DMS) test facility was erected on the Foxtrot Property close to Renard 2 and Renard 3, to treat the material from the bulk sampling.

In December 2008, an economic assessment study jointly undertaken by Agnico-Eagle Mines Limited (Agnico-Eagle) and AMEC Americas Limited (AMEC), was completed and formed the 2008 NI 43-101 Technical Report on the Preliminary Assessment of the Renard Project authored by Scott Wilson Roscoe Postle Associates Inc. (SWRPA).

Agnico-Eagle was responsible for mining, cost estimating and financial analysis. Elements of the Agnico-Eagle mining study, such as cost estimation, were completed to a pre-feasibility standard. AMEC designed the DMS facility, and prepared the Mineral Resource Estimate. Given the more conceptual nature of the mine plan, and inclusion of Inferred Resources from AMEC's Mineral Resource Estimate used in the study, the two studies combined comprise a Preliminary Assessment under the definitions contained within NI 43-101. An audit of the Agnico-Eagle study and economic analysis was completed by SWRPA, as independent Qualified Persons for the 2008 NI 43-101 report.

The December 12, 2008 Technical Report was revised on March 25, 2009 to clarify that the QPs were taking responsibility for the WWW International Diamond Consultants valuation and that diamond valuations are not amenable to data verification procedures that can be performed by a QP. None of the scientific and technical information in the report changed.

Golder Associates Ltd. (Golder) completed a 2009 updated resource estimate which formed the NI 43-101 2009 Mineral Resource Update for the Renard Project (Farrow, 2010). The effective date of the report was December 8, 2009. The initial report was dated January 21, 2010, and revised on January 27, 2010. The revision did not make any material changes to the initial report.

SWRPA was then retained by Stornoway Diamond Coroporation (Stornoway) to prepare an update of the March 25, 2009 Independent Technical Report on the Project. The purpose of the report was to provide an updated document for the disclosure of a Preliminary Assessment of a potential mining and processing operation at the Project.

A Preliminary Assessment of the Project was completed on March 22, 2010. This study comprised a conceptual mine plan, capital and operating cost estimates, and cash flow model prepared by SWRPA; a diamond processing plant design, with capital and operating cost estimates, prepared by AMEC; and social, environmental and permitting aspects contributed by Stantec. The conceptual mine plan was based upon the National Instrument (NI) 43-101 compliant mineral resource estimate prepared by Golder. Golder, AMEC and Stantec prepared or contributed to sections of the Technical Report. This May 5, 2010 Technical Report conformed to NI 43-101 Standards of Disclosure for Mineral Projects (Lecuyer et al., 2010).

In 2011, Stornoway completed a Feasibility Study for the Project. The lead consultant for the Feasibility Study was SNC-Lavalin Inc. The study, and subsequent National Instrument 43-101 compliant technical report, were prepared by the following independent consultants: SNC-Lavalin Inc., AMEC Americas Limited, Golder Associates Limited, G Mining Services Inc., GeoStrat Consulting Inc., Itasca Consulting Canada Inc., and Roche ltée Groupe-Conseil. The technical report summarizing the Feasibility Study was filed on Jan 3, 2012.

Stornoway retained GeoStrat Consulting Services Inc., (GeoStrat) to provide an independent mineral resource estimate update for the Renard Project (Farrow and Farrow, 2011). This estimate represents the third reporting of mineral resources from the Renard Project. Despite

7,654 m of additional core drilling and 3.6 t of sampling since the last disclosure, the latest indicated resource has remained relatively unchanged. The increase of the inferred resource is primarily due to the upgrade of a portion of the Renard 65 kimberlite from potential mineral deposit (PMD) to Inferred.

No commercial production has been carried out on the property.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The Project area is located within the eastern portion of the Superior Craton (Superior Structural Province) Figure 7.1. The Superior Province forms the Archaean core of the Canadian Shield, and is the largest contiguous region of Archaean crust remaining on the globe. It is considered to be an amalgamation of small continental fragments of Meso-Archaean age and Neo-Archaean oceanic plates, with a complex history of aggregation between 2.72 Ga and 2.68 Ga. Since about 2.6 Ga, the Province has been tectonically stable (Percival, 2006).

The Superior Province is surrounded by provinces of Paleo-Proterozoic age on the west, north and east (Churchill Province), and Meso-Proterozoic age (Grenville Province) on the southeast. Margins of the Superior Province were affected during Paleo-Proterozoic and Meso-Proterozoic tectonism. Proterozoic and younger activity is limited to rifting of the margins (Mid Continent Rift System), emplacement of numerous mafic dyke swarms, compressional reactivation and large-scale rotation at ca. 1.9 Ga, and failed rifting at ca. 1.1 Ga (Percival, 2006). Figure 7.2 shows the Archaean and Proterozoic rocks in the general region of the Foxtrot Property.

Quaternary glacial cover in the area was controlled by the New Québec Ice Divide. From the divide, ice flowed north and northeast toward Ungava Bay and west to southwest toward Hudson Bay. Glacial lineaments are well developed and widespread. Eskers and hummocky to discontinuous, unmoulded, ground moraine deposits are also common.

There are five known episodes of kimberlitic volcanism in Québec (Moorhead et al., 2003); from south to north, the kimberlite fields are:

- Témiscamingue: six diatreme facies pipes intruding the Pontiac Subprovince. Two age dates, 125 Ma (Rb–Sr) and 142 Ma (U–Pb) have been obtained. Kimberlites are hosted in the northwest-trending Témiscaming structural zone.
- Desmaraisville: five hypabyssal pipes and numerous dykes located in the central portion of the Abitibi Subprovince. Age date of 1104 Ma (Rb–Sr fromphlogopite). Hosted in the Waswanipi–Saguenay Tectonic Zone; pipes are in close proximity to northeast-trending Proterozoic dykes.
- Otish: at least 12 pipes intruding the northeast portion of Opatica and Opinaca Subprovinces. Age dates range from 550.0 + 3.5 Ma at Beaver Lake to 640.5 + 28 Ma at Renard. The kimberlite field is associated with the southern end of the Mistassini-Lemoyne structural zone, and near-northwest, and northeast-trending Proterozoic diabase dykes.
- Wemindji: kimberlitic sills intruding Archaean-age gneisses of La Grande Subprovince, located at the western end of the Wemindji–Caniapiscau structural zone where it intersects the northeasterly projection of the Kapuskasing zone.
- Torngat: diamond-bearing dykes recognized in the Paleo-Proterozoic Rae Province near the Archaean Nain Craton. These dykes were classified as carbonatized ultramafic lamprophyres and dated at 550 Ma.

The Renard Cluster is considered to be part of the Otish kimberlitic volcanic event.

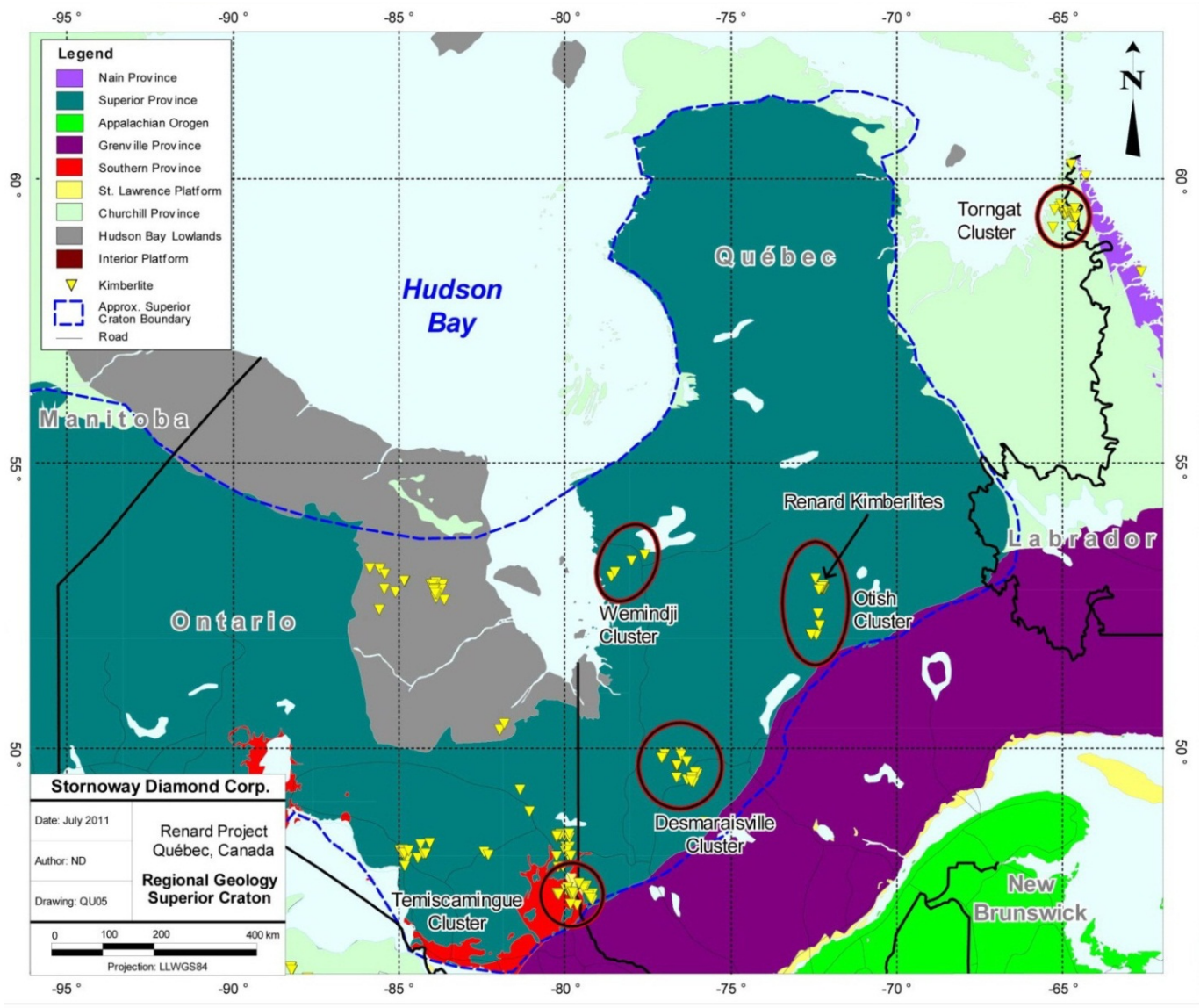

Figure 7.1: Regional Geology – Superior Craton

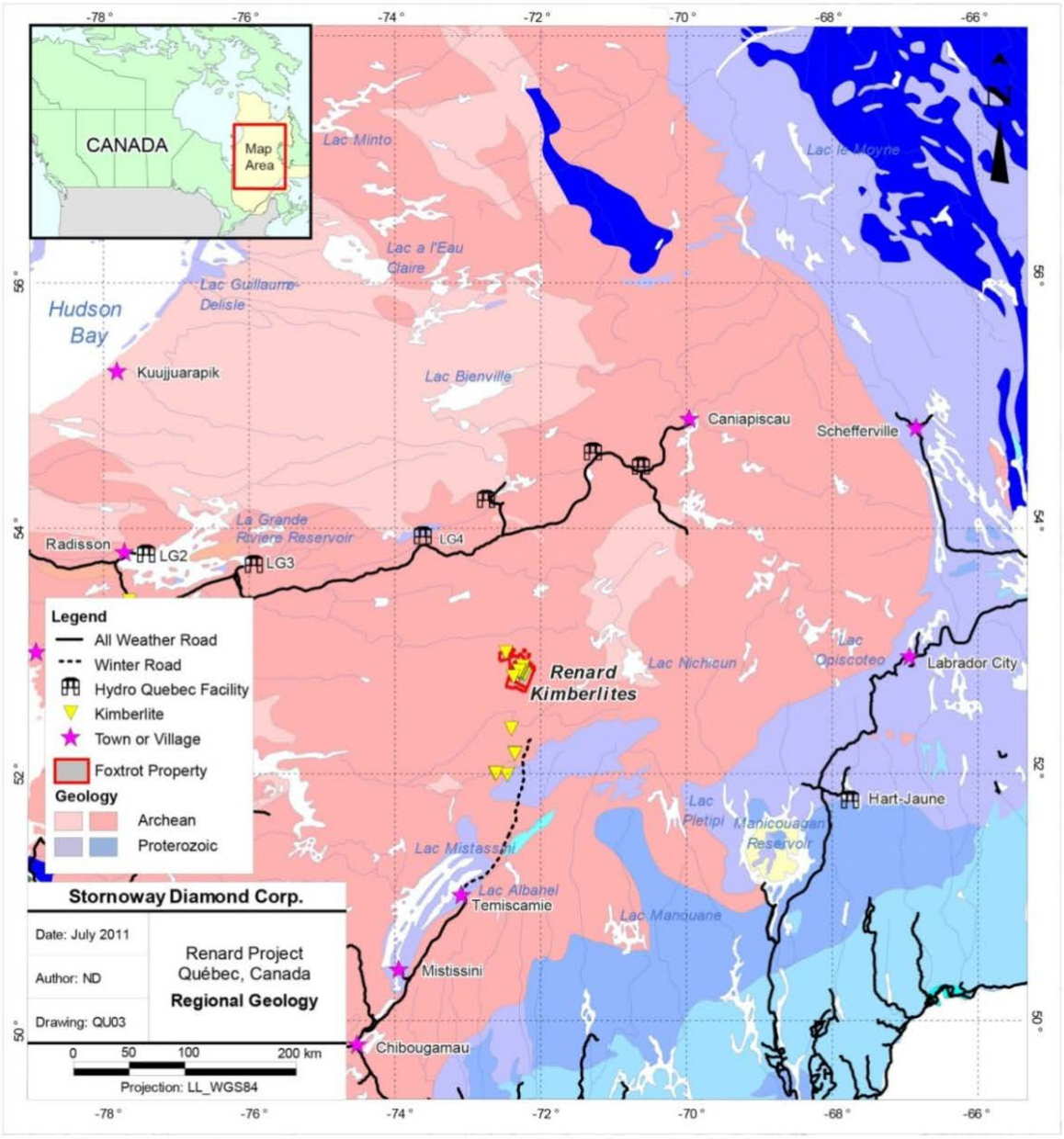

Figure 7-2: Regional Geology - Foxtrot Area

Figure 7.2: Regional Geology – Foxtrot Area

7.2 Project Geology

The Project area is located on the south-eastern portion of the Superior Structural Province bordered by Proterozoic rocks of the Labrador Trough in the east and the Grenville Province in the south (Figure 7.1). This portion of the Superior Craton is sometimes referred to as the "Ungava Craton". Proterozoic rocks of the Labrador Fold Belt in the east, the Cape Smith Fold Belt in the north and the Grenville Province in the south surround the Project area. Northern portions of the Project area consist of north-northwest trending, plutonic and gneissic terranes. Based on metamorphic grade, mineralogy, lithology and aeromagnetic observations, the terranes appear to vary in width from 70 km to 150 km (Percival et al., 1994).

The Foxtrot Property is situated between the La Grande greenstone (volcanic) belt to the north and the Eastmain greenstone (volcanic) belt to the south. Granite-gneiss and retrograde granulite gneiss are the predominant lithologies, with lesser amounts of granite and granodiorite. Contained within the gneiss are relict metasedimentary and metavolcanic rock assemblages along with associated mafic and ultramafic intrusive rocks. The Otish Mountain and Mistassini groups of Proterozoic, clastic, metasedimentary rocks overlie the Archean lithologies, marginal to the Grenville Province. Mafic and ultramafic intrusive rocks of variable affinities are more common in the southeast than in the southwest.

Granite–gneiss and retrograde granulite gneisses of sedimentary origin are the predominant lithologies in the Property area; however, lesser granite and granodiorite may also be present. The gneisses may contain relict metasedimentary and metavolcanic rock assemblages, as well as associated mafic and ultramafic intrusive rocks. Minor linear belts of supracrustal metavolcanic rocks occur throughout the area, generally trending east-west or west-northwest. Northwest-trending, Proterozoic Mistassini Swarm diabase and gabbro dykes up to 30 m wide cross-cut all lithologies. Isolated outliers of Proterozoic clastic metasedimentary rocks are present in the area (O'Connor and Lépine, 2006).

Metamorphic grade within the Foxtrot area is primarily amphibolite facies with local granulite facies being reported near Lac Minto (Percival et al., 1994). Higher-grade lithologies in the north are interpreted as supracrustal relicts dating to 3.1 Ga. Granite and granite gneiss are dated at 2.7 Ga and local felsic and intermediate intrusive rocks are dated at 2.5 Ga.

Glacial overburden within the Foxtrot Property can be up to 30 m thick, but is on average 12 m thick in the area of the Renard Cluster. Glacial deposits consist of till, eskers, moraine and post-glacial sediments, and their orientation reflects ice transport from the north-northeast.

In this report, reference is made to the Renard Core Area, which is defined as a 37-ha area that contains the Renard 2, Renard 3, Renard 4, Renard 65 and Renard 9 kimberlite pipes as well as the Lynx and Hibou dyke systems. Kimberlites are discussed individually in the following sections.

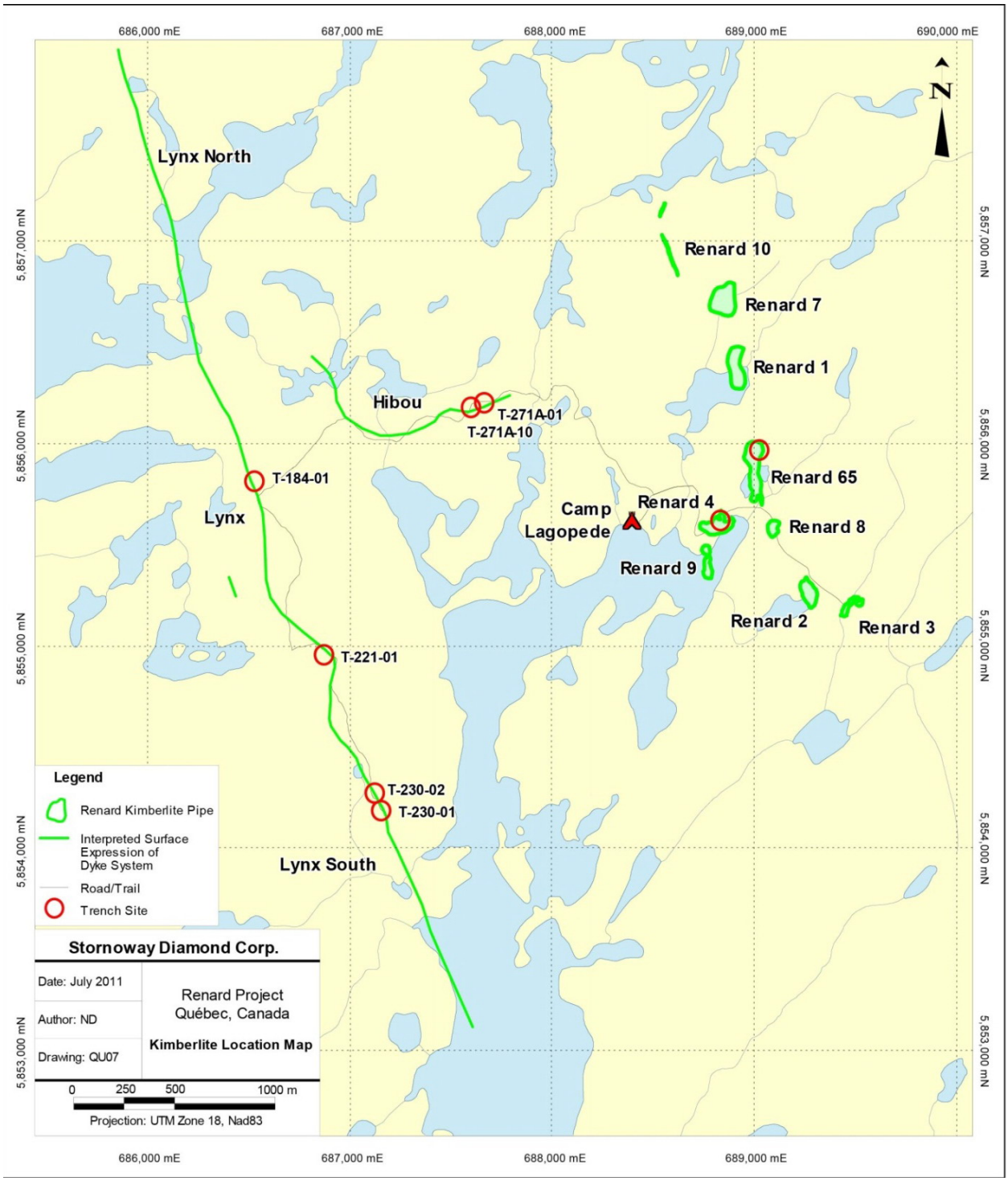

Figure 7.3: Kimberlite Location Map

7.3 Kimberlite Mineralization at Renard

The Renard kimberlite pipes and Lynx and Hibou dykes were emplaced into granitic and gneissic host rocks, and contain diamonds of potential economic interest. The bodies comprise a late Neo-Proterozoic to Cambrian kimberlite field in Québec (Girard, 2001; Moorhead et al., 2002; Letendre et al., 2003).

To date, nine kimberlite pipes have been identified over a 2 km2 area in the Renard Cluster (Renard 1 to Renard 10; with Renard 5 and Renard 6 forming one body, referred to as Renard 65). The kimberlite pipes are typically spaced between 50 m and 500 m from each other (Figure 7.3). Geophysical data and drill information from delineation and bulk sampling programs indicate that, in general, most of the Renard kimberlites are irregular and elliptical in plan view. Surface areas of the kimberlite portion of the pipes range from 0.1 ha to 2.01 ha. A summary of each pipe is presented in Table 7.1. Renard 2, Renard 3, Renard 4, Renard 9 and Renard 65 are the subject of the Mineral Resource Estimate in Section 14 of this report. At the present time, the other kimberlite pipes are considered either too small, have low apparent diamond content, or are not sufficiently sampled or understood to support mineral resource estimation. Additional work on these other bodies may be undertaken at a later date; however, no work plan or budget has been prepared at this time. Two laterally extensive kimberlite dyke systems, known as the Lynx and Hibou dykes, have been identified to the west and northwest of the pipe cluster, respectively (Figure 7.3). Portions of both dykes are included in the mineral resource estimation. Additional dyke-like kimberlites have been discovered elsewhere on the property. These are not included in mineral resource estimation but may warrant additional work at a later date. No work plan or budget has been prepared for this work at this time.

This following sections focus on the internal geology of the pipes which is fundamental to the resource estimation process.

7.3.1 General Geology

Kimberlite nomenclature has evolved several times throughout the work carried out on the Renard kimberlites. The terminology used at this time to describe the rock types in these kimberlites is in accordance with that used in the most recent scientific literature (Field and Scott Smith, 1999; Sparks et al., 2006; Cas et al., 2009). Within this report the following terms and definitions are used:

- Massive Volcaniclastic Kimberlite (MVK): a general term that includes kimberlite classified texturally as tuffisitic kimberlite breccia.
- Coherent Kimberlite (CK): a general term that refers to kimberlite that has not been fragmented (i.e. the magma broken apart as a result of emplacement processes). In general, the term coherent kimberlite is used to refer to large, pipe-infilling events of this nature.
- Hypabyssal Kimberlite (HK): a more specific textural term for CK. Typically used here to describe the detailed texture of a CK rock and commonly used when referring to dykes or irregular intrusions.
- Tuffisitic Kimberlite Breccia (TKB): a more specific textural term for MVK. Characterized by microlitic clinopyroxene in the matrix of the rock.

- Transitional Kimberlite: this refers to kimberlite that shows textures of both volcaniclastic kimberlite and coherent kimberlite. A small “t” denotes a transitional textured HK or TK when describing rock-types (e.g., HKt or TKt).

The Renard kimberlite pipes comprise root zone to diatreme facies rocks characterized by a complex internal geology. These pipes can be classified as “typical” South-African-style kimberlites (Hawthorne, 1973, Clement and Skinner, 1979, Clement, 1982 and Field and Scott Smith, 1999). In most pipes, with the exception of Renard 3 and Renard 10, the dominant phase is a MVK that can be classified as TKB. In general, these TKBs are extensively altered and have a massive texture. They consist of varying amounts of olivine, juvenile clasts and country rock xenoliths that are poorly sorted, typically loosely packed and less commonly clast supported, all set within a highly altered interclast matrix. In many pipes a secondary pipe-filling phase is present that is typically a more coherent or transitional kimberlite characterized by lower country rock xenolith content and higher olivine content set within a dominantly crystalline groundmass. In all bodies, HK is present as both dykes and irregularly shape intrusions that are found within each pipe infilling phase, between contacts of phases and along pipe margins. These are typically considered later stage intrusions. The HK intrusions can vary in thickness from a few centimetres to several metres and, in the case of the Lynx and Hibou dyke system, for example, they are laterally extensive.

The Renard bodies are all associated with extensive cracked country rock (CCR) and with the exception of Renard 3, Renard 7 and Renard 8, have a significant marginal country rock breccia (CRB). The marginal CCR consists of both broken and solid country rock with minor amounts of HK dykes and veins throughout. Minor zones of extensively fractured to brecciated country rock are also present in this unit. The CRB typically lies between the main kimberlite units and the CCR and is characterized by dominantly broken and pulverized country rock, with an overall dilution of 95% or greater. CRB contains from 0% - 5% kimberlite, present as olivine in the pulverized country rock matrix. Locally, the CRB may also contain more than 10% diamond-bearing kimberlitic material, in the form of late-stage, cross-cutting HK dykes.

Each Renard kimberlite contains a variety of phases that are distinguishable from one another by differing macroscopic and microscopic properties as well as diamond grades. A summary of the various kimberlite lithologies present in the Renard bodies is provided in Table 7.2.

Whole rock trace element compositions suggest that the Renards have a close affinity to Group I kimberlite (after Skinner, 1989), with some melnoitic overlap, likely due to contamination by country rock (Birkett et al., 2004). However, petrographic analyses of these rocks support Group I kimberlite classification.

Previous U-Pb dating of groundmass perovskite in HK dykes within Renard 1 suggested an emplacement age of 631.6 ± 3.5 Ma (Birkett et al., 2004). Recent data obtained for the main rock-types in Renards 2 and 3 using the same method suggest an emplacement age of 640.5 ± 2.8 Ma.

Table 7.1: Renard Kimberlite Pipes and Significant Dyke Systems

Pipe (year of discovery)	Surface Area of Kimberlite* (ha)	Dimensions of Kimberlite at Surface* (m)	Vertical Extent of Kimberlite Modelled (m)	Depth of Cover (m)	Major Kimberlite Units	Exploration Potential	Comments
Renard 1 (2001)	1.3	210 x 70	280	6	MVK, transitional CK and large HK intrusion	No additional work planned	Diatreme-zone kimberlite with overall regular shape. Slightly elongate north-south, sub-circular shape at surface.
Renard 2 (2001)	0.8	153 x 70	688	22	MVK and transitional CK	Remains open at depth	Diatreme-zone kimberlite with irregular shape near surface. Slightly elongate north-south at surface. Size of kimberlite portion of pipe increase with depth. Partly covered by lake.
Renard 3 (2002)	0.3	120 x 24	445	12	Transitional CK, MVK, large HK intrusions	Remains open at depth	Root-zone kimberlite with overall irregular shape.
Renard 4 (2002)	1.1	178 x 110	635	11	MVK, transitional CK	Remains open at depth	Partly covered by shallow lake (about 0.4 ha). Diatreme-zone kimberlite with overall regular shape. Slightly elongate, east-west trending sub-circular shape at surface.
Renard 65 (2002)	2.01	330 x 90	585	8	Transitional CK, MVK	Remains open at depth	Diatreme zone kimberlite. Slightly elongate north-south trending irregular shape at surface.
Renard 7 (2002)	1.6	285 x 124	255	5	MVK	No additional work planned	Diatreme-zone kimberlite with overall regular shape and sub-circular shape at surface.
Renard 8 (2002)	0.4	80 x 50	240	8	MVK, large HK intrusion	No additional work planned	Diatreme-zone kimberlite with overall regular shape and sub-circular shape at surface.
Renard 9 (2003)	0.5	163 x 35	374	10	MVK, transitional CK	Remains open at depth	Lower diatreme to root-zone kimberlite. Elongate, north-south trending, elliptical shape at surface. Body shape dips to the east with depth. Entirely covered by 2–4 m deep lake.

Pipe (year of discovery)	Surface Area of Kimberlite* (ha)	Dimensions of Kimberlite at Surface* (m)	Vertical Extent of Kimberlite Modelled (m)	Depth of Cover (m)	Major Kimberlite Units	Exploration Potential	Comments
Renard 10 (2003)	0.1	370 x 12	210	3	CK	No additional work planned	North-northwest trending dyke-like kimberlite. Possible blind intrusion. Almost entirely covered by 3 m deep lake.
Lynx (2003)	na	4230 (strike length only)	95	5	HK	Remains open along strike and down dip	North-northwest trending dyke, variable dip from 10o to 50o to the east.
Hibou (2005)	na	952 (strike length only)	92	8	HK	Remains open along strike and down dip	East-west trending dyke; shallow dip approximately 10o to the north-northeast.

* Based on 3D model; excludes CCR and CRB.

Table 7.2: Major Geological Units

Kimberlite	Major Geological Unit	Dominant Colour	Textural Classification	Textural Classification Codes	Distinguishing Characteristics
Renard 2	Kimb2a	blue to blue green	volcaniclastic kimberlite	TK	high abundance and size of country rock xenoliths (CRx), blue clay matrix
	Kimb2b	brown	magmatic > volcaniclastic kimberlite	HK-HKt toTKt	coarse olivine sizes, lower CRx % and size, abundant perovskite
	Kimb2c	dark green to black	magmatic kimberlite	HK	uniform distribution of crystalline groundmass, low dilution
	CRB-2a	white with blue matrix	na	breccia	highly diluted Kimb2a characterized by very large, tightly packed country rock xenoliths with minor amounts of blue kimberlite matrix.
Renard 3	Kimb3b	blue to blue green	volcaniclastic kimberlite	TK-TKt	high abundance and size of CRx, abundant juvenile clasts
	Kimb3c	dark green to grey/black	magmatic kimberlite	HK	uniform distribution of crystalline groundmass, low dilution
	Kimb3d	black to dark brown	magmatic kimberlite	HK-HKt	strongly altered CRx with black to green alteration rims and bleached centres
	Kimb3g	mottled green-brown to dark brown	magmatic > volcaniclastic kimberlite	HKt-TKt	texturally complex, similar to kimb3f but has lower abundance of HK autoliths and mantle nodules
	Kimb3f	light-dark brown with mottled blue-green zones	volcaniclastic > magmatic kimberlite	TKt-HKt	higher abundance of HK autoliths and higher % of larger CRx than in 3d, mantle nodules present
	Kimb3h	black	magmatic kimberlite	HK	uniform distribution of crystalline groundmass, 15% - 25% commonly bleached CRx
	Kimb3i	black	magmatic kimberlite	HK	uniform distribution of coarse crystalline groundmass, indicator minerals more common
Renard 4	Kimb4a	blue grey to grey green	volcaniclastic kimberlite	TK	high abundance and size of white-pink CRx, common carbonatized olivines
	Kimb4b	brown	magmatic > volcaniclastic kimberlite	HK-HKt to TKt	coarse olivine sizes, lower CRx % and size, common mantle nodules, abundant perovskite
	Kimb4c	black to dark green	magmatic kimberlite	HK (to HKt)	uniform distribution of crystalline groundmass, low dilution
	Kimb4d	dark blue	highly variable - volcaniclastic > magmatic kimberlite	TK-TKt with HKt-HK zones	dark blue clay matrix, common indicator minerals, higher abundance of HK autoliths and HK dykes
Renard 65	Kimb65a	pale blue-grey to dark grey-green	volcaniclastic kimberlite	TK-TKt	high abundance and size of CRx, abundant juvenile clasts
	Kimb65b	black to dark brown	magmatic kimberlite	HK-HKt	strongly altered CRx dark green to partially bleached centres, common mantle nodules

Kimberlite	Major Geological Unit	Dominant Colour	Textural Classification	Textural Classification Codes	Distinguishing Characteristics
	Kimb65c	dark green to grey/black	magmatic kimberlite	HK	uniform distribution of crystalline groundmass, low dilution, common flow banding of olivines
	Kimb65d	light-dark brown with mottled blue-green zones	volcaniclastic > magmatic kimberlite	TKt-HKt	dirty brown appearance, creamy green and yellow rimmed CRx, common HK autoliths
	Kimb65e	dark brown	magmatic > volcaniclastic kimberlite	HKt	complex magmatic transitional unit in the southern part of the body
Renard 9	Kimb9a	greyish green	volcaniclastic kimberlite	TK	high abundance and size of CRx, abundant juvenile clasts, pyrite replaces olivines
	Kimb9b	black to brownish black	magmatic > volcaniclastic kimberlite	HK-HKt (rare TKt)	varies in texture from east (HK) to west (TKt)
	Kimb9c	dark green to grey/black	magmatic kimberlite	HK	uniform distribution of crystalline groundmass, two types – non-magnetic carbonate rich and magnetic
Lynx		dark green to black	magmatic kimberlite	HK	uniform distribution of crystalline groundmass, coarse olivines, abundant ilmenite
Hibou		dark green to black	magmatic kimberlite	HK	uniform distribution of crystalline groundmass, very coarse olivines common, abundant ilmenite and chrome diopside
All	CR	variable	na	na	country rock
	CCR	variable	na	na	large country rock blocks separated by thin kimberlite-filled fractures and HK dykes situated around the main kimberlite phases
	CRB	variable	na	breccia	brecciated country rock
	CRB+K	variable	na	breccia	brecciated country rock with up to 5% kimberlite material usually identified as olivine macrocrysts

Note:
HK = hypabyssal kimberlite; HKt = transitional hypabyssal kimberlite; HK-HKt = hypabyssal to transitional kimberlite; HKt-TKt = magmatic to tuffisitic kimberlite; TK = tuffisitic kimberlite; TK-TKt = tuffisitic to transitional tuffisitic kimberlite.

7.3.2 Renard 2

The Renard 2 kimberlite is a mid-sized kimberlite pipe within the Renard Cluster. It is interpreted as a diatreme-zone kimberlite with irregularities in the external shape of the kimberlite near the surface, but an overall smooth and tapering shape when considering the emplacement envelope (the CRB and CCR). The shape of the kimberlite portion of the pipe expands with depth. The internal geology of Renard 2 (Figure 7.4 and Figure 7.5) was established using geological logs of both core and reverse circulation drill holes combined with detailed mapping of the underground drift and petrographic and geochemical studies. A detailed description of the geology and emplacement history of Renard 2 is described in Fitzgerald et al. (2009).

Renard 2 consists of two main pipe-infills: an MVK referred to as Kimb2a and a transitional CK referred to as Kimb2b. The two main infills exhibit contrasting primary textures: olivine abundances and populations, country rock xenolith abundances and populations, and diamond contents. The kimberlite is surrounded by extensive marginal CRB and CCR. In addition to these, HK dykes and intrusions (referred to as Kimb2c) of varying thickness are found throughout the body, along the pipe contacts, within the marginal breccia and in the cracked country rock (Fitzgerald et al., 2009).

Each kimberlite unit can be described as follows:

Kimb2a is volumetrically the most significant kimberlite rock type infilling the pipe, accounting for 56% of total kimberlite by volume. It is extensively altered, massive and can be texturally classified as a TKB (Clement and Skinner, 1979; Clement, 1982; Clement and Reid, 1989; Field and Scott Smith, 1999; Hetman, 2008). This blue to blue-green rock consists of olivine, juvenile clasts and country rock xenoliths that are poorly sorted, typically loosely packed and less commonly clast supported and set within a highly altered interclast matrix. Olivine macrocrysts comprise 10% of the rock and are typically medium-grained to rarely coarse-grained. Juvenile clasts are common, have sharp margins and are of both cored and uncored varieties (Webb, 2006). Primary groundmass minerals in juvenile clasts include phlogopite, spinel and perovskite. This unit commonly contains 50% - 75% fresh to moderately altered granitoid and lesser gneissic country rock xenoliths (granitoid dominate). These are set within a non-crystalline matrix that comprise clays, serpentine and microlitic clinopyroxene. This rock can be classified mineralogically as a phlogopite kimberlite.

Kimb2b is the second most volumetrically significant kimberlite rock type, accounting for 44% of total kimberlite by volume. This rock is a moderately altered, massive and texturally variable CK that displays both coherent and rarer volcaniclastic (i.e. TKB) textures. This brown rock consists of olivine, country rock xenoliths and rare juvenile clasts that are poorly sorted and set within a dominantly crystalline groundmass. Olivine macrocrysts comprise 15% - 30% of the rock and are medium- to coarse-grained with common very coarse grains. Juvenile clasts are relatively rare and, where observed, typically have diffuse margins. Primary groundmass minerals include phlogopite, perovskite, spinel, carbonate and rare monticellite. Secondary minerals in the groundmass include serpentine, clays and microlitic clinopyroxene. This unit commonly contains 20% - 50% moderately to strongly altered granitoid and gneissic country rock xenoliths (gneissic appear to dominate). This rock can be classified mineralogically as a monticellite phlogopite kimberlite.

Contacts between these two rock types are sharp and commonly highlighted by HK dyke intrusions, as well as changes in the proportion of country rock xenoliths in Kimb2a. The contrasting textural and component features described above and the sharp nature of the contacts, combined with the difference in diamond grade, supports interpretation of the two pipe infills as two distinct rock-types emplaced during two separate volcanic events. The emplacement of Kimb2b is thought to have preceded that of Kimb2a based on the contact relationships between these two phases and the relative distribution of these within the pipe, as it is often identified as remnant blocks within Kimb2a near surface and is present marginally to Kimb2a at depth.

Coherent kimberlite also occurs in Renard 2 in the form of late-stage HK dykes and irregular intrusions (referred to as Kimb2c), ranging in thickness from a few centimetres up to 15 m and contributing greater than 15% by volume. Kimb2c is black to dark-green coloured with medium- to coarse-grained and, commonly, very coarse-grained macrocrystic olivine. The various HK intrusions include a spectrum of primary groundmass mineral assemblages, ranging from monticellite dominated to phlogopite dominated, each additionally consisting of common carbonate, spinel and perovskite. This unit does not commonly contain country rock xenoliths but, where present, comprises < 5% of the unit and are strongly altered.

Also modelled in Renard 2 is the CRB-2a unit. This internal country rock breccia was identified during mapping of the underground exposure and drill core logging, and consists of large fresh blocks of white granitoid and gneiss with a minor matrix component manifested as < 5 cm wide, blue coloured veins and Kimb2a intersections. Country rock xenolith content is 95% or greater. The presence of the Kimb2a intersections, overall blue colour of the matrix, and similarities in appearance, type and alteration state of the country rock xenoliths, suggest that this CRB represents a highly diluted zone of Kimb2a. Petrographic examination of matrix zones supports this interpretation.

The kimberlite in Renard 2 is surrounded entirely by extensive marginal CRB and CCR. The CRB has a maximum width of approximately 100 m near surface and extends the full depth of the body on the north side of the pipe. The CCR is present in the upper portion of the pipe and is spatially associated with the Kimb2a unit. More HK dykes are present within the CCR unit on the western side of the body as compared to the eastern side.

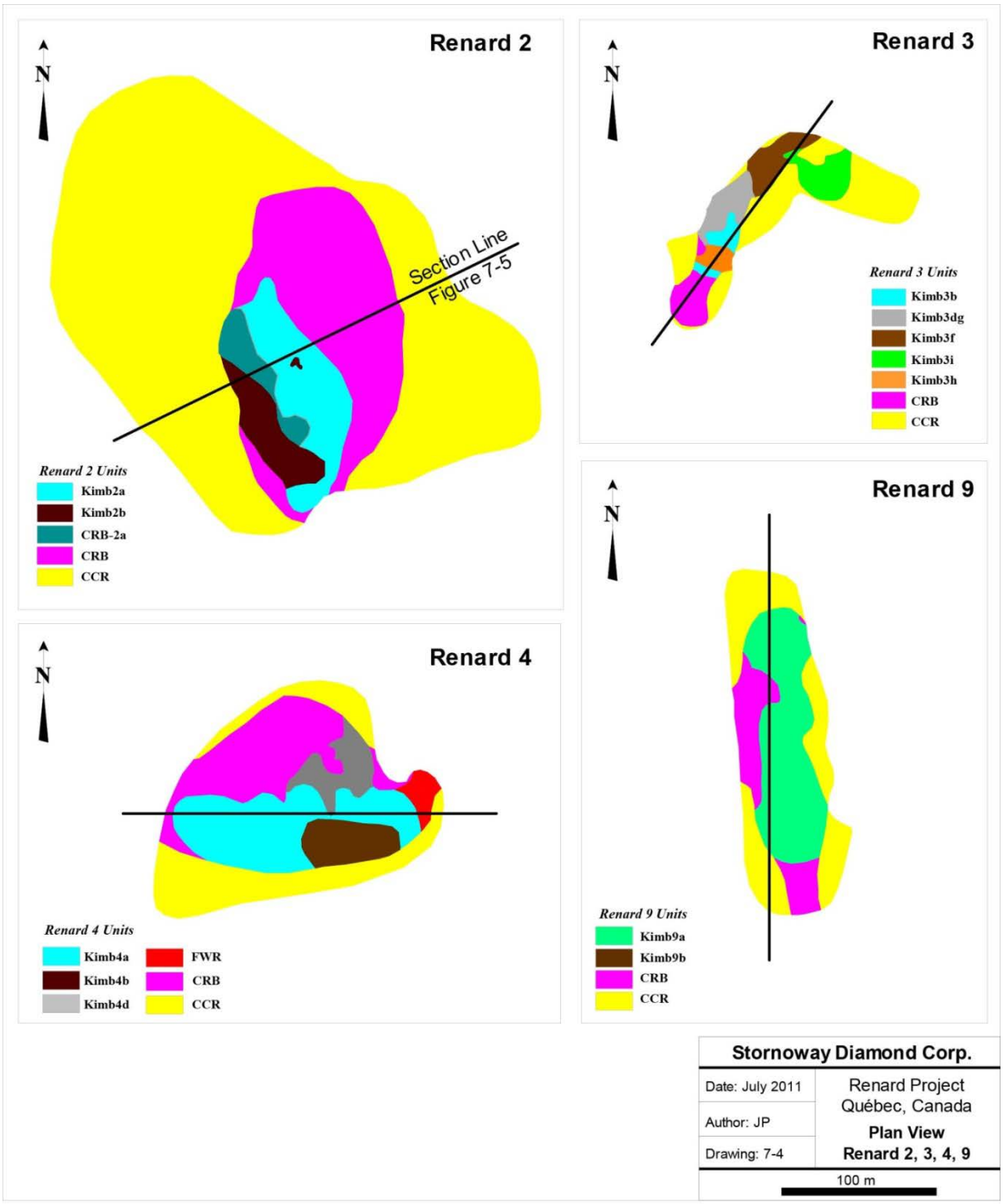

Figure 7.4: Plan View – Renard 2, Renard 3, Renard 4 and Renard 9

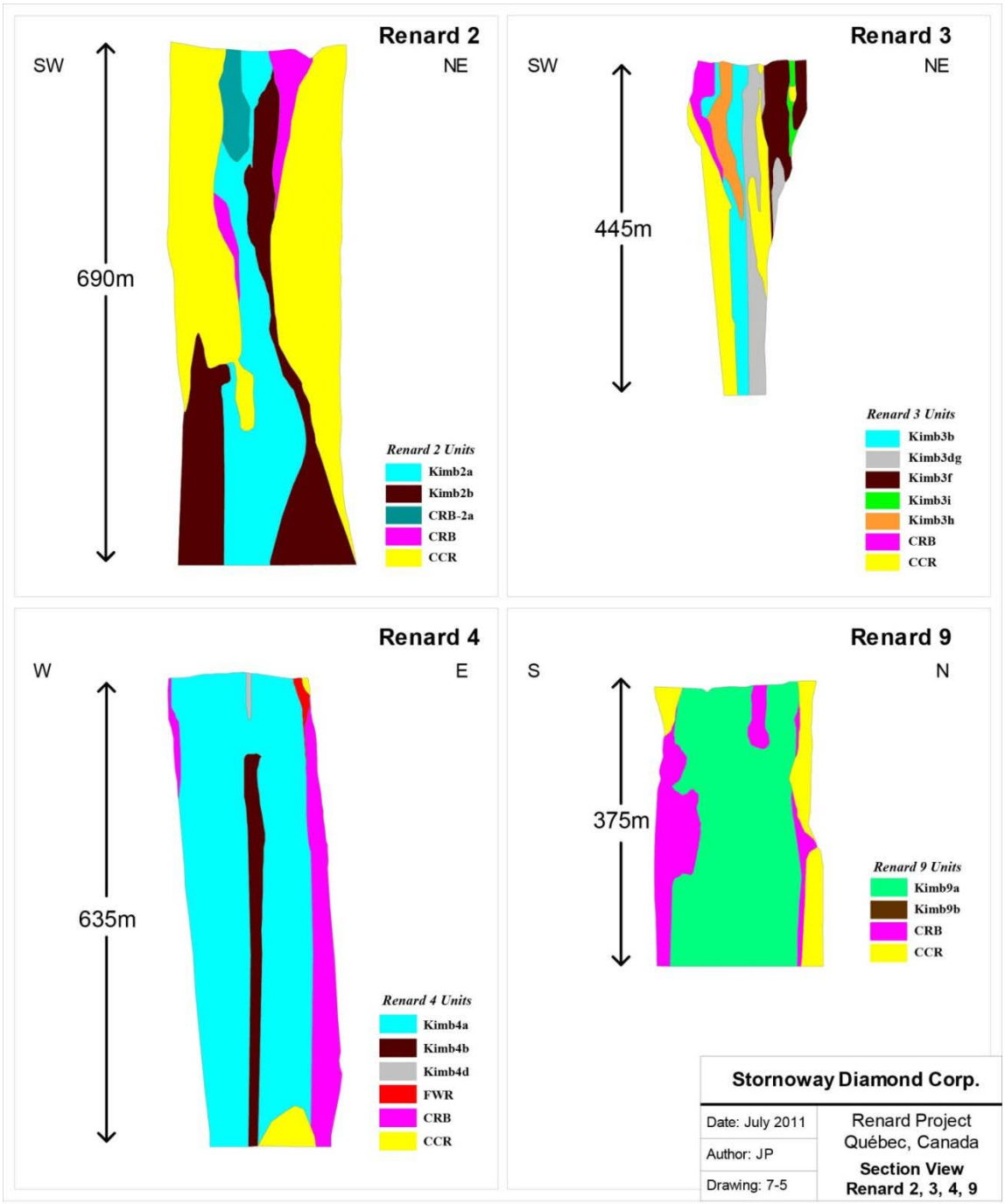

Figure 7.5: Section View – Renard 2, Renard 3, Renard 4 and Renard 9

7.3.3 Renard 3

The Renard 3 kimberlite is one of the smallest kimberlite pipes within the Renard Cluster. It is interpreted as a complex, steep-sided, root-zone kimberlite, with several irregularities in the external shape. The geology of this kimberlite has been determined using detailed logging of drill core, mapping of the underground decline walls, and results of petrographic and geochemical studies (Figure 7.4 and Figure 7.5).

The current three-dimensional (3D) model of Renard 3 has two possible feeder zones and consists of six kimberlite rock types in order of volumetric significance: Kimb3d, Kimb3g, Kimb3i, Kimb3b, Kimb3h, and Kimb3f. Currently, the Kimb3d and Kimb3g units are modelled together as one 3D solid, based on their comparative mineralogy and abundances of groundmass minerals observed through thin section analysis, as well as similar grade and diamond counts (stones per tonne) obtained from underground sampling. A small CRB zone exists on the upper, southern portion of the pipe, in association with the Kimb3b phase. More extensive CCR discontinuously surrounds the majority of the kimberlite pipe margin. In addition to the six main kimberlite rock types infilling the pipe, a number of HK dykes and intrusions (Kimb3c) occur throughout the body, along the pipe contacts, within the marginal breccia and in the cracked country rock.

Each kimberlite unit can be described as follows:

Kimb3d is a coherent (HK/HK(t)), black to dark brown kimberlite with medium- to coarse-grained and common very coarse-grained macrocrystic olivine. This unit commonly contains 20% - 40% strongly altered, dark green to black-rimmed granitoid and lesser gneissic country rock xenoliths with bleached centres. The matrix is crystalline and dominated by primary groundmass minerals consisting of phlogopite, carbonate, perovskite, spinel and rare monticellite. Texturally this rock is classified as a CK and mineralogically as a phlogopite kimberlite.

Kimb3g is a transitional volcaniclastic (HKt/TKt), dark brown to mottled-brown-green kimberlite with medium- to coarse-grained and rare very coarse-grained macrocrystic olivine. This unit contains approximately 30% - 50% granitoid and gneissic country rock xenoliths moderately altered with creamy green and yellow rims. Juvenile clast abundance is variable and exhibits common thick selvages. The matrix is non-crystalline to crystalline and contains both primary groundmass minerals and patchy zones of secondary alteration minerals. Texturally this rock is classified as an HKt/TKt and mineralogically as a phlogopite kimberlite. Kimb3d and Kimb3g are modelled as one unit and together account for 47% of the kimberlite by volume.

Kimb3i is a coherent (HK), black kimberlite with medium- to coarse-grained and common very coarse-grained macrocrystic olivine. This unit contains on average 10% - 20% strongly altered granitoid and gneissic country rock xenoliths, with alteration style varying from entirely bleached and dark-green coloured to having black rims with bleached centres. The matrix is uniform, crystalline and dominated by primary groundmass minerals consisting of carbonate, phlogopite, perovskite, spinel and rare apatite. Texturally this rock is classified as a CK and mineralogically as a phlogopite carbonate kimberlite. Kimb3i represents 15% of total kimberlite by volume.

Kimb3b is a blue to blue-grey volcaniclastic (TK/TKt) kimberlite breccia with fine- to medium-grained, and less commonly coarse-grained macrocrystic olivine. This unit contains altered pink to patchy green granitoid xenoliths, with lesser gneissic xenoliths that are fresh to moderately altered green. Xenolith content varies from 55% - 75%. Juvenile clasts are common and are

both cored and uncored. The matrix is dominated by clinopyroxene, serpentine and clay, contributing to its blue-grey colour. Texturally this rock can be classified as a TKB and mineralogically as a phlogopite kimberlite. Kimb3b represents 17% of total kimberlite by volume.

Kimb3h is a coherent (HK) black kimberlite with medium- to coarse-grained and rare, very coarse-grained macrocrystic olivine. This unit contains 15% - 25% strongly altered, dark green to black-rimmed granitoid and gneissic country rock xenoliths with bleached centres. The matrix is crystalline with primary groundmass minerals consisting of phlogopite, carbonate, spinel and perovskite. Texturally this rock is classified as a CK and mineralogically as a carbonate phlogopite kimberlite. Kimb3h represents 11% of total kimberlite by volume.

Kimb3f is a volcaniclastic (TKt/HKt) brown to mottled green-blue kimberlite with medium-grained and common coarse-grained macrocrystic olivine. This unit contains 35% - 50% gneissic and granitoid country rock xenoliths, moderately altered with creamy yellow to green rims. Juvenile clasts with thick selvages are abundant, however, are commonly diffuse and are not always easily observed, particularly in darker, more magmatic zones. Autoliths of HK are abundant and, in general, this unit contains the highest percentage of autoliths compared to other units in the Renard 3 body. Texturally this rock is classified as a TKt/HKt and mineralogically as a phlogopite kimberlite. Kimb3f represents 10% of total kimberlite by volume.

Coherent kimberlite also occurs in Renard 3 in the form of late-stage HK dykes and irregular intrusions (Kimb3c). They range in thickness from a few centimetres up to 20 m. Kimb3c is a black to dark-green HK with medium- to coarse-grained, and commonly, very coarse-grained macrocrystic olivine. This unit does not commonly contain country rock xenoliths, but where present, they comprises < 5% of the unit.

CRB is found only in the southern, uppermost part of the pipe and therefore comprises only a minor portion of the entire kimberlite pipe. It has a maximum width of 45 m, wrapping around the southern end of the pipe, and extends vertically for almost 175 m. The kimberlite is surrounded from top to bottom by discontinuous CCR.

The emplacement history of Renard 3 is complex and not fully understood at this time. However, textural features of the rocks, contact relationships and distribution of the rock types in the pipe suggest that Kimb3b and Kimb3f were emplaced early during pipe development, and Kimb3d, Kimb3g and Kimb3i were emplaced later.

7.3.4 Renard 4

The Renard 4 kimberlite is one of the larger kimberlite pipes within the Renard Cluster. It is interpreted as a diatreme-zone kimberlite with minor irregularities in the external shape. The geology of this kimberlite has been determined using detailed logging of drill core and mapping of the surface trench (Figure 7.4 and Figure 7.5).

The Renard 4 pipe contains three kimberlite geological units: an MVK, referred to as Kimb4a, a transitional CK, referred to as Kimb4b, and a texturally complex MVK, referred to as Kimb4d. In the current 3D model there is also a unit referred to as FWR (further work required) on the eastern edge of the body that is currently unclassified. It may represent a more highly diluted Kimb4a or Kimb4d. A significant marginal CRB and CCR surround the main kimberlite. The

Renard 4 CCR adjoins at depth with the CCR of Renard 9, along the southern margin of the pipe. In addition to the three main pipe infills, a number of HK dykes and irregular intrusions occur throughout the body, along the pipe contacts, within the marginal breccia and in the cracked country rock.

Each kimberlite unit can be described as follows:

Kimb4a is an MVK that can be further classified as a TKB. It is blue to green coloured with medium-grained and, less commonly, fine-grained macrocrystic olivine. A characteristic feature of this unit is that the olivine macrocrysts are commonly carbonatized, which is not typical of other Renard TKB's. Juvenile clasts are common and are of both cored and uncored varieties. The unit contains 50% - 80% granitoid and gneiss country rock xenoliths that are fresh to moderately altered. The matrix is dominated by clay, serpentine carbonate and clinopyroxene, contributing to its blue-green colour. Kimb4a has been divided into two sub-types based on subtle differences in xenolith character and mineralogy: Kimb4a-1 is a blue-grey rock with fresh xenoliths and is mineralogically classified as a phlogopite monticellite kimberlite; Kimb4a-2 is a green rock with dark pink, hematized xenoliths that can be classified mineralogically as a phlogopite spinel monticellite kimberlite. Further work is required in order to determine if in fact these are two separate phases, or if the differences in the rock characteristics are due to variations within one phase. Kimb4a represents 71% of total kimberlite by volume.

Kimb4b is a transitional volcaniclastic (HK/HKt/TKt), mottled light to dark brown kimberlite with medium- to coarse-grained macrocrystic olivine. Juvenile clasts are present and typically appear as diffuse magmatic domains within a more altered groundmass. Their abundance is variable and they exhibit thick selvages. Country rock xenoliths are fresh to more commonly, strongly altered granitoid and gneiss, with content ranging from 25% - 50%. The groundmass varies from crystalline to non-crystalline. Mantle nodules are very common and characterize this unit. Mineralogically this rock is classified as a perovskite monticellite phlogopite kimberlite. Kimb4b represents 25% of total kimberlite by volume.

Kimb4d is a volcaniclastic to transitional (TK/TKt), dark blue kimberlite with medium-grained and, less commonly, fine- and coarse-grained macrocrystic olivine. It is a highly variable kimberlite texturally that contains common coherent (HK) and transitional coherent (HKt) intervals within an overall MVK texture. These more coherent zones occur as both dykes and possible autoliths of undiluted Kimb4d. Juvenile clasts are both uncored and cored varieties, some with diffuse margins. Country rock xenoliths are fresh to moderately altered granitoid and gneiss and represent 40% 75% of the rock. The matrix is dominated by serpentine, carbonate, clay minerals, clinopyroxene microlites and mica. Mantle indicator minerals are common and notably more abundant than in the other units. Mineralogically this rock can be classified as a phlogopite spinel monticellite kimberlite. Kimb4d represents 4% of total kimberlite by volume.

Contacts between these rock types are sharp and commonly highlighted by HK dyke intrusions. Contrasting textural and component features described above and the sharp nature of the contacts, combined with the difference in diamond grade, supports interpretation of the three pipe infills as distinct rock-types emplaced during three separate volcanic events. The emplacement of Kimb4d is thought to have preceded that of Kimb4a based on the contact relationships between these two phases and the fact that Kimb4d appears as a remnant block of kimberlite that does not extend to depth. A second pipe kimberlite emplacement event resulted in the removal of a significant volume of Kimb4d and infilling of the diatreme by

Kimb4a. Finally, the Kimb4b unit removed a portion of the Kimb4a unit and infilled the south eastern portion of the pipe. Emplacement of Kimb4a is thought to have preceded formation of Kimb4b based on the presence of a remnant block of Kimb4a within the Kimb4b unit.

CK also occurs in Renard 4 in the form of late-stage HK dykes and irregular intrusions (Kimb4c) contributing in the order of 8% by volume. They range in thickness from a few centimetres up to 34 m. Kimb4c is a black to dark-green HK with medium- to coarse-grained and, commonly, very coarse-grained macrocrystic olivine. This unit does not commonly contain country rock xenoliths but, where present, they comprise < 5% of the unit and are strongly altered.

CRB is present dominantly in the north and west portion of the body, with smaller blocks in the south and east. The marginal breccia is both vertically and horizontally extensive, reaching a maximum width of 60 m adjacent to the body near surface and extending almost 650 m vertically. CCR surrounds the kimberlite from top to bottom discontinuously, however it is more extensive on the southern side of the body. The southern part of the Renard 4 CCR merges with the northern side of the Renard 9 CCR at approximately 210 m depth below surface and was modelled as one continuous unit.

7.3.5 Renard 9

The Renard 9 kimberlite is one of the smaller kimberlite pipes within the Renard Cluster. It is interpreted as a lower diatreme to root-zone kimberlite with an irregular external shape that dips to the east with depth. The internal geology of Renard 9 has been established using geological logs of both drill core and reverse circulation drill holes (Figure 7.4 and Figure 7.5).

Renard 9 consists of two main pipe-infills: an MVK referred to as Kimb9a and a volumetrically minor (2%) transitional CK referred to as Kimb9b. In general, the texture of these kimberlite phases change from east to west across the body: Kimb9b changes from more HK-like in the east to more transitional TK-like to the west; and the dilution of Kimb9a changes significantly from east to west. An extensive CRB is present on the western side of the body, across both the length and height of the body. The CRB is spatially associated with the Kimb9a unit. Also peripheral to the kimberlite is significant CCR, which joins with that of Renard 4 at depth. In addition to these units, hypabyssal kimberlite (HK) dykes and intrusions (referred to as Kimb9c) of varying thickness are found throughout the body, along the pipe contacts, within the marginal breccia and in the cracked country rock. There are two sub-types of HK in Renard 9: Kimb9c 1 is a black, strongly magnetic HK with dyke-like contacts, whereas Kimb9c 2 is a grey, very carbonate rich, non-magnetic HK with irregular contacts. Kimb9c-2 may in fact represent undiluted Kimb9a, occurring as autoliths throughout the TKB.
Each kimberlite unit can be described as follows:

Kimb9a is volumetrically the most significant kimberlite rock type infilling the pipe, accounting for 98% of total kimberlite by volume. It is extensively altered, generally massive and can be texturally classified as a TKB. This pale-green to grey-green rock consists of olivine, juvenile clasts and country rock xenoliths that are poorly sorted, typically loosely packed and less commonly clast, supported and set within a highly altered interclast matrix. Olivine macrocrysts are typically fine- to medium-grained with rare coarse grains and are commonly altered to serpentine, carbonate and rare pyrite. Two mineralogical types of juvenile clasts are observed in Kimb9a. Both have sharp margins and include both cored and uncored varieties. Type one is

classified as a spinel phlogopite kimberlite and type two as a spinel phlogopite monticellite kimberlite. This unit contains on average 60% - 75% fresh to moderately altered granitoid and gneissic country rock xenoliths set within a non-crystalline matrix comprising clays, serpentine and microlitic clinopyroxene.

Kimb9b accounts for only 2% of total kimberlite by volume. The relative abundance of this rock type in the body, position and contact relationships suggest that Kimb9b is a remnant kimberlite phase emplaced before Kimb9a. This rock is a moderately altered, massive and texturally variable CK that displays both coherent and more rare volcaniclastic (i.e. TKB) textures. This brown kimberlite consists of olivine, country rock xenoliths and rare juvenile clasts that are poorly sorted and set within a dominantly crystalline groundmass. Olivine macrocrysts are medium- to coarse-grained. Juvenile clasts are relatively rare and, where observed, typically have diffuse margins. Groundmass minerals consist of perovskite, monticellite and phlogopite, set within an inhomogeneous, variably crystallized, interclast matrix dominated by carbonate, clays and microlitic clinopyroxene. This unit contains 20% - 50% moderately to strongly altered granitoid and gneissic country rock xenoliths. This rock exhibits textures of both MVK and a CK and can be classified mineralogically as a perovskite monticellite phlogopite kimberlite. The abundance and character of perovskite and phlogopite are diagnostic for this unit.

Two types of HK occur in Renard 9; late-stage HK dykes (Kimb9c 1) and irregular intrusions or autoliths (Kimb9c 2). These are typically less than 1 m thick in drill core and range from black to dark green to grey coloured. Kimb9c 1 can be classified mineralogically as a phlogopite monticellite kimberlite and Kimb9c 2 can be classified as a phlogopite spinel monticellite kimberlite with extensive carbonate throughout. This unit does not commonly contain country rock xenoliths, but where present, they comprise < 5% of the unit and are strongly altered.

The Renard 9 kimberlite has an extensive marginal CRB on the western and southern sides of the body, spatially associated with Kimb9a. It has a surface width of approximately 30 m and extends the full depth of the body. Contacts between this and the Kimb9a unit are typically gradational. Peripheral CCR with rare HK dykes discontinuously surrounds the kimberlite and joins with the CCR of Renard 4 at depth

7.3.6 Renard 65

The Renard 65 kimberlite is the largest kimberlite in the Renard Cluster. It is interpreted as a diatreme-zone kimberlite with a slightly irregular shape at surface, and tapering gradually with depth. The internal geology of this kimberlite has been determined using geological logs of both core and reverse circulation drill holes and in a preliminary fashion using examination of field logs of drill core (Figure 7.6).

Renard 65 consists of four main pipe-infilling kimberlite units: Kimb65a, Kimb65b, Kimb65d and Kimb65e. In addition, CRB and CCR surround the main kimberlite pipe infills. In the current 3D geological model, Kimb65e was modelled with Kimb65a as its distribution was such that it was hard to group it in 3D, suggegsting that it was a remnant unit in Kimb65a. In addition, a number of HK dykes and irregular intrusions (Kimb65c) occur throughout the body as late-stage intrusions along the pipe contacts and within the CRB and CCR.

Each kimberlite unit can be described as follows:

Kimb65a is a pale blue-grey to green-grey volcaniclastic (TK/TKt) kimberlite breccia with fine- to medium-grained and, less commonly, coarse-grained macrocrystic olivine. This unit contains 50% 75% fresh to altered, pink to patchy-green-coloured granitoid xenoliths, with lesser amounts of gneiss. Juvenile clasts are common and are both cored and uncored. Mineralogically this rock can be classified as a phlogopite kimberlite, and contributes 78% by volume to the kimberlite.

Kimb65b is a coherent (HK/HKt) black to dark brown kimberlite with coarse to very coarse-grained macrocrystic olivine. This unit contains 5% 35% strongly altered, dark-green to black-rimmed gneissic and lesser granitoid country rock xenoliths. Mantle nodules are abundant and include harzburgite, peridotite and possible minor dunite types. The matrix is crystalline with minor patches of alteration from the digestion of country rock xenoliths. Kimb65b represents 8% of the kimberlite by volume.

Kimb65d is a volcaniclastic (HKt/TKt) light to dark brown, occasionally mottled green kimberlite with medium- to coarse-grained and, rarely, very coarse-grained macrocrystic olivine. This unit contains 30% - 50% (may reach up to 70%) granitoid and gneissic country rock xenoliths. These are moderately altered with pale green and yellow rims. Juvenile clasts are present, are both cored and uncored, and are typically < 3 cm in size. HK autoliths are also present and are round with sharply defined margins. The matrix varies from crystalline to non-crystalline, with patchy zones of secondary alteration. Kimb65d represents about 14% by volume.

Kimb65e is a complex, volcaniclastic to coherent (HKt), dark-brown kimberlite with medium- to coarse-grained macrocrystic olivine. This unit contains 25% - 55% gneissic and granitoid country rock xenoliths (granite may dominate), fresh to moderately altered pale green, with stronger alteration in smaller xenocrysts. The matrix is crystalline to patchy crystalline with alteration minerals from xenolith digestion. The primary groundmass mineralogy includes phlogopite, carbonate, spinel and perovskite. Due to the irregular nature of the contacts, Kimb65e is currently modelled in Kimb65b.

The HK dykes (Kimb65c) that occur in Renard 65 drill core range in thickness from a few centimetres up to 35 m. Kimb65c is a coherent black to dark-green kimberlite with medium- to coarse-grained and, commonly, very coarse-grained macrocrystic olivine. This unit does not commonly contain country rock xenoliths, but where present, they comprise < 5% of the unit.

Renard 65 kimberlite is surrounded by extensive CRB and CCR units that extend to depth but are discontinuous around the pipe. CRB is dominant on the western side of the body whereas CCR dominates on the east and north sides.

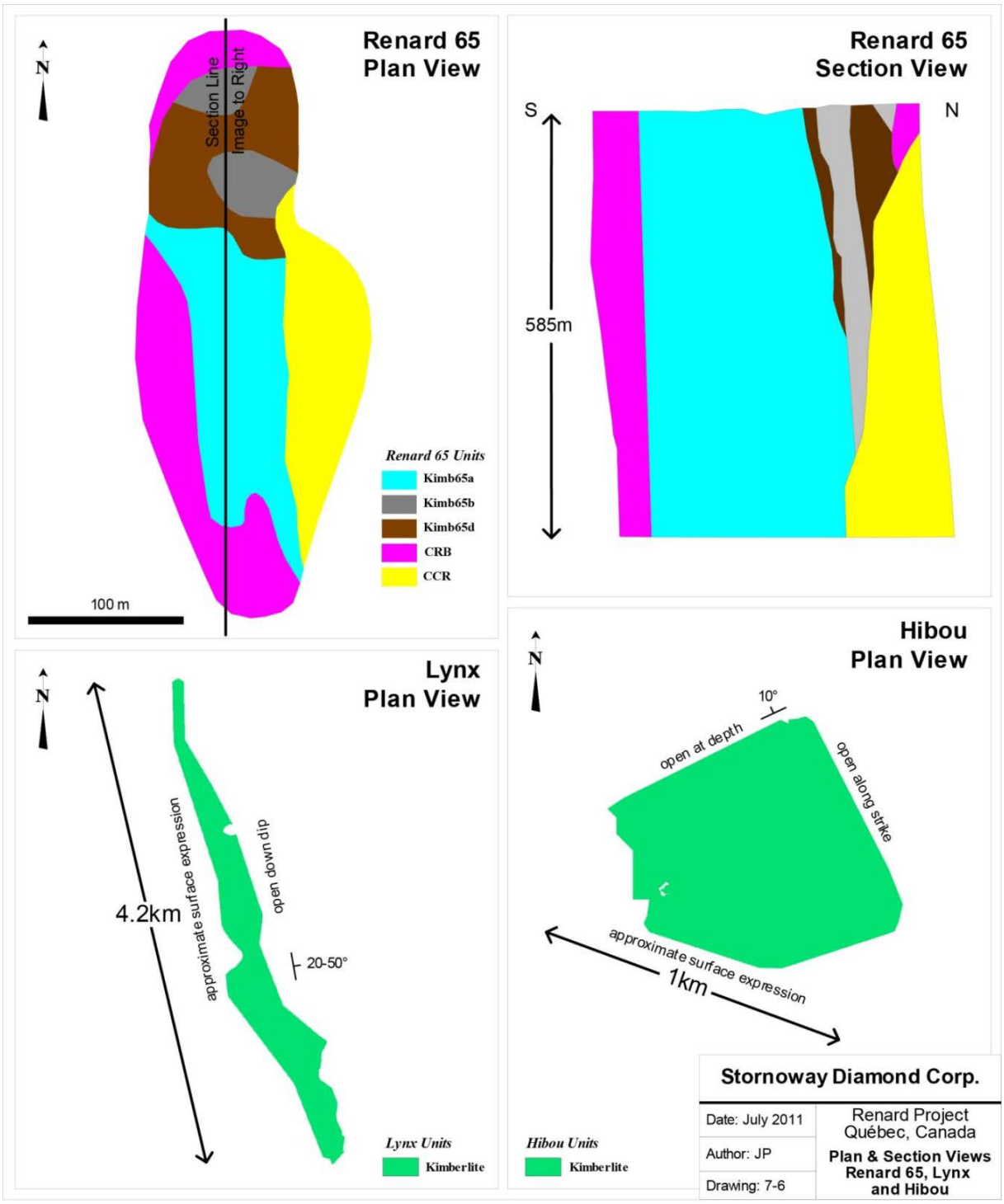

Figure 7.6: Plan and Section Views: Renard 65, Lynx and Hibou

7.3.7 Other Renard Kimberlite Pipes

The remaining Renard kimberlite pipes (Renard 1, Renard 7, Renard 8 and Renard 10) are currently not part of the economic assessment but are included in the following summary as future work may be undertaken on these bodies. Plan and Section views are shown in Figure 7.7 and Figure 7.8.

Renard 1

The Renard 1 kimberlite is the second largest kimberlite in the Renard Cluster. It is interpreted as a diatreme-zone kimberlite with a slightly irregular shape at surface, and tapering gradually with depth. The geology of this kimberlite has been determined in a preliminary fashion using 11 drill cores. A twelfth hole (537 m), completed in 2010, crossed the body from south to north at an inclination of -58° and intersected kimberlite CRB and CCR from 195 to 508 m downhole, and supports the model shown in Figure 7.7 and Figure 7.8. At the present time insufficient diamond data exists to allow the classification of Renard 1 as a resource.

Renard 1 consists of three main pipe-infilling kimberlite units: an MVK referred to as Kimb1a (comprising 61% of total kimberlite); a transitional CK referred to as Kimb1b (comprising 27% of total kimberlite) and a large, irregular HK intrusion referred to as Kimb1c (comprising 11% of total kimberlite). It is unclear at this time how these kimberlites relate temporally, and whether or not they represent a textural change within one kimberlite phase or are three distinct phases. In addition to these main infills, a CRB and CCR almost completely surround the main kimberlite pipe infills. The CRB is much larger on the western side of the body and the CCR is much larger on the eastern side. These are extensive and extend to depth in the kimberlite, however are discontinuous around the pipe. A number of HK dykes and irregular intrusions occur throughout the main phases of kimberlite, along the pipe contacts and within the marginal CRB and CCR units. These are also referred to as Kimb1c because at this time they cannot be distinguished geologically from HK in the large intrusion when examining drill core.

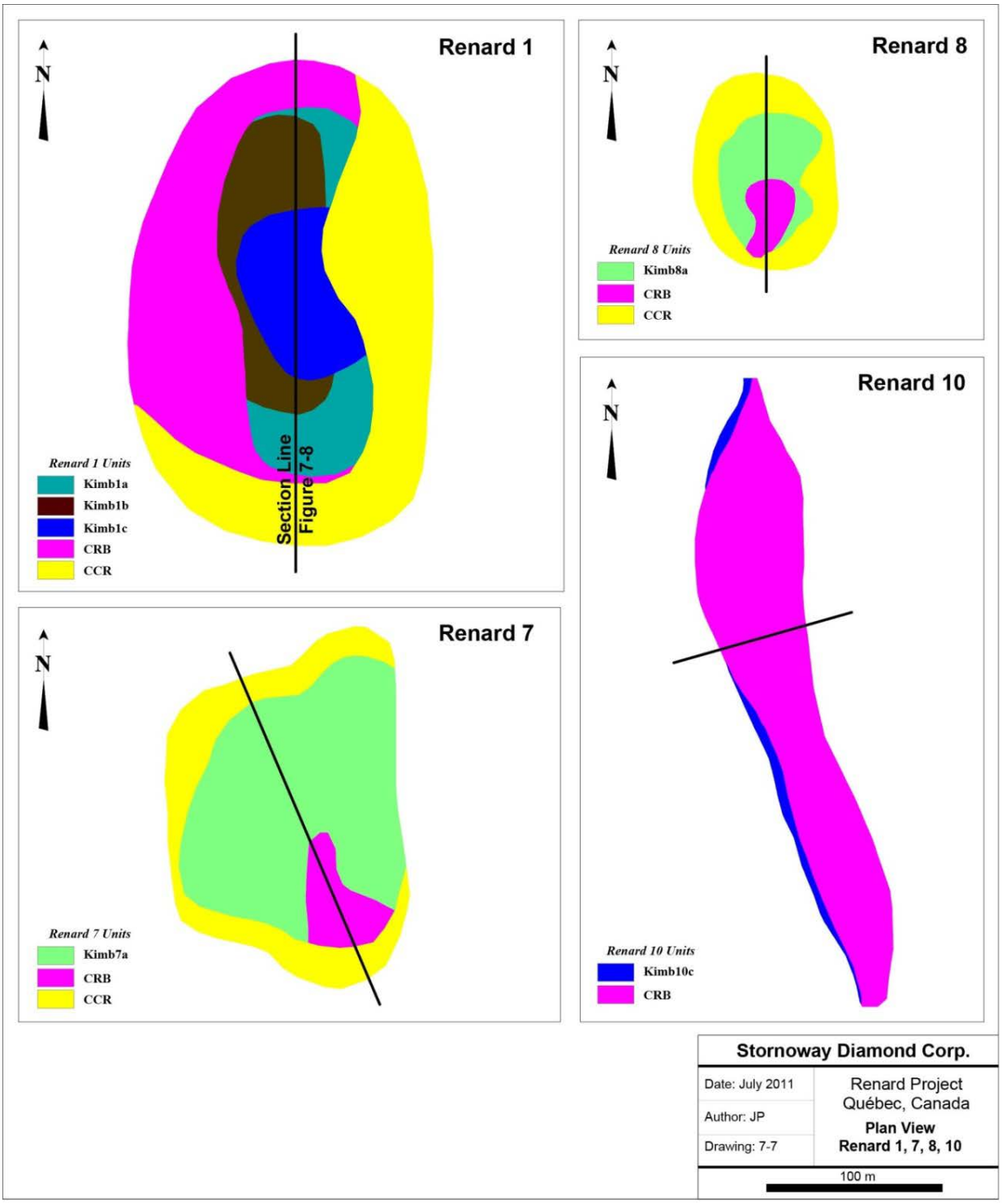

Figure 7.7: Plan View – Renard 1, Renard 7, Renard 8 and Renard 10

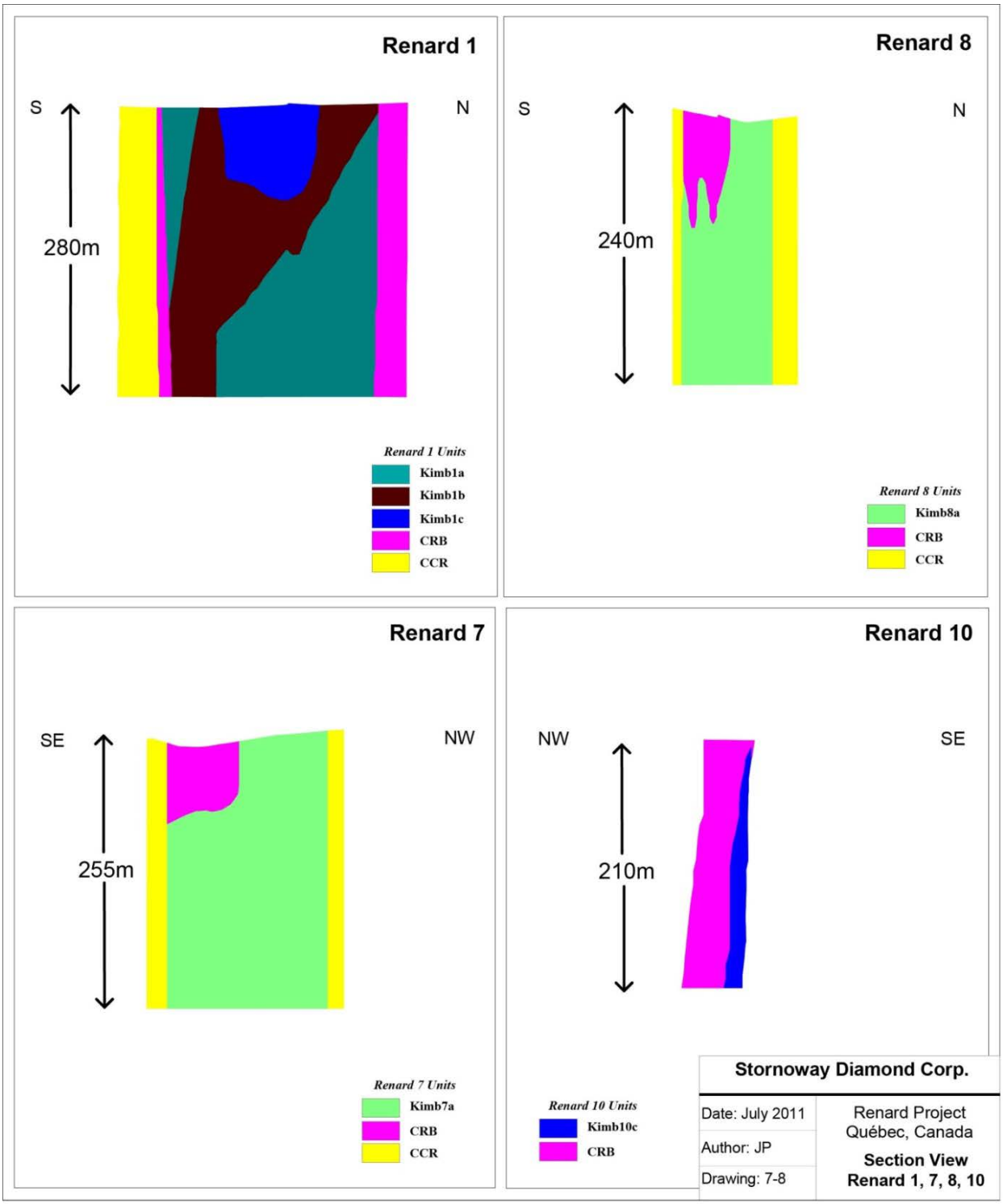

Figure 7.8: Section View – Renard 1, Renard 7, Renard 8 and Renard 10

Renard 7

The Renard 7 kimberlite is a moderate to large sized kimberlite in the Renard Cluster. It is interpreted as a diatreme-zone kimberlite with a slightly irregular shape at surface, and tapering gradually with depth. The geology of this kimberlite has been determined in a preliminary fashion using seven drill holes.

The current 3D model of Renard 7 consists of two main pipe-infilling kimberlite units: a highly diluted grey-green MVK that is referred to as Kimb7a and is classified as TKB; and an extensive amount of HK localized in the upper southern portion of the body referred to as Kimb7c. However it is unclear at this time if this HK represents a main pipe infilling event. In the current 3D model, separate phases of kimberlite were not modelled, therefore their relative abundances are unknown. CCR almost completely surrounds the main kimberlite pipe infills along with a CRB block located in the southeast. This CRB is more internal than marginal to the Kimb7a unit. Additional, smaller but significant zones of CRB are also noted within the Kimb7a unit. A number of HK dykes and irregular intrusions occur throughout the body, along the pipe contacts, within the marginal breccia and in the cracked country rock. These are also referred to as Kimb7c because at this time they cannot be distinguished geologically from HK in the large intrusion.

Renard 8

The Renard 8 kimberlite is one of the smallest kimberlite pipes within the Renard Cluster. It is interpreted as a steep-sided diatreme-zone kimberlite with a regular external shape. The geology and geomoetry of this kimberlite has been determined in a preliminary fashion using ten drill holes.

The current 3D model of Renard 8 consists of one main pipe infill kimberlite unit: a highly diluted grey-green MVK that is classified as TKB, referred to as Kimb8a. In addition, CCR completely surrounds the main kimberlite pipe infills along with a CRB block located in the south. The CRB is more internal than marginal to the kimberlite. In addition, a number of HK dykes and intrusions referred to as Kimb8c occur throughout the body, along the pipe contacts, within the marginal breccia and in the cracked country rock.

Renard 10

The Renard 10 kimberlite is the smallest “pipe” within the Renard Cluster. It cannot technically be classified as a pipe per se, as its morphology is more dyke-like. However, the kimberlite (17% of the body by volume) in this body is surrounded by a significant marginal CRB (83% of the body by volume). This is a feature that has not been noted to be associated with other large dykes on the Foxtrot property. Only five diamond drill holes have been completed to date on Renard 10 and it is on this data that these preliminary interpretations are based.

The kimberlite in Renard 10 is classified as HK and the morphology of this HK is dyke-like. Kimberlite represents 17% of the modelled body by volume and is typically undiluted. Contacts between this HK and the adjacent CRB are sharp. The CRB has an overall appearance and texture similar to that of the other Renard bodies. This, and the presence of the extensive CRB associated with the relatively small amount of kimberlite suggests that Renard 10 may actually represent a blind intrusion.

7.3.8 Lynx and Hibou Kimberlite Dykes

Lynx Kimberlite Dyke

The Lynx dyke consists of CK that can be further classified as HK. Trenching and mapping of Lynx and the logging of 73 drill holes reveals that the dyke comprises semi-continuous thick intersections of kimberlite with many thin (from < 1 cm to 40 cm) discrete sheets adjacent to it. Lynx extends for a minimum of 4.2 km along strike and has a variable dip (10^o to 50^o) to the east. The dyke may reach cumulative thicknesses of up to 3 m, however the average thickness is 1.8 m. The main kimberlite intersection appears to pinch and swell and, where thin, is strongly altered and replaced by clays. Overall, Lynx displays varying levels of alteration, often showing zonation with portions of the main intersection being highly altered and friable on one half, but massive and relatively unaltered in the other. Fracturing is present throughout the dykes and varies in intensity (Figure 7.6).

Macroscopically, the Lynx HK is dark grey to green coloured, strongly magnetic and consists of two generations of olivine (macrocrysts and phenocrysts) set in a carbonate rich groundmass. Olivine macrocrysts are medium- to coarse-grained with common, very coarse grains and are completely serpentinized. Olivine comprises 40% - 50% of the rock, however local variations range from 10% 90%. Olivine megacrysts and olivine dominated mantle nodules larger than 10 cm are present. Mantle indicator minerals are very common throughout this HK, representing up to 3% of the rock by volume and include coarse- to very coarse-grained picroilmenite, chrome diopside and pyrope garnet. Country rock xenoliths are rare, representing less than 2% of the rock. Xenoliths are rounded to sub-rounded and are strongly altered granitoid or gneiss fragments. Contacts with the surrounding country rock are marked in the kimberlite by local minor fracturing and sometimes extensive carbonate veining. Chilled margins with flow aligned olivine macrocrysts are common and typically less than 5 mm in width. Fracturing parallel to the contact, and joints perpendicular to the contact, are present within the country rock and decrease in intensity away from the contact. The Lynx dyke typically cuts the foliation of the country rock.

Microscopically, the Lynx HK consists of olivine macrocrysts and phenocrysts and evenly distributed, crystalline groundmass dominated by carbonate, serpentine and phlogopite with an average modal for carbonate of 15% - 35%. Olivine are completely pseudomorphed to a pale-yellow green serpentine and more rarely carbonatized, and commonly show a preferred orientation with their long axes sub-parallel to the dyke margins. Groundmass minerals consist of carbonate, phlogopite, monticellite, serpentine and ilmenite with minor dolomite. Accessory minerals (5% modal) include perovskite, spinel and apatite. Distinctive magmatic textures are observed in Lynx, where calcite occurs as millimetre scale oikocrysts that enclose small olivine and other matrix minerals (Patterson et al., 2009).

Hibou Kimberlite Dyke

The Hibou dyke is a CK that can be further classified as an HK. Hibou consists of one main kimberlite intersection with thin HK veins and dykes (from 1 cm to 30 cm) adjacent to the main intersection. Trenching and mapping of Hibou and the logging of 35 drill holes reveals that the HK may reach up to 3.5 m thick, but is on average 2 m thick. The dyke has a west-northwest strike extent of at least 850 m with a shallow dip of approximately 10^o to the north-northeast. It is open, down dip and along strike to the northwest. The HK pinches and swells and, where thin, is strongly altered to clays. Overall Hibou displays varying levels of alteration and/or weathering that is zoned. Fracturing is present throughout the dyke and varies in intensity (Figure 7.6).

Macroscopically, Hibou HK is grey-green to green coloured and strongly magnetic. Olivine macrocrysts are medium- to coarse-grained with very common coarse grains set in a carbonate rich groundmass. Olivine macrocrysts comprise approximately 50% - 60% of the rock but can vary from 25% - 70% locally. Olivine size frequently continues beyond the size range of macrocryst classification with individual olivine grains measuring more than 3 cm in length. Olivine dominated mantle nodules are also present and reach up to 10 cm in size. Olivines are typically completely serpentinized. Mantle indicator minerals such as pyrope garnet, chrome diopside and picroilmenite are readily observable, representing up to 3% of the rock by volume. Local calcite segregations are observed and calcite veining is present through the dyke and can reach up to 2 mm in thickness. Country rock xenoliths are relatively rare, representing < 2% of the rock by volume. Xenoliths are strongly altered, are sub-rounded to rounded and consist of gneiss or granitoid fragments that measure up to 7 cm in diameter. Contacts with the surrounding country rock are sharp and marked in the kimberlite by local minor fracturing and carbonate veining. Chilled margins with flow-aligned olivine macrocrysts are common and typically less than 5 mm in width. Fracturing parallel to the contacts, and joints perpendicular to the contacts, are present within the country rock. Hibou is cut by rare black hematite veins less than 5 mm thick that appear to follow the jointing in the dyke. Flow banding of groundmass minerals is evident and tends to be somewhat erratic in orientation. The dyke is variably weathered and in some areas alteration is intense.

Microscopically, Hibou HK consists of olivine macrocrysts and phenocrysts set within an evenly distributed, crystalline groundmass dominated by carbonate, serpentine and phlogopite with an average modal carbonate estimate of 15% - 35%. Olivine macrocrysts are fresh to completely pseudomorphed to a dark green to black serpentine and are rarely carbonatized. They commonly display a preferred orientation with their long axes sub-parallel to the dyke margins. Unlike Lynx, fresh olivine macrocrysts are more common in Hibou. Groundmass minerals consist of carbonate, phlogopite, monticellite, serpentine and ilmenite with minor dolomite. Accessory minerals (5% modal) include perovskite, spinel and apatite. Calcite oikocrysts (as in Lynx) and dolomite rhombs occur in fresh samples of Hibou (Patterson et al., 2009).

8.0 DEPOSIT TYPES

There are two types of diamond deposits: primary and secondary. Primary deposits are those in which the diamonds remain inside the original host rock (usually kimberlite) that conveyed them to the surface. Secondary deposits are formed when the diamonds are eroded from the host rock and concentrated by the action of water into alluvial deposits (in rivers) or marine deposits (in beaches). The Renard kimberlites are primary deposits.

8.1 Overview of Primary Diamond Deposits

Primary diamond deposits such as kimberlites and lamproites have produced over 50% of the world's diamonds. The remainder was derived from recent to ancient placer deposits that originated from the erosion of kimberlite and/or lamproite. Although diamondiferous kimberlite and lamproite comprise most of the economic diamond deposits, other diamond-bearing rocks have also been discovered and are the subject of numerous academic papers. Such diamond-bearing rocks include ultramafic lamprophyres (aillikites) in Canada and volcaniclastic komatiites in French Guiana (Capdevila et al., 1999). It has been established by the scientific community that diamonds are not genetically related to kimberlite or lamproite but that kimberlite and lamproite intrusives serve as a transport mechanism for bringing diamonds to surface (Kirkley et al., 1991) from the mantle.

Clifford (1966) and Janse (1991) stated that a majority of economic diamondiferous kimberlites occur in stable Archaean age cratonic material that has not undergone any thermal or deformational event since 2.5 Ga. Such Archaean age cratons include the Kaapvaal, Congo and West African cratons (Africa), Superior and Slave Provinces (Canada), East European Craton (Russia, Finland), and the West, North and South Australian cratons. The only exceptions to date are the Argyle and Ellendale mines of Australia, which occurred in Proterozoic-age, remobilized, cratonic material.

To date, over 6,000 known kimberlite and lamproite occurrences have been discovered, of which over 1,000 are diamondiferous. Some of the well-known diamondiferous kimberlites/lamproites currently being mined include Argyle (lamproite) in Australia; Orapa and Jwaneng (kimberlites) in Botswana; Jubilee, Udachnaya and Mir (kimberlites) in Russia; Venetia (kimberlite) in South Africa, and the Ekati and Diavik clusters (kimberlites) in Canada.

Economic diamond kimberlite and/or lamproite pipes generally range from less than 0.4 ha to 146 ha in surface area, with the maximum size being more than 200 ha (for example, Catoca, Angola). Economic diamond grades can range from 3.5 cpht to 600 cpht.

8.2 Kimberlite-Hosted Deposits

Much of the following discussion of kimberlite types and deposits is taken directly from a publication on ore deposit models by Mitchell (1991).

Kimberlites remain the principal source of primary diamonds despite the discovery of high-grade deposits in lamproites. Recent mineralogical and Rb–Sr isotopic studies have shown that two varieties of kimberlite exist:

- Group 1, or olivine-rich monticellite serpentine calcite kimberlites; and
- Group 2, or micaceous kimberlites (predominantly occur in southern Africa).

"Group 1" kimberlites are complex, hybrid rocks consisting of minerals that may be derived from:

- Fragmentation of upper mantle xenoliths (including diamond);
- Megacryst or discrete nodule suite; or
- Primary phenocrysts and groundmass minerals.

The contribution to the overall mineralogy from each source varies widely and significantly influences the petrographic character of the rocks. Consequently, Group 1 kimberlites comprise a petrological clan of rocks that exhibit wide differences in appearance and mineralogy as a consequence of the above variation, coupled with differentiation and diverse styles of emplacement of the magma. Currently, three textural-genetic groups of kimberlite are recognized, each being associated with a particular style of magmatic activity in such a system. These are:

- Crater facies;
- Diatreme facies; and
- Root Zone facies.

Rocks belonging to each facies differ in their petrology and primary mineralogy, but may contain similar xenocrystal and megacrystal assemblages.

With a few exceptions, such as the Finsch Kimberlite Mine in the Republic of South Africa and the Dokolwayo Kimberlite Mine in Swaziland, most of the well-known diamondiferous kimberlites in South Africa and elsewhere are Group 1 kimberlites. The Renard kimberlites are considered to be Group 1 kimberlites.

The Renard kimberlites are interpreted to be steep-sided, pipe-like structures with irregular to elongate shapes in plan view. Surface expressions of the kimberlite vary between 0.1 ha and 2.0 ha although there are larger haloes of broken country rock. The Renard Cluster is mainly composed of diatreme-like kimberlitic breccia lithologies and hypabyssal kimberlitic material. No crater material is noted in the Renards.

The Lynx, Hibou, North Anomaly, Southeast Anomaly and G04-296 hypabyssal dykes are interpreted to be intrusions of kimberlitic material that did not vent to the earth's surface at the time of emplacement.

9.0 EXPLORATION

All exploration has been carried out by Stornoway Diamond Corporation (Stornoway).

9.1 Geological Mapping

Structural geological mapping was undertaken in the area of the Lynx dyke in 2004, and over the Renard bodies in 2006 and 2010. The objective of the mapping programs of 2004 and 2006 was to identify structural controls that could help locate more kimberlitic intrusions or dykes. The program results proved to be inconclusive. It was noted, however, that the rock fracturing intensity increased near the kimberlitic bodies. The objective of the geological mapping in October 2010 was to identify large-scale feature of interest (ie. faults, shear zones) interpreted from geophysical data and aerial photographs. Overburden in targeted areas was cleared to expose the bedrock for mapping. The program was successful in highlighting two large faults and two smaller ones within the proposed mine site.

Sampling trenches were excavated on Renard 4, Renard 65, Lynx, Hibou, and the North Anomaly. Each trench was mapped in detail prior to sampling. Mapping was undertaken with grid control and reference points were surveyed by a registered surveyor. Mapping notes were made on the contacts and included the mineralogy, relationship of the dyke to the host rock, and structural measurements in the host rocks. Plan and section maps were completed for the majority of the trenches.

During the underground bulk sampling program at Renard 2 and Renard 3, geological mapping was completed on all workings. The face was mapped after each round during the development of the ramp and drifts and, after the underground excavation was completed, the ramp and drift walls were mapped in the kimberlites

9.2 Heavy Mineral Sampling

Since the inception of the Foxtrot Project, approximately 12,000 heavy mineral samples have been collected over a 400,000 km2 area of which some 8,140 lie within the current land holdings. Heavy mineral samples collected to date are summarized by year in Table 9.1. This work has been reported in previous technical reports (Clements and O'Connor, 2001, 2002; Lucas et al., 2003; Lépine and O'Connor, 2004; O'Connor and Lépine, 2005, 2006).

Table 9.1: Heavy Mineral Sampling

Year	Foxtrot Property Number of Samples	Comments
1996–1999	13	Regional samples within the area that became the Foxtrot Property
2000	48	One sample collected during this program yielded a highly anomalous indicator mineral count
2001	252	Completed to further define mineral trains and to prioritize geophysical anomalies for drilling
2002	785	Collected to define existing mineral trains and to increase the sample density within the property
2003	914	Collected along detailed grids with dense sample spacing and as wider spaced, reconnaissance samples
2004	2,000	Collected along detailed grids with dense sample spacing and as wider spaced, reconnaissance samples
2005	1,412	Collected along detailed grids with dense sample spacing and as wider spaced, reconnaissance samples
2006	1,203	Collected along detailed grids with dense sample spacing and as follow up samples
2007	959	Collected along detailed grids with dense sample spacing and as follow up samples
2008	554	Collected along detailed grids with dense sample spacing and as follow up samples
Totals	8,140	

9.3 Geophysical Surveys

Since 2000, several ground and airborne geophysical surveys were completed on the Foxtrot Property. This work has been reported in previous technical reports (Clements and O'Connor, 2001, 2002; Lucas et al., 2003; Lépine and O'Connor, 2004; O'Connor and Lépine, 2005, 2006). Geophysical surveys that have been completed to the effective date of this report are summarized in Table 9.2.

Table 9.2: Geophysical Surveys

Year	Airborne Geophysical Surveys (total line km)	Number of Ground Magnetic Surveys (total line km)	Number of Electromagnetic Surveys (total line km)
2000	1,419	0 (0)	0 (0)
2001		5 (38.4)	0 (0)
2002	900	21 (140.5)	14 (20.9)
2003	8,900	52 (452.3)	10 (19.4)
2004	4,778	58 (505.4)	33 (67.7)
2005	19,491	25 (215.2)	41 (84.2)
2006		29 (308.0)	51 (81.6)
2007		52 (441.5)	43 (41.9)
2008	1,969	32 (385.0)	12 (11.8)
Totals	37,457	274 (2,486.3)	204 (327.5)

9.4 Drilling

Drilling is discussed in Section 10.0.

9.5 Sampling for Diamonds

Three basic levels of progressively larger diamond sampling procedures are summarized below (caustic fusion sampling, mini-bulk sampling and bulk sampling), followed by descriptions of the comparable core, reverse circulation, trenching and underground sample programs.

9.5.1 Caustic Fusion Sampling

The caustic fusion process is used to evaluate, characterize and correlate the diamond potential of individual kimberlite lithologies, and to provide data to facilitate the grade estimation process. The objective of this type of test is to extract all diamonds greater than 0.1 mm in size, through chemical dissolution of the host rock sample. Individual samples may vary in size from a few kilograms to hundreds of kilograms, depending on the available material and the specific purpose of the testing. Kimberlite may be collected from drill core, float boulders, subcrop, outcrop, underground exposures and subsamples of material in a process facility or a combination thereof. Individual sample results from comparable kimberlite units may be merged together to provide larger, statistically more representative, samples.

Kimberlite is collected, described and recorded by the site geologists following protocols in place at the time. Samples are individually numbered, weighted, sealed in a tamper-resistant container appropriate for the volume of material, and transported to the test facility by a combination of charter aircraft and commercial couriers.

9.5.2 Mini-Bulk Sampling

Although there is no formal industry-accepted definition of a 'mini-bulk' sample, many companies would agree that the term is generally used to refer to the processing of kimberlite material up to several tens of tonnes. This material may be derived from drill core, RC chips, boulders, subcrop, outcrop, trenches or underground workings. Mini-bulk samples are usually processed through Dense Media Separation (DMS) equipment that, depending on specifications and diamond recovery objectives of a particular program, may be configured to recover diamonds of greater than 0.5 mm, 0.85 mm or 1.18 mm on square-mesh screens. In some cases caustic dissolution or other extraction techniques may be utilized to recover the diamonds. All of Stornoway's mini-bulk samples reported herein were processed through DMS equipment, and the reported diamond content is based upon stones retained on either 1.18 mm square-mesh screens or +1 DTC screens.

Stornoway's mini-bulk sampling programs reported herein have used drill core, RC chips, boulders, and surface trenches to source kimberlite material. Drill core was collected, described and recorded by the site geologists following protocols in place at the time. Core boxes were sealed, weighted and secured to pallets for shipping, then forwarded to the North Vancouver laboratory facilities. Upon receipt samples were verified, then composited and processed through an onsite DMS plant. The mini-bulk sample data has not been used for mineral resource estimating.

During the various RC drill programs (see Section 10), screen openings of either 0.98 mm or 1.18 mm were used to dewater the RC kimberlite chip recovery flow and to remove the undersize fraction. Recovered kimberlite was separated at regular downhole intervals into individual samples and collected in either large 1.5 t bulk sample bags or individually lined 205 l steel drums. Bulk bags and/or drums were clearly identified with a unique numbering system, fastened with tamper-proof seals and transported to secured Stornoway storage facilities. Charter aircraft and bonded freight services were organized as required to transport sample materials to the process facility. In 2004, RC mini-bulk samples were dispatched to the external Thunder Bay Mineral Processing Laboratory (TBMPL) in Ontario for DMS processing, whereas RC mini-bulk samples from 2006 were sent to Stornoway's North Vancouver laboratory and those from 2007 were processed by the Lagopède-based DMS.

9.5.3 Bulk Sampling

Although there is no formal industry-accepted definition of a 'bulk' sample, many companies would agree that the term is generally used to refer to the processing of kimberlite material exceeding several tens of tonnes. This material may be derived from drill core, RC chips, boulders, subcrop, outcrop, trenches or underground workings. Bulk samples are usually processed through DMS equipment that, depending on specifications and diamond recovery objectives of a particular program, may be configured to recover diamonds of greater than 0.85 mm or 1.18 mm on square-mesh screens. In some cases larger screen sizes or other extraction

techniques may be utilized for diamond recovery. All of Stornoway's bulk samples reported herein comprise either surface trench or underground sample material, and were processed through DMS equipment. The reported diamond content is based upon stones retained on either 1.18 mm square mesh or +1 DTC screens.

To maintain the integrity of the bulk samples, a sampling protocol was established before the exploration work took place in order to ensure that the final diamond recovery data can be linked to the correct kimberlite, sample location within the body and rock type. Each surface or underground round was assigned a sample number and a colour code by the mine geologist. For surface samples, individual loads were directed to specific dump piles in a secure, access-controlled, storage area located near the DMS facility. The storage area was monitored 24 hours per day by an independent security force (Groupe Canaprobe of Montréal, Québec) and by CCTV surveillance.

For underground samples, a sample number placard was posted in the remuck bay and on the dump truck so the underground operators and process plant staff were constantly aware of which sample was being handled. On surface, a member of the DMS staff at the controlled access storage site directed the loads to the correct stockpile locations and kept a log of loads that were delivered from underground. Once all loads were hauled to surface, the mine geologist checked that the remuck bay was cleaned out in preparation for the next sample. Sample number placards were taken to the surface with the final dump truck load and placed on the appropriate stockpile. A location plan of the surface or underground samples was updated each time a new pile was initiated, in order to easily identify sample location.

9.5.4 Core Sampling

Certain drill core collected during historical drill programs (discussed in Section 10) was composited and treated for macrodiamond recovery. Results are summarized in Table 9.3.

Table 9.3: Summary of Macrodiamond Sampling Results

Kimberlite Body	Sample Type	Year Collected	Number of Samples	Total Weight (t)	Total Carats (+1 DTC/+1.18 Tyler)
Renard 1	Drill Core	2002	1	0.3	0.00
Renard 1	Drill Core	2003	11	10.0	0.58
Renard 2	Drill Core	2002	7	4.9	2.89
Renard 2	Drill Core	2003	8	8.6	4.53
Renard 2	Drill Core	2004	9	11.5	13.30
Renard 2	Drill Core	2005	16	6.7	4.75
Renard 2	Drill Core	2006	7	2.8	2.47

Kimberlite Body	Sample Type	Year Collected	Number of Samples	Total Weight (t)	Total Carats (+1 DTC/+1.18 Tyler)
Renard 2	RC Chips	2004	12	171.2	145.38
Renard 2	RC Chips	2007	15	86.8	70.95
Renard 2	Underground	2006/2007	15	2448.8	1601.94
Renard 3	Drill Core	2002	5	4.9	6.47
Renard 3	Drill Core	2004	13	13.7	13.70
Renard 3	RC Chips	2004	10	157.0	184.53
Renard 3	RC Chips	2007	13	59.4	34.86
Renard 3	Underground	2006/2007	13	2113.7	2799.85
Renard 4	Drill Core	2002	6	4.8	2.94
Renard 4	Drill Core	2003	15	12.4	5.32
Renard 4	Drill Core	2004	36	32.3	13.62
Renard 4	Drill Core	2005	1	0.5	0.48
Renard 4	RC Chips	2004	17	141.8	52.57
Renard 4	RC Chips	2006	14	41.4	34.67
Renard 4	Surface Sample	2004	2	1.8	3.09
Renard 4	Surface Sample	2005	6	9.8	17.27
Renard 4	Surface Sample	2006/2007	7	2104.2	2721.88
Renard 65	Drill Core	2002	2	0.8	1.19
Renard 65	Drill Core	2003	23	19.8	8.29
Renard 65	Drill Core	2004	22	17.8	4.05
Renard 65	RC Chips	2004	18	147.2	32.49
Renard 65	Surface Sample	2007	2	266.0	51.77
Renard 65	Surface Sample	2012	1	5080.8	962.8

Kimberlite Body	Sample Type	Year Collected	Number of Samples	Total Weight (t)	Total Carats (+1 DTC/+1.18 Tyler)
Renard 7	Drill Core	2005	4	4.1	0.10
Renard 8	Drill Core	2005	4	6.1	0.47
Renard 9	Drill Core	2004	6	6.0	5.56
Renard 9	Drill Core	2005	4	6.2	6.24
Renard 9	RC Chips	2006	19	70.3	34.09
Renard 9	RC Chips	2007	5	27.3	11.97
Hibou	Surface Sample	2005	5	19.8	4.58
Hibou	Surface Sample	2006	2	31.4	38.52
Hibou	Surface Sample	2008	1	543.9	781.41
Lynx	Surface Sample	2003	3	3.9	3.67
Lynx	Surface Sample	2004	2	10.3	14.47
Lynx	Surface Sample	2005	6	34.7	40.40
Lynx	Surface Sample	2007	3	494.3	528.93
North Anomaly	Surface Sample	2006/2008	2	46.4	43.00

9.5.5 Reverse Circulation (RC) Sampling

RC chip sampling programs on the Renard Project were undertaken with objectives that varied over time from simply collecting a large amount of kimberlite to create representative samples, to characterizing the grade over various depth intervals, to regular sampling intervals. Early work on the Project also considered that some minimum quantity of kimberlite should comprise each sample to accommodate processing. The data set of sampling results represents:

- Sampling intervals from 10 m to more than 50 m;

- Combination of chips from adjacent holes over similar intervals in order to double the volume of kimberlite from that interval;
- In some areas, regular sampling at uniform downhole spacings; and
- Sampling governed by internal geological contacts, completed during the 2007 program.
- Results are summarized in Table 9.3.

9.5.6 Trench Sampling

Since 2005, several thousand tonnes of kimberlitic material have been excavated from trenches on the Renard 4 and Renard 65 bodies and Lynx, Hibou and North Anomaly dykes. Trench locations are shown in Figure 7.3. Trench details are summarized in Table 9.4 and macrodiamond results in Table 9.3.

Table 9.4: Trenching

Area	Trench Name	Date of sampling	Dimension (m)	Sample Collected (t)
Renard 4	Renard 4	2004	5 x 5	1.8
	Renard 4	2005	25 x 25	9.8
	Renard 4	2006	35 x 27	2,345.0
Renard 65	Renard 65	2007	20 x 16	266.0
Renard 65	Renard 65	2012	26 x 32	5080.8
Lynx Dyke	T-184-01	2005	8 x 35	6.2
	T-221-01	2005	6 x 25	12.9
	T-221-01	2007	8 x 40	364.1
	T-230-01	2005	12 x 10	12.7
	T-230-02	2005	10 x 25	3.0
	T-230-02	2007	10-15x 106	130.2
Hibou	T-271A-01	2005	10 x 6	19.8
	T-271A-10	2006	31 x 10	31.4
	T-271A-10	2008	40 x 40	543.9
North Anomaly	T-222-01	2006	1.5-3.5 x 13	46.4

9.5.7 Underground Bulk Sampling

The underground exploration program at the Renard Project was designed to extract a minimum of 2,000 t of kimberlite from each of Renard 2 and Renard 3 at a depth of approximately 55 m below the present surface. Supplies and equipment were mobilized from Chibougamau via Buffalo aircraft to the ice airstrip at Camp Lagopède in March and April 2006.

Preparatory work included establishing powder magazines, constructing office and dry facilities for the miners, erecting a maintenance shop and generator shed, constructing a mine water sump, installing a waterline, and assembling the equipment. Equipment consisted of a two-boom jumbo, one haul truck, two scoops, a scissorlift truck, and a crew transport vehicle. Genivar Inc. supervised all aspects of the operations, while mining contractor Monterie Expert Inc. was engaged to provide equipment and personnel to complete the work.

The portal was located on the southwest side of a hill immediately north of Renard 2. An 8 m, near-vertical outcrop face was present at the location, reducing the need for an extensive portal trench. Following mobilization and assembly of the equipment, cleaning of the portal area commenced at the end of June to expose the bedrock. A small airtrack drill was used to bore the portal blast holes. The first blast occurred on July 16, 2006. Four blasts were required to obtain a suitable face for the portal. To provide additional ground support for the brow of the portal, chain link fencing and strapping were bolted to the rock.

The first round in the ramp was taken on August 1, 2006, and work was completed on February 22, 2007. The ramp dimensions are 3.8 m by 4.2 m, with a maximum grade of 15%. The back was screened and bolted, generally using a 1.2 m by 1.2 m pattern with Swellex bolts or resin rebar and galvanized, welded wire mesh. A total of 749.1 m of underground workings was completed, comprising a portal, ramp, ore drifts, safety bays, sump pump stations, an electrical substation and a powder magazine. Excavations were completed to a depth of approximately 55 m below surface. The main ramp length totalled 435.6 m with lateral excavation in waste of 111 m (from base of ramp to Renard bodies) and lateral excavation in ore of 202 m. Horizontal drifts to access the Renard 2 and Renard 3 kimberlites were each about 100 m in length. Ground conditions within the cross-cut leading into Renard 2 were challenging, but ground conditions within the kimberlites were excellent. Production rates averaged 3.59 m per day; but varied depending upon whether single or double faces were being advanced.

Underground, the faces were surveyed using a Leica total station instrument equipped with a reflectorless measuring device. Details of each side of the gallery and the front face were surveyed with a variable number of survey points depending on the regularity of the surface. Each survey was divided into five different surfaces: floor, back, left and right walls, and front face. These surveys were used to measure the rate of advance and to calculate the volume of material removed.

Sampling consisted of collecting the kimberlite material blasted from each round. After each blast, material was mucked from the face with scoops and collected in a remuck bay. From the remuck bay, the ore was loaded into a haul truck and transported to the secure ore stockpile area located adjacent to the DMS facility on the surface, some 200 m southeast of the portal (Figure 9.1). A typical round was three to five metres in length, or approximately 150 t. A total of 60 individual samples were collected: 29 from Renard 2 (5,010 calculated tonnes) and 31 from Renard 3 (5,161 calculated tonnes). A representative sub-sample from each of Renard 2 (15

round; 2,449 t) and Renard 3 (13 rounds; 2,114 t) were processed through the Lagopède DMS for macrodiamond recovery (Table 9.3). The remaining unprocessed rounds are stored if future work is required.

The underground workings were allowed to flood late in 2007, and the portal is currently closed.

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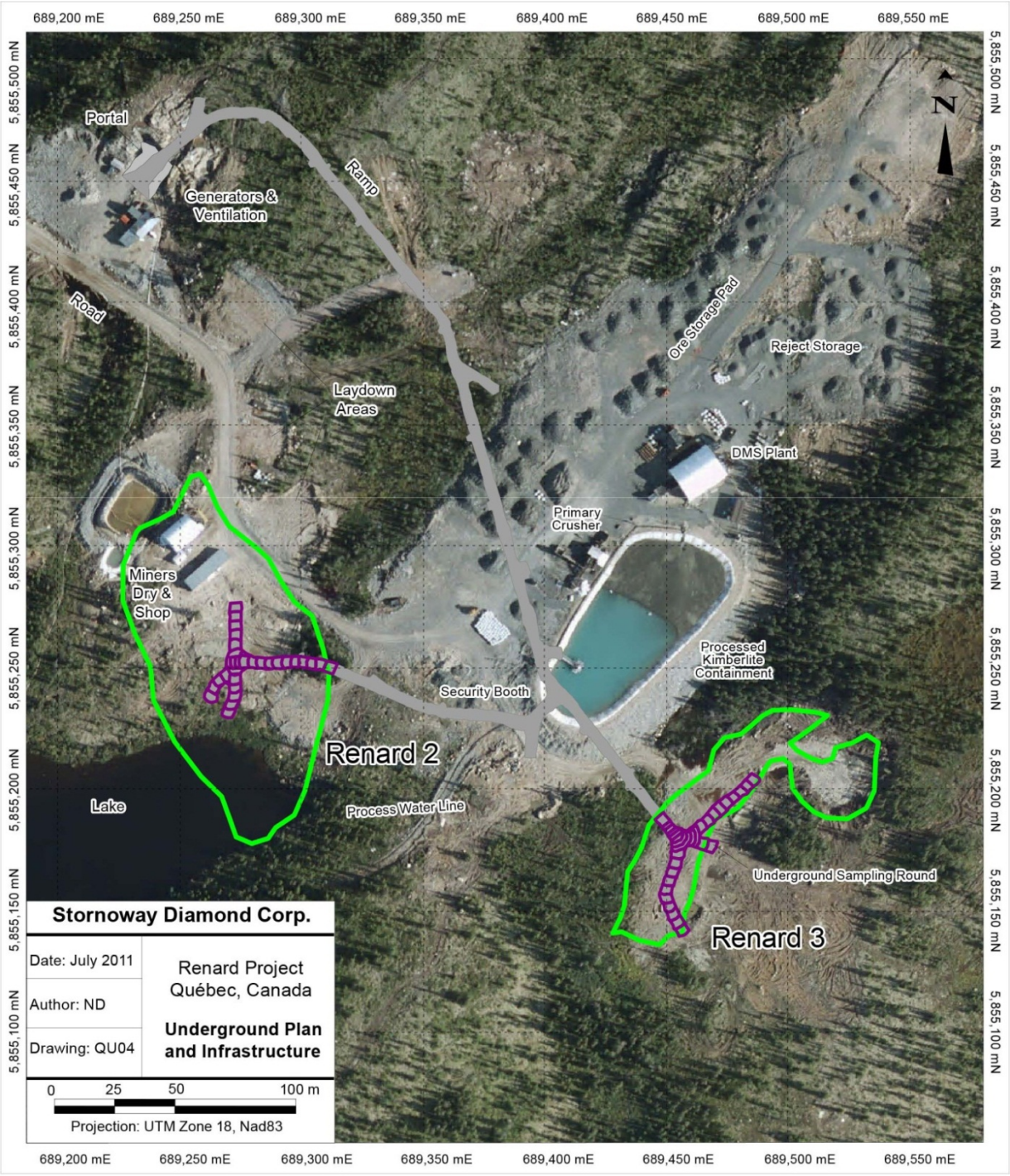

Figure 9.1: Underground Plan and Infrastructure

9.6 Bulk Density Determinations

The dry bulk density database for the Foxtrot Property comprises 1,641 bulk density records, consisting of 1,186 measurements from drill core and 455 from bulk sampling. When multiple measurements from the same sample, and multiple subsamples from the same rock are averaged, and the laboratory Quality Control (QC) checks removed, there are 1284 spatially discrete density samples.

Morris Magnetics Inc. performed the first density tests in 2004 on Renard 2, Renard 3 and Renard 4 using the water displacement method. Between 2004 and 2009, additional bulk density samples were measured by Morris Magnetic Inc. and SGS Canada Inc., primarily through the immersion/water displacement method, as follows:

1. Dry the sample in the oven at 49ºC overnight.
2. Weigh each sample in air.
3. Weigh each sample suspended in water.
4. Calculate the displaced volume of the dry rock in water.
5. The difference between the dry rock weight and the water displaced gives the calculated bulk density.

Quality control checks comprising duplicate measurements with water and wax immersion techniques, repeat density determinations, blind checks between laboratories, and multiple subsamples of the same rock demonstrate small density variances on a sample-by-sample, measurement-by-measurement and process-by-process basis, but there does not appear to be a consistent shift from laboratory to laboratory. Variability is generally in the order of 0.00 g/cm3 to 0.05 g/cm3 (or from 0% - 2% assuming an average bulk density of 2.60 g/cm3), and should not be a concern for data interpretation. Density variations did not show a correlation with country rock dilution, nor was there a clearly demonstrable change of density with increasing depth in the kimberlite pipes.

Country rock (CR) consists of predominantly gneissic, lesser granitic and minor pegmatitic phases. The relationship between granitic and gneissic units is commonly cyclic, such that large intersections of drill core are best described as mixed granite-gneiss. Given the proportion of mixed units, and the overlapping density ranges for granite and gneiss, a calculated average bulk density of 2.72 g/cm3 has been applied to the country rock of all bodies.

Bulk density ranges for both the country rock breccia (CRB) and country rock breccia with minor kimberlite as thin matrix/vein fillings (CRB+K) are a little lower than the country rock (CR) units, likely as a function of fracturing and alteration associated with brecciation, with overlapping ranges. An average bulk density of 2.61 g/cm3 for both CRB and CRB+K units has been applied.

Table 14.7 shows the average bulk density values for the various geological units within the Renard kimberlites.

9.7 Moisture Content

Determining the moisture content of each sample prior to processing through caustic fusion, DMS or bulk density is necessary to allow an accurate dry weight of the kimberlite to be calculated. The drying of bulk density samples is part of the standard routine used to measure density, as documented above.

Samples of drill core processed through the Stornoway North Vancouver laboratory as part of the historical mini-bulk programs were stored inside for extended periods of time prior to DMS processing (23 days to 598 days; average 141 days). As a result, the core had already dried out and no moisture content tests were performed.

Moisture content tests were performed on the RC chip samples to allow an estimate of percentage chip recovery during drilling. These moisture contents were not used in the determination of RC mini-bulk sample masses, or during RC diamond grade calculations, as tonnages were based upon theoretical sample volumes of each contributing geological unit and the appropriate density.

During bulk sampling programs, moisture content tests were completed on kimberlite from both the surface and underground samples. As each individual sample was being run through the DMS plant, between five and 15 aliquots of raw feed (one to two kilograms each) were taken from the feed conveyor. Each aliquot was weighed, then dried in the oven at 105ºC for two hours and reweighed. Results for each aliquot were recorded and averaged to represent the assumed moisture content for the complete subsample. Average moisture values of 5.8% (range 3.0% - 8.5%) and 7.8% (range 3.9% - 15.4%) were obtained for the underground samples at Renard 2 and Renard 3, respectively. The large tonnage surface samples varied from an average of 7.1% (range 6.5% - 7.7%) at Lynx to an average of 10.8% (range 5.4% - 21.1%) at Renard 4. Average moisture content at Renard 65 was 8.3% (range of 4.8% - 11.9%) and the Hibou sample was 9.8% (treated as one large sample).

9.8 Petrography, Mineralogy and Other Research Studies

Since 2001, several petrographical and mineralogical studies have been undertaken on the Renard kimberlitic bodies and the Lynx, Hibou and North Anomaly dykes. These studies include:

- Detailed trench mapping;
- Detailed core logging in the Renard Core Area, and the Lynx and Hibou dykes;
- Petrographical thin section descriptions;
- Indicator mineral and geochemical studies on the various kimberlite phases at Renard 2, Renard 3, Renard 4, Renard 65 and Renard 9, and Lynx and Hibou dykes;
- Age dating HK dykes from the Renard 1 kimberlite, on the main phases at Renard 2 and Renard 3 and the Lynx dyke; and
- Characterization of geophysical signatures on a unit by unit basis.

The macroscopic and petrographic studies were used to distinguish the different lithological units within the kimberlite pipes, and were essential for the construction of 3D geological models.

9.9 Further Exploration

Several unexplained kimberlite indicator mineral trains and geophysical anomalies remain within the Foxtrot Property. To explore for additional kimberlite occurrences, the Company may collect additional heavy mineral samples, undertake ground and airborne geophysical surveys, prospect areas of anomalous indicator mineral results and drill exploration targets generated from the heavy mineral sample, geophysical and prospecting work. No work is planned or budgeted at the present time.

10.0 DRILLING

10.1 Background and Summary

All drilling has been carried out under the control of Stornoway and predecessor company Ashton. A total of 855 drill holes (120,278 m) has been drilled on the Property since 2001, comprising 36 RC holes (6,151 m) and 630 exploration core holes (109,237 m), 35 geomechanical holes (3,213 m), 108 geotechnical holes (1,034 m) and 46 hydrogeological holes (643 m). Total surface drilling on a year-to-year basis for exploration work is summarized in Table 10.1. Total drilling for geomechanical, geotechnical and hydrological purposes is summarized in Table 10.2. During 2007, and as part of the underground bulk sample work, 22 holes were drilled from underground on Renard 2 (1,508 m) and 21 holes on Renard 3 (874 m). These 22 holes are included in the totals above and in Table 10-1.

Vertical and angled holes were drilled through the kimberlite bodies, from which three dimensional geological models were constructed for resource estimation. Drilling intersections are therefore not related to true thickness of mineralization.

Between 2001 and 2002, drilling was completed for early-stage, exploration-focused programs for all the bodies except for Renard 9 and Renard 10, which were discovered in 2003 and 2005, respectively. From 2003, drilling was used primarily to support advanced-stage project evaluation and deposit delineation by providing bulk and mini-bulk samples. Target exploration drilling was undertaken between 2001 and 2010. Drilling in 2011 and 2012 was focussed on collecting data to support the proposed mine plan and infrastructure design.

Table 10.1: Summary of Exploration Drill Programs

Program by Year	Number of Core Holes	Number of Extended Holes	Drilled Metres	Number of RC Holes	Drilled Metres
2001	6	0	554	0	0
2002	33	0	4,688	0	0
2003	70	0	12,642	0	0
2004	125	0	17,489	23	4,157
2005	138	3	26,099	0	0
2006	96	0	11,426	5	805
2007	105	2	12,455	8	1,189
2008	16	0	2,160	0	0
2009	29	3	16,506	0	0
2010	12	2	5,218	0	0
Total	630	10	109,237	36	6,151

Of the exploration drill holes listed above, 456 holes totalling 88,585 m were used in the 2010 resource estimate.

Table 10.2 Summary of Geomechanical, Geotechnical and Hydrogeological Drill Programs

Year	Purpose	Number of Drill Holes	Drilled Metres
2006	Hydrogeological	8	122
2010	Geomechanical	26	2,436
2010	Hydrogeological	12	109
2010	Geotechnical	39	200
2011	Geotechnical	20	493
2012	Geomechanical	9	777
2012	Geotechnical	49	341
2012	Hydrogeological	26	412
Total		189	4,890

10.2 Reverse Circulation Drilling

Since 2004, Foundex Exploration of Surrey, BC, has been the drilling contractor for all reverse circulation (RC) work on the Renard Project, employing an HT-2000 drilling system rig. RC holes ranged from 88 m to 211 m depth and measured 27.7 cm to 28.8 cm diameter. All RC boreholes were drilled vertically. In 2004, Stornoway completed a two part 23-hole, RC drilling program on the Foxtrot Property, for 4,157 m of drilling. Another five and eight RC holes were drilled in 2006 and 2007, for a total of 805 m and 1,189 m, respectively. Due to the complex geology of the Renard bodies and the reduced geological information gathered by using RC drilling methods, all RC borehole locations were twinned by core drill holes.

The winter 2004 RC drilling program was undertaken using a mix of water and air to bring the kimberlitic chips to the surface, whereas the summer 2004 program used a “reverse flood” technique. Reverse flood drilling uses much larger volumes of water, which is thought to reduce diamond breakage by flushing the cuttings away from the drill head in a timely manner, and facilitating a more rapid, less aggressive transport to surface. In the 2006 and 2007 drill programs, the reverse flood technique was used for all RC drilling, with the exception of one hole on Renard 4, where the air and water method was required due to technical difficulties.

A downhole calliper survey was performed when each RC hole was completed. The goal of the RC calliper survey was to measure the diameter of the hole for the calculation of volume (in cubic metres) of material drilled along the length of the RC hole for diamond grade estimation.

The calliper system comprises a mechanical three-arm calliper with a winch and cable system. Each arm of the calliper can extend to a maximum distance of 0.2 m in length. The survey

methodology consists of lowering the calliper to the bottom of the RC hole, extending the arms until they contact the RC hole wall, and then raising the instrument at a constant rate so that the calliper arms can measure the RC hole profile in real time. Each RC hole was surveyed twice so that the repeatability of the data is consistent and reliable. The information is recorded on a laptop computer and subsequently processed and interpreted. The data were presented as a graphic 3-D downhole log and a downhole Excel spreadsheet.

RC drilling recovers kimberlite as a continuous stream of rock chips ranging in size from a few millimetres to several centimetres depending on the characteristics of the rock being drilled and the type of bit being used. Since a certain percentage of rock material is discarded as under-sized material, or lost in fractures in an RC borehole, the weight of the sample is calculated rather than measured directly. The weight equals the volume of the hole multiplied by the bulk density of the kimberlitic material. Volumes were determined mathematically as a function of hole diameter and sample length. Tonnages were determined on a sample-by-sample basis using the proportion of each geological unit and the measured density as determined from the same interval in the twinned core hole (refer to Section 9).

Rock chips are collected at regular depth intervals, logged geologically, and archived for later use. Rock chip size, morphology, and granulometry are monitored constantly and recorded in drill logs. The Foundex drillers also recorded technical information during the drilling, including rotary pressure, bit pressure, revolutions per minute, air pressure and production rates. This RC data is used to compare with diamond recovery data and to determine the best bits for particular rock types.

RC chip recovery is monitored on a sample-by-sample basis by estimating the theoretical tonnage (calliper volume times sample length times density) minus the approximate dry weight of the samples processed. RC recoveries typically vary from 51% - 78%. Recoveries are lower when the rock is strongly diluted by country rock xenoliths. Stornoway observed that certain RC bit types and drilling techniques did not give good recoveries.

10.3 Core Drilling

Chibougamau Diamond Drilling of Chibougamau, Québec, has been the drill contractor for all core drilling (both surface and underground) on the Renard Project since 2001.

The diesel-powered surface drill rigs used for the majority of the work are unitized HC150's (essentially equivalent to a Boyles 44), housed in metal shacks, skid-mounted and moved by bulldozer. Drill rods and ancillary equipment are transported by skid-mounted sloop (or sled). The rigs, as currently configured, are capable of drilling up to PQ diameter core (8.51 cm). Prior to the start of the 2009 summer drill program, the aluminum tower on one of the rigs was replaced with a stronger steel tower to allow deeper drilling. When required to test outlying exploration targets, the drill rig is configured for helicopter transport by removing it from the metal shack. Surface holes ranged from 12 m to 843.5 m in length, and were drilled at a variety of orientations with dips from -45º to -90º.

The underground drill is a lightweight rig manufactured by Chibougamau Diamond Drilling and driven by air pressure. It is mounted on a steel frame and drills ATW (3.0 cm) core. The shortest underground hole was 7.5 m in length, and the longest 118 m in length.

From 2001 to 2006, geologists provided by SOQUEM and Stornoway logged core using a method termed by Stornoway as “lithological” logging. During this period, all core logging was completed at the secure core shack at the Lagopède camp prior to being shipped to North Vancouver for diamond analysis. In 2006, the logging style changed to a more detailed procedure termed by Stornoway as “petrological” logging. This method involves laying out one (or more) drill hole(s) in its (or their) entirety to allow the logger to compare geology between sections within an individual drill hole or across several drill holes more efficiently and effectively. In addition, with the petrological logging method, more emphasis is placed on petrographic and thin section work to identify and correlate geological units. Beginning in 2006, all drill cores containing or suspected to contain kimberlite were logged using the lithological technique in camp and then shipped from the Project site to Stornoway’s North Vancouver facilities for petrological logging. In addition, core remaining from drill programs completed prior to 2006 was re-logged using the petrological logging technique. In 2009 and 2010, core was logged using the petrological method in the field. To allow core holes to be laid out in their entirety, core logging was completed in a larger facility adjacent to the core shack.

In addition to a traditional drill core log which noted rock type, mineralogy, colour, texture, structure, and alteration, a modal analysis spreadsheet was completed for holes drilled between the summer of 2003 and the end of 2008. Estimates of the country rock xenolith, kimberlitic matrix, olivine macrocryst, magmaclast, and autolith content, in percent, were noted for each 3 m core interval, and used to produce a visual representation of the estimated percent kimberlite content versus depth. Beginning in 2009, the percentage of country rock xenoliths within the kimberlite units was measured directly using a line scan method. Line scans are completed by measuring all country rock xenoliths greater than 0.5 cm along a line drawn on the core. The country rock xenolith percentage is calculated by dividing the total of the country rock xenolith measurements by the length of the section measured, which is typically 1 m. Line scans have also been undertaken on archived drill core.

Core boxes were labelled with aluminium embossing tape and stacked on wooden pallets. A portion of the core was labelled “waste” and stored at the camp. Between 2001 and 2008, kimberlitic material was palletized, strapped, numbered, and shipped to Chibougamau or Mirage by air. For the core designated for diamond testing, pallets were immediately loaded onto a locked van provided by Fastfreight of Montréal. Once full, the van was sealed and shipped directly to the North Vancouver laboratory. If no tests were required on the core, the pallets were shipped across the country using regular commercial transport carriers. Since 2009, only core required for petrographic or diamond analysis and telescoped holes are shipped to Stornoway’s North Vancouver facility.

Core recovery and other geotechnical parameters are collected for all drill cores on the Project kimberlites. Core recovery in the area of the Renard Cluster, Lynx, Hibou and North Anomaly dykes is typically greater than 90%. Each box of core is photographed with a digital camera at the Lagopède core shack. Photographs are then identified with the drill hole and core box numbers and stored on Stornoway’s network in the North Vancouver office.

10.4 Geotechnical, Geomechanical and Hydrogeological Drilling

While drilling for delineation or mini-bulk samples in the Core Area, detailed geotechnical observations have been recorded from exploration drill core. All holes are logged for geotechnical parameters such as total core recovery (TCR), rock quality designation (RQD), intact rock strength, weathering/alteration, joint orientation, joint condition rating and fracture frequency in order to obtain rock mass quality values. Point load and Schmidt Hammer measurements have been completed on selected holes for Renard 2, Renard 3, Renard 4, Renard 9 and Renard 65. Beginning in 2009, holes have been drilled to produce oriented core for the purpose of obtaining orientation data from the core. Azimuth (Alpha) and inclination (Beta) measurements for all fractures in the oriented drill core were recorded to aid in the development of a geotechnical model of the Renard mine site.

Eight holes (122.7 m) were drilled for hydrological studies in the vicinity of Renard 2 and Renard 3 before the beginning of the underground operations (Barbeau, 2006). Seven of these drill holes were subsequently converted to water wells. The holes were advanced through the overburden down to the bedrock. Soil sampling and soil identification were performed every 1.5 m. Two to three permeability tests were carried out in each borehole. A pumping test was performed to evaluate the hydraulic conductivity of the ground. This drilling was done using a regular diamond drill with PQ diameter drill rods, allowing the construction of observation wells with 50 mm PVC pipes and screen.

A total of 26 geomechanical drill holes to test the proposed open-pit walls and deep underground at Renard 2, Renard 3, and Renard 4 were completed in March 2010 for a total of 2,436 m. These core holes were also used for geological modelling purposes.

Packer testing was carried out in March 2010 on two historical boreholes to test the hydraulic conductivity of both country rock and kimberlite at depth at Renard 2 and Renard 4.

Twelve boreholes (109 m) were drilled in 2010 for hydrological studies in the area for future Processed Kimberlite Containment (PKC) facility, Plant, Airstrip and Landfill sites. The holes were advanced through the overburden and a short distance into bedrock to verify competency. Soil sampling and soil identification were performed at regular intervals during drilling, and hydraulic conductivity tests were also carried out within the wells. Drilling was done using an NQ-size (4.75 cm) split-spoon-hammer setup to advance through the overburden. Observation wells were then constructed using 50 mm PVC pipes and screens, sand filters, with bentonite plugs at the base of the well.

Thirty-nine overburden “profiling” boreholes were drilled in October 2010 totalling 200 m. The objective of this drill program was to map overburden thicknesses in proposed areas for future PKC facility, process plant, airstrip and landfill sites. Drilling was done using a standard diamond drill rig to advance through the overburden down to bedrock.

Twenty boreholes totalling 493 m were drilled in winter 2011 to characterize sediment and bedrock conditions along a proposed dyke that was considered at one point in the study; the final Project design does not incorporate the proposed dewatering dyke. A triple tube core rig was mobilized to site to conduct the drilling campaign. Twelve of the holes were for triple-tube soil coring and were advance 3m into bedrock, six holes were advanced 15 m to 50 m into bedrock, and two holes were drilled to define bedrock profile with soil samples recovered. Four

of the holes were equipped to perform hydraulic conductivity testing and static water level measurements. One of the holes was equipped with a Casagrande-type piezometer.

During the summer of 2012, a series of geotechnical and geomechanical investigations have been performed in order to obtain additional information on soil and bedrock for the detailed engineering design. Nine geomechanical holes (777 m) have been drilled in the bedrock to obtain structural data in the vicinities of the portal and the underground ramp. Two holes (69 m) were drilled to sample the bedrock in the vicinity of R65 in order to perform tests to determine the cement aggregate potential of this rock. Three holes (159 m) were drilled to perform packer tests, to assess the hydraulic conductivity of the rock, The hydraulic conductivity has also been verified at different areas along fault 28 using packer testing inside two existing holes and without packer testing inside another three existing holes. A total of 47 boreholes have been performed in soil in order to determine the properties (soil type, thickness, etc.) in areas where infrastructure will be built, such as the airstrip, mine roads, buildings, etc. Monitoring wells have been installed in 26 of the boreholes to monitor water levels through the seasons. Some boreholes in soil have been extended below the rock contact to determine the rock unit type and its potential bearing capacity for infrastructures.

10.5 Collar Surveys and Down Hole Surveys

Before the fall of 2010, all positional work on the Property was carried out using a transformed UTM 18 NAD 27 local coordinate system (Canada Mean). Since then the transformed UTM 18 NAD 83 local coordinate system (Canada Mean) has been the reference system on the Foxtrot Property.

Exploratory drill holes prior to 2004 were located relative to marked grids constructed using either Global Positioning System (GPS) units or chain and compass. In addition to using the marked grid, all exploratory borehole locations were verified by hand-held GPS units with no differential correction. Exploration drill hole azimuths and inclinations are set using a compass and protractor, respectively. Final exploration borehole inclinations are surveyed using an “acid test” system.

To position delineation and mini-bulk hole collar location pickets and front sights for holes drilled within the Renard Cluster, a registered surveyor uses GPS equipment with sub-centimetre accuracy. Technicians use a theodolite to align the drill rig with the collar picket and front sight pickets to ensure the holes are started accurately. After the drill programs are completed, the surveyor returns to locate the drill collar positions accurately by surveying the casings for locations and orientations. During 2003 and 2004, Corriveau J.L. and Associates Inc. of Val d’Or, Québec, surveyed all the delineation and mini-bulk (as well as several of the exploratory) borehole collars using a Leica System 500 RTK real-time, differentially corrected GPS system (DGPS) with ± one centimetre accuracy. In addition, cut-and-chained stations on the ground grids covering Renard 2, Renard 3, Renard 4, Renard 65 and Renard 8 were accurately located using the same survey equipment, and a series of control points were established on the Property.

From the winter of 2005 until present, Paul Roy Surveyor of Chibougamau, Québec confirmed locations of all survey collars. The downhole track of core holes drilled at Renard 2, Renard 3, Renard 4, Renard 65, Renard 7, Renard 8, Renard 9, and Renard 10 and several exploratory

holes drilled outside the Renard Cluster, were surveyed using the FlexIt borehole survey instrument to determine the azimuths and inclinations more accurately. The FlexIt instrument is an electronic downhole compass tool that measures several parameters, such as azimuth, dip, magnetic field, and dip of the magnetic field. Once the data are downloaded into a computer, a program corrects for magnetic declination and rejects readings above the average magnetic field (56,000 nanoTeslas) for the area of the Renard bodies. The readings above average generally correspond to areas of magnetic rock. In addition to the FlexIt system, a gyroscope system from Gyrosmart was used for the summer 2007 drilling campaign. This particular survey tool (IBG10), known as Imego's digital Butterfly Gyroscope, uses a MEMS (Micro-Electro-Mechanical System) gyro sensor and allows the downhole orientation survey to be carried out inside the drill string. For the drill programs completed during 2009 and 2010, in addition to using the FlexIt tool, a DeviFlex system from Devico A.S. was used to determine borehole geometry. The DeviFlex is a non-magnetic, electronic, multishot tool that uses accelerometers and strain gauges to calculate changes in inclination and azimuth. It is not affected by magnetic fields and is used for surveying inside casings and drill strings.

All the core holes drilled on the Property are surveyed for magnetic susceptibility. From 2001 to 2006, an Exploranium KT-9 hand held instrument was used to manually measure the core at select intervals. Since 2006, magnetic susceptibility readings have been taken using a continuous reading, multi-parameter probe (MPP) from Instrumentation GDD.

10.6 Drill Programs

Between 2001 and 2002, drilling in the Renard Core Area was undertaken primarily as early-stage, exploration-focused programs. From 2003, most of the drilling was used to support delineation work and advanced-stage project evaluation through the collection of mini-bulk and bulk samples for all bodies except Renard 9 and Renard 10, which were discovered in 2003. Additional target exploration drilling has been ongoing since 2001 on the larger Foxtrot Property. These drilling programs have been discussed in detail in earlier technical reports (Clements and O'Connor, 2001, 2002; Lucas et al., 2003; Lépine and O'Connor, 2004; O'Connor and Lépine, 2005, 2006). Location plans for drilling not used to support mineral resource estimation can be found in these reports. Details of the 2009 program can be found in the May 2010 Preliminary Assessment (Lecuyer et al., 2010) while the 2010 program is discussed in the technical report for the Feasibility Study filed in 2011 (Bedell, et al., 2011). Drilling in 2011 and 2012 was conducted for geomechanical, geotechnical and hydrogeological purposes and is described in section 10.4 of this report.

Tables 10.1 and 10.2 summarize the drilling on the Foxtrot Property. Drill hole collar locations and outlines of the kimberlite bodies contributing to the Mineral Resource Estimate are discussed in Section 14.

11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Laboratories

Three Dense Media Separation (DMS) process facilities have been used as primary macrodiamond extraction laboratories during the Renard exploration programs to date: an external unrelated 10 tph commercial facility at Thunder Bay, Ontario (owned and operated by Kennecott Canada Exploration Inc. and doing business as the Thunder Bay Mineral Processing Laboratory (TBMPL)); an internal 5 tph facility situated in North Vancouver, British Columbia (owned and operated by Stornoway); and, an internal 10 tph facility situated at Camp Lagopède, Québec (owned by Stornoway and SOQUEM, and operated by Stornoway on behalf of the joint venture). The TBMPL is accredited by the Standards Council of Canada to the ISO/IEC 17025 standard as a testing laboratory for specific tests. Neither of the internal DMS facilities are accredited, but have been independently audited and are subjected to ongoing QA/QC testing. One non-accredited laboratory (Microlithics Laboratories Inc., Thunder Bay, ON) was used as a secondary facility for auditing tailings from mini-bulk and bulk samples. Kimberlite concentrate generated by the TBMPL and Lagopède DMS facilities was shipped to the North Vancouver laboratory for final treatment and recovery of diamonds.

During the Renard exploration programs, microdiamonds were recovered by one internal facility situated in North Vancouver, British Columbia (owned and operated by Stornoway) and four external unrelated commercial facilities: Microlithics Laboratories Inc. located in Thunder Bay, ON; Saskatchewan Research Council Geoanalytical Laboratories (SRC), Saskatoon, SK; TBMPL (owned and operated by Kennecott Canada Exploration Inc.), Thunder Bay; ON; and, SGS Lakefield Research Ltd. (Lakefield) in Lakefield, ON. Neither Stornoway's internal facility nor Microlithics Laboratories are accredited, but have been independently audited and are subjected to ongoing QA/QC testing. Both Lakefield and SRC are accredited by the Standards Council of Canada under ISO/IEC 17025 "General Requirements for the Accreditation of Calibration and Testing Laboratories (CAN-P-4D)". The TBMPL is accredited by the Standards Council of Canada to the ISO/IEC 17025 standard as a testing laboratory for specific tests. Microlithics, SRC and SGS Lakefield were further utilized as secondary laboratories to cross-check and verify recoveries from other microdiamond facilities.

11.2 Dense Media Separation (DMS) Facilities

Dense media separation is a standard industry process for the liberation and extraction of macrodiamonds from large volumes of sample material (commonly tens to thousands of tonnes). Rock samples are progressively crushed and the disaggregated material passed over a series of size sorting screens before being mixed with a slurry of ferrosilicon and water. A cyclone is used to split the heavy minerals, including diamonds, from the lighter waste rock. The heavy mineral concentrate is removed from the DMS plant and stored under secure conditions until the diamonds can be extracted. Waste material is recycled through the plant and recrushed to liberate finer and finer diamonds. The minimum and maximum diamond size that can be recovered by the process is determined by the plant configuration. For the Renard Project, all DMS plants targeted stones of +1 DTC screen size (essentially equivalent to a +1.18 mm square-mesh screen size).

The non-accredited, internal Stornoway laboratory in North Vancouver provides mineralogical and geochemical analyses for diamond exploration in support of Stornoway's exploration projects. The laboratory includes a 5 tph Bateman/Van Eck & Lurie DMS facility for the recovery of commercial-sized diamonds from core, mini-bulk and bulk samples. The diamond recovery circuit includes a sizing circuit, an X-ray flow-sorting machine and grease table equipment.

A 10 tph Bond Equipment DMS facility was mobilized, assembled and commissioned on site at Camp Lagopède in Québec during 2006 and 2007. This internal facility was used in 2007 and 2008 to process material from mini-bulk and bulk sampling programs.

Kennecott's TBMPL owns and operates a 10 tph Bateman/Van Eck & Lurie DMS facility, initially constructed in 1993 and rebuilt during 2003. This facility was used during 2004 to process RC chip samples collected from the Renard Project.

11.3 Caustic Fusion Sampling

The caustic fusion technique, also known as caustic dissolution, utilizes chemical processing to provide total liberation of all diamonds within a given sample in order that an accurate diamond distribution can be determined. Caustic dissolution processes are usually applied to recover microdiamonds from relatively small volume samples (tens to hundreds of kilograms). Rock samples are loaded into large steel pots and caustic soda is added to dissolve the mineral matrix hosting the diamonds. Dissolution takes place over an extended period of time in temperature-controlled kilns. Once the reaction is complete, the residue is cooled and poured through stainless steel wire mesh screens at the required size to avoid loss of small diamonds. Depending on the size of the residue, further standard dissolution may be required. In cases where abundant oxides remain in the residue, a variety of other chemicals may be used to reduce the size of the concentrate, without harming the diamonds. Residues are then observed under microscopes by trained personnel, and the diamonds recovered, counted, sized and weighed.

To assure the integrity of the process, a chain of custody is established between the customer and the laboratory. Customer samples are processed in a controlled environment to ensure that confidentiality is maintained at all times. All samples are handled with due diligence during processing stages, according to previously defined protocols. Quality control grains are added to each aliquot undergoing the caustic dissolution process to monitor recovery. Similar caustic dissolution processes are used by all four external unrelated commercial facilities.

Historically, Stornoway's internal laboratory facility applied a process of attrition milling, followed by heavy mineral separation and caustic fusion dissolution, as the initial method of testing kimberlite discoveries for microdiamonds. Initial processing was a complex process of controlled and iterative crushing followed by fractionation, magnetic separation techniques and heavy liquid separation to produce diamond-enriched concentrates. Caustic reagents are mixed with a portion of the concentrate and melted by heating in a muffle furnace at high temperature. Diamonds are then recovered from the resulting fusion residue.

11.4 Database

Data collected from the various exploration, mini-bulk, and sampling programs were collated into a SQL Server relational database. Data requests are processed through the database administrator. Access to the data in SQL Server is restricted to the database administrator only.

The database is stored on the server in the North Vancouver office, with backups being performed every day. One copy of the database is removed from site on a regular basis. Processing and diamond results hard copy data are stored in fire resistant filing cabinets in the North Vancouver office as are hard copy data of the Renard core field logging. In addition, these hardcopies have been scanned as digital PDF files which are stored on the server.

11.5 Sample Security

All sample processing is undertaken by qualified operators who conduct their work in secure laboratory areas with restricted access and following strict sample handling protocols. Diamonds recovered from the caustic dissolution process are generally very small (<0.85 mm), have limited commercial value and the focus is to extract and retain all available stones. Security, while present, does not form as large a component of process as it does for the DMS work which can recover hundreds of carats of diamonds, many of which may be potentially attractive for theft.

DMS operations, post-processing treatment of DMS concentrates, observing, and post-observation handling of concentrates and diamonds, from 2004 to the effective date of this report, were conducted under approved security protocols and procedures, which include but are not limited to:

- Chain of Custody documentation;
- Dual locking containers;
- Uniquely numbered, single use, tamper resistant seals;
- Monitoring and control of sample weights;
- Limited access or dual access to certain laboratory premises;
- Closed-circuit TV surveillance; and
- External (third party) security guards.

Comparative analysis of diamond size distribution is checked against historical and external laboratory results.

11.6 Drill Core

In the case of drill core collected for microdiamond analyses (i.e. caustic dissolution) during the 2009 and 2010 drill programs, once sample intervals had been established in the field, core was extracted from the core boxes and packed into 20 l, white plastic pails lined with heavy polyethylene sample bags. Bar coded sample tags with unique identification numbers were inserted into each sample bag and the bags sealed. Pails were clearly labeled inside and out

with the same sample number, and then sealed with tamper resistant single use lids. Individual pail weights were recorded and this information forwarded to the company's expeditor in Chibougamau and the office in North Vancouver. The expeditor was notified once the pails left camp on a charter aircraft, met the aircraft at the landing site and transferred the sample pails to a secure location. The pails were then consigned to a commercial courier service for delivery to either the contract caustic fusion laboratory or to North Vancouver. Upon receipt of the samples the caustic fusion facility would take possession of the samples, document the number and weight of the pails and notify both the camp and the North Vancouver office. In certain cases diamond spikes were added to the core samples to assist in the sample tracking and QA/QC verification procedures.

Historically, during drilling operations for mini-bulk sampling, specific security measures were instituted to minimize the potential for tampering and to maintain the integrity of the drill core from initial coring on site to delivery of the core to the North Vancouver laboratory.

Once a full core tray has been reviewed by a joint venture geologist at the drill rig, the core boxes are sealed for transport by helicopter or ground transport to the core logging facility located at the Lagopède base camp, approximately 500 m west of the Renard kimberlite cluster. The core boxes are opened in the core library for detailed inspection, photographing and logging. The core library is a secure, locked facility without windows. Only authorized personnel are allowed inside.

Once logging is complete, core boxes are resealed and sent by various types of charter aircraft to LG-4 or Chibougamau where a quality control officer, employed by the joint venture, receives them. Core boxes are transferred by forklift truck from the aircraft to a semi-truck trailer and organized on palettes for delivery to Vancouver. The boxes are metal-strapped in a minimum of two locations to avoid movement during transport. Locks with hardened steel shackles secure the semi-trailer when the quality control officer is not present. The on-site quality control officer, the project manager located at the remote base camp, and the North Vancouver laboratory manager possess keys for the trailer locks. Once the semi-trailer is fully loaded, it is locked a final time and numbered security seals are placed on the lock hasp. The trailer is then dispatched to North Vancouver, where it is received by the laboratory manager. All the information such as seal number, trailer and container numbers, and date of departure is transmitted to the Vancouver office. Once the shipment arrives in Vancouver, the seal is verified and the van is off-loaded by a technician who verifies the contents against the chain of custody.

11.7 Reverse Circulation (RC) Chips

Reverse Circulation sample drilling operations occur under strict security measures to minimize potential tampering and to maintain the integrity of the samples while being shipped from the drill site to the North Vancouver or commercial laboratory. A fenced area with controlled access at the Lagopède camp and/or the LG-4 site, was used for temporary storage of RC chip samples during the summer 2004 and 2006 drilling programs, prior to shipping the samples south. For the winter and fall 2007 RC drilling program, RC bags were stored in a restricted access area adjacent to the Lagopède DMS facility, before being processed on site.

Joint venture geologists supervised the sample collection. During drilling, access to kimberlitic chips was limited to authorized joint venture staff and drillers. Once filled, 1.5 t sample bags or

205 l steel drums were sealed immediately with cable ties and a security seal was applied to deter opening the bags or the drums. The bags and the barrels were loaded on pallets and transported by forklift to the camp's restricted loading area until loaded into aircraft.

During the 2004 and 2006 winter drilling program, large capacity aircraft were used to transport the material to Chibougamau or Mirage. Once at the destination, the bags were unloaded from the aircraft for storage at a secure location where only authorized personnel were permitted, before being shipped to the Thunder Bay DMS facility or the North Vancouver facilities. During the summer of 2004, float-equipped airplanes were used to carry secured 205 l barrels to Mirage, a tourist outfitter lodge near LG-4, before being shipped to the Thunder Bay DMS facility. Seal number, trailer and container numbers and date of departure were recorded on the chain of custody and were transmitted to the Vancouver office. Once the shipment was in Vancouver, the seal was verified and the van was off-loaded by a technician who verified the contents against the shipping papers and checked sample weights.

11.8 Bulk Sampling

11.8.1 Underground Bulk Samples

Access to the underground workings was restricted to ensure safety of personnel, as well as security and integrity of the samples. Only personnel involved in the underground program had access to the kimberlite, such as geologists, surveyors, equipment operators and drillers. On surface, the sample storage areas, treatment compound and DMS facility were considered as restricted areas and subject to access control, camera surveillance and security monitoring. An independent Québec-based security firm, Canaprobe, was mandated to implement site security protocols and provided bonded security officers. During bulk sample processing, only authorized personnel had access to the restricted areas. Security personnel escorted all approved visitors. All personnel involved in the operations at the DMS facility were subject to criminal record checks and random daily searches. Surveillance by the security force included use of overlapping and task specific CCTV coverage, as well as a continuous physical presence within the DMS plant. Locked dual custody provisions were in force in the vicinity of the DMS concentrate.

The concentrate was discharged from the DMS into 205 l steel drums lined with a heavy polyethylene sample bag. Once sample processing was complete, or the drum was full, the sample bag was sealed with a numbered tamper-resistant security tag by both a processing representative and a security representative. Each drum was mechanically sealed, locked and secured with two additional numbered tags. Sealed concentrate drums were weighted, labelled and moved to a secure, access controlled, storage site under constant camera surveillance and chain of custody documentation. Once enough concentrate drums had been collected, they were removed from site on a charter aircraft and transported under chain of custody to Chibougamau or Mirage, where they were temporarily stored in locked containers or secure vans before being shipped to North Vancouver with a bonded trucking company. Concentrate was transported in a locked, sealed and documented tractor-trailer on a "non-stop, 24/7" continuous basis from Chibougamau or Mirage directly to North Vancouver. Concentrate reception in North Vancouver was under dual custody conditions and included monitoring by external security officers, drum/external security seal number verification, integrity inspections and weight control. Concentrate drums were then placed in a locked, access-controlled storage

cage under CCTV surveillance until they could be processed. Paper records of access/transfer points were also maintained.

11.8.2 Renard 4, Lynx and Hibou Trench Bulk Samples

To ensure the security and integrity of the large tonnage surface samples, and the sample process itself, access to the trenches and the DMS stockpiles was limited to personnel specifically involved in the sampling, such as geologists, field assistants and heavy equipment operators. A log of personnel accessing the sites was recorded daily by Security Personnel. Detailed records of the sample collection were kept including times of various activities and any incidents such as spillages. In addition, records were kept of all sample transfers from the trenching site to the DMS stockpile.

11.9 Final Diamond Treatment and Recovery

Diamond bearing concentrates generated by DMS processing of underground bulk samples, large tonnage trench samples and RC chip samples from the Renard Project were all subjected to final processing at Stornoway's North Vancouver laboratory facilities. The diamond recovery circuit includes a sizing circuit, an X-ray Flow-Sort machine and grease table equipment.

All processing of concentrates was undertaken in secured, controlled access, CCTV monitored areas of the North Vancouver facilities. An independent, external and bonded security force (FBIG Investigations) monitored CCTV equipment, provided a physical presence at critical points of the diamond extraction process and recorded both routine activities and any abnormal incidents (sample spillage, etc.). These security personnel also checked sample seals, sample weights and provided key control services for dual-locked storage areas, concentrate canisters and restricted areas. Processing personnel were subject to random searches at various times.

12.0 DATA VERIFICATION

12.1 Stornoway Quality Assurance and Quality Control Programs

The descriptions of the Stornoway QA/QC programs are taken from O'Connor and Lépine (2005; 2006) and Lecuyer et al. (2009).

12.1.1 Caustic Fusion and DMS Sampling

QA/QC testing is conducted on 5% of all samples passing through the North Vancouver caustic fusion dissolution circuit and Dense Media Separation (DMS) circuit. GeoStrat Consulting Services Inc. (GeoStrat reviewed the QA/QC procedures but did not independently verify the QA/QC procedures for the caustic fusion and DMS samples.

QA/QC programs conducted by Stornoway include:

- Blind spiking of samples in processing, using "known" natural and synthetic diamonds prior to processing with recovery determined at the observation stage;
- Blind spiking of samples in observing, using "known" diamonds prior to observation;
- Regular testing of all machines and equipment;
- Calibration and verification procedures;
- Routine audits of non-observable fractions and reject materials;
- Use of internal standards and reference materials which are calibrated to provide traceability and reproducibility;
- A record-keeping system of documentation, which retains in archives all original records and data, with all amendments clearly marked, initialled and dated for reference;
- Corrective actions which are implemented immediately when any aspect of laboratory analysis, or chain of custody documentation does not conform to procedural standards;
- The investigation and verification of any result which appears to be a potential statistical anomaly, to ensure laboratory results fit within the geological context; and
- Use of external laboratories for check samples. Up to 5% of all samples are routinely sent for external analysis.

The caustic fusion and DMS diamond recovery facilities are governed by a series of detailed procedures that are appropriate to ensure the security and integrity of samples and the final results. All samples received in the laboratory are accompanied by a chain of custody document and with security seals that must be verified prior to processing any sample. Upon receipt, the samples are stored in a secure facility with restricted access. The diamond recovery circuits are in restricted areas and all samples, concentrates, diamonds and data are locked in safes, cabinets, drying ovens, or secure rooms when not being handled.

12.2 GeoStrat Verification

As part of the independent expert review, David Farrow and Darrell Farrow conducted the following verification checks on the Foxtrot Property:

- Site visits by David Farrow and Darrell Farrow from March 5 to March 8, 2009 and by David Farrow on August 21, 2012;
- Review of the surface and underground geological and mineralization interpretations (Section 7);
- Review of the historic and current exploration programs (Sections 6 and 9);
- Deposit model (Section 8);
- Review of data that are supporting mineral resource models (Sections 9, 10, and 11). The review covered drill core inspection, review of core logging, sampling and assay protocols and methods, and review of sample security measures and sample storage;
- Review of QA/QC data protocols and methods, data integrity and validation of RC, drill core and underground data (Sections 11 and 12), and
- Review of diamond valuation methodologies.

GeoStrat has visited the Project site and the North Vancouver offices in order to audit procedures at Stornoway. Independent samples were not collected and treated by GeoStrat since this is not practical for diamond sampling.

The audit process requires matching of raw data from field copies for the various data collection areas to final copies of data to be used in public reporting and resource estimation.

GeoStrat has further reviewed documentation of procedures and verified that activities in the field conform to Stornoway's published internal procedures for those activities.

GeoStrat is of the opinion that Stornoway's published and practiced procedures for collection of data in the field and transposition of these data into data 'products' to support resource evaluation work and initial costing exercises meet industry best practice guidelines.

12.2.1 Special Considerations for Diamond Resource Determination

Unlike commodities such as gold or base metals, diamonds do not have a standard value per unit weight that can be used to calculate value of a deposit. A one carat diamond can be worth from less than one dollar to tens of thousands of dollars, depending on the shape, colour and quality. A parcel of diamonds must be individually examined to establish an average value. Diamond values also change with the mix of diamonds over time, however, as a whole, diamond values have tended to increase with time. As this can be a somewhat subjective exercise, multiple valuations from different professional diamond valuers, or diamantaires, are necessary, and are usually averaged to give an estimate of the probable true price of the goods in question. Diamond price estimates can differ between valuers by as much as ± 20%, especially on smaller parcels of diamonds. These differences are simply due to the fact that

different diamantaires will perceive the value of a stone or parcel of stones differently. Their price guidelines will differ somewhat as well.

In a valuation exercise, it is necessary to involve a number of diamantaires to obtain a range of valuations that can be averaged to get an accurate price estimate and to use these data to model an average price. Often, in early stage evaluations of diamond projects, diamond price modelling is undertaken. In price modelling, the small sample size is compensated for by estimation of what the diamond population in a larger sample would be. By doing this, the valuer attempts to predict the likelihood of finding larger stones and what their effect on the overall value of the parcel would be and, as such, estimate more closely what the run-of-mine (ROM) value would be. Modelling involves study of the diamond parcel on hand, including size distributions and valuations, to statistically estimate the upper and lower limits of a production parcel at certain confidence levels based upon the small parcel on hand. To accomplish this, Stornoway contracted WWW Diamond Consultants International (WWW) of London to obtain valuations and perform price modelling. WWW are recognized international leaders in this field.

Small parcels of diamonds are difficult and time consuming to value, so individual sample goods are generally combined on the basis of geology or some other parameter. Valuation parcels are generally sieved into DTC sieve classes (+1, +3, +5, +7, +9, +11) and grainer and carater categories. A cumulative 8,315.58 carat diamond parcel acquired by bulk sampling underground, trench sampling and RC drilling completed by Stornoway between 2003 and 2007 was used for value modelling.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

Dense media separation (DMS) is a standard industry process for the liberation and extraction of commercial sized macrodiamonds from large volumes of core, mini-bulk and bulk sample material. Rock samples are progressively crushed and the disaggregated material passed over a series of size-sorting screens before being mixed with a slurry of ferrosilicon and water. A cyclone is used to separate the heavy minerals, including diamonds, from the lighter waste rock. The heavy mineral concentrate is removed from the DMS plant and the diamonds are extracted. Waste material is recycled through the plant and recrushed to liberate finer and finer diamonds. The minimum and maximum diamond size that can be recovered by the process is determined by the plant configuration. For the Renard Project, all DMS plants targeted stones of +1 DTC screen size (essentially equivalent to a +1.18 mm square-mesh screen size) or above, although smaller stones have been recovered.

Three DMS process facilities have been used as primary macrodiamond extraction laboratories during the Renard exploration programs to date: an internal 10 tph facility situated at Camp Lagopède, Québec (owned by Stornoway Diamond Corporation (Stornoway) and SOQUEM Inc. (Soquem), and operated by Stornoway on behalf of the joint venture); an external unrelated 10 tph commercial facility at Thunder Bay, Ontario (owned and operated by Kennecott Canada Exploration Inc. and doing business as the Thunder Bay Mineral Processing Laboratory (TBMPL); and an internal 5 tph facility situated in North Vancouver, British Columbia (owned and operated by Stornoway). By far the largest proportion of DMS processing has been through the Lagopède plant, as shown by facility and sample type in Table 13.1. Table 13.2 shows the sample breakdown for the individual Renard kimberlite bodies comprising the current resource calculations.

Table 13.1: DMS Facilities – Sample Processing Breakdown

Sample Type	Lagopède		TBMPL		North Vancouver	
	# Samples	Weight (t)	# Samples	Weight (t)	# Samples	Weight (t)
RC	33	173.5	57	617.3	33	111.7
Surface	16	8538.2	0	0	27	109.0
Core	0	0	0	0	200	174.5
Underground	28	4562.5	0	0	0	0
Totals	76	13274.2	57	617.3	260	395.2

Table 13.2: DMS Facilities – Allocation of Resource Sample Processing

Resource Body	DMS Plant Total Weight Processed (tonnes)		
	Lagopède	TBMPL	North Vancouver
Renard 2	2535.6	171.2	34.6
Renard 3	2173.1	157.0	18.6
Renard 4	2104.2	141.8	103.0
Renard 65	5346.8	147.2	38.4
Renard 9	27.3	0	82.5
Lynx	494.3	0	48.9
Hibou	564.9	0	30.1
Totals	13246.2	617.2	356.1

The basic flow sheet used for each of the three mineral processing facilities is described in the following subsections, as well as an outline of the QA/QC procedures.

13.2 DMS Processing - Lagopède

A 10-tonne-per-hour Bond Equipment DMS facility was mobilized, assembled and commissioned on site at Camp Lagopède in Québec during 2006 and 2007. Stornoway operated this facility in 2007 and 2008 to process material from mini-bulk and bulk sampling programs. Trained DMS process operators supervised each shift in a secure building. Sample processing followed procedures similar to those of the Thunder Bay and North Vancouver facilities, with some exceptions due to the nature of remote field operations, as noted below.

After excavation, kimberlite samples were stockpiled outside the facility in a secure compound with restricted access. Individual samples were prepared through a primary jaw crusher, and 50.0 mm crushed product fed directly into the DMS facility where a scrubber trommel unit removes the +20.0 mm oversize. The +20.0 mm oversize material was reduced using a secondary cone crusher set at 10.0 mm. Coarse rejects were recrushed to -6.0 mm and re-circulated through the facility. The first seven subsamples of the 2007 bulk sample program used a 1.2 mm by 13.0 mm slotted screen. Subsequent samples employed a 1.0 mm by 12.0 mm slotted screen to increase the recovery of smaller diamonds. Contract security services were provided on site by an independent third party (Groupe Canaprobe of Montréal, Québec).

DMS concentrates generated by the Lagopède facility were sealed in drums and dispatched in a locked container under chain of custody protocols to Stornoway's internal laboratory in North

Vancouver for final treatment and recovery of diamonds. The laboratory owns and operates integrated diamond recovery equipment including a sizing circuit, X-ray flow-sort machine and grease table. Upon arrival at the North Vancouver diamond recovery facility, concentrates are screened into four size fractions, passed through an X-ray sorter twice, and the tailings diverted to a grease table circuit to generate a final diamond concentrate. Diamond-enriched concentrates are stored in sealed lock boxes within high security cages before being transferred to the observation laboratory for final diamond recovery and reporting.

A team of trained mineral observers and mineralogists undertakes final diamond recovery using a combination of hand-sorting and binocular microscopy techniques. All diamonds recovered are routinely verified, described, weighed, photographed and recorded by trained mineralogists. Results are reported as diamonds retained on a +1 DTC sieve, or as stones greater than 1.18 mm using Tyler square mesh sieve classes when required. All diamond recovery is carried out with dual custody handling provisions under video surveillance in restricted areas. An independent external security team monitors and records all operations, transfers and seal control. FBIG Investigations (Vancouver, BC) has provided these contract services since 2004.

Neither the internal DMS facility nor the diamond recovery circuit are accredited, but both have been independently audited and are subjected to ongoing QA/QC testing.

13.3 DMS Processing – Thunder Bay Mineral Processing Laboratory

Kennecott's TBMPL owns and operates a 10-tonne-per-hour Bateman/Van Eck & Lurie DMS facility, initially constructed in 1993 and rebuilt in Thunder Bay during 2002 and 2003. This facility was used on two occasions in 2004 to process RC chip samples collected from the Renard Project. Operators from Stornoway's North Vancouver facility supervised contract DMS processing in Thunder Bay to verify the quality and continuity of work being performed. The TBMPL is accredited by the Standards Council of Canada to the ISO/IEC 17025 standard as a testing laboratory for specific tests.

RC chip samples were sent to the facility in sealed pails, drums, or bulk bags, each with an individual sample number and security seal. Security seals were verified as being intact upon arrival at the laboratory, recorded in a chain of custody document and isolated in a secure storage area with controlled access overseen by an external independent security service (Apex Security, Thunder Bay). The same security team also monitored DMS processing and maintained video surveillance in restricted areas.

Since the kimberlite material was already broken into chips from the RC drilling process, it was directly fed to the DMS scrubber unit, to remove the +16.0 mm oversize fraction which was sent to a secondary rolls crusher set at 8.0 mm. Kimberlite was processed through the facility then the tails screened at 6.0 mm. Tails greater than 6.0 mm were recrushed in a roll crusher set at 3.8 mm and repassed through the plant. Finer tails were discarded. A 0.85 mm by 14.0 mm slotted product screen was used within the Thunder Bay DMS facility.

DMS concentrates were dried, then screened into six size fractions before being sent to the North Vancouver laboratory in secure shipments for further processing by heavy liquid and magnetic separation techniques. Final diamond recovery, classification and reporting were also carried out in North Vancouver under secure, independently monitored conditions (FGIB

Investigations (pre-2012) and Securigard Services Ltd. (2012-2013), both based in Vancouver, B.C.).

13.4 DMS Processing – North Vancouver Facility

Stornoway's internal laboratory in North Vancouver provides mineralogical and geochemical analyses for diamond exploration in support of Stornoway's exploration programs. The laboratory includes a 5-tonne-per-hour Bateman/Van Eck & Lurie DMS facility and a diamond recovery circuit. Kimberlite samples are received at the laboratory as small diameter drill core in sealed core boxes, reverse circulation drill chips in sealed pails, bags or drums, or as surface material from outcrop or boulders packed in drums or bulk bags. Each box, pail, bag or drum is given an individual sample number and sealed in the field prior to shipment. Bags and drums are identified with security seals that are verified as being intact upon arrival at the laboratory in a locked and sealed shipment. Once the chain of custody documents, sample numbers, and security seals have been checked and verified, the sample is isolated in a secure storage area with restricted access.

Trained operators conduct DMS processing in a secure portion of the laboratory. Samples are initially reduced through a primary jaw crusher at a nominal gap of 20.0 mm to 25.0 mm. This material is then passed through a combination of jaw and roll crushing at a nominal gap of 10.0 mm to 14.0 mm. The crushed product is fed into the five-tonne-per-hour DMS facility where a scrubber trommel unit removes the +14.0 mm oversize for recrushing.

After primary feeding and sizing, all sample material is processed through the cyclone. DMS floats are screened at 4.0 mm with the undersize going to tails, and the oversize to iterative recrushing and repassing at 4.0 mm and 2.0 mm to minimize the potential for diamond breakage. On early samples, a 1.0 mm by 12.0 mm slotted screen was employed within the DMS facility. In 2006, this was changed to a 0.85 mm by 14.0 mm slotted screen to better recover small diamonds, ensure a bottom diamond cut-point of 1.18 mm using a Tyler square mesh screen, and produce a minimum stone size smaller than that generally retained on a +1 DTC sieve.

Resultant DMS concentrates are screened into four or six size fractions, passed through an X-ray sorter twice, and the final tailings diverted to a grease table circuit for final recovery. Heavy liquid and magnetic separation recovery methods were employed on samples prior to 2006. Diamond-enriched concentrates are stored in sealed lock boxes within high security cages before being transferred to the observation laboratory for final diamond recovery and reporting.

Final diamond recovery, classification and reporting were also carried out in North Vancouver under secure, independently monitored conditions (FBIG Investigations, Vancouver, BC).

Neither the internal DMS facility nor the diamond recovery circuit are accredited, but both have been independently audited and are subjected to ongoing QA/QC testing.

13.5 QA/QC

Quality control testing was routinely conducted on drill core, RC chip, surface and underground samples processed through the three DMS facilities. Tailings and various processing residues are also subject to audit. QA/QC measures include but are not limited to:

- Adherence to established/documented processing and handling protocols;
- Systematic density bead tracer testing prior to operating plant to ensure efficiency and diamond recovery;
- Addition of luminescent density tracers prior to DMS to monitor processing and X-ray sorter efficiency. Recovery is measured after X-ray sorter recovery;
- Addition of identifiable natural diamond spikes prior to any DMS processing; recovery is measured at the diamond sorting stage;
- Addition of natural diamonds prior to observation to determine the efficiency of diamond sorting and for security control purposes;
- Audit of representative coarse DMS tailings from select kimberlite subsamples;
- Re-crush of DMS tailings and concentrate tailings to monitor processing efficiency, X-ray/grease recovery, and diamond liberation;
- Regular DMS feed size analysis;
- Testing of processing circuit clean-out residues;
- Audits of processing rejects using secondary laboratories and DMS facilities;
- Monitoring of diamond recovery statistics, including size frequency analyses;
- Independent third party audit processes (spiking, process review, etc.); and
- Review and audit of DMS and diamond data, operating procedures and QA/QC programs.

13.6 Metallurgical Testing

13.6.1 Mini Bulk Sample Processing

During advanced-stage evaluation of the Project, mini-bulk samples from drill core and RC chips (Renard 2, Renard 3 and Renard 4) were processed using small scale diamond processing plants located in Stornoway's facilities at North Vancouver and on site at Camp Lagopede in Québec. Some useful dense medium separation (DMS) concentrate yield data were generated. The information is based on dry concentrate weight and dry plant feed weight expressed as a percentage and summarized in Tables 13.3 and 13.4.

Table 13.3: DMS Yield Data – North Vancouver Plants

	1 tph Plant DMS Yield			5 tph Plant DMS Yield		
	Max %	Ave %	Min %	Max %	Ave %	Min %
Renard 2	3.07	1.59	0.22	1.63	0.61	0.12
Renard 3	3.46	2.38	1.47	2.14	0.72	0.25
Renard 4	5.78	2.3	0.46	1.06	0.23	0.05

Table 13.4: DMS Yield Data – Lagopède Plant

	10 tph Plant DMS Yield		
	Max %	**Ave %**	**Min %**
Renard 2	0.2	0.13	0.06
Renard 3	0.67	0.34	0.04
Renard 4	0.24	0.14	0.08

The 1 tph plant was operated to recover diamonds greater than 0.5 mm and DMS rejects were recycled and re-crushed to liberate additional diamonds and produce final plant rejects of less than 2 mm. Compared to a commercial plant where recovery of diamonds greater than 1 mm is usually targeted and final plant rejects are crushed to less than 6 mm, or in some cases higher, the DMS yields generated in the 1 tph plant are unrealistically high for commercial plant design.

The 5 tph plant was operated to recover diamonds greater than 1 mm and to produce final plant rejects less than 4 mm. Considering that the commercial plant will employ a high pressure grinding roll (HPGR) unit, which is proven to liberate diamonds and hence heavy minerals better than conventional cone crushing, it is reasonable to assume DMS yields achieved in the 5 tph plant will equate to the expected higher yields with the inclusion of a HPGR.

The 10 tph plant was operated with commercial plant settings; recovery of diamonds greater than 1 mm and production of final plant rejects less than 6 mm. However, because liberation was achieved with conventional cone crushing the DMS yields will be understated if these data are applied to a circuit with a HPGR.

Consideration of the DMS yield data generated produced plant design data presented in Table 13.5.

Table 13.5: Design SMA Yields

	Max %	**Ave %**	**Min %**
Design DMS Yield	2.14	1.63	1.06

AMEC considers the above design data to be conservative, leading to a moderately sized, easy to operate, recovery plant.

13.6.2 Bulk Sample Processing - 2007

Bulk samples from Renard 2 (2,449 t), Renard 3 (2,114 t) and Renard 4 (1,659 t) were processed on site at Camp Lagopède in Québec during 2007. DMS concentrate was shipped off-site to Stornoway's recovery plant in North Vancouver. The recovered diamonds were used for diamond price estimation and grade determinations through the analysis of Size Frequency Distribution models of the Renard diamond population. The data listed below was collected during this period to support future process design.

- Plant feed particle size distribution;
- DMS concentrate yield;
- Diamond recovery;
- Diamond size frequency distributions;
- Granulometry (rejects and DMS concentrate);
- Plant mass balance and operating data; and
- X-ray sorter concentrate yields.

Data from the bulk sample processing indicated the following:

- The production of fines (-1 mm), a predictor of ore hardness, ranged from hardest to softest; Renard 3 at 26% Renard 2 at 31% and Renard 4 at 39%;
- The kimberlites have a low DMS yield using conventional cone crushing for liberation;
- Approximately 95% of the diamonds by weight were recovered after two stages of x-ray sorting. The remainder were recovered using a grease table;
- Grease scavenging of x-ray rejects will be worthwhile in a commercial plant; and
- Renard 2 exhibits potential to host larger diamonds than the standard commercial plant top size cut-off of 25 mm.

13.6.3 Additional Testwork Completed to Support Process Design

Testwork as shown in Table 13.6 was completed to support process design.

Table 13.6: Summary of Testing Completed during Feasibility Study to Support Process Design

Tests Undertaken / Data Generated	Test Location
HPGR bench scale tests to product fineness. Results were modeled to predict full scale performance	SGS, Ontario
HPGR pilot scale tests to determine product fineness, influence of moisture in feed and expected operating gap	BC Mining, Vancouver
Scrubber tests to determine resistance to abrasion breakage	SGS, Ontario
Grindability tests to characterize ore hardness	SGS, Ontario
Settling tests to permit thickener sizing, select suitable flocculant and estimate flocculant consumption	Outotec, Ontario
Filtration tests to determine dewatering characteristics of fine kimberlite particles	Outotec, Ontario
Centrifuge tests to determine dewatering characteristics of fine kimberlite particles	Decanter, Tennessee
Wet x-ray and magnetic separation tests using DMS concentrate	Deb Tech, South Africa

Testwork findings were:

- Renard 2 kimberlites exhibit similar impact breakage as country rock breccia containing significant granite xenoliths. The crusher work index for these samples indicates moderately hard ore.
- Renard 2 kimberlites and country rock breccia are resistant to comminution by abrasion as indicated by t_a values of 0.37 to 0.55, where t_a is the measure of resistance to abrasion breakage. This indicates that a scrubber or a mill could be utilized for washing rather than comminution.
- Laboratory scrubbing results indicate that comminution attributed to scrubbing would generate limited additional fines; less than 1% and 5.5% for Renard 2 and Renard 3 kimberlites respectively.
- BC Mining generated process design data for a HPGR unit (750 mm diameter x 220 mm wide) in a tertiary crushing and recrush mode with a full feed size distribution. In the commercial plant design the required product size for treating a mixture of fresh feed (-50 mm, 135 tph) and DMS rejects (-30 +6 mm, 100 t/h) will be achieved with a specific pressing force of 2.5 N/mm2 and with a net specific energy consumption of about 2 kWh/t. For these operating conditions a large operating gap of 34 mm is predicted thus protecting against possible diamond damage or breakage. The HPGR product flake was found to be soft and easily broken.

- Outotec generated settling test and scale-up data for kimberlite slimes (-0.25 mm) thickening. Samples from Renard 2 settled easily requiring low flocculant dosage rates. In the commercial plant design, utilizing a high rate thickener, overflow clarities of 100 ppm will be achieved with a flocculant dosage of 30 g/t or slightly higher and a high density underflow of 45% w/w solids will easily be achieved and maintained.
- Outotec performed filtration evaluation tests using -0.25 mm processed kimberlite. The material proved difficult to filter such that filtration of fine processed kimberlite for the purpose of dry stacking is unlikely to be successful.
- Decanter Machine Inc performed pilot scale testing to determine the dewatering capabilities of an 18 x 60 solid bowl centrifuge with -1 mm processed kimberlite as feed. The tests proved that this material can be successfully dewatered for the purpose of dry stacking thus eliminating the traditional processed kimberlite containment dam with a central water pond.
- HPGR product generated from Renard 2 samples was processed in Stornoway's 5 t/h plant to produce concentrate. This material was tested in a dual wavelength x-ray machine to determine probable yields that could be obtained under production conditions. The tests demonstrated that satisfactory yields are possible with the dual wavelength feature. AMEC recommends further yield optimization work in the next phase of the Project.
- Although no formal wet magnetic separator tests were performed, preliminary test runs completed by Deb Tech provided anticipated mass reductions. Wet magnetic separators may be used in the recovery plant design to provide DMS concentrate mass reductions. AMEC believes that employing x-ray sorters for bulk mass reductions instead of the combination of magnetic separators and x-ray sorters will provide greater benefit to the Project from a reduction in building height and cost.

13.7 Basis for Recovery Estimates

DMS rejects (-6 +1 mm) were collected after the treatment of sub-samples from Renard 2, 3 and 4 and reprocessed in a small scale DMS plant and in the recovery plant at Stornoway's North Vancouver facility. From this audit exercise the lowest recovery efficiency of liberated diamonds in the Lagopede bulk sample plant was estimated to be 97.5% by carat.

Further plant efficiency audits were completed by adding 30 identifiable natural diamonds to the bulk sample plant feed and in a separate audit, 20 identifiable natural diamonds were added. In the first audit, 29 out of 30 diamonds were recovered giving a recovery efficiency of 96.7% by stone and 98.2% by carat. In the second audit 100% recovery was achieved.

Also in accordance with established QA/QC procedures 10 diamond simulants were added to each sample. A total of 149 simulants from 150 simulants introduced into 15 Renard 2 samples were recovered for an indicated recovery efficiency of 99.3%. All 120 simulants in 12 Renard 3 samples and 70 simulants in 7 Renard 4 samples were recovered for an indicated recovery of 100% in both cases.

The recovery efficiency for the commercial plant is estimated to be not less than 96% by carat (97% for DMS and 99% for recovery).

13.8 Metallurgical Variability

Renard 2 will account for 83% of diamond production, Renard 3, 8% and Renard 4, 9%. All major geological units from each deposit were processed during the bulk sampling campaign at Lagopede. Also the major geological units from Renard 2, the dominant kimberlite source, were tested as described in section 13.1 above.

13.9 Comments

AMEC believes that sufficient and appropriate test work has been completed to support the feasibility study process design.

14.0 MINERAL RESOURCE ESTIMATES

14.1 Introduction

This section documents the process by which the updated Mineral Resource Estimate for the Renard 2, Renard 3, Renard 4, Renard 65 and Renard 9 kimberlites and the Lynx and Hibou Dykes within the Renard Core Area (Figure 7.3) was established

The 2010 Renard Mineral Resource Estimate Update was prepared by an independent Qualified Person, David Farrow, P.Geo., of GeoStrat. The Mineral Resource Estimate comprises the integration of kimberlite volumes, bulk density, petrology and diamond content data obtained from 88,585 m of diamond drilling, 6,151 m of reverse circulation (RC) drilling, 16.4 t of samples submitted for microdiamond analysis, 600.8 cts of diamonds (6,457 stones) recovered from RC drilling and 8,611.6 cts of diamonds (84,381 stones) recovered from surface trenching and underground bulk sampling.

The methodology of the Mineral Resource Estimate has been reviewed by Darrell Farrow of GeoStrat and Dr Paddy Lawless of Dr Paddy Lawless and Associates CC.

14.2 Previous Work

A previous Mineral Resource Estimate was produced by Golder in 2009. The results were detailed in the previous Preliminary Assessment (Lecuyer et al, 2010) and are summarized in Table 14.1

Table 14.1: Historical Mineral Resource as of December 2009

Mineralization Category	Renard 2	Renard 3	Renard 4	Renard 9	Lynx	Hibou	Total
Indicated							
Tonnage	17,475,000	1,705,000	7,315,000	0	0	0	26,495,000
Total Carats	17,957,000	1,806,000	3,199,000	0	0	0	22,962,000
Average cpht	103	106	44	0	0	0	87
Average Dilution %	37	36	56	0	0	0	na
Inferred							
Tonnage	5,365,000	154,000	4,572,000	5,747,000	1,798,000	178,000	17,814,000
Total Carats	6,415,000	189,000	1,874,000	2,634,000	1,924,000	256,000	13,292,000
Average cpht	120	122	41	46	107	144	75
Average Dilution %	29	37	55	66	na	na	na

14.3 Methodology

The Mineral Resource Estimate for Renard kimberlite pipes 2, 3, 4, 9 and 65 is based upon the determination of undiluted grade models for each kimberlite lithofacies, as derived from diamond sampling, and their application to a dilution block model derived from diamond drilling. This methodology has been selected as appropriate based on the geology of the Renard kimberlites ("transitional", TK/HKt-TKt kimberlites), the recognition that country rock dilution is the principal driver of grade variability in the Renard kimberlites, and the observed stability of the diamond population on an undiluted basis within each kimberlite lithofacies. Resource block models created in this manner can then be tested against observed diamond sampling data, which includes RC drilling and diamond drilling for each of Renard 2, 3, 4, 9 and 65, surface trenching in Renard 4 and 65, and, underground bulk sampling in Renard 2 and 3.

The Mineral Resource estimation process comprised the following steps:

1) Establish geological models for each kimberlite pipe using diamond drill logs, petrographic analysis, and emplacement modeling;
2) Analyse the diamond populations;
    a) Review macrodiamond populations to ensure they are robust for mineral resource valuation;
    b) Develop the relationship between macrodiamond and microdiamond populations; and
    c) Establish that the microdiamond population is stable with depth.
3) Establish undiluted grade models for each lithofacies;
    a) Undertake diamond size frequency analysis using combined macrodiamond and/or microdiamond data from representative samples;
    b) Where appropriate, establish individual population grades for mixed samples; and
    c) Determine a factor to account for the presence of discontinuous, but mappable, hypabyssal units present within each lithofacies (units Kimb2c, Kimb3c, Kimb4c and Kimb65).
4) Establish the parameters of the block model.
5) Establish the dilution model;
    a) Establish distributions for country rock dilution in the various bodies and lithfacies based on line scan and/or modal means; and
    b) Undertake geostatistical analysis and estimation of the dilution per rock type, per kimberlite.
6) Undertake bulk density analysis;
    a) Establish bulk density per lithofacies; and
    b) Demonstrate that bulk density is constant with depth and dilution.
7) Integrate undiluted grade models and bulk density values with country rock dilution in the block model.

8) Validations ;

   a) Calculated dilution in the block model against measured dilution in drilling; and

   b) Calculated grade in the block model against observed diamond recoveries in sampling.

The Mineral Resource estimate for the Lynx and Hibou dykes was established by applying average grades from surface trench samples within a geological model established by drill hole data, given the different emplacement model and simpler internal composition of kimberlite dykes.

14.4 Geological Models

The geological investigation of the Renard kimberlites is based on the megascopic observation (logging/mapping) of drill cores, reverse circulation drill holes, underground drifting, magnetic susceptibility and the macroscopic and/or microscopic observation of representative petrographic samples of kimberlite. The combined data were used to differentiate geological units and in turn identify distinct internal phases of kimberlite, required for any economically relevant three dimensional (3D) geological model.

The Renard 3D geological models were constructed using the software program Gemcom Surpac and an access database with a drill hole mappable units table containing a summary of the above mentioned data. An initial 3D solid was constructed for the entire kimberlite body (including CCR and CRB). Following this, 3D clipping solids were created for each geological unit and phase of kimberlite and systematically intersected or clipped with the initial kimberlite pipe, clipping from the outside units inward. Each phase of kimberlite is typically separated by steep, near-vertical and predominantly sharp, cross-cutting internal contacts.

Geological solid models were created by Stornoway for each of the Renard bodies included in this study: Renard 2; Renard 3; Renard 4; Renard 9; Renard 65; Lynx and Hibou.

14.5 Diamond Sampling Analysis and Grade Models

Diamond sampling of the Renard Project kimberlites includes core sampling, RC chip sampling, small tonnage trench sampling, and bulk sampling from both surface trenches and the underground exploration workings. These data have been collected, compiled and analyzed by a combination of methodologies in order to cater to the spatial distribution of sampling and the amount of information available for each kimberlite domain in each body.

14.5.1 Macrodiamond Sample Analysis

Table 14.2 summarizes diamond recoveries from bodies that were used for size frequency analysis and estimation of average stone size.

Table 14.2: Recovery Data Used in Diamond Size Frequency Analysis

Kimberlite	Weight (t)	Total Stones (+3 DTC)	Carat Wt (+3 DTC)	Sieve
Renard 2 UG	2,448.8	13,246	1,590.58	DTC (+3)
Renard 3 UG	2,113.7	24,407	2,753.995	DTC (+3)
Renard 4 Trench	2,104.2	25,480	2,688.71	DTC (+3)
Renard 65 Trench	266.0	450	63.625	DTC (+3)
Lynx Trench	494.3	5,271	515.32	DTC (+3)
Hibou Trench	543.9	8,401	781.49	DTC (+3)

These data and results for size frequency and carats per stone are considered representative of each body for the purposes of resource estimation.

14.5.2 . Diamond Grade Models

The relationship between microdiamonds and macrodiamonds can be used to establish diamond content for a kimberlite through the creation of a grade model (Chapman and Boxer, 2004). Diamond grade models were created for each lithofacies in each kimberlite pipe where representative macro and micro diamond sampling was available. For Renard 2, representative samples weighing approximately 200 kg were collected for microdiamond recovery from each underground bulk sample round and placed in sealed 205 litre drums for storage. Archived material from specific underground rounds, representative of specific kimberlite facies, were submitted for standard caustic fusion microdiamond recovery. Dilution estimations were made for each caustic sample and were complemented by measured line scan data for each underground round. Dilution estimates were combined with microdiamond and macrodiamond results to determine undiluted stones per 10 kg, undiluted stones greater than 0.6 mm per 10 kg, undiluted total stones and an undiluted model grade (Table 14.3). The model grade was adjusted to reflect non-recovery of small stones due to DMS plant inefficiencies (i.e. the adjusted model grade is lower than the unadjusted model grade). No attempt has been made to account for potential higher stone recoveries in a more efficient commercial plant setting, or grade loss during sampling through diamond breakage.

The grade for the discontinuous hypabyssal units referred to as ‘c’ units was determined in a similar manner as the other diamond grade models. Due to the discontinuous nature of the ‘c’ units, macro diamond data were not always available for modeling with microdiamond data. In such instances, the microdiamond curves were used in conjunction with selected macrodiamond data from more magmatic samples to provide guidance in the larger stone sizes. Macro diamond data from unit kimb2b, in conjunction with UG rounds that host appreciable

mappable contributions from unit kimb2c, were used to assist in forming the shape of the distribution curve. Kimb4c utilized microdiamond data and a 600.2 tonne macro diamond sample with 2.0% dilution from a surface exposure. A total of 245.32 kilograms of caustic fusion data from drillcore, with a dilution value of 2.3%, was used to define the microdiamond portion of the curve. No macro diamond sample is available for unit 65c, the coarse portion of the Kimb65b, therefore the macrodiamond distribution was utilized to assist in defining the macro portion of the curve. A total of 256 kg of drillcore caustic fusion data, assigned a dilution value of 4%, comprises the microdiamond dataset used for unit 65c.

14.5.3 Microdiamond Sample Analysis

To test for the continuity of mineralization and stability of the diamond population with depth in the Renard 2 body, sampling for microdiamond analysis was undertaken on 50 m levels (horizontal slices) from beneath the previous RC and underground sampling to the base of the body at ~700 m (-200 masl) below the land surface. From each 50 m level, approximately 200 kg of Kimb2a and Kimb2b was collected from the 2009 drill core and treated for microdiamond recovery. A total of 2,046.93 kg of unit Kimb2a and 1,889.20 kg of unit Kimb2b were analyzed by standard caustic fusion processing. The results for Kimb2a and Kimb2b are presented in Figure 14.1 and Figure 14.2 respectively, as stones per hundred tonnes per unit interval plots which demonstrate that grades, on an undiluted basis, remain consistent with depth.

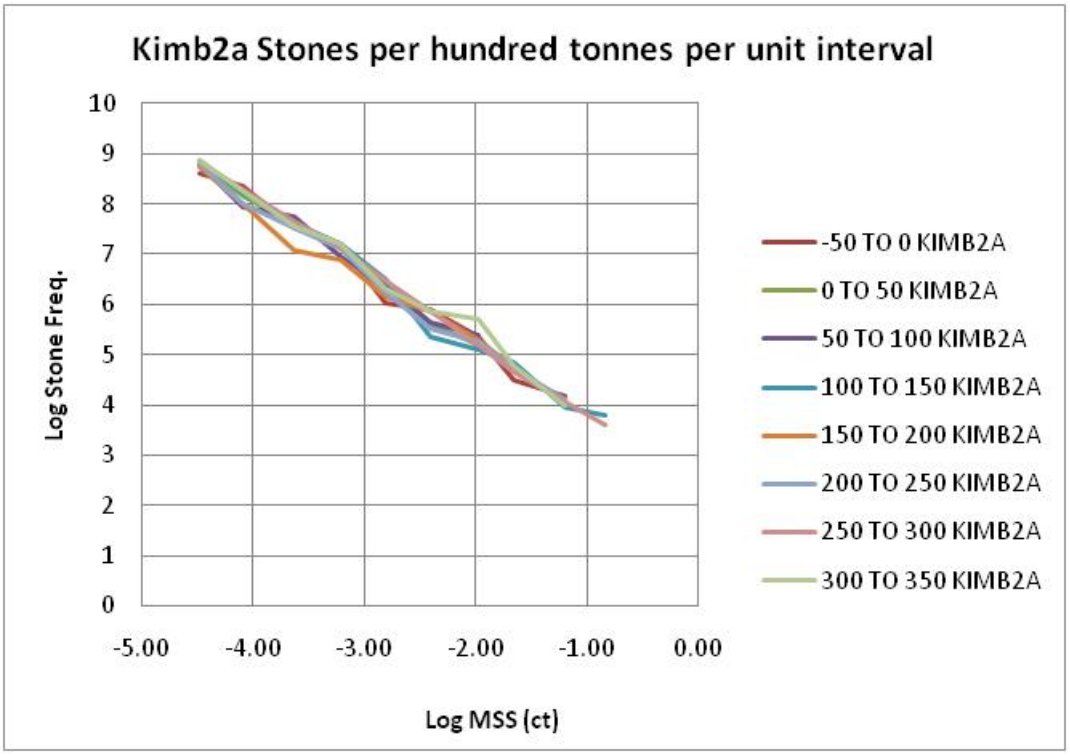

Figure 14.1: Kimb2a Microdiamond Sampling

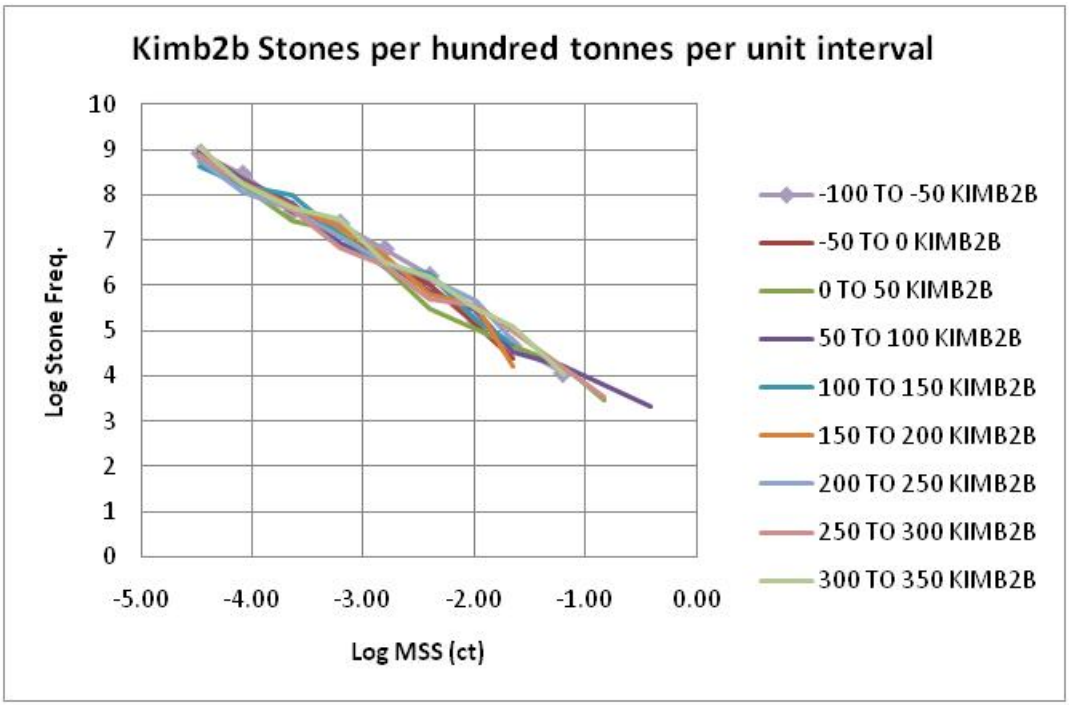

Figure 14.2: Kimb2b Microdiamond Sampling

Table 14.3: Macrodiamond and Microdiamond Samples Used to Determine the Micro/Macro Relationship in Renard 2 and Renard 3

	Kimb2a	Kimb2b	Kimb2c	Kimb3d/g	Kimb3f	Kimb3h	Kimb4a	Kimb4b	Kimb4c	Kimb4d	Kimb65a	Kimb65b	Kimb65c	Kimb65d
Microdiamond Sample wt (kg)	207.93	486.86	579.65	419.55	222.64	102.1	869.43	276.61	245.3	44.65	599.34	445.6	256.0	607.44
Microdiamond Dilution %	50	32	11.4	14	24.7	2.9	63.1	34.4	2.9	49.63	56.7	19.4	5.0	34.4
Microdiamond Total Stones	107	503	566	249	80	156	194	160	296	14	55	50	105	143
Undiluted Stones per 10kg	10.29	15.1	11.02	6.9	4.77	15.7	6.05	8.82	12.42	6.23	2.12	1.39	4.32	3.6
Undiluted Stones >0.6mm per 10kg	0.58	1.14	0.99	0.582	0.12	1.31	0.5	0.99	1.0	1.78	0.19	0.22	0.37	0.35
Macrodiamond Sample wt (t)	140.42	515.7	-	384.16	317.9	143.38	158.55	158.55	600.21	1504	185.64	198.6	-	165.03
Macrodiamond Dilution %	35	16	-	14	25	2.9	51	37	2.0	60	49.4	18.5	-	44.37
Macrodiamond Total Carats	58.17	646.69	-	559.14	278.62	274.81	61.48	61.48	1049.7	1672.11	45.63	40.33	-	36.23
Macrodiamond Total Stones	512	5646	-	5207	2197	2931	927	927	10697	16376	639	360	-	544
Model Grade Adjusted (cpht)	83	181	261	168	119	214	81	67	186	250	41	47	85	57

14.5.4 Macro Diamond Size Frequency Modelling

Diamond size frequency modelling was undertaken for all remaining kimberlite lithologies to determine undiluted grades for all units (Table 14.4). This was carried out primarily using the macrodiamond samples and fitting the curves to remain parallel to the microdiamond curves established for each pipe.

Table 14.4: Summary of Macrodiamond Data Modelling

	Kimb3i	Kimb9a	Kimb9b
Macrodiamond Sample wt (t)	28.5	27.6	na
Macrodiamond Dilution %	42	59	na
Macrodiamond Total Carats	9.39	20.23	na
Macrodiamond Total Stones	77	150	na
Model Grade (cpht)	45	136	136

14.5.5 Macrodiamond Sample Analysis

Table 14.2 summarizes diamond recoveries from bodies that were used for size frequency analysis and estimation of average stone size.

Table 14.5: Recovery Data Used in Diamond Size Frequency Analysis

Kimberlite	Weight (t)	Total Stones (+3 DTC)	Carat Wt (+3 DTC)	Sieve
Renard 2 UG	2,448.8	13,246	1590.58	DTC (+3)
Renard 3 UG	2,113.7	24,407	2753.995	DTC (+3)
Renard 4 Trench	2104.2	25,480	2688.71	DTC (+3)
Renard 65 Trench	266.0	450	63.625	DTC (+3)
Lynx Trench	494.3	5,271	515.32	DTC (+3)
Hibou Trench	543.9	8,401	781.49	DTC (+3)

These data and results for size frequency and carats per stone are considered representative of each body for the purposes of resource estimation.

14.5.6 Microdiamond Sample Analysis

The relationship between microdiamonds and macrodiamonds can be used to establish diamond content for a kimberlite (Chapman and Boxer, 2004). Once established, that relationship can be applied to microdiamond sampling for grade determination. Representative samples weighing approximately 200 kg were collected from each underground bulk sample round and placed in sealed 205 l drums for storage. Archived material from specific underground rounds, representative of specific kimberlite facies, were submitted for standard caustic fusion microdiamond recovery. Dilution estimations were made for each caustic sample and were complemented by measured line scan data for each underground round. Dilution estimates were combined with microdiamond and macrodiamond results to determine undiluted stones per 10 kg, undiluted stones greater than 0.6 mm per 10 kg, undiluted total stones and a model grade Table 14.3 The model grade was adjusted to reflect non-recovery of small stones due to DMS plant inefficiencies (i.e. the adjusted model grade is lower than the unadjusted model grade).

Table 14.6: Macrodiamond and Microdiamond Samples Used to Determine the Micro/Macro Relationship in Renard 2 and Renard 3

	Kimb2a	Kimb2b	Kimb2c	Kimb3d/g	Kimb3f	Kimb3h	Kimb4a	Kimb4b	Kimb4c	Kimb4d	Kimb65a	Kimb65b	Kimb65c	Kimb65d
Microdiamond Sample wt (kg)	207.93	486.86	579.65	419.55	222.64	102.1	869.43	276.61	245.3	44.65	599.34	445.62	256.05	607.44
Microdiamond Dilution %	50	32	11.4	14	24.7	2.9	63.1	34.4	2.9	49.63	56.7	19.4	5.0	34.4
Microdiamond Total Stones	107	503	566	249	80	156	194	160	296	14	55	50	105	143
Undiluted Stones per 10kg	10.29	15.1	11.02	6.9	4.77	15.7	6.05	8.82	12.42	6.23	2.12	1.39	4.32	3.6
Undiluted Stones >0.6mm per 10kg	0.58	1.14	0.99	0.582	0.12	1.31	0.5	0.99	1.0	1.78	0.19	0.22	0.37	0.35
Macrodiamond Sample wt (t)	140.42	515.7	-	384.16	317.9	143.38	158.55	158.55	600.21	1504	185.64	198.6	-	165.03
Macrodiamond Dilution %	35	16	-	14	25	2.9	51	37	2.0	60	49.4	18.5	-	44.37
Macrodiamond Total Carats	58.17	646.69	-	559.14	278.62	274.81	61.48	61.48	1049.78.0	1672.11	45.63	40.33	-	36.23
Macrodiamond Total Stones	512	5646	-	5207	2197	2931	927	927	10697	16376	639	360	-	544
Model Grade Adjusted (cpht)	83	181	261	168	119	214	81	67	186	250	41	47	85	57

14.5.7 Macro Diamond Size Frequency Modelling

Diamond size frequency modelling was undertaken for all remaining kimberlite lithologies to determine undiluted grades for all units (Table 14.4). This was carried out primarily using the macrodiamond samples and fitting the curves to remain parallel to the microdiamond curves established for each pipe.

Table 14.7: Summary of Macrodiamond Data Modelling

	Kimb3i	Kimb9a	Kimb9b
Macrodiamond Sample wt (t)	28.5	27.6	na
Macrodiamond Dilution %	42	59	na
Macrodiamond Total Carats	9.39	20.23	na
Macrodiamond Total Stones	77	150	na
Model Grade (cpht)	45	136	136

14.5.8 Mixed Sample Deconvolution

Deconvolution is the process of calculating individual lithofacies grades from mixed samples. Representative microdiamond and macrodiamond data for essentially pure Kimb3b and Kimb3c units were not available for grade modeling. Undiluted grades have been estimated through a back calculation methodology. Based on volume calculations from underground survey and mapping data, the proportion of undiluted Kimb3h, Kimb3d/g and Kimb3c have been calculated for underground rounds 3107, 3006, and 3008. The proportional contribution of total carats attributable to unit Kimb3d/g and Kimb3h was calculated based on pure grade calculations (see Table 14.3). This carat contribution was removed from the overall recovered carat total for each bulk sample round and the remainder attributed to the Kimb3c unit. The resulting weighted average grade for the Kimb3c unit is 230 cpht (Table 14.5). A similar exercise was conducted for the Kimb3b unit and involved bulk sample rounds 3104 and 3105 which are mixtures of Kimb3h, Kimb3d/g and Kimb3b. The undiluted proportion of carats was calculated for Kimb3h, and Kimb3d and removed from the total carats recovered from the bulk sample rounds. The remaining carats were assigned to unit Kimb3b and the weighted average grade for Kimb3b is 101 cpht (Table 14.5).

Table 14.8: Calculated Macrodiamond Grades

	Kimb3b*	Kimb3c*
Macrodiamond Sample wt (t)	348.3	521.7
Macrodiamond Dilution %	50.61	11.72
Macrodiamond Total Carats	244.98	938.19
Proportional Carat Contribution (3h, 3dg)	136.06	476.57
Proportional Weight (Kg)	108.2	201.1
Proportional Carat Contribution	108.92	461.62
Model Grade (cpht)	101	230

* Grades back calculated by stripping out contributions from Kimb3h and Kimb3d/g

14.5.9 Intrusions of Hypabyssal (Coherent) Kimberlite

Hypabyssal (Coherent) Kimberlite is present in each kimberlite pipe as coherent but discontinuous units (eg Kimb2c, Kimb4c, Kimb65c and Kimb3c). Where encountered in the underground decline on Renard 2 and Renard 3, the hypabyssal kimberlite units tend to be irregular in shape, orientation and in lateral and vertical continuity. Analysis of drill core shows a consistent proportion of hypabyssal kimberlite in each lithofacies unit of each body. In Renard 2, the average contribution of Kimb2c into the Kimb2a unit is 15.7%. For Renard 3, the percentage of Kimb3c present in 3b, 3dg, 3f, 3h and 3i is 15.9% for each unit. For Renard 4, the percentage of Kimb4c present in the dominant Kimb4a was calculated at 9%. For Renard 65, the percentage of Kim65c in Kimb65a unit is 10.7%. Hypabyssal kimberlite is also observed within the peripheral Country Rock Breccia ("CRB") and to a lesser extent in the Cracked Country Rock ("CCR") units, outside of the Mineral Resource.

14.6 Dilution Models

14.6.1 Dilution Sampling Analysis

Table 14.6 lists the sampling undertaken to establish the models of country rock dilutions in each kimberlite body. Different methods of dilution determination were employed by Stornoway over time. The most recent method used being "Line Scans", where the amount of dilution in each metre of core was measured in a quantitative and consistent manner. For Renard 2, only the line scan data was utilized to establish the dilution models. The older method, known as "Modal", employed a visual estimate per 3 m geological unit. Whilst coarser in resolution, where sufficient data coverage exists, modal data yields similar overall dilution figures to the line scan data. Due to the paucity of line scan data in bodies other than Renard 2, modal data was used to develop the dilution models. It is recommended that all available core from the other bodies be line scanned. In Renard 4 and Renard 65, the newer line scan data were averaged to mimic modal data and improved both representative and spatial coverage.

Table 14.9: Line Scan and Modal Dilution Data

Lithology	Line Scan			Modal		
	Count	Mean	Std Dev	Count	Mean	Std Dev
Kimb2a	3996	49.6	28.1	2249	49.5	29.9
Kimb2b	2560	27.4	20.5	437	35.1	23.5
Kimb3b	18	16.1	17.2	440	54.1	33.1
Kimb3d/g	157	34.2	18.1	725	38.0	24.9
Kimb3f	-	-	-	205	39.1	27.6
Kimb3h	-	-	-	123	27.4	25.7
Kimb3i	11	3.5	3.4	281	34.3	36.5
Kimb4a	-	-	-	1783	52.0	28.8
Kimb4b	-	-	-	846	37.5	20.69
Kimb4d	-	-	-	402	51.6	35.1
Kimb65a				1048	53.9	27.6
Kimb65b				166	31.0	16.2
Kimb65d				349	47.2	17.4
Kimb9a	-	-	-	1694	61.21	31.7
Kimb9b	-	-	-	63	26.91	22.8

14.6.2 Dry Bulk Density

Table 14.7 summarizes the bulk density results. A single bulk density per lithofacies type is used in this resource evaluation.

Table 14.10: Summarised Density Data

Body	Lithology	Number of Samples	Average Bulk Density	Standard Deviation
Renard 2	Kimb2a	110	2.61	0.05
	Kimb2b	50	2.73	0.07
	Kimb2c	38	2.67	0.07
Renard 3	Kimb3b	11	2.61	0.04
	Kimb3c	17	2.65	0.03
	Kimb3d+Kimb3g	61	2.74	0.07
	Kimb3f	10	2.68	0.04
	Kimb3h	2	2.58	0.09
	Kimb3i	9	2.75	0.07
Renard 4	Kimb4a	66	2.59	0.05
	Kimb4b	20	2.73	0.06
	Kimb4c	14	2.59	0.05
	Kimb4d	12	2.60	0.04
Renard 65	Kimb65a	7	2.57	0.05
	Kimb65b	5	2.72	0.06
	Kimb65d	6	2.77	0.08
	Kimb65c	6	2.59	0.06
Renard 9	Kimb9a	59	2.56	0.06
	Kimb9b	2	2.60	0.07
	Kimb9c	9	2.60	0.03
Lynx	HK	20	2.54	0.05
Hibou	HK	20	2.53	0.08
CR all bodies	GN, GN-GR, GR-GN, GR, PEG, CCR	77	2.72	0.09
CRB all bodies	CRB	17	2.61	0.09
CRB+K all bodies	CRB+K, CRB+/-K, CRB-3, CRB-3+K, CRB-2a	33	2.61	0.07

In certain kimberlites, the average bulk density of kimberlite lithologies may change with increasing depth due to a variety of factors, including alteration, dilution, magmatic variation, etc. The surface of the Renard 2 body is at 510 masl and the individual samples have been positioned at the center of the same 50 m level slices being used for ongoing microdiamond sampling (e.g., 450 masl to 500 masl). The second layer down, at 445 m, represents the level of underground sampling. For the most part the samples for each of the three Renard 2 kimberlite lithologies tend to cluster in a narrow density range, and it is considered that, for this study, density is constant with depth.

Much of the Renard kimberlite material contains a significant contribution of country rock xenoliths. These xenoliths could be expected to have an impact on the density of the samples, for example, if the density of country rock is greater than that of the host kimberlite. However, alteration of xenoliths in the kimberlite can be quite pronounced in some cases, which would tend to lower the overall bulk density of the kimberlite. A series of some 91 bulk density samples from recent core drilling in and around the Renard 2 kimberlite were line scanned prior to density determinations in an effort to establish a quantitative dilution estimate. There is minimal correlation between the sample results, suggesting that country rock dilution has a minimal effect on density.

14.6.3 Model Setup

A block model size of 5 m x 5 m x 5 m was selected as most appropriate to represent the small size of the kimberlite pipes, and as best suited to accommodate open pit mine planning, which is currently based on increments of 5 m bench heights (e.g., 10 m, 15 m). The block model is not rotated, and incorporated each of Renard 2, Renard 3, Renard 4, Renard 65 and Renard 9.

14.6.4 Estimation Process

The Ordinary Kriging (OK) interpolation method was used to estimate the country rock dilution into the block model using variography parameters defined from the geostatistical analysis and summarized in Table 14.8. Inverse distance squared (ID2) and nearest neighbour (NN) models were run to validate the kriging interpolation.

Table 14.11: Kriging Parameters

			Range				Range		
	Nugget	**x1**	**y1**	**y2**	**sill1**	**x2**	**y2**	**z2**	**sill2**
Kimb2a	250	4	4	4	245	75	75	50	295
Kimb2b	180	3	3	8	130	30	30	50	100
Kimb3b	600	1	1	10	100	10	10	41	400
Kimb3d/g	380	4	4	4	65	20	20	40	170
Kimb3f	350	8	8	8	250	40	40	40	150
Kimb3h	580	20	19	20	600	20	35	50	200
Kimb3l	580	20	19	20	600	20	35	50	200
Kimb4a	285	45	45	69	170	-	-	-	-
Kimb4b	164	25	25	25	40	45	45	80	90
Kimb4d	1114	21	21	42	192	-	-	-	-
Kimb65a	510	42	30	65	245	-	-	-	-
Kimb65b	18	25	25	25	67	35	35	64	110
Kimb65d	120	10	10	40	10	20	20	83	145
Kimb9a	760	42	42	80	250	-	-	-	-
Kimb9b	400	25	25	50	135	-	-	-	-

14.6.5 Integration Process

For each kimberlite lithology, the undiluted grade for the block was multiplied by the estimate of the dilution for the block. For each rock type, if a factor was required to account for the Kimb2c/Kimb3c/Kimb4c/Kimb65c component, this was applied. In the block model the resulting grade was multiplied by the proportion of the rock type in the block, the relevant density was applied to each rock type and the results summed to establish the total carats and tonnes in the block.

Available information for the Lynx and Hibou dykes was examined and average grades from the bulk samples assigned as appropriate grades for the bodies.

14.7 Validation

14.7.1 Dilution Validation

Proper estimation of country rock dilution is a key determinant of the Mineral Resource confidence. Amongst many issues checked, dilution is the crucial key to this estimation. In Figure 14.3 and Figure 14.4 estimates of dilution are shown against the measured dilution.

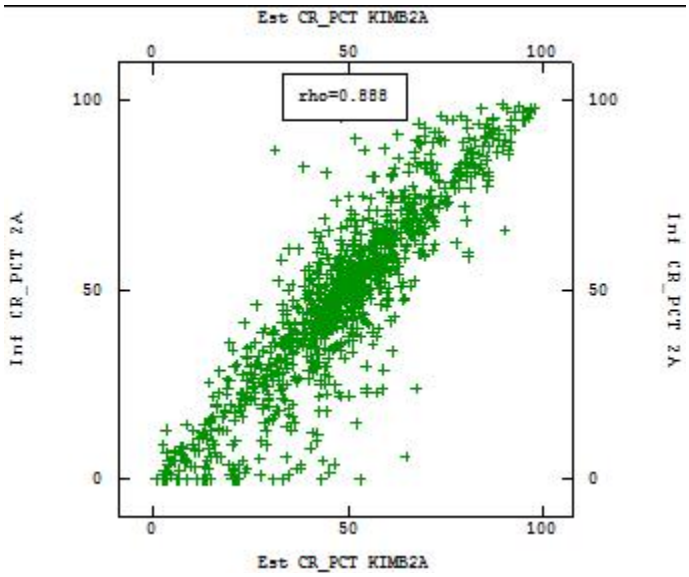

Figure 14.3: Estimated vs. Actual Dilution for Kimb2a

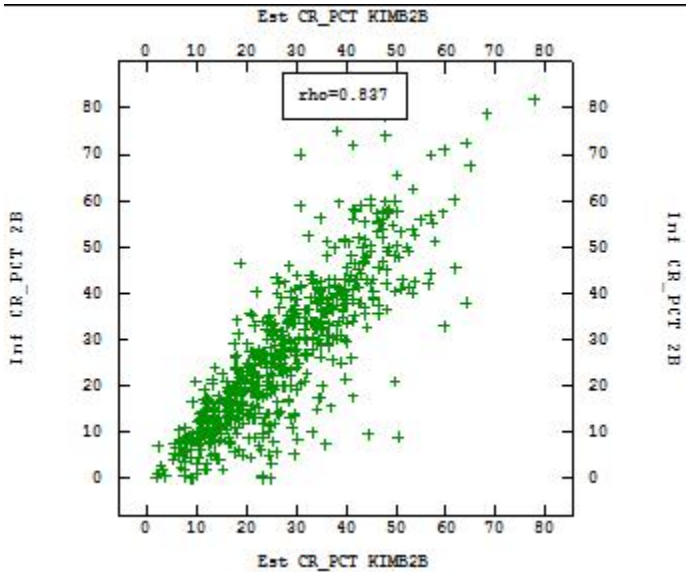

Figure 14.4: Estimated vs. Actual Dilution for Kimb2b

14.8 Diamond Price Estimates

A valuation of the Renard diamonds was performed under the supervision of WWW International Diamond Consultants Ltd. (WWW) in Antwerp Belgium between May 9th-13th, 2011. The results were announced by Stornoway on June 13, 2011, are presented in Table 14.9 and Table 14.10 and discussed below.

Table 14.12: Renard Kimberlite Pipe Diamond Valuations and Diamond Price Models[1]

Kimberlite Body	Weight of Valuation Sample (Carats)[6]	September 2009 Re-Price		May 2011 Valuation		Diamond Price Estimate for Mine Planning Purposes[2]
		Observed Average Diamond Price (US$/carat)	Diamond Price Model (US$/carat)	Observed Average Diamond Price (US$/carat)	Diamond Price Model (U$/carat)	
Renard 2	1580.06[3]	US$108	US$117 (with sensitivities of $103 to $131)	US$173	US$182 (with sensitivities of $163 to $236)	US$182
Renard 3	2,752.26[4]	US$130	US$117 (with sensitivities of $103 to $131)	US$171	US$182 (with sensitivities of $153 to $205)	
Renard 4	2,674.49[5]	US$67	US$75 (with sensitivities of $65 to $82)	US$100	US$112 (with sensitivities of $105 to $185) US$164	US$164[7]
Renard 65 Renard 9	na					US$164[8]

Notes 1 As determined by WWW Diamond Consultants International Ltd. at a +1 DTC sieve size cut-off.
2 As recommended by GeoStrat (refer to text Section 14.6).
3 Comprised 1,589.57 cts at the September 2007 valuation.
4 Comprised 2,651.17 cts at the September 2007 valuation.
5 Comprised 2,191.73 cts at the September 2007 valuation.

6 Carats submitted for 2011 valuation represent large rolled samples derived from bulk sampling, and do not include stones available from the earlier smaller scale sampling.
7 Price recommend for Renard 4 utilizes the recommended WWW Renard 4 value model price combined with Renard 2 + Renard 3 Size Frequency Distribution (SFD), refer to text in Section 14.6.
8 Recommend price utilizes the Renard 2 + Renard 3 Size Frequency Distribution (SFD) and the Renard 4 value model for a conservative Renard 9 and Renard 65 diamond price

Consistent with prior valuations, in the 2011 work WWW created a single diamond price model for the Renard 2 and Renard 3 valuation samples given the similarity of the diamonds in terms of diamond qualities and size distribution. This base case price model is US$182/carat, a 56% increase compared to the price model of US$117/carat established in September 2009. Also consistent with past practice, a separate diamond price model has been adopted for the Renard 4 valuation sample given its apparently finer distribution of diamond sizes. The Renard 4 base case price model is US$112/carat, a 49% increase compared to the previously established price model of $75/carat. Model price sensitivities are US$163 to US$236/carat for Renard 2, US$153 to US$205/carat for Renard 3 and US$105 to US$185/carat for Renard 4. Diamond price models represent the diamond price that might reasonably be expected based on commercial-scale recoveries of all diamond size classes. WWW's sensitivity limits are set such that it is highly unlikely that an actual diamond price achieved for each kimberlite body would fall below the "Minimum" sensitivity, but it is possible that the actual diamond price achieved may be higher than the "High" sensitivity number, which is not a maximum price.

As recommended by WWW, Stornoway will utilize a base case diamond price model of US$182/carat for the Renard 2 and 3 kimberlite pipes in the financial model of the Renard Feasibility Study. Prior analysis has suggested that the apparently finer size distribution of diamonds in the Renard 4 valuation sample, which accounts for that sample's lower base price model of US$112/carat, is attributable to increased diamond breakage and poor plant performance during recovery. In addition to the recent analysis as described in Farrow (2010) and Lecuyer et al. (2008) an overall large degree of similarity is observed for the Renard 2, Renard 3, and Renard 4 diamond distributions by AMEC (Oosterveld, 2008). WWW stated that Manson (2009) makes a compelling argument that the in-situ size frequency distribution ("SFD") for Renard 4 is in fact similar to that of Renard 2 and Renard 3 and the difference is due to diamond loss or breakage. If this is the case then the Renard 4 SFD model will be much coarser than the sample SFD and would increase the modelled SFD significantly. WWW therefore created a "high" model using the SFD model for Renard 2 and Renard 3 (WWW, 2011). On this basis, and taking into account alternate interpretations of diamond quality and the compelling discussion regarding size distribution, a diamond price of US$ 164/carat was adopted for Renard 4, which is slightly more conservative than the Renard 2 and Renard 3 diamond price. During the 2010 Resource Update, the Renard 2 and Renard 3 diamond price was adopted for Renard 4, as shown above in Table 14.9. Size frequency analysis of the Renard 9 and Renard 65 samples indicates an overall similarity with the the other pipes of the Renard cluster. The recommended diamond price for Renard 9 and Renard 65 utilizes the Renard 2 + Renard 3 SFD and the Renard 4 diamond value model.

Table 14.13: Lynx and Hibou Dyke Diamond Valuations and Diamond Price Models[1]

Kimberlite Body	Weight of Valuation Sample (Carats)[5]	September 2009 Re-Price		May 2011 Valuation		Diamond Price Estimate for Mine Planning Purposes[2]
		Observed Average Diamond Price (US$/carat)	Diamond Price Model (US$/carat)	Observed Average Diamond Price (US$/carat)	Diamond Price Model (U$/carat)	
Lynx	537.49[3]	US$55	US$57 (with sensitivities of $48 to $85)	US$99	US$119 (with sensitivities of $99 to $144)	US$119
Hibou	771.28[4]	US$38	US$57 (with sensitivities of $48 to $85)	US$71	US$118 (with sensitivities of $88 to $136)	

Notes 1 As determined by WWW Diamond Consultants International Ltd. at a +1DTC sieve size cut-off in May 2011.
2 As recommended by GeoStrat.
3 Parcel valued in May 2011 by WWW includes 19 carat parcel of Lynx Boulder Float.
4 Parcel valued in May 2011 by WWW.
5 Carats submitted for May 2011 valuation represent large rolled samples derived from bulk sampling.

In addition to their valuation work on the Renard 2, Renard 3 and Renard 4 diamond samples, WWW also provided an updated valuation for the Lynx and Hibou diamond samples. WWW has recommended a base case modeled price of US$119/carat be adopted for the Lynx diamond sample with sensitivities of US$99 to US$144/carat, and a base case modeled price of US$118/carat be adopted for the Hibou diamond sample with sensitivities of US$88 to US$136/carat. The proposed mine plan for Renard presently does not include a plan to mine the Lynx and Hibou dykes.

14.9 Mineral Resource Classification

The reported Mineral Resources for the Renard Project kimberlites were constrained by GeoStrat using drill density and sample distribution. Consideration was also made of the confidence with which diamond sampling results and the location of drill hole intercepts could be forecast. Classification of the resources in diamonds is an inclusive process, looking at aspects of geology, grade, revenue, density, diamond size frequency and continuity.

For the Renard kimberlite pipes, the Indicated Resource classification incorporates areas where substantial sampling has been undertaken. For Renard 2 this includes the RC, underground bulk samples and microdiamond sampling to 600 m. For the other pipes it is limited to the RC hole depths which indicate grade continuity. These depths are supported by the more extensive core drilling which is sufficient to delineate the resources at an Indicated level from the geological/lithological/volumetric/density aspect.

Sufficient carats have been recovered for revenue analysis and support that the revenue is at the Indicated level of confidence for Renard 2, Renard 3 and Renard 4.

14.9.1 Reasonable Prospects of Economic Extraction

Canadian Institute of Mining and Metallurgy (CIM) standards and Securities Commission disclosure regulations require that a mineral resource can only be declared on a mineral deposit which has "reasonable prospects of economic extraction". Work completed for the Feasibility study supports Geostrat's opinion that the mineral resource presented herein has reasonable prospects of economic extraction.

14.9.2 Uncertainty of Mineral Resources

Mineral Resources for the Renard Project documented in Section 14 are estimates and no assurance can be given that the anticipated tonnages and grades will be achieved or that the indicated level of recovery will be realized. Market fluctuations and diamond prices may render the mineral resources uneconomic. Moreover, short term operating factors relating to the diamond deposits, such as the need for orderly development of the deposits or the processing of new or different grades of diamonds, may cause any mining operation to be unprofitable in any particular accounting period. Until they are categorized as "Mineral Reserves" under NI 43-101, the known mineralization at Renard is not determined to be economic ore. The Company's ability to put the Renard Project into production will be dependent upon completion of necessary engineering studies and the receipt of all required permits. In addition, substantial expenditures are required to develop the mining and processing facilities and infrastructure at any site chosen for mining, and depend upon the Company's ability to raise those funds.

14.10 Mineral Resource Statement

The Mineral Resource Estimate for the Renard Project is summarized in Table 14.11 and Table 14.12.

The Mineral Resource Estimate takes into account geological, mining, processing and economic constraints and is classified in accordance with the 2005 CIM Definition Standards for Mineral Resources and Mineral Reserves.

The Mineral Resource Estimate is based on the continuity of geology between kimberlite at depth and kimberlite nearer surface and the generally low variation in sample results for the different kimberlite phases with depth. GeoStrat notes that there is potential for additional volume at depth for the Renard Project, as the geological models have been constructed conservatively in areas of limited drilling.

The Indicated tonnage reported in Table 14.11 and the Inferred tonnage in Table 14.12 lie within the solid model shells. Tonnage and grade estimates include estimation of internal dilution (barren material) within each body.

Table 14.14: January 2011[1] Indicated[2] Mineral Resources[3] Renard Diamond Project

Deposit	Total Tonnes	Total Carats	Average cpht[4]	Average Dilution %
Renard 2	17,632,000	18,092,000	102.6	37.4
Renard 3	1,751,000	1,849,000	105.6	36.8
Renard 4	7,246,000	3,813,000	52.6	49.1
Renard 9	0	0	0	0
Lynx	0	0	0	na
Hibou	0	0	0	na
Total	26,629,000	23,755,000	89.2	na

Notes 1 Effective Date is January 24, 2011.
2 Classified according to CIM Definition Standards.
3 The Mineral Resources have reasonable prospects of economic extraction.
4 The diamond size cut off was +1 DTC or +1.18mm square mesh screen.

Table 14.15: January 2011[1] Inferred[2] Mineral Resources[3] Renard Diamond Project

Deposit	**Total Tonnes**	**Total Carats**	**Average cpht[4]**	**Average Dilution %**
Renard 2	5,208,000	6,142,000	117.9	29.1
Renard 3	542,000	637,000	117.6	39.2
Renard 4	4,756,000	2,087,000	43.9	55.5
Renard 9	5,700,000	2,686,000	47.1	65.4
Renard 65	12,935,000	3,722,000	28.8	48.7
Lynx	1,798,000	1,924,000	107	na
Hibou	178,000	256,000	144	na
Total	31,116,000	17,454,000	56.1	na

Notes 1 Effective Date is January 24, 2011.
2 Classified according to CIM Definition Standards.
3 The Mineral Resources have reasonable prospects of economic extraction.
4 The diamond size cut off was+1DTC or +1.18mm square mesh screen.

14.11 Target for Further Exploration (formerly Potential Mineral Deposit)

In addition, Targets for Further Exploration (TFFE) were identified on the various kimberlites. [Note: prior to changes in the NI43-101 legislation that came into effect June 30, 2011, the TFFE were known as Potential Mineral Deposits (PMD)]. TFFE are derived from geological volumes based on projection at a standard 85° of the pipe margin in pipes or to within 50 m of known borehole intersections in case of the dykes. These are detailed in Table 14.13 as low and high ranges, both of which are considered to be geologically realistic. Total TFFE was identified as being between 55 and 76 million tonnes, containing between 23.5 and 48.5 million carats of diamonds, at an average grade of 43 cpht to 64 cpht. These were defined on a basis of outcrops, limited delineation drilling and surface sampling. The potential quantity and grade of any target for further exploration is conceptual in nature, there is insufficient exploration to define a mineral resource and it is uncertain if further exploration will result in the target being delineated as a mineral resource.

Table 14.16: January 2011 Target for Further Exploration[1] Renard Diamond Project

Low Range				**High Range**		
Deposit	**Total Tonnes**	**Total Carats**	**Average cpht**	**Total Tonnes**	**Total Carats**	**Average cpht**
Renard 2	4,012,000	4,132,000	103	4,586,000	8,621,000	188
Renard 3	771,000	825,000	107	1,644,000	2,762,000	168
Renard 4	11,077,000	4,209,000	38	15,300,000	12,087,000	79
Renard 9	3,858,000	1,736,000	45	6,327,000	3,163,000	50
Renard 65	29,517,000	6,789,000	23	41,591,000	13,725,000	33
Lynx	3,089,000	2,966,000	96	3,199,000	3,839,000	120
Hibou	2,745,000	2,855,000	104	2,853,000	4,309,000	151
Total[2]	55,069,000	23,512,000	43	75,500,000	48,506,000	64

Notes 1 Previously known as Potential Mineral Deposit prior to June 30, 2011 changes to NI43-101 legislation.
2 Totals may not equal the sum of the individuals due to rounding.

Substantial increases to the volumes of the Renard kimberlite bodies, due to additional drilling and subsequent modelling, are responsible for the major differences between the results of the previous Mineral Resource Estimate (Farrow, 2010) and the current estimate (Farrow and Farrow, 2011).

15.0 MINERAL RESERVE ESTIMATE

15.1 Basis of Estimate

The requirements for Mineral Resources to be converted to Mineral Reserves are that:

- Only Measured and Indicated Resources can be included;
- The resources are part of a mining plan;
- Modifying factors of dilution and mining recovery are applied; and
- The mine plan passes an economic test.

Each of these requirements was addressed in establishing the Mineral Reserves for the Renard Project.

As part of the Feasibility Study, a detailed mine plan was developed to extract the Indicated Resources of the Renard Project, dilution and recovery assumptions were applied, and, cut-off grades were calculated using preliminary costs and diamond valuations.

AMEC undertook a high level review of the Mineral Resource models and geological wireframes that had been prepared by Geostrat Consulting. The review was completed in sufficient detail to assess whether the Mineral Resource geological and block models were complete, appropriate to the selected mining methods, and suitable to support feasibility study mine planning, such that it would be reasonable for AMEC to rely on them in preparing the underground Mineral Reserve estimates. Based on these review activities, AMEC concluded that the Mineral Resource geological and block models were complete and suitable to support feasibility study mine planning and design.

Open pit and underground Mineral Reserves were estimated independently based on criteria specific to each method. A reconciliation of the Mineral Resources included in the open pit and underground mine plans was completed and confirmed that all available resources were included.

15.2 Open Pit

The reserves for the surface mine design are reported according to the Canadian Institute of Mining, Metallurgy and Petroleum's (CIM) standards. According to these standards, resource model blocks classified as measured and indicated are reported as proven and probable reserves respectively. Owing to the above reporting standards, Inferred Resources cannot be included as reserves and so have not been included in the life of mine (LOM) schedule. No resource blocks are classified as measured and therefore no part of the Mineral Reserve classifies as proven.

The total probable Mineral Reserve includes an ore dilution factor and an ore recovery factor to estimate a recoverable mining reserve.

The ore outline mining shapes consisted of expanding the modelled kimberlite solids to incorporate a 1 m dilution envelope (Figure 15.1). Specific dilution factors were estimated for each kimberlite orebody through compilations performed on a bench by bench basis. The dilution envelope is assumed not to contain diamonds and is therefore considered to be zero grade.

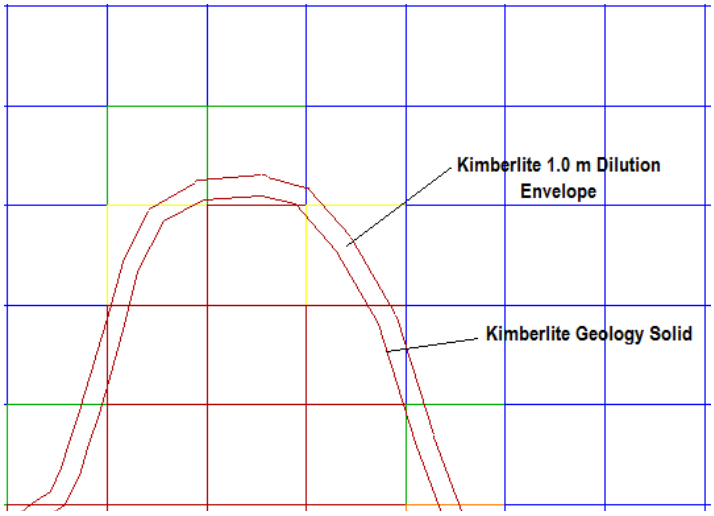

Figure 15.1: Illustration of Dilution Envelope

The dilution factors by bench per kimberlite pipe are shown in the following tables (Table15.1 and Table 15.2).

Table 15.1: R2 & R3 Dilution by Bench

R2	
Bench Level	**Bench Dilution**
508	-
498	36%
488	9%
478	8%
468	8%
458	8%
448	10%
438	8%
428	7%
418	4%
408	2%
403	1%

R3	
Bench Level	**Bench Dilution**
508	32%
498	14%
488	11%
478	11%
468	11%
458	14%
448	15%
438	8%
428	6%
418	4%
413	2%

Table 15.2: Average Dilution Factors

Dilution	R2	R3
In-situ Ore Tonnage	1,276,881	681,007
Diluted Ore Tonnage	1,367,224	752,760
Dilution Factor	7.1%	10.5%

A 96% mine recovery factor is then applied to account for ore missed at the edge of the orebody. It is also possible to lose some of the ore during transportation to the process plant due to misdirected loads or operator error.

15.3 Underground

15.3.1 Estimation Procedure

The first step in the estimation of the Mineral Reserves was to create 3-D shapes for each of the stopes based on the Mineral Resource model and the planned mining method. The criteria used was to create practical shapes that were compatible with the planned development of the drawpoint levels and drill drifts, the drill patterns available, the type of drilling equipment selected, and likely post blast outlines. The outlines of the pipes in some areas were irregular and in these areas practical (smoothed) mining outlines were created, which typically encapsulated some waste rock and occasionally excluded some kimberlite.

The stope outlines were created by cutting a series of resource plans every 10 m vertically and then planning and digitizing a stope outline for each level. The individual stope outlines created over the full height of each pipe were created in Vulcan software and a 3-D solid was created. Stope solids were subdivided into stopes by mining zone and then further subdivided into undercuts and main stope body. The main stope body was then further subdivided to coincide with potential sequencing of blasting and production. A number of remnant solids were also created for kimberlite left in the walls of the open pit. As part of this process, the block model grades within the pipes were examined and, while there was some variation in grade across the levels, in AMEC's opinion there wasn't any real opportunity, given that BHS is a bulk mining method, to eliminate local areas of lower grade kimberlite from the stope shapes.

As part of this process, the block model grades within the pipes were examined and, while there was some variation in grade across the levels, in AMEC's opinion there wasn't any real opportunity, given that BHS is a bulk mining method, to eliminate local areas of lower grade kimberlite from the stope shapes.

The tonnage and grade of all the Mineral Resources contained within each stope shape was calculated using the resource block model and Vulcan software. The stopes included in the underground Mineral Reserves are shown in Figure 15.2.

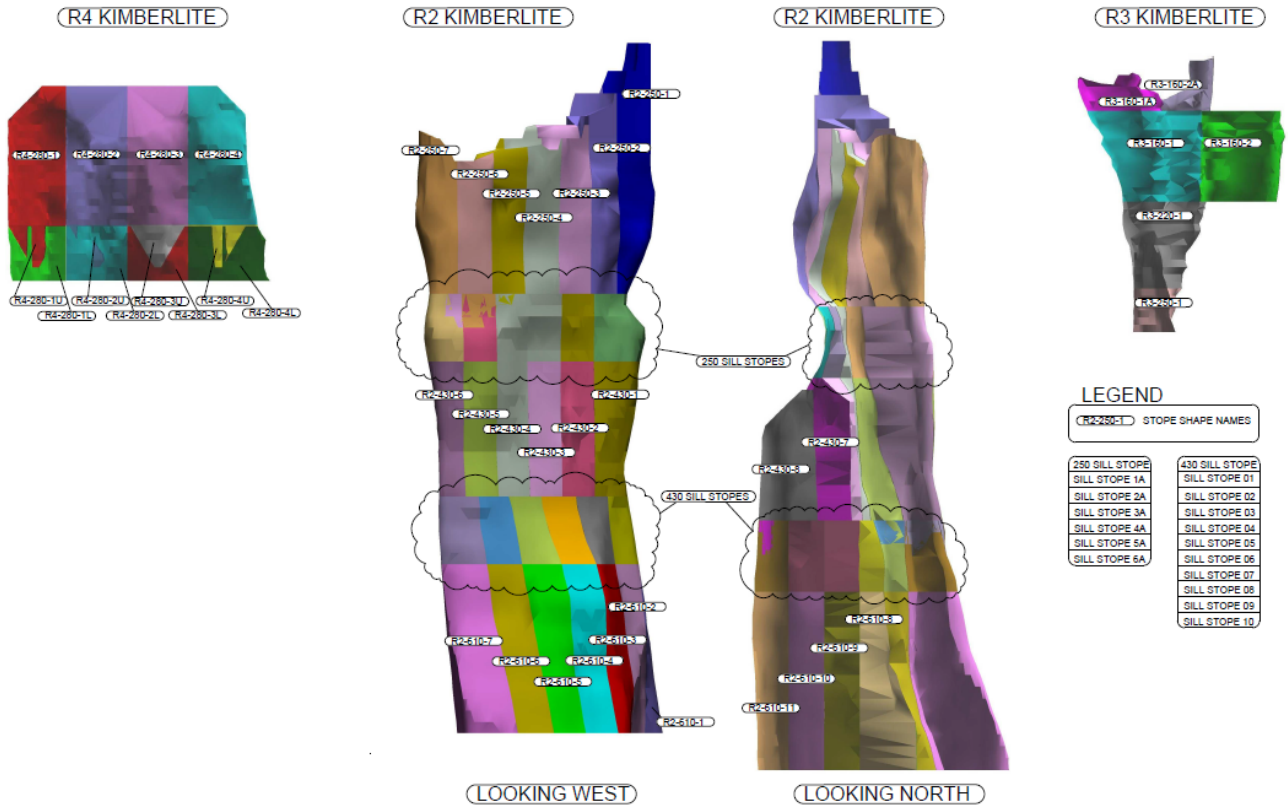

Figure 15.2: Stopes Included in the Underground Mineral Reserves

15.3.2 Modifying Factors

As discussed in Section 15.3.1 the stope shapes include any waste contained within them as internal dilution. The modifying factors of mining recovery and external dilution have been applied to the contained Mineral Resource and waste in the stope shapes.

The assumptions of recovery and dilution for BHS mining used for this study are listed in Table 15.3:

Table 15.3: Assumptions of Recovery and Dilution of BHS Mining

Modifying Factor	Pipe		
	R2	R3	R4
Mining Dilution	20%	14%	14%
Mining Recovery	82%	85%	78%

The grade of all dilution has been assumed to be zero.

Considerations for recovery and dilution estimate assumptions for BHS mining are:

- With BHS, the first 35% of the production is swell, which will have very low dilution. When the stope is full of blasted ore the first 50% to 60% of the drawdown will also have very low dilution and high recovery. When backfill waste starts mixing with the ore as it is drawn down, dilution for each tonne drawn will start increasing significantly. The build-up of the dilution and recovery assumptions are shown in Table 15.4.

Table 15.4: Assumptions for Dilution and Recovery of BHS Mining for R2 pipe

Drawdown Stage in Stope	Portion of Total Ore in Stope	Recovery Estimates	Dilution Estimates
Shrinkage Blasting	35%	98%	5%
Middle Drawdown	32% (50% of ore remaining after blasting)	85%	12%
Final Drawdown	31% (47% of ore remaining after blasting)	71%	40%
Final Clean-up	2% (3% of ore remaining after blasting)	40%	105%
Subtotal	**100%**	**84%**	**20%**
Mining losses[1]		2.0%	-
Total		**82%**	**20%**

Note 1: mining losses due to (a) irregular geometry of the walls, (b) miss-holes and blasting problems, (c) ore stranded above arches in bottom drawpoint level, (d) failure of brows in drawpoints.

- Placement of drawpoints and planning of drawdown is critical to achieve mass flow and minimize “rat-holing” of the blasted ore in the stope during drawdown.
- It is known that the CCR and CRB units, which surround the pipes in many locations, contain diamonds however the grade is unknown, consequently no grade has been

assumed for the dilution. Further, much of the dilution will come from the open pit waste rock to be placed on top of the blasted kimberlite, and this will contain no grade.

At the very bottom level (610L for Renard 2 and 280L for Renard 4), the kimberlite remaining on the underside of the drawpoint troughs will not be recovered as part of BHS mining. To recover this ore, sublevel retreat mining (SLR) will be used at the very end of the mine life. Upholes will be drilled from all the drawpoints and scram drifts located below the troughs and cones and then the holes will be blasted and the ore mucked in a retreat fashion. The mining recovery for this final stage of mining has been assumed to be 50%.

The dilution and recovery factors as listed in Table15.4 were applied on a stope-by-stope basis to calculate the reserve for each stope. A final step was to review the mineability and the calculated Mineral Reserve tonnes and grade for each stope. Three stope blocks totalling 315,000 tonnes, located just outside the open pit stope walls, were initially included in the Mineral Reserve but upon review were excluded since they were judged too problematic to mine.

The grade of all the stopes was compared to the cut-off grade (33.0 cpht for Renard 2/Renard 3 and 36.6 cpht for Renard 4) and all stopes were found to exceed this cut-off.

15.4 Mineral Reserves

15.4.1 Open Pit Mineral Reserve Statement

The total open pit probable Mineral Reserve is estimated at 2 million tonnes of ore with an average diluted grade of 0.94 cpt for 1.91 million carats. The Renard 2 pipe has the highest average diamond grade at 0.95 cpt followed closely by Renard 3 at 0.93 cpt. The open pit Mineral Reserves are summarized by bench in Table 15.5 and by kimberlite pipe in Table 15.6.

Table 15.5: R2/R3 Open Pit Mineral Reserves by Bench

Bench Level	Ore Tonnes (x 1000)	Carats (x 1000)	Avg. grade (cpht)	OVB Tonnes (x 1000)	Waste Rock Tonnes (x 1000)	Strip Ratio (OVB + WST)/ ORE	Total Tonnes (x 1000)
518	-	-	-	25	0.9	-	26
508	1.3	0.7	53,8	890	172	827.01	1,063
498	37	26	70,3	1,030	875	52.02	1,942
488	154	141	91,6	314	1,511	11.84	1,978
478	266	252	94,7	17	1,479	5.63	1,762
468	271	255	94,1	-	1,387	5.11	1,658
458	259	241	93,1	-	895	3.45	1,155
448	260	244	93,8	-	708	2.73	968
438	246	243	98,8	-	400	1.63	645
428	229	224	97,8	-	274	1.19	503
418	177	168	94,9	-	100	0.56	276
408	115	102	88,7	-	50	0.43	165
398	20	16	80,0	-	6	0.28	26
Total	2,035	1,913	94.0	2,277	7,857	4.98	12,168

Table 15.6: Open Pit Mineral Reserves by Category by Kimberlite Pipe

Pipe	Classification	Tonnes (x 1000)	Grade (cpht)	Carats (x 1000)
R2	Probable	1,312	94.6	1,242
R3	Probable	723	92.8	671
Total	**Probable**	**2,035**	**94.0**	**1,913**

15.4.2 Underground Mineral Reserve Statement

The underground Mineral Reserves are summarized by pipe and mining zone in Table 15.7. All Mineral Reserves are in the "probable" category. Of the three pipes in the Mineral Reserve, Renard 2 contains 78% of the tonnes and 85% of the carats, Renard 4 contains 17% of the tonnes and 10% of the carats and Renard 3 contains 5% of the tonnes and carats.

Table 15.7: Underground Mineral Reserve Statement

Zone/Level	Mining Parameters			Proven Reserve			Probable Reserve		
	Internal Dilution (%)	Mining Recovery (%)	External Dilution (%)	Tonnes (t x 1000)	Grade (cpht)	Carats (x 1000)	Tonnes (t x 1000)	Grade (cpht)	Carats (x 1000)
R2 Pipe									
R2-250	24.7%	82%	20%	-	-	-	3,265	64.1	2,093
R2-430	7.7%	82%	20%	-	-	-	5,075	73.0	3,703
R2-610	8.8%	82%	20%	-	-	-	8,689	90.0	7,820
Subtotal R2	**11.5%**	**82%**	**20%**	**-**	**-**	**-**	**17,029**	**80.0**	**13,616**
R3 Pipe									
R3-250	20.5%	85%	14%	-	-	-	112	83.0	93
R3-220	20.7%	85%	14%	-	-	-	294	89.8	264
R3-160	21.4%	85%	14%	-	-	-	596	81.4	486
Subtotal R3	**21.1%**	**85%**	**14%**	-	-	-	**1,003**	**84.1**	**843**
R4 Pipe									
R4-280	**1.4%**	**78%**	**14%**	-	-	-	**3,718**	**42.4**	**1,575**
Totals	**9.9%**	**81%**	**19%**	**-**	**-**	**-**	**21,750**	**73.7**	**16,034**

15.4.3 Consolidated Mineral Reserve Statement

A consolidated summary of the Mineral Reserves for the Renard Project including both underground and open pit mining is presented in Table 15.8. The table includes a breakdown of the Mineral Reserves by pipe and mining method as well as totals by pipe and mining method. In summary, 91% of the diamonds are mined by underground methods and overall 83% of the diamonds are contained in the Renard 2 kimberlite pipe.

Table 15.8: Mineral Reserves Summary – Open Pit and Underground

Mine	Probable Reserve (t x 1000)	Grade (cpht)	Carats (x 1000)	% of Total Tonnes	% of Total Carats
Renard 2 Open Pit	1,313	94.6	1,242	5.5%	6.9%
Renard 3 Open Pit	723	92.8	671	3.0%	3.7%
Renard 2 Underground	17,029	80.0	13,616	71.6%	75.9%
Renard 3 Underground	1,003	84.1	843	4.2%	4.7%
Renard 4 Underground	3,718	42.4	1,575	15.6%	8.8%
Total	**23,785**	**75.5**	**17,947**	**100.0%**	**100.0%**
Subtotals by Pipe and Mining Method					
Renard 2 Pipe	18,342	81.0	14,858	77.1%	82.8%
Renard 3 Pipe	1,725	87.7	1,514	7.3%	8.4%
Renard 4 Pipe	3,718	42.4	1,575	15.6%	8.8%
Open Pit	2,035	94.0	1,913	8.6%	10.7%
Underground	21,750	73.7	16,034	91.4%	89.3%
Mine	**Probable Reserve (t x 1000)**	**Grade (cpht)**	**Carats (x 1000)**	**% of Total Tonnes**	**% of Total Carats**
Renard 2 Open Pit	1,313	94.6	1,242	5.5%	6.9%
Renard 3 Open Pit	723	92.8	671	3.0%	3.7%
Renard 2 Underground	16,17,029	80.0	13,616	70.6%	75.9%
Renard 3 Underground	1,003	84.1	843	4.2%	4.7%
Renard 4 Underground	3,718	42.4	1,575	15.6%	8.8%
Total	**23,785**	**78.75.5**	**17,947**	**100.0%**	**100.0%**
Subtotals by Pipe and Mining Method					
Renard 2 Pipe	17,18,342	84.81.0	14,858	76.77.1%	82.8%
Renard 3 Pipe	1,725	87.7	1,514	7.3%	8.4%
Renard 4 Pipe	3,718	42.4	1,575	16.15.6%	8.8%
Open Pit	2,035	94.0	1,913	8.6%	10.7%
Underground	21,750	76.73.7	16,034	91.4%	89.3%

Note: All Mineral Reserves are in the Probable Category

15.4.4 Reconciliation of Mineral Resources and Reserves

A reconciliation of Mineral Resources and Mineral Reserves is shown in Table 15.9.

Table 15.9: Reconciliation of Mineral Resources and Mineral Reserves

Mineral Resources1,2	Indicated (t x 1000)	Grade (cpht)	Carats (x 1000)	% of Total Tonnes	% of Total Carats
Renard 2	17,632	102.6	18,092		
Renard 3	1,751	105.6	1,849		
Renard 4	7.246	52.6	3,813		
Total	**26,629**	**89.2**	**23,755**	**100.0%**	**100.0%**
Resources in Mineral Reserves (probable)					
Renard 2	17,357	102.7	17,818		
Renard 3	1,535	110.1	1,690		
Renard 4	4,112	49.0	2,015		
Sub Total	**17,516.1**	**93.6**	**21,523**	**86%**	**91%**
Excluded Resources					
Stopes & Crown Pillar	3,346	59.6	1,995	14%	9%
Outside Stope Boundaries	278	85.5	237	1%	1%
Sub Total	3,624	61.6	2,232	13%	8%
Total	**26,629**	**89.2**	**23,755**	**100%**	**100%**
Mineral Resources1,2	**Indicated (t x 1000)**	**Grade (cpht)**	**Carats (x 1000)**	**% of Total Tonnes**	**% of Total Carats**
Renard 2	17,632	102.6	18,092		
Renard 3	1,751	105.6	1,849		
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Total	**26,629**	**89.2**	**23,755**	**100.0%**	**100.0%**
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Renard 4	4,112	49.0	2,015		
Sub Total	**17,516.1**	**93.6**	**21,523**	**86%**	**91%**
Excluded Resources					
Stopes & Crown Pillar	3,346	59.6	1,995	14%	9%
Outside Stope Boundaries	278	85.5	237	1%	1%
Sub Total	3,624	61.6	2,232	13%	8%
Total	**26,629**	**89.2**	**23,755**	**100%**	**100%**
Mineral Resources[1,2]	**Indicated (t x 1000)**	**Grade (cpht)**	**Carats (x 1000)**	**% of Total Tonnes**	**% of Total Carats**
Renard 2	17,632	102.6	18,092		
Renard 3	1,751	105.6	1,849		
Renard 4	7.246	52.6	3,813		
Total	**26,629**	**89.2**	**23,755**	**100.0%**	**100.0%**

Resources in Mineral Reserves (probable)					
Renard 2	17,357	102.7	17,818		
Renard 3	1,535	110.1	1,690		
Renard 4	4,112	49.0	2,015		
Sub Total	**17,516.1**	**93.6**	**21,523**	**86%**	**90%**
Excluded Resources					
Stopes & Crown Pillar	3,335	61.1	2,039	13%	9%
Outside Stope Boundaries	307	74.3	228	1%	1%
Sub Total	**3,624**	**62.2**	**2,232**	**14%**	**10%**
Total	**26,629**	**89.2**	**23,755**	**100%**	**100%**

Note: 1 Renard Mineral Resource Update - February 3, 2011
2 Includes both Open Pit and Underground Mining

15.5 Risk to Mineral Reserves

15.5.1 Open Pit Risks

The risk to the open pit Mineral Reserves is minimal in that the open pits are small, relatively shallow and represent only 11% of the overall reserve. Mining dilution remains a risk since ore/waste separation will rely on visual placement of dig limits by experienced geologists.

15.5.2 Underground Risks

The main risks to the underground Mineral Reserves are higher dilution and lower recovery than planned. Ore drawdown management during production is critical to limiting dilution and maintaining high recovery. A key aspect of managing drawdown is maintaining an even drawdown over the entire blasted ore column in the stope to promote mass flow of the ore. One of the risks resulting from poor draw management is the development of rat-holes above individual drawpoints, ahead of their neighbours, right up to the waste backfill placed on top of the blasted ore. A good drawdown plan in combination with strict draw procedures and regular monitoring is required for success. Identifiable markers placed in the blasted ore, either via placement in blastholes or on drill levels will provide good data on the effectiveness of the drawdown system.

16.0 MINING METHODS

A mine plan has been developed to exploit the Indicated Resources in the Renard 2, Renard 3 and Renard 4 kimberlite pipes. The mine plan for Renard 2 and Renard 3 combines open pit and underground mining methods with the pit depth determined through a trade-off study between open pit and underground mining costs. The Renard 4 mine plan employs underground methods only. Since the Renard 4 kimberlite is located below Lake Lagopède, a 100 m thick crown pillar will be maintained throughout the life of mine.

Mine operations will commence at Renard 2 and Renard 3 with open pit mining and preparation of the underground mine development and infrastructure. Following completion of the open pit at Renard 2 and Renard 3, the underground mine will be extended to surface through the pit floor. Waste rock from the open pit and underground development will be placed into the Renard 2 and Renard 3 underground workings via the open pit to provide support for the stope walls during mining. Waste material to support the Renard 4 underground stopes will be backfilled through backfill raises from surface.

The mine design was developed by AMEC (underground) and G Mining Services Inc. (open pit). The overburden slope assessment was performed by Golder Associates and the rock mechanics and hydrogeology elements of the mine design were supplied by Itasca Consulting Canada Inc.

16.1 Overburden Pit Slope Assessment

This section summarizes the overburden slope assessment used to provide guidelines on the overall slope design for the open pits. The design of the open pits is discussed in Section 16.4.

The stability analysis of the overburden slopes for the open pit portions of Pit R2/R3 was performed. The geotechnical properties and parameters for the overburden used in the analyses were based on the geotechnical investigations prior to 2012 discussed in Section 18.1. The overburden thickness over Pit R2/R3 was determined with a model developed using the exploration drilling logs. The overburden thickness ranges from 0 m to 27 m with an average of about 10 m.

A stability analysis for Pit Renard 65 was not performed. Similar soil conditions are expected; however, soil thickness is expected to be less. The overburden slope configuration discussed below is considered to be applicable to Renard 65; however, a geotechnical inspection of the site-specific conditions is required during excavation to confirm the applicability of the overburden slope configuration.

The analysis did not include the section of the pit perimeter within the small lake as information on the soil conditions in the lake is unavailable. A two-dimensional stability assessment was performed. Selected cross-sections of differing overburden thicknesses were analysed. The position of the groundwater table was varied to assess its effect on the stability of the slopes. The groundwater level used in the assessment varied between El. 504 m and El. 514 m. The range of groundwater levels is based on results of site-specific monitoring.

The stability of the slopes is affected by the overburden conditions (thickness, stratigraphy, strength, etc.) and the position of the groundwater table. The typical overburden slope

configuration is shown on Figure 16.1. The maximum slope angle is 2 horizontal to 1 vertical (2H:1V). Slope benches are limited to a maximum height of 10 m. The offset from the overburden slope crest in which no equipment or infrastructure is allowed varies based on the total overburden thickness. The pit rim offset (i.e. distance from the toe of the overburden slope to the crest of the bedrock pit slope) also varies based on the overburden thickness. A bench width of 5 m is to be provided. A 3 m thick (perpendicular to the slope face) layer of waste rock or free-draining granular material is to be placed on the overburden slope face to provide additional stability against erosion and seepage from the overburden. A surface water diversion ditch will be excavated along the crest of the overburden slope. A surface water diversion ditch is to be excavated in the bedrock near the crest of the bedrock slope to divert and collect water away from the open pit. Piezometric monitoring of the overburden slopes during excavation and throughout the life-of-mine is recommended. This monitoring will allow the determination of the phreatic surface to enable confirmation and assessment of the stability of the overburden slopes.

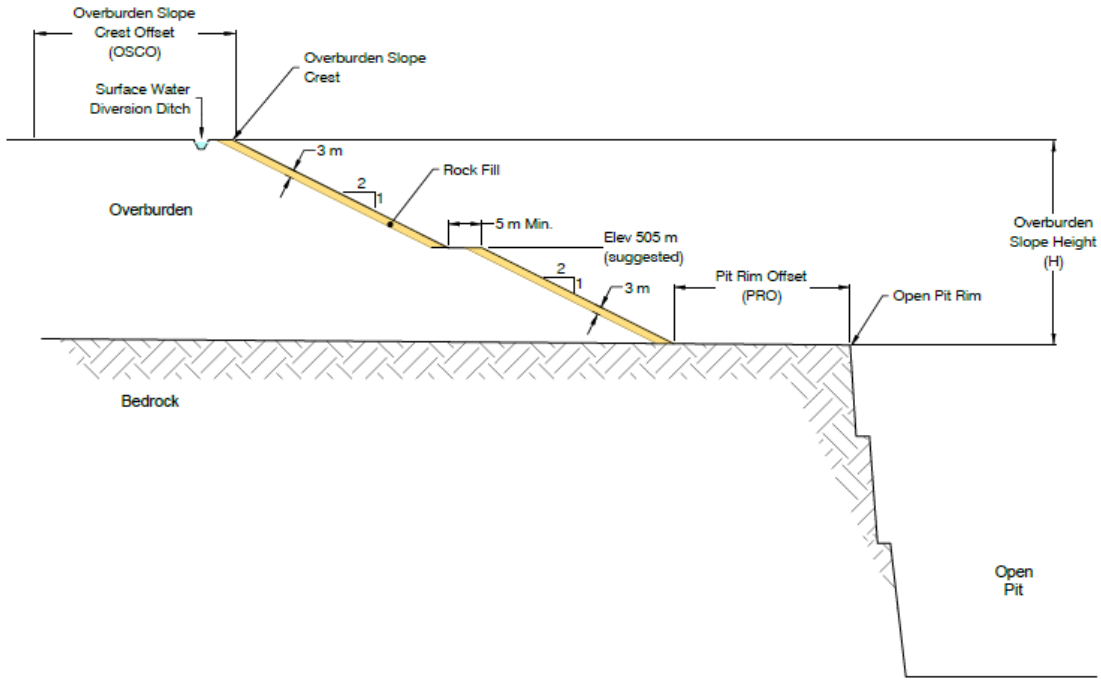


Figure 16.1: Typical Overburden Slope Configuration

16.2 Rock Mechanics

This chapter has been prepared by Richard K. Brummer, PhD., P.Eng. based on work completed by engineers and geologists at Itasca Consulting Canada Inc. (Geomechanics) and Itasca Denver (Hydrogeology). Rock mechanics laboratory testing was completed by the Rock Mechanics Laboratory at the University of British Columbia and Université Laval's Rock Mechanic Laboratory, while Stornoway personnel were responsible for all on-site characterization, borehole logging, etc. Detailed individual reports have been completed on

each of the aspects described below. A complete list of relevant reports prepared by Itasca has been included in the References list (Section 27 - References).

16.2.1 Site Investigations

Various field investigations for the feasibility level study have been conducted (by others) since 2005. These included on site geological mapping (both at surface and underground), and several campaigns of geomechanical data collection, including collecting joint orientation data from oriented drill holes.

During the 2009 drilling program a total of 24 geomechanical/exploration boreholes were drilled between June and September for a total of 12,717 m. Of the 24 drill holes, 13 were oriented for further geomechanical investigation. During the 2010 field program a total of 36 geomechanical/exploration boreholes were drilled between February and November for a total of 7367 m. All 36 boreholes were oriented for further geomechanical investigation. A select group of drill holes from past drill programs (prior to 2009) were chosen for supplementary geomechanical study and were updated to the same set of core logging standards as in recent drill programs. The updated drill holes were then added to the geomechanical database. As of August 22, 2012, 25,525 m of core has been logged using Laubscher's Rock Mass Rating (RMR_{L90} - Laubscher, 1990) and 18,297 m for Barton's Q system (Barton, 1974).

All recovered core was photographed and logged.

16.2.2 Laboratory Testing and Geomechanical Database

The descriptions, terms and standards used by Stornoway to construct the laboratory testing and geomechanical database follow the terminology recommended by the International Society for Rock Mechanics (ISRM) and in particular the Rock Characterization and Testing: ISRM Suggested Methods (1981).

The geomechanical field assessment identified the following geomechanical units for the purposes of mine design:

- ***Blue Massive Volcaniclastic Kimberlite (Kim2A)*** is extensively altered and generally massive. This blue to blue–green rock is comprised of olivine, juvenile clasts and country rock xenoliths that are poorly sorted, typically loosely packed and less commonly clast supported and set within a highly altered interclast matrix. Country rock xenoliths are comprised of locally derived granite and gneiss that are fresh to moderately altered.
- ***The Brown Coherent Kimberlite (Kim2B)*** is a moderately altered, massive and texturally variable unit that displays both coherent and less common volcaniclastic (i.e. Tuffitic Kimberlite Breccia) textures. Macroscopically, the Brown CK is pale brown to black and is comprised of two generations of olivine set within a crystalline groundmass with common country rock xenoliths that typically show significant reaction to the host kimberlite. Country rock xenoliths include both granitoid and gneiss, although the latter predominates.
- ***Coherent Kimberlite (Kim2C)*** occurs in the form of late stage dykes and irregular intrusions and both are classified as Hypabyssal Kimberlite (HK). HK dykes are found

throughout the body and typically occur along contacts between types of main pipe infill, along the pipe walls and throughout the marginal breccia. Dykes range in thickness from a few centimetres to 1m and contacts are often characterized by flow alignment of elongated minerals and carbonate veining. The HK is dark green to black and is comprised of two generations of olivine (macrocrysts and phenocrysts) that are evenly distributed and set within a homogeneous, crystalline groundmass.

- ***Country Rock Breccia (CRB)*** can encompass both brecciated gneiss or brecciated granite or pegmatite. All types are present with the CRB of the Renard deposit. Granitic-type is most common. It is described as rock that consists of more than 95% country rock fragments with < 5 % to no kimberlite component.
- ***Cracked Country Rock (CCR)*** is fractured country rock with varying degrees of fracturing. The breakage of this country rock can be related to kimberlite emplacement – whether as emplacement of the kimberlites or of a peripheral HK dyke.
- ***Gneiss (GN)*** consisting of fresh, competent, fine grained, banded, quartz-feldspar-biotite gneiss with less common coarse grained sections with less banding.
- ***Granite (GR)*** consisting of fresh, competent, equigranular granite; primary mineralogy includes: quartz, feldspar and biotite.
- ***Pegmatite (PEG)*** consisting of fresh, competent, equigranular pegmatite; primary mineralogy includes: quartz, feldspar and biotite.

Samples from selected boreholes and rock types were collected for laboratory testing. All samples were tested at the UBC Rock Mechanics Laboratory in Vancouver and Université Laval's Rock Mechanic Laboratory. Laboratory testing included triaxial, uniaxial compression (including modulus of elasticity) and Brazilian indirect tension tests on intact rock samples. The laboratory tests were complemented by direct shear tests on rock fractures from core. Slake durability tests were also undertaken for the kimberlites.

16.2.3 In Situ Stress

During the early stages of design it is necessary to have reasonable estimates of in situ stress in order to provide the required input to numerical stress analysis models. Stresses were selected for the Renard Project based on information available from the World Stress Map. The current design considers a conservative range of stresses, within the range for comparable mines to Renard in the Canadian Shield.

16.2.4 Caveability Considerations

Caving methods were initially evaluated to determine whether they could be employed for Renard. Prior to the analysis, Itasca reviewed the provided rock mass classification results. Based on the empirical and laboratory rock data it was decided that all kimberlite units could be treated as equivalent in their geomechanical properties. Furthermore, there was no major variation in rock quality with depth within the kimberlites. The results of the slake durability tests justified the assumption that only slight weathering was to be expected in the kimberlites. The Mine Rock Mass Rating (MRMR) empirical approach (Laubscher, 2000) was employed to determine whether the kimberlite domains are amenable to caving. Results from the various analyses concluded that in all cases considered for Renard, natural progressive caving was

unlikely. It was concluded that the Renard orebodies will require some form of blast-assisted method.

16.2.5 Fault Analysis

Investigations into the presence of faults have been ongoing since 2010 thru to present day. The fault analysis work included a review of prior work undertaken at Renard (by others), orthophoto lineament analysis in July 2010, and groundtruthing and structural mapping in October 4-13, 2010. A first round of Datamine modelling of faults (as wireframe surfaces incorporating drill hole fault intercepts) was carried out in November 2010. Then modeling of faults was revised in May 2011 based on validations carried out by Stornoway. The most recent fault modelling work, conducted in 2012 and updated with all currently available information,, included a detailed review of the previous model.

Presently, eight fault segments have been identified and modelled from drill core logging and geophysical information. All are dominated by brittle deformation. Modelled strike lengths range from 76 to 440m, dip lengths are 116 to 736m, and modelled thicknesses range from 0.3-18.4m, based on the true thicknesses of the drill hole fault intercepts used to construct the fault wireframe solids. Two weak zones have been newly identified, and have been modelled based on 5-90m thick intervals of broken to rubbly material. The zones are shallow and have possibly been subject to near surface weathering and potentially influenced by forceful kimberlite emplacement. The focus for fault analysis has been on R3/R3 pipes given they will be mined first. Fault analyses will be ongoing as more data becomes available from the site.

16.2.6 Pit Slope Assessment

A series of open pit bedrock slope analyses for the Renard 2 and Renard 3 pits were carried out in 2010. Following a series of iterative analyses the slope stability analysis was revised (in 2011) to take into consideration the updated and optimised final layout, geology and wall orientations of the pits.New information with respect to orientation data and the newly discovered weak zones (described in the previous section) will be considered for the pit-slope design task in the detailed engineering phase.

16.2.7 Subsidence Assessment

A subsidence assessment was carried out at Renard with the main goal of validating the proposed placement of surface infrastructure. Any critical life of mine or surface infrastructure should be located outside of the influence of mining-induced subsidence.

Subsidence limits were initially established using an empirical approach. A break angle of 85º was determined for the limit of discontinuous subsidence and a limit angle of 80º was assumed as the limit of continuous subsidence. Detailed numerical analyses were completed in 2012 using *FLAC*^{3D}, which incorporated a more rigorous sensitivity analysis while considering a number of parameters, e.g. rock mass strength, influence of faulting, effect of backfill etc. The empircal limits were confirmed with the numerical models. Furthermore, analogues to similar Kimberlite deposits were investigated. Based on the empirical and numerical analyses, as well as the analogue deposits, the subsidence limits that have been determined at this stage of

study appear to be adequate. As more information becomes available, the analyses should be revisited to ensure these limits will still be valid, particularly at the start of the detailed engineering and execution phases.

16.2.8 Panel Retreating Sequence

The approach to mining at Renard is a hybrid method with two major phases: 1) a drilling and blasting phase, and; 2) an ore extraction phase. The drilling/blasting phase will be very similar to operations commonly undertaken in blasthole stoping mines. The ore extraction phase should be thought of in the same context as a caving style operation (minus the natural caving component), where draw control strategies are considered paramount for ensuring optimal dilution control to maximise recovery. In order to safely implement this method, a panel retreating sequence has been recommended whereby a mining horizon is subdivided into smaller panels, and mined in a retreating, pillarless, fashion.

Only swell tonnes are to be pulled during the drilling/blasting process inside an active panel, in order to promote panel/wall stability. Blasted waste material is planned to be placed at the bottom of the pit on top of the blasted ore. The rationale behind this measure is based on the fact that broken rock (ore or waste):

- will always provide some level of confinement to the pipe walls contributing to all aspects of stability for the mine;
- will minimize the surface disturbance footprint due to subsidence;
- will improve overall wall stability and minimize the risk of air blast which may possibly be caused by large falling blocks or hung up material which suddenly falls.

16.2.9 Panel Dimensioning

Individual panels have been dimensioned with empirical stability methods, using the geomechanical logging data. It was found that the panel dimension (footprint) must not exceed a maximum hydraulic radius of 12m. Detailed 3D numerical modelling, incorporating the geometrical complexities and sequencing detail is planned for the detailed engineering phase. Also, it has been recommended to implement a monitoring strategy which will allow early detection of any movements occurring in the temporary sills while working in active areas and a means to monitor air gaps (beneath temporary sills) to help keep these at a minimum.

16.2.10 Draw Level Design Guidelines

As part of ongoing optimisation studies conducted in late 2011 and in early 2012, the draw level design was updated to help improve recovery. A conventional style caving draw level layout was proposed. This layout would lend itself to the panel retreat concept, meet operational and production requirements, and provide optimal recovery while minimising waste entry.

In order to ensure the layout would provide adequate recoveries, kinematic flowmodelling was conducted using Itasca REBOP software. A parametric study was initiated which considered two widely accepted variations on the Offset Herringbone layout, while testing sensitivity to

various parameters, e.g. orebody width, mean fragment diameter etc.. The two layouts only varied slightly, and it was decided to adopt a 15m x 30m (drawpoint spacing x haulage drift spacing) Offset Herringbone layout.

16.2.11Fragmentation Considerations

Fragmentation is one of the key controlling parameters in terms of recovery and dilution for the planned mining method at Renard. Because of the importance of fragmentation as a design parameter, empirical analyses were undertaken using the Kuz-Ram blast fragmentation model to help anticipate the fragmentation of the kimberlites resulting from the current blast design.

Based on successful R65 bulk sample trials, the Kuz-Ram model was used to anticipate the range of fragmentation that might be achieved during production blasting, with specific emphasis on R2. Sensitivity analyses were undertaken with consideration given to explosive strength, drill hole spacing, and to some degree, rock mass character. The results from this work were encouraging, and demonstrated that a fair amount of fragmentation control would be possible through blast design. The results of these analyses were used as a key input parameter for the global dilution and recovery analyses.

16.2.12Global Recovery and Dilution Estimate for R2

Itasca's kinematic flow simulation software REBOP was used to simulate the recovery of the R2 kimberlite pipe. The REBOP simulations provided reasonable recovery values ranging from ~81% to ~87%. These simulations incorporated a number of the previously mentioned design elements giving due consideration to the anticipated fragmentation ranges for Renard, however, the values used must be verified once more fragmentation data becomes available from the site.

16.2.13Ground Support Standards

The design of ground support for Renard was mostly based on the empirical guidelines published by Barton (2002) and Grimstad and Barton (1994). These guidelines are based on rock mass quality, size of the opening, and intended use of the excavation. The prescribed Ground support specifications were based on standards currently adopted in Canadian hard rock mines and are in compliance with good engineering practice. There are plans to revisit the support design during detailed engineering (and in the future) to ensure the current design's validity and/or to modify it as needed.

16.2.14Crown Pillars

The Renard 4 kimberlite pipe is located below a lake. Consequently it is important to ensure the stability of the Renard 4 crown pillar since an uncontrolled failure could lead to water inflow into the mine during production. This would represent a significant risk to the safety of personnel working underground.

Empirical studies have been undertaken using the approach proposed by Carter (1992). This work suggested that a very thick crown pillar would be required for stability unless the spans

could be reduced. Because of the importance of the stability of the crown pillars, a 3D global stress numerical modelling was undertaken, whereby Stornoway requested that the stability of a 100m thick crown pillar be evaluated.

The modelling that was conducted for the crown pillar in the Renard 4 pipe indicates that under base case assumptions the crown pillar is stable. It was further shown that within the range of reasonable assumption with regards to material properties or stress, there are combinations in which the crown pillar is unstable. The greatest sensitivity in the modelling performed to date is the presence of a horizontal stress at surface.

The models suggest that if a flat-back profile were to be used, the depth of failure into the crown pillar will be between 25% and 35% of the span of the orebody. This coincides well with the general rule of thumb in which arching is assumed to occur at 1/3 of span. The back of the stope should, within reasonable bounds, be formed to create such an arch with the blasted profile to limit the ability of the rockmass to define its own stable arch (which can be quite different depending on local jointing). The peak of the arch should not be above the 100m limit defined by the crown pillar.

16.2.15Limitations

For any mining project, there are always unknown rock mechanics and hydrogeological conditions that cannot be predicted ahead of actual mining. These unknown conditions, such as faulting, zones of weak rock, or zones of unanticipated water inflow, may only be discovered during mining and may require significant changes to the mining plan. While additional laboratory testing and characterisation work should always be ongoing at every stage of project development to reduce uncertainty, it is never possible to carry out enough drilling/characterisation work ahead of time to identify all of these potential risks. This may in turn have significant effects on cost, as relates for example to slope angles, dewatering and/or underground development, excavation size, etc.

It must be understood that there are limitations in the ability of numerical model results to assess all potential failure modes (effect of faults, effect of rockmass variability, the effect of draw, and the stabilizing effect of leaving ore in the stope are some examples of factors beyond the explicit capabilities of the models run for this Project to date). Further work should be performed to reduce the uncertainties in the analysis and to examine other potential modes of failure. This could be done in more detailed engineering studies at a later stage of the Project and would include the following.

- An analysis to represent the effects of jointing on the rockmass and the effects of major faults either through the Ubiquitous Joint Rockmass Model in FLAC3D or by using 3DEC;
- An analysis to consider rockmass variability in which the entire spectrum of rockmass strengths is respected in the model in a statistically valid way; and
- If additional faults are found to intersect the region of the Renard 2 pipe, they should be explicitly considered in both the stress modelling and the hydrogeology modelling.

As underground experience is gained at this mine site, these analyses should be continually reviewed to determine if they are still valid over time and to ensure that new information is properly considered into the analysis.

16.3 Hydrogeology

Hydrogeological analyses were completed by Itasca Denver, Inc., in coordination with Itasca Consulting Canada, Inc., (Itasca). The work included: compiling the existing hydrogeologic data, constructing a numerical, three-dimensional (3-D) groundwater flow model of the site, and estimating the potential groundwater inflows to the mine working for the feasibility case mine plan.

The geometry of the open pits and underground mine workings was provided by Stornoway. The boundaries of the groundwater flow model domain were selected far away from the mining area where the hydraulic stress induced by mining is insignificant. The water levels of the major lakes were obtained from Stornoway.

During 2010, WESA conducted packer testing in two existing boreholes that were selected by Stornoway and Itasca for the purpose of determining the hydraulic conductivity of the rock units. Data was also obtained from various sources, including the previous hydrogeologic field investigations that have been conducted at the Renard site (by GENIVAR, WESA, and Roche) between 2005 and 2010. The hydraulic conductivity of the country rock generally decreases with depth, as expected, but shows high variability within 50 m of the ground surface.

The groundwater flow model constructed for this investigation utilises the numerical code MINEDW (Azrag et al., 1998) developed by Itasca, which solves 3-D groundwater flow problems with an unconfined (or phreatic) surface using the finite-element method. MINEDW has several attributes that were specially developed to address conditions often encountered in mine dewatering. The modelling code has been used and verified for numerous mine dewatering projects throughout the world.

The excavation of open pits, ramp, shaft, and the draw drifts of the underground mine was simulated according to the mining schedule. The subsidence and crack zones associated with underground mining were also simulated. These regions were simulated as areas of enhanced hydraulic conductivity values that develop over time, according to the mining schedule.

16.3.1 Prediction of Inflow

Three predictive simulations were conducted, including a base case scenario and two additional scenarios designed to assess the effects of uncertainties in the key model parameters. The model has been updated in the fall of 2012 following the mining sequence update.

The Itasca model considered the mining sequence and schedule and estimated the inflows over time. The inflows reach an initial peak in 2017 with 2,300 m^3/day (421 USgpm) after 37 months of development and then gradually drop down to a long term inflow varying between 1,200 m^3/day (220 USgpm) and 1,600 m^3/day (294 USgpm) before reaching a life-of mine overall peak of 2,475 m^3/day (3454 USgpm) late in the mine life when mining on R2 610L horizon begins.

It was found that the predicted inflow is highly sensitive to the hydraulic conductivity of the country rock. Increasing the hydraulic conductivity value by one order-of-magnitude would result in the predicted inflow rate three times greater than the value predicted from the base case.

16.3.2 Limitations

The confidence level of the groundwater flow model prediction depends on the available hydrogeologic data obtained at the site. Currently, there are limited data on the hydrogeologic conditions of both the overburden and the country rock at the site, the interaction between the surface-water bodies and the groundwater system, and the hydrogeologic conditions of the faults. The model used the existing data in the predictive simulations, the results of which are subject to change depending on any new data collected in future investigations. These limitations should be addressed in future model updates.

16.4 Open Pit Design

The mining strategy for the Renard orebodies consists of recovering the upper portions of Renard 2 and Renard 3 through open pit mining methods and the deeper portions of Renard 2, Renard 3 and Renard 4 using an underground blasthole shrinkage mining method. The size of the open pit was established through a trade-off study that optimized the profit from mining by open pit versus underground mining techniques.

Material will be mined from the Renard 65 kimberlite body to supply construction aggregates and underground backfill material during operation once the surface waste rock stockpiles are exhausted. Country rock from the Renard 65 pit will be used as backfill and aggregate while the kimberlite, which can be reclaimed for processing in the future, will form the platform of the ore stockpile. The resulting Renard 65 pit will be used as a catch basin to manage water on the project site.

16.4.1 Open Pit Optimization

Open pit optimization was carried out using the Whittle software package implementing the Lerch-Grossman algorithm. The Whittle optimizations were performed using the Renard 2010 resource model provided by GeoStrat Consulting Services Inc. This model was used to produce an updated mineral resource estimate which was included in a press release by Stornoway on January 24, 2011. The NI 43 101 technical report was filed with SEDAR on February 7, 2011. This feasibility level study only considers mineralization classified in the Indicated Resource category which limits open pit mining to the Renard 2 and Renard 3 kimberlite ore bodies.

A mining dilution factor was estimated for each pipe by including a 1 m envelope around the ore. The resulting dilution factor, which was employed in the optimization process, was utilized to calculate the diluted grade. As the Renard 2 and Renard 3 open pits merge on surface, an average dilution factor of 10% was used in the optimization process. A 96% mine recovery was applied to account for ore misdirected or missed at the edge of the ore body.

The optimization parameters utilized to generate the Whittle shells are summarized in Table 16.1. These parameters were based on preliminary operating cost estimates which were subsequently refined during the course of the Feasibility Study.

Table 16.1: Summary of Open Pit Optimization Parameters

Parameters		Renard 2	Renard 3
Diamond price	US$/carat	139	
Exchange rate	C$/US$	1.00	
Selling Cost	%	3.0%	
IBA payment	%	1.0%	
Royalty	%	2.0%	
Metallurgical recovery	%	100%	
Open pit mining dilution	%	10%	
Open pit mining recovery	%	96.0%	
Total processing cost	C$/t processed	12.45	
G&A cost	C$/t processed	13.68	
Total ore based cost	C$/t processed	26.13	
Cut-off grade	cpt	0.21	0.19
Reference mining cost		Owner	
Kimberlite ore	C$/t mined	3.60	
Granite & gneiss waste rock	C$/t mined	3.45	
Overburden	C$/t mined	2.90	
Incremental bench cost	C$/t mined/10m	0.045	
Whittle overall slope angles		Renard 2	Renard 3
Kimberlite ore	Degrees	45	
Granite & gneiss waste rock	Degrees	45	
Overburden	Degrees	20	

The optimum pit size selected to guide the pit design yielded an estimated incremental mining cost per tonne of ore equal to C$28.38/t. The optimum pit shell was the first shell to yield a higher cost than that achievable using the estimated blasthole shrinkage mining cost C$23.13/t. The optimum pit shell is presented in Table 16.2.

Table 16.2: Summary of Optimal Shell used for Detailed Pit Design

Whittle Results		R2/R3
Shell number		6
Revenue factor		0.48
Total tonnage	Mt	9.20
Waste tonnage	Mt	7.30
Ore tonnage	Mt	1.90
Diamonds	carats	1,776,616
Grade	cpt	0.94
Strip ratio	Mt:Mt	3.85 : 1

16.4.2 Bench Geometry

Bench heights of 10 m were selected to facilitate efficient drilling and blasting activities. The loading units will mine the benches in two 5 m high cuts. The pit slope profile was determined using geotechnical recommendations from Golder Associates (overburden) and Itasca Consulting Canada Inc. (bedrock).

The modelled overburden thickness varies from 5 m at the edges to > 25 m over the center of the orebodies. The slope configuration recommendations vary based on overburden thickness and are presented in Table 16.3.

Table 16.3: Overburden Slope Configuration Recommendations (refer to Section 16.1)

OVB slope height (m)	Overburden Slope Configuration				
	Upper slope (max 10 m)	Berm width (m) min.	Lower slope	Min OVB slope crest offset	Min pit rim offset (m)
0 - 6	2H : 1V	---	---	10	8
6 - 10	2H : 1V	---	---	10	8
10 – 20	2H : 1V	5	2H : 1V	20	15
> 20	2H : 1V	5	2H : 1V	20	20

Itasca was tasked with investigating the geotechnical parameters, as they relate to mining, and to recommend feasibility level slope angles for the open pits. Itasca's pit slope recommendations did not involve geotechnical domains therefore the pit slope profiles are uniform for all pit walls and for all the pits studied. The recommended rock slope configuration is presented in Table 16.4.

Table 16.4: Rock Slope Configuration Recommendations (Itasca Consulting)

Pit	Rock Slope Configuration			
	Final bench height (m)	Catch bench width (m)	Bench face angle (°)	Inter-ramp angle (crest to crest)
R2	20	9.0	80	57.9
R3	20	9.0	80	57.9

16.4.3 Ramp and Haul Road Design

The ramp was designed to accommodate double lane traffic. The largest equipment considered for the Project was a 60-ton class rigid truck. The maximum ramp gradient is 10%. Single lane ramps are used to access the pit bottoms. Out-of-pit haul roads are designed for double lane traffic.

A road network will connect the pits to the crusher and stockpile area, waste dump, overburden stockpile and processed kimberlite facility.

16.4.4 Open Pit Design

The Renard 2 and Renard 3 open pit excavations result in a single pit at surface with two pit bottoms centred on the respective orebodies. The excavation is therefore treated as a single pit and is referred to as the Renard 2/Renard 3 pit. Renard 65 will be mined to obtain sufficient backfill and construction material for the Project; therefore the pit design is not constrained by economical limits.

The Renard final pits are too small to allow for internal pit phases. The dimensions of the pits are summarized in Table 16.5 and are presented in Figure 16.2 and Figure 16.3.

Table 16.5: Renard Pit Dimensions

Pit	Pit dimensions (m)	Pit depth (m)	Pit footprint surface area (ha)
R2/R3	455 x 310	113	126.2
R65	450 x 315	61	8.5

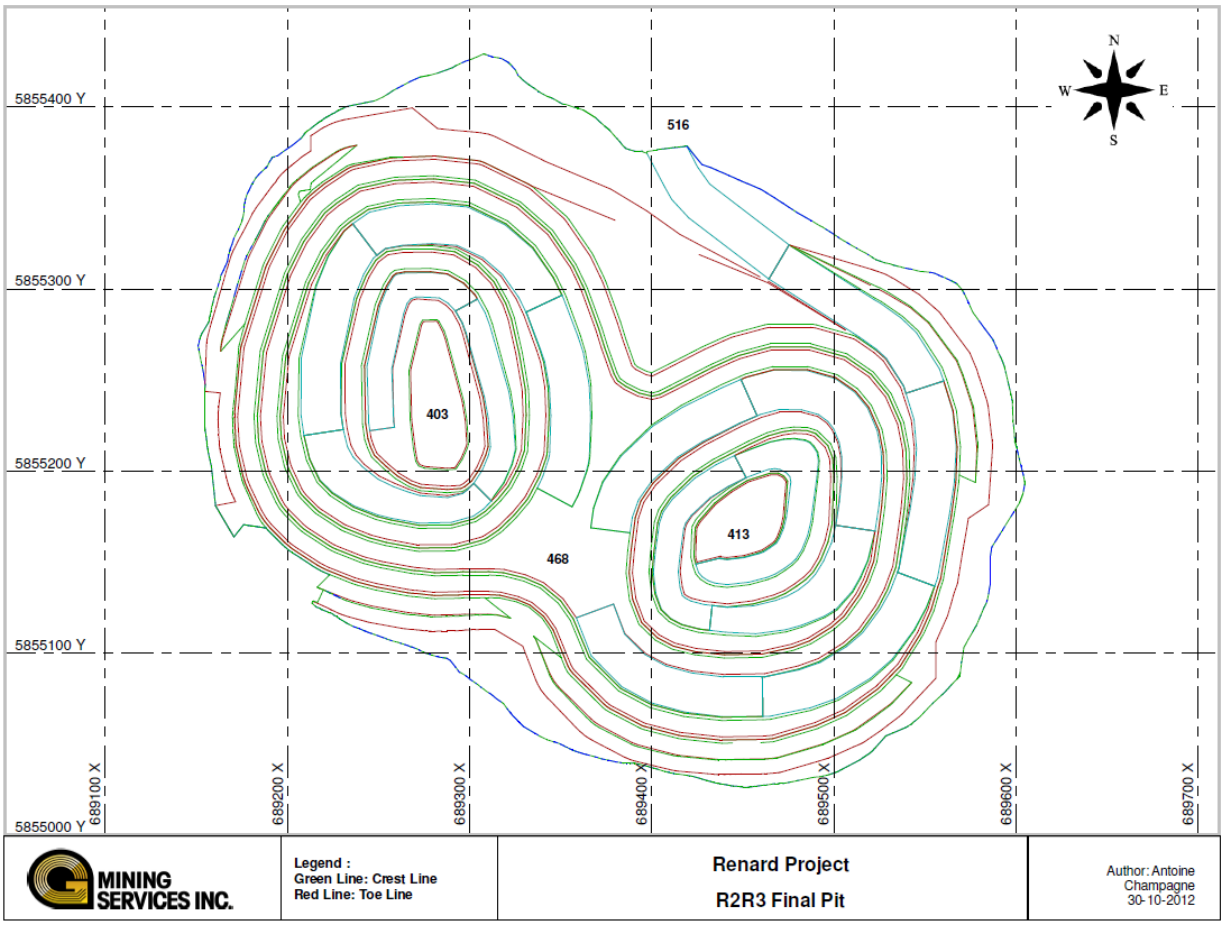


Figure 16.2: Final Renard 2/Renard 3 Pit Design

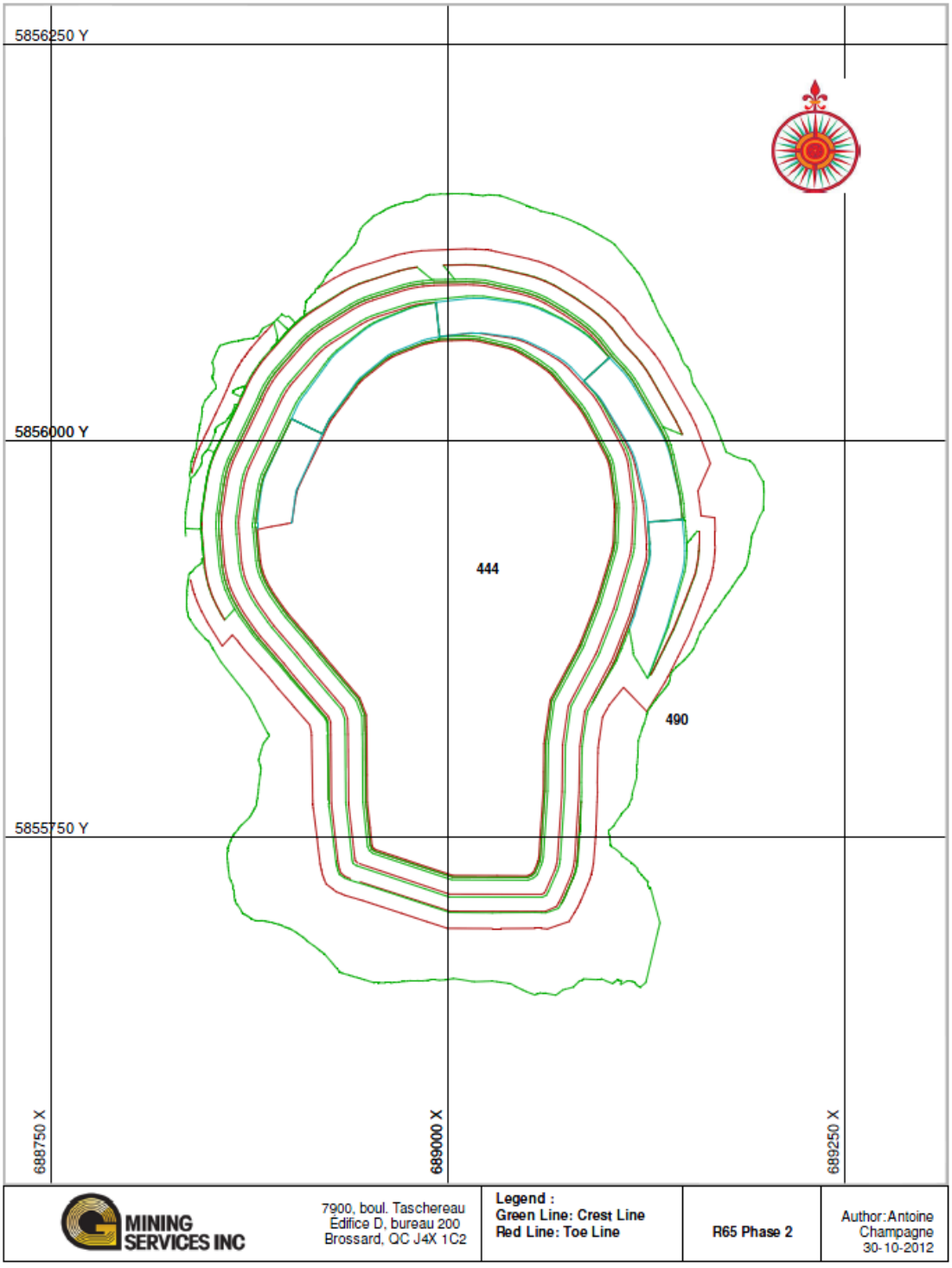


Figure 16.3: Renard 65 Pit Design

16.4.5 Drilling, Blasting and Surface Explosive Storage

Production drill patterns were proposed for both waste rock and kimberlite. The waste rock is a granitic gneiss, whereas the kimberlite ore is an extensively altered material. The drill pattern proposed for the kimberlite ore is tighter than the pattern proposed for waste rock to achieve a better energy distribution and a higher explosives powder factor, as presented in Table 16.6.

Table 16.6: Production Blast Pattern Parameters

Design ratios	Units	Pattern 1 Waste Rock	Pattern 2 Kimberlite
Explosive type		Emulsion	Emulsion
Explosive density	g/cm3	1.20	1.20
Hole diameter	mm	165	165
Burden	m	5.0	4.0
Spacing	m	5.0	4.0
Sub-drill	m	1.5	1.5
Stemming	m	3.5	3.2
Bench height	m	10.0	10.0
Blasthole length	m	11.5	11.5
Powder factor	kg/t	0.30	0.50

The blast pattern yields presented in Table 16.7 are based on an average density of 2.70 t/m^3 for waste rock, and 2.65 t/m^3 for kimberlite. The tonnage per metre of drilling ranges from 37t/m for kimberlite to 59 t/m for waste rock.

Table 16.7: Blast Pattern Yields

Pattern yield	Units	Pattern 1 Waste Rock	Pattern 2 Kimberlite
Rock density	t/m^3	2.70	2.65
BCM/hole	BCM/hole	250	160
Yield per hole	t/hole	675	424
Yield per metre drilled	t/m drilled	59	37

Pre-split blasting is recommended to maximize stable bench face and inter-ramp angles along final walls. The pre-split consists of a row of closely spaced holes along the design excavation limit. The holes are loaded with a light charge and detonated simultaneously or in groups separated by short delays. Firing the pre-split row creates a crack that helps define the excavation limit and helps prevent wall rock damage by venting explosive gases and reflecting production blast shock waves.

The pre-split design parameters are outlined in Table 16.8. The hole diameter is identical to that of the production patterns which can be drilled with the same rig.

Table 16.8 Pre-split Design Parameters

Pattern yield	Units	Pre-split
Hole diameter	mm	165
Spacing	m	1.5
Stemming	m	3.4
Bench height	m	20
Blasthole length	m	20.7
Face area	m2	31.1
Total explosive charge	kg	30.0
Toe charge	kg	5.0
Decoupled charge	kg	25
Decoupled charge length	m	17
Decoupled charge loading	kg/m	1.47

Drill rig requirements were established from the yield per metre drilled estimated from the proposed drill patterns, the instantaneous penetration rates and the actual drilling time described by an overall drilling factor. The instantaneous penetration rate is a function of the bit size, compressor size and the rock mass unconfined compressive strength (UCS). The granite-gneiss waste rock UCS averages 140 MPa and the kimberlite average is lower and variable ranging from 78-100 MPa. Table 16.9 presents the drilling productivity assumptions used to estimate drill rig requirements.

Table 16.9: Drill Productivity Assumptions

Drill productivity	Units	Pattern 1 Waste Rock	Pattern 2 Kimberlite	Pre-split
Pure penetration rate	m/h	55.0	80.0	55.0
Overall drilling factor	%	50.0	50.0	50.0
Overall penetration rate	m/h	27.5	40.0	27.5
Drilling efficiency	t/h	1,614	1,475	-
Drilling efficiency	holes/h	2.75	4.00	1.33

Due to the limited tonnage to be mined, a single versatile drill rig model is proposed to drill pre-split and production holes with a diameter of 165 mm. A diesel powered, self-contained crawler mounted drilling rig designed for DTH blasthole drilling with angle drilling capability is required. The rig will be equipped with a high pressure 350psi/1000 CFM compressor.

A high energy bulk emulsion explosive suitable for surface mines in wet or dry conditions was selected. The bulk emulsion is booster sensitive with an average density of 1.20 g/cc that can be varied. A packaged pre-split explosive is planned for blasting pre-split holes and minor quantities of packaged emulsion will be kept on-hand (Table 16.10).

Table 16.10: Explosive Hole Loading and Cost

Explosives Products	Units	Pattern 1 Waste Rock	Pattern 2 Kimberlite	Pre-split
Bulk emulsion	kg	205	213	5
NONEL 7.3m	ea	1	1	
NONEL 18.0m	ea	1	1	
NONEL 9.0m	ea			
NONEL 30.0m	ea			1
Booster 450gr	ea	2	2	1
Pre-split explosive	kg			25
Explosives cost/hole	$/hole	172	177	233
Explosives cost	$/tonne or $/m^2	$0.27/t	$0.44/t	$7.50/m^2

Blasting activities are planned on day shift only and will be outsourced to an explosives provider who will be responsible for supplying and delivering explosives in the hole through a shot service contract. The explosives supplier crew would consist of two men working 12 hour shifts on rotations.

The surface explosives storage site consists of a 40,000 kg capacity, heated explosive magazine with stairway and loading ramp for storage of 1.1D class explosives such as packaged product, detonating cord and boosters. An additional 10,000 kg capacity detonator magazine which can contain 100,000 detonators is required for storage of 1.1B class explosives such as detonators.

The explosives storage site will store 20,000 kg of emulsion ISO containers which will be electrically heated as required. From this container, emulsion will be pumped into an emulsion re-pump truck for distribution to the open pit or into 1,500 kg bins for underground distribution.

16.4.6 Open Pit Mine Production Schedule

The mine production schedule was developed to supply the plant at a nominal rate of 2.16 Mt/yr or 180,000 t/month. A ramp up schedule over nine months is planned starting in December 2015, including 3 months of commissioning is presented in Table 16.11.

Table 16.11: Plant Ramp-up Schedule

Month	Planned Tonnes (t)	Percent of Nameplate Capacity (%)
1 (Dec 2015)	5,500	Commissioning
2	5,500	Commissioning
3	5,500	Commissioning
4	63,000	35%
5	108,000	60%
6	135,000	75%
7	157,000	87%
8	171,000	95%
9	180,000	100%

The open pit production rate in the Renard 2/Renard 3 pit is less than the mill capacity. An ore stockpile will be accumulated which can be used to balance the open pit and plant schedule until such time as sufficient ore is available produced by underground mining operations.

The production schedules were developed on a monthly basis until the end of 2018; quarterly until the end of 2022 and annually thereafter. The monthly scheduling provides a high level of detail on how the operation will transition from open pit to the underground.

The Renard 65 phase 1 borrow pit will be mined to provide construction materials and to establish a water catchment basin. Mining of the phase 1 borrow pit will occur from July 2013 to June 2014 after which mining activity is transferred to the R2/R3 open pit. Mining of the Renard 2/Renard 3 open pit will commence in March 2014 and be concluded in February 2017. Prior to the start of mining production, the mine department will perform various site preparation tasks. Extensive clearing and grubbing will be required on the Renard 65 borrow pit, Renard 2/Renard 3 pit, waste dump, overburden storage, and PKC storage sites. Both permanent and temporary mine roads will need to be constructed, and a network of drainage ditches will be built around the major roads and working areas for water management purposes.

16.4.7 Equipment Fleet

The primary loading tools will consist of one hydraulic excavator and one wheel loader. The hydraulic excavator will be a Caterpillar 390D, or equivalent, fitted with a 6.0 m^3 bucket which is equivalent to an average actual payload of 11.6 tonnes. The wheel loader will be a Caterpillar 990 H, or equivalent, with an 8.6 m^3 bucket and an average actual payload of 14.8 tonnes. A Caterpillar 345D L, or equivalent, will also be on site for construction purposes, secondary rock breakage, ditching and various support functions and can act as a back-up unit if required.

Both rigid-body and articulated haul trucks will be used for open pit production. The use of both types of trucks will result in greater versatility and lower capital costs. A total of seven units are required: three articulated trucks and four rigid body haul truck. The articulated truck is the Caterpillar 740, or equivalent, fitted with a 24.0 m^3 box. The rigid-body haul truck is the Caterpillar 775F, or equivalent, fitted with a 41.9 m3 box. Given the productivity parameters of

the Project, the articulated haul trucks will maintain a productivity of 771 tph when matched with the excavator and 842 tph when paired with the wheel loader. The rigid body trucks will maintain a productivity of 793 tph when matched with the excavator, and 850 tph when paired with the wheel loader.

Major support equipment will consist of track dozers, motor graders, and wheel dozers. Additional minor support equipment includes compactors, tool carriers, service trucks, fuel & lube trucks, water trucks, and an excavator.

Motor Grader – A motor grader with a 14 ft blade will be required for haul road maintenance such as removing spill debris, spreading surface material, clearing snow, and maintaining an appropriate level grade. One unit will be required during open pit production periods.

Track Dozer – Caterpillar D6 track-type tractor, or equivalent, will be required for all dozing activities. These include but are not limited to, waste dump construction, bench & pit clearing, stockpile management, overburden stripping, and road construction. Three units will be required during the peak open pit production periods.

Compactor – A smooth drum compactor (Caterpillar CS56 or equivalent) is required. One unit will be required for 4-5 months per year for maintenance and construction purposes on the haul roads, waste dumps, and PKC storage facilities.

Tool Carrier – A tool carrier (Caterpillar IT62H or equivalent) is required for maintenance support, material handling and surface support. One unit is planned.

Service Truck / Fuel & Lube Truck / Water Truck – A service truck fleet (Kenworth T800 or equivalent) is equipped to fulfil these various functions. Three trucks will be required; one as a maintenance service truck, another for refuelling and lubricating the track-type equipment, and third for watering the haul roads for dust suppression.

Service Excavator – A hydraulic excavator (Caterpillar 345D L or equivalent) will be used for various construction projects, most notably drainage ditch and road construction, as well as maintenance on the PKC storage facility and secondary rock breakage.

16.5 Underground Design

16.5.1 Mining Methods

At the outset of the feasibility study mine design, a trade-off study was conducted to select the preferred underground mining method or methods to be used to extract the Renard 2, Renard 3 and Renard 4 pipes. In the study, potential mining methods were identified, evaluated and compared considering safety, dilution and recovery, production capacity, schedule, economics, risks and opportunities. As part of this study, a rock mechanics evaluation was conducted by Itasca Consulting Canada Inc. which determined that the Renard 2 and Renard 4 pipes were unlikely to support natural progressive caving, since the “caving” region is reached only for the lowest rock properties and largest hydraulic radii. As a result, block caving was eliminated as a potential mining method. Four underground mining methods were shortlisted and evaluated including: blasthole shrinkage (BHS), sublevel retreat (SLR), longhole panel mining with backfill (LHP), and blasthole shrinkage with pillars (BHSP). Based on this study, the “Blasthole

Shrinkage" (BHS) mining method was selected to mine the underground portions of the Renard 2, Renard 3 and Renard 4 kimberlite pipes.

The BHS method consists of drilling and blasting the ore with long large diameter blastholes and extracting the ore at the base of the stopes from drawpoints. During the blasting phase, ore is left in the stope to support the walls until the complete stope has been blasted. Since ore expands when blasted due to the voids in the broken rock, it is necessary to draw out this swell on an ongoing basis, approximately 35% of the total in situ ore. Thus with this method, there is a continual supply of ore production throughout the blasting period, and once the blasting is complete, the remaining broken ore can be drawn from the stope – generally at a very high rate with few restrictions.

The advantages of BHS are:

- High production capacity capable of producing 6000 tpd;
- Minimizes overall development requirements with large sublevel spacing (60 m);
- Blasted ore left in stope (shrinkage) provides support to stope walls until the blasted ore is drawn down;
- Modern large equipment 12t LHD's and ITH drills with 60+ m blasthole lengths) can be used resulting in high equipment productivity;
- Initial production of the stopes can begin before completion of the Renard 2/Renard 3 open pit;
- Over half of the blasted ore can be recovered with low dilution before higher dilution in the final drawdown phase begins.

The BHS method has been designed to progress in three stages from the top of the pipe (bottom of the open pit) downwards. As the ore is drawn from the stope, a large open void will be created if no backfill is used. The mine plan is to fill this void using open pit waste as backfill which will be placed into the stope by dumping it from trucks on surface onto the top of the broken ore column being drawn down in the pipe. For Renard 2/Renard 3, the open pit waste will be dumped over the side of the pit,, while for Renard 4 the waste will be dumped into backfill raises leading to the top of the underground stope.

16.5.2 Mine Design

Access to the mine will be provided by a ramp to exploit the mineral reserves which have been defined down to a depth of 610 m for the Renard 2 pipe, 250 m for the Renard 3 pipe and 235 m for the Renard 4 pipe. The underground mine has been planned so that a single common ramp will provide access to all three Renard pipes. The ramp has beensized to allow efficient ore extraction with an underground mobile equipment fleet, such as 60t haul trucks and 12t LHD's. It will be driven from surface to the 610L and will connect to all six drill levels and three production levels. Since only a ramp and a supported raise with a manway have access through to surface, all workers, equipment and materials will be transported in and out of the underground mine via the ramp facility.

As part of their scope of work, Itasca established empirical subsidence zone limits based on an 80^{o} subsidence cone from the base of the Renard 2, Renard 4, and Renard 9 pipes at a 700 m depth. These limits created two "no-go" zones for any long term permanent structures, including

a potential shaft and the ramp and all other surface facilities. In addition, Itasca completed a global scale numerical geotechnical model which indentified areas from which long term permanent underground infrastructure should be excluded. As part of this exercise, an ellipse was created within which future shaft could potentially be located, and plans showing horizontal strain and total displacement within that ellipse were produced. Except for one corner of the ellipse closest to the Renard 2 pipe, all values were within acceptable limits. The global numerical model was also used to generate similar data at the various elevations and this data was used to guide the placement of the ramp, permanent levels and other infrastructure underground.

A 3-D view of the planned development and stopes is shown in Figure 16.4.

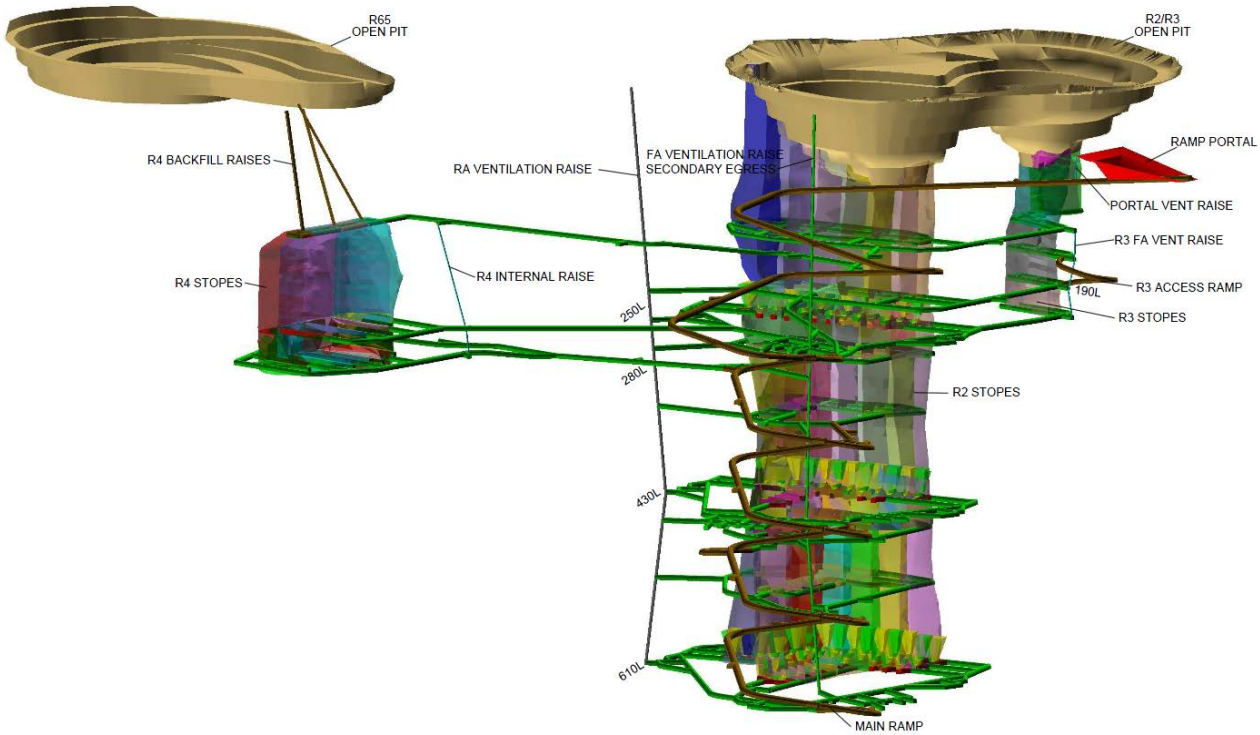


Figure 16.4: 3-D View of the Renard Mine

16.5.3 Stoping Design

The Renard 2 pipe has the largest indicated resource and has the highest quantity of contained carats in the three pipes included in the feasibility study and, consequently, has the greatest influence on selecting the stoping design. The indicated portion of the Renard 2 pipe has a vertical height of 610 m, of which the upper two thirds is somewhat irregular in outline.

For production mining, the Renard 2 pipe has been divided vertically into three production stages (zones), with the bottom two stages each 180 m high and the uppermost stage 150 m high to the bottom of the pit. The resulting production levels with this design are at level 250L, 430L and 610L (all level designations are depth below surface in metres). Drawpoints and cones will be located at the base of each of the stages. A top-down mining sequence is planned for the three mining zones in Renard 2. First, the upper 250L zone will be mined, followed by the 430L zone and finally the 610L zone. Each mining zone has been sub-divided into vertical panels; seven panels on 250L, ten panels on 430L and eleven panels on 610L. Mining of the kimberlite will proceed horizontally from the North panels towards the South panels in a stair fashion. During the transition between two zones, production of the level below will start from the North panels as the South panels of the level above are in the progress of being deplinished. On a panel scale, the shrinkage aspect of the BHS method comes into play here, whereby the blasted lower section of the panel stope will be kept full of blasted ore during this period, thus limiting the potential for unravelling of the crown pillar. Once the crown/sill pillar level has been reached and blasted, full mucking activity of the panel can initiate. As the blasting progresses towards the crown or sill pillar inside a particular panel, the blasting can start in the next adjadent stope to the south in a stair fashion. The blasting sequence for Renard 2 is shown on Figure 16.5.

The base of the indicated resource in the Renard 4 pipe is 235 m below surface and will be mined below a 100 m thick crown pillar for a 135 m vertical height. The interpretation of the shape of the Renard 4 pipe is oblong, and very regular in its outline, so this height was deemed to be suitable for BHS mining. One production level and two drill levels will be used to extract all reserves from the Renard 4 orebody, which will also be subdivided into vertical panels as Renard 2. The bottom production level for Renard 4 pipe is at the 280L as shown on Figure 16.4.

The Renard 3 pipe is fairly large close to surface but then necks down to a fairly small pipe below the open pit and further necks down at the base of the indicated resource at a depth of 250 m. This pipe will be mined bottom up by BHS with drill/production levels at 30 meters intervals, which will be developed from Renard 2 existing development at that time. The bottom production level for the Renard 3 pipe is at the 250L and mining will extend up to the bottom of the pit as shown on Figure 16.4.

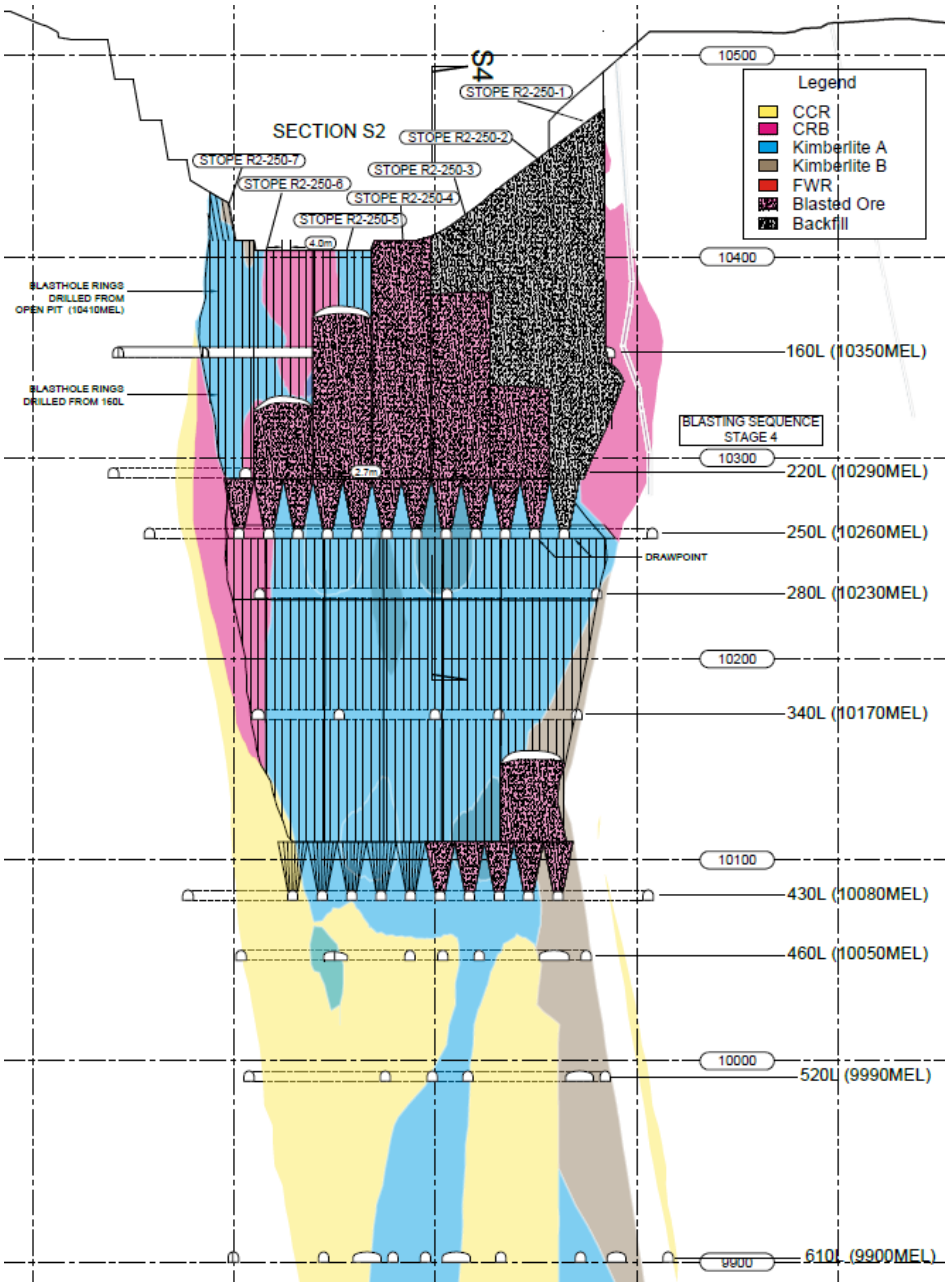


Figure 16.5: Typical Section Illustrating the BHS Method and Mining Zones

16.5.4 Drilling and Blasting

The kimberlite will be drilled and blasted by a combination of large diameter downholes for the body of the stopes and smaller diameter upholes for the undercut drawcones and drawpoint level pillar blasting.

The primary production drills will be in-the-hole (ITH) rigs drilling 165 mm diameter blastholes with a toe spacing of 4.0 m and a burden of 4.0 m for an average drill factor of approximately 22 t/m drilled. Typical hole lengths will be 60 to 75 m. The ITH drills will be compressed air-driven with each unit having its own booster compressor to raise the air pressure to 400 psi from the nominal mine compressed air pressure of 125 psi. The undercut drilling will be by a rubber tired electric hydraulic production drill unit drilling 75 mm diameter blastholes with a toe spacing of 3.0 m and a burden of 2.7 m, for an average drill factor of approximately 10 t/m drilled. Hole length will vary between 25 to 30 m.

Emulsion explosives will be used almost exclusively for both development and production blasting.

16.5.5 BHS Mining

With the BHS method, the kimberlite in a panel is blasted in successive vertical lifts over the height of the panels, in a benching fashion. A slot raise will be excavated over the height of the stope to provide an initial free face from which to initiate the blasting. This slot raise will be mined during the drilling phase using a “Machine Roger” ITH drill head attached to an ITH drill rig. While the panel is being blasted, sufficient blasted ore will be left in the stope (shrinkage) to provide support to the walls and limit the extent of sloughing or caving from the back if that should be experienced. This is illustrated in Figure 16.5.

The blasting and production phase will start by blasting the undercut drawcones from the drawpoint level and advance in a retreat blasting and mucking sequence. After each blast, sufficient ore will be drawn (mucked out) to provide for a free face and ample void before proceeding with the next blast. The blasting strategy will be to blast out the drawcones below each panel before beginning the blasting of the downholes.

The downholes are typically 60 m long, drilled from the drill level above. Blasting will be done by vertical benching, initiated towards a central slot raise, in vertical lifts. Following each blast, the blasted ore will be mucked out from the drawpoints below, until the void or gap at the top of the blasted ore to the back is sufficient for choke blasting the next lift. During the blasting phase, 35% of the ore will be drawn from the stope through drawpoints to accommodate the swell of broken ore.

Once the crown pillar of the north panel of the 250 mining zone is blasted, the Renard 2 pipe will be open to surface and, at this point, production mucking in this panel can be ramped up to the full production rate. As blasting of the other panels progresses towards the South, the opening in the pit bottom will get bigger until the crown pillar of the last panel is blasted.

As the blasting inside the south panels of 250L is nearing completion, the blasting sequence can progress in the north panels of the 430L. The mass blasting of the 430L mining zone residual pillar (pillar below the 250L drawpoint level) will include not only the pillar, but also the

underside of the drawpoint troughs on the 250L. Detailed modeling will confirm the thickness of the pillar to be kept in situ between the above mucking horizons and the blasted stope below. Mucking directly above the panel will have to be completed before the sill/crown pillar of the panel below can be removed. After this blast, the drawpoints on the 250L will have been destroyed and mucking from the nnorth panel on 430L can be ramped up to full production. This sequence will be repeated for each individual panel as mining progresses towards the South on this level. The blasting sequence on 610L will be the same as on the 430L.

16.5.6 Backfilling

After the 250L North panel residual pillar is blasted through to the base of the open pit and production and full drawpoint production begins from the 250L drawpoint level, the column of blasted ore will be gradually drawn down, leaving an open hole. As the crown pillar of each panel is being blasted, ore drawdown will progress and void will be filled by the waste dumped from surface. Consequently, waste rock backfill will be used as an integral part of the mining plan. Waste rock for this purpose will be available from the stripping of the open pit as well as from development in waste rock underground. This use of the open pit waste will also enable the rehabilitation of the waste rock pile during mine operation rather than at the end of the mine life.

The method of placing the waste rock backfill into the mine is straightforward, it will be dumped over the edge of the open pit, cascade down to the stope opening at the base of the pit and then onto the blasted ore column. This concept is illustrated in the mining method drawing shown in Figure 16.5. The backfilling method for the Renard 4 pipe will be the same with the exception that three raises will be driven up from the base of the crown pillar, and the waste rock backfill placed into the stope through these raises.

As the ore column is drawn down during production, waste rock will be steadily dumped into the stope to maintain a constant level of backfill in the stope at surface. Waste rock for the backfill will be generated from the Renard 2/Renard 3 open pit stripping, Renard 65 pit excavation for water collection, and underground mine development. Overall, there is sufficient waste rock available for backfilling.

16.5.7 Drawpoint Mucking and Production

Ore production will be from blasted ore pulled from drawpoints situated at the base of each mining zone. With the BHS method, waste rock backfill will be placed on top of the ore column and consequently, it is critical that the drawpoint design provide good draw coverage and a draw management system be instituted that will result in an even drawdown of the blasted ore column. The drawpoint layout proposed for the R2 pipe at Renard is based upon a commonly used arrangement from block caving operations, which is the Offset Herringbone layout. The drawpoints are arranged on a 30 m x 15 m spacing with the drawpoint accesses having a 40 degree break angle from the access drift. With a drawcone spacing of 15 m center to center and a height of roughly 30 m, the resulting drawpoint pattern is a 10 m pillar and a 5 m wide drawpoint. Even distribution of drawpoints allows for maximum draw control and minimizes the likelihood of increased dilution and backfill ingress into the ore column. The drawpoint brows in the R2 pipe will be sufficiently supported with cable bolts and shotcrete to ensure the durability of the drawpoint during the mining life cycle of the stope.

Ore drawdown management during production is critical to limiting dilution and maintaining high recovery. There are two aspects to managing drawdown: the first is to maintain an even drawdown over the whole blasted ore column in the stope and promote mass flow of the ore. One of the big dangers resulting from poor draw management is the development of rat-holes above individual drawpoints, ahead of their neighbors, right up to the waste backfill placed on top of the blasted ore. A good drawdown plan in combination with strict draw procedures and regular monitoring is required for success. Identifiable markers placed in the blasted ore, either via placement in blastholes or on drill levels will provide good data on the effectiveness of the drawdown system.

During stope production mucking, waste material that appears in a drawpoint will be sorted from the ore stream to reduce the amount of diluted material sent to the surface crusher. The sorting will be visually done by the geologist and scooptram operator as the surrounding country rock and the waste inclusions in the pipe is significantly lighter in color than the kimberlite. Sorted waste material will be mucked to surface for dumping either to the waste pile or into the pit as backfill material.

The objective on the drawpoint levels is to create and maintain a high production capacity through efficient mucking and ore transportation. A key element of the planned system is the use of high capacity mine haulage trucks to transport the ore to surface. To maximize the effectiveness of the trucking system, a series of truck loadouts are planned on each production horizon to facilitate the loading and queuing of trucks to assist in the prevention of production bottlenecks. The truck loadouts are placed to reduce the ore haul from the drawpoints by scooptrams to a minimum and utilize the haulage capacity of the mine trucks.

On the 250L and 430L, two truck loadouts are planned, one on the north side of the R2 pipe and one on the south side. The average haul distance from drawpoint to loading station is approximately 95 m. On the 610L, an additional truck loading station is planned for the west side of the R2 pipe to minimize the haul distance from the “T” section of the pipe.

Drawpoint mucking will be done with 12 t capacity LHD’s, for which presently there are three major mine equipment manufacturers that supply this class of unit. These units will load from the drawpoints and haul directly to the truck loading station. To achieve 6,000 tpd, the LHD’s will

haul a total of 540 buckets per day or 270 per shift. Generally, this activity will be split between two producing haulage drifts each with between 6 to 20 drawpoints depending upon the advance of the stoping sequence. Based on mucking and tramming simulation, a total of 3 productions LHD's are required, except for the final years of mine operation where 4 units will be required.

The 430L drawpoint level design and location of the drawpoint drawdown cones are shown in Figure 16.6.

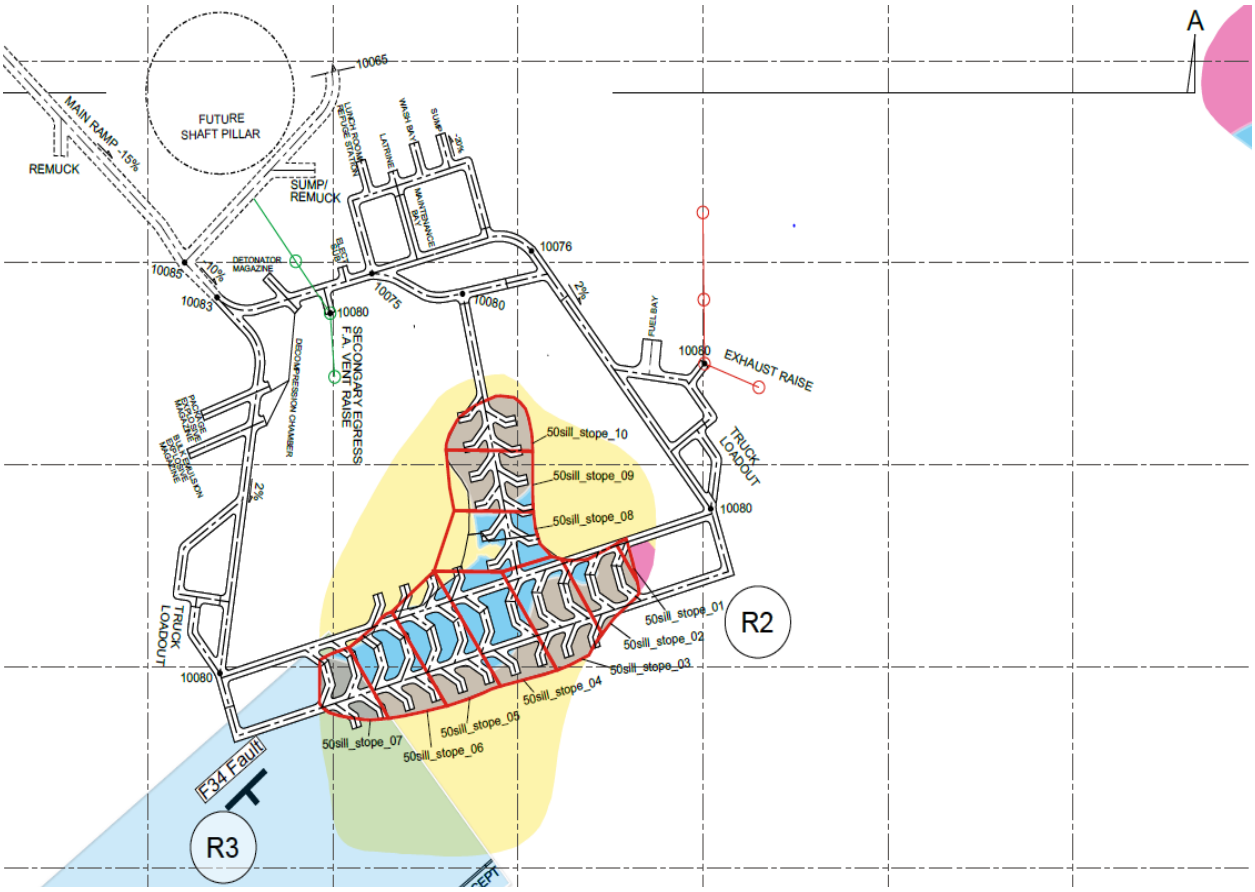


Figure 16.6: 430L Drawpoint Level Layout and Drawdown Coverage

16.5.8 Underground Mine Development

The mine will be accessed by a ramp and it will be driven at a size of 7.5 m wide by 5.6 m high and at a grade of 15% from surface to the 610L. When complete, the ramp will extend from surface down to the bottom 610 production level with a total length of 4.6 km and connecting to all production and drill levels. To facilitate smooth traffic in the ramp during the ore haul, passing bays will be incorporated into the ramp design to allow loaded ore trucks to travel the ramp uninterrupted.

All drill and production levels will be driven from the ramp and typically will be driven at a grade of +2%. Development waste from the levels will be trucked to surface via the ramp. All levels are planned with connections to the fresh air ventilation raise close to the ramp and to the exhaust raise in order to allow the main air flow to circulate at or near most workplaces. The sizes of the headings for ramps, production levels (5.6mW x 4.6mH), drill levels (5.3mW x 5.3mH), drill drifts (5.3mW x 4.8mH) and drawpoints (5.0mW x 4.2mH) were selected based on the equipment to be used and to accommodate ventilation ducting during excavation. The haulage drives and drawpoints are aligned to minimize haulage distances. The maintenance and support facilities for the mine, such as permanent refuge stations, are located on the main production levels.

The R3 and R4 pipes will be accessed off levels developed for R2 and only the 190L requires connecting through an internal ramp from 220L for the R3 development.

The main exhaust raises will be twinned unsupported vertical raisebore holes, 4.1 m in diameter, extending from surface to 610L. These raises will be bored in three sections, the first from surface to 250L at an inclination of 70^{o} so that the surface collar can be situated away from the R2/R3 pits and as close as possible to the surface facilities. The second and third sections will be vertical and will be bored from 250L to the 430L and from the 430L to the 610L. The main fresh air raise will be a 6.1m diameter supported raise bored from surface to the 250L and then in separate legs between the 250L, 430L and 610L. The fresh air raise will also serve as the secondary egress from the mine in the event of emergency where the primary egress through the ramp is unavailable. Mine services such as electrical cables, mine water and communications will be provided through the raise as well. A 3.0 m diameter bored ventilation raise will be excavated during development of the ramp from surface down 30 m to the ramp. The raise will be used to supply fresh air in the ramp. Internal raises with a diameter of 3.0m will be excavated in the vicinities of R3 and R4 ore bodies in order to allow the main flow to circulate closer to those workplaces.

The main mine development infrastructures are shown in longitudinal projection in Figure 16.7.

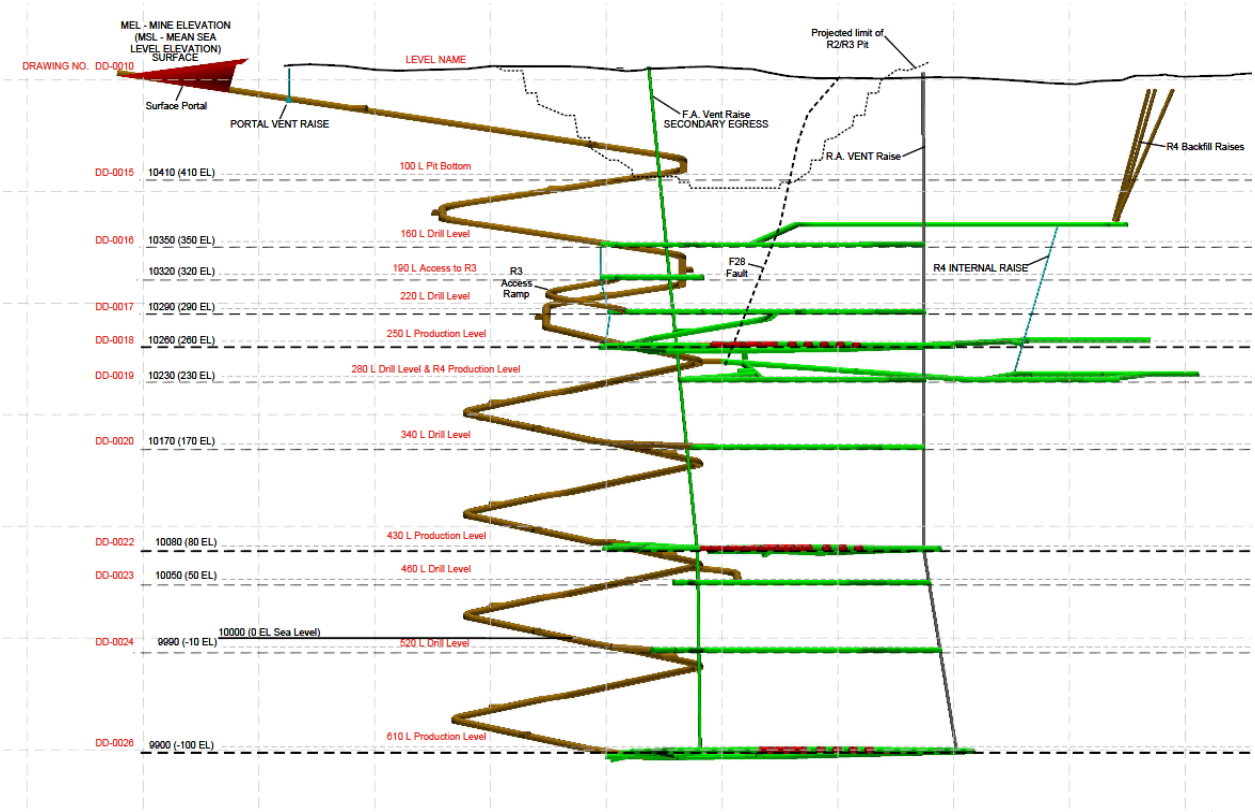


Figure 16.7: Longitudinal Section Showing Pre-production Capital Development

16.5.9 Ventilation and Heating System

Based on the planned diesel-operating equipment fleet, the mine ventilation requirement at full production is estimated to be 400 m^3/s (850,000 cfm). This quantity of fresh air will meet the requirements of Québec's Occupational Health and Safety Act. The mine will have two fresh air intakes, one via a 6.1 m diameter supported raise (main Fresh Air Raise - FAR) and the other a 3.0 m diameter raise (portal raise) which will be connected to the ramp immediately after the portal airlock doors. The maximum ventilation requirement occurs in the year 2021, when production will be transitioning to the deepest level of the mine and two development crews will be active. Fresh air will be distributed on every drill and production level by the ramp and main fresh air raise. The contaminated air will be exhausted through two 4.1 m diameter exhaust raises (RAR) excavated in parallel. The two main fans will be located on the surface on top of these two raises. The raises will be connected to every level to minimize the recirculation of contaminated air. A schematic of the primary ventilation network is shown in Figure 16.8.

Air distribution within the mine will be controlled by a combination of ventilation doors and regulators. This will allow complete flow coverage of the levels with minor use of auxiliary ventilation and ventilation curtains.

The R3 and R4 ore bodies will each be ventilated by 3.0 m diameter internal raises which will connect all related levels, allowing the main level air flow to circulate closer to the workplaces.

The main exhaust fans will be equipped with variable speed drive motors (VFD) to enable a high degree of control over the airflow rates produced by the fans. This degree of control will be used to maintain required airflow balances and ventilation pressure differentials throughout the mine. It should be noted that the fresh air requirement varies over the mine life, ranging from a minimum of 141 m^3/s in 2016 to a maximum of 400 m^3/s in 2021, mainly due to a larger production truck fleet as haul distances increases with depth.

Two 75 hp fans will be installed in series in the airlock system near the portal in order to counter the aspiration effect of the pull system. This will also minimize the formation of ice in the ramp access. A maximum of 25 m^3/s (52,000 cfm) of fresh air will be pushed by these fans through the airlock system.

Since the fresh air will be flowing in through two raises, two heating systems will be installed at these locations. A propane heating system with a capacity of 32 MBTU/hr (suitable for 142 m^3/s) will be installed on top of the portal raise. For the main fresh air raise, two propane heating systems with a total capacity of 64 MBTU/hr (284 m^3/s) will be installed. A glycol heating system will be installed in series ahead of the propane heaters near the main fresh air raise entrance. The capacity of this glycol heat recovery system is estimated to be in the magnitude of 21.6 MBTU/hr. The glycol heating system will use the waste heat from the power plant generators which will be transported via a propylene glycol loop to the heater location just ahead of the propane heaters.

During the development phase, the ramp will be ventilated by auxiliary fans and rigid ventilation ducting to the face. A ventilation raise will be excavated alongside the ramp, which is designed such that the raise will connect with the ramp on every level. Auxiliary fans and ducting will be moved lower into the mine as new raise connections are developed.

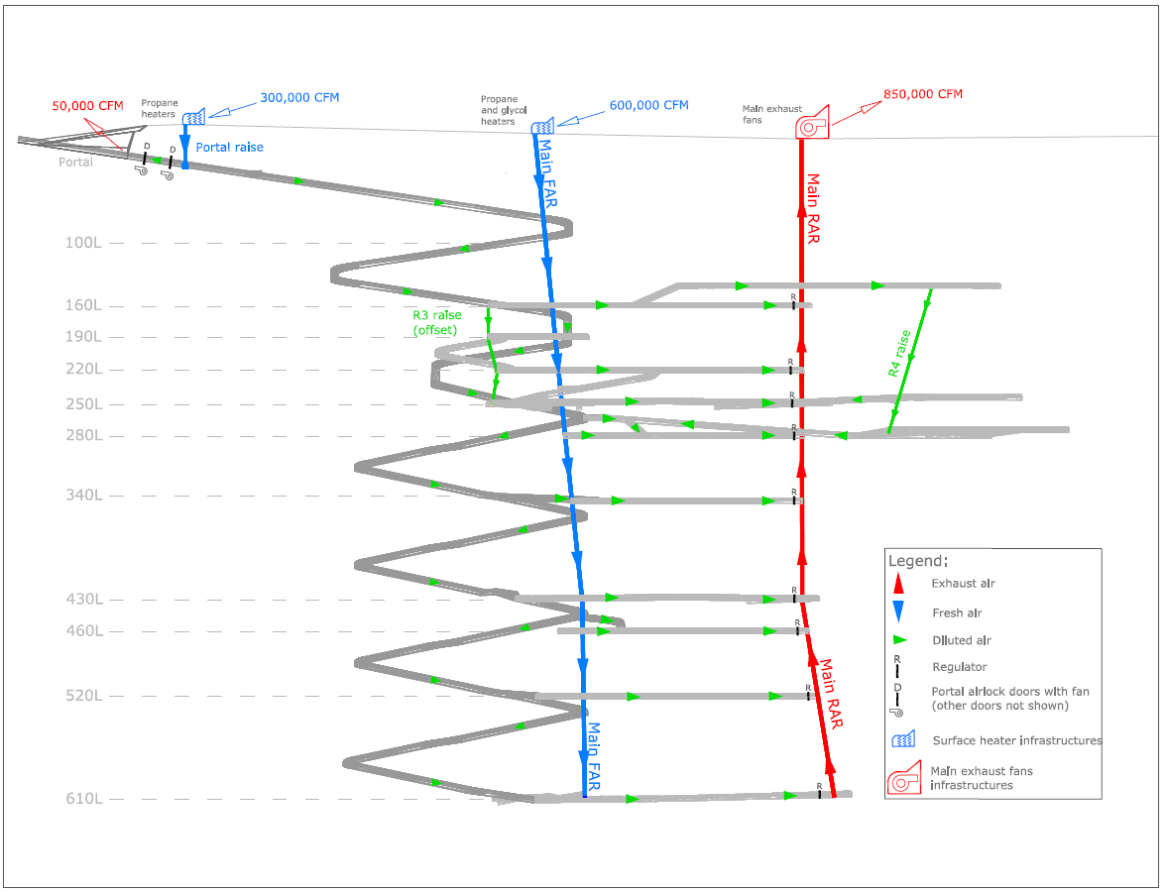


Figure 16.8: Main ventilation network schematic

16.5.10 Underground Infrastructure and Services

Electrical power will be provided throughout the mine for drills, jumbos, fans, pumps, lighting, and other miscellaneous loads. The power will be distributed to the mine at 4.16 kV through three primary feeders, two located in the fresh air raise (3C 250 MCM) and the third in the ramp. All three feeders will be supplied from the power station electrical room. A substation located on surface will be fed from the power house electrical room containing BUS A and B and having 2000 amp capacity. From there, power will be distributed to the mine surface facilities including the main exhaust fans.

Compressed air will be provided from a surface compressor only for the underground refuge stations as required by Quebec mining regulations. A single two inch line will provide compressed air from surface through the fresh air raise to the permanent refuge stations on the 250L, 430L and the 610L. Compressed air for production and development equipment such as ITH production drills, and small equipments such as hand-held drills, pumps, etc. will be provided by portable compressors located near the working areas underground. The required compressor capacity is estimated at 6,200 cfm. The portable compressors will be electrically

powered for the ITH drill compressors and diesel powered for the mobile compressors assigned to other development or production activities. Development equipment, such as jumbos and bolters, will utilize onboard compressors for compressed air generation.

Process water is required underground for drilling, dust suppression, and wash-down of equipment and rock faces for geology and mining. The underground requirements have been estimated to be 5.2 l/s. Process water will be supplied from the project process water distribution system on surface, and will be fed underground through 102 mm (4") pipelines in the fresh air raise and ramp. Potable water for underground use will be provided in bottles and will be available at the refuge stations.

Water will enter the underground mine from several sources including ground water inflows into mine openings, precipitation into the R2 and R3 open pits and process water. A hydrogeological study based on hydrogeological modeling was completed by Itasca to estimate ground water inflows. The Itasca model considered the mining sequence and schedule and estimated the inflows over time. The inflows reach an initial peak in 2017 with 2,300 m^3/day (421 USgpm) after 37 months of development and then gradually drop down to a long term inflow varying between 1,200 m^3/day (220 USgpm) and 1,600 m^3/day (294 USgpm) before reaching a life-of mine overall peak of 2,475 m^3/day (3454 USgpm) late in the mine life when mining on R2 610L horizon begins. Process water requirement is estimated at 450 m^3/day (82 USgpm). Precipitation is variable and could add 100 to 400 m^3/day. Based on these estimates, the pumping system has been designed with a nominal pumping capacity of 2,725 m^3/day (500 USgpm) operating at 60% utilization, so that peak inflows of up to 4,500 m^3/day could be accommodated on a short term basis.

A three-stage dewatering system is planned. Sumps and pumping stations will be located on the 250L, 430L and 610L with water pumped from the lower pump station being rehandled at the upper station. It is anticipated that approximately 30% of the mine water inflows will be intercepted and contained at the 250L and 430L as mining progresses deeper. Consequently, the 610L lower pumping station have been designed for 1960 m^3/day (360 USgpm) while the 250L and 430L upper stations will be capable to pump the full predicted flow of 2,725 m^3/day (500 USgpm). All pump stations will have two pump arrangements installed, one operational and the second on standby for maintenance. The 610L pump station will be a four pump arrangement capable of pumping water the vertical distance between the 610L and the 430L. Both the 250L and 430L pump stations will be five pump arrangements, the 430L capable of pumping to the 250L and the 250L pumps to surface. The sumps on the 250L, 430L and 610L will be horizontal sumps in series, with a dirty water sump to permit settling of fines overflowing to a clean water sump feeding the pumps. Clean water will be pumped from the 430L and 610L to the sump on 250L, and then pumped to surface. Tertiary sumps will be provided on the ramp just down grade from each level entrance and additional sumps will be provided on the individual levels as required. A gravity drainage system will be established whereby drainage holes will be drilled to connect the levels and channel mine water down to the sump on the nearest level.

Communications will be provided by a leaky feeder radio system to be installed in the ramp and the main levels. The system will be capable of transmitting both data and voice as well as high speed internet and telephone. In addition, conventional telephone service will be provided to the refuge stations, maintenance facilities and fueling stations.

Emulsion explosives will be used almost exclusively for both development and production blasting. Most of the emulsion will be supplied as a bulk product with a minor amount as packaged explosive. The bulk emulsion will be delivered from surface storage to underground in 1,500 kg containers on specialty trucks provided for this purpose. From surface, the trucks will deliver loaded containers directly to the workplace or to one of the explosive magazines underground. Two sets of explosive and detonator magazines will be provided underground. The main magazines will be located off the ramp close to surface and the second set will be located off the ramp just below the 430L. Separate explosive magazines will be provided for bulk explosives and packaged explosives. The bulk explosive magazines have been designed to store 40 containers for a total capacity of 60,000 kg of emulsion. The packaged explosive magazines are designed to hold just over 1,000 cases of explosive for a capacity of just under 20,000 kg. Detonators will be stored in a separate magazine.

A diesel fuel storage and distribution fuelling station will be provided on surface to service the site. All underground vehicles coming to surface via the ramp will be able to fuel up from this facility. In addition to this surface facility, a fuelling system to deliver fuel underground through a pipeline will be provided. On surface, the fuel delivery system will consist of a 100,000-litre tank and a 500-litre surface transfer tank located in the vicinity of the gensets. Underground fuel bays will be constructed on all main production levels; 250L, 430L and 610L. Each fuel bay will be equipped with a 1,200-litre receiving tank and dispensing equipment. The transfer of fuel underground will be done through a 25 mm diameter double wall piping system, on a batch basis and will be fully instrumented to ensure safe operation. The pipeline on surface will be a buried line with at least a -1% grade to the fresh air raise intake structure and will be installed down the fresh air raise with a valve located such that it can be delivered to either of the three fuelling stations. At each of the levels, the fuel line will be installed in the drift to the fuelling station location. The delivery system will be a “dry” system in which there is no fuel in the pipeline except when a batch of fuel is being transferred. The underground fuelling stations have been located close to the main exhaust raise as a fire safety precaution and will be equipped with fire doors and a fire suppression system. Mobile vehicles such as service vehicles and trucks will fuel directly at the fuelling stations but for slow moving or more remote equipment, a fuel truck will be used to deliver fuel directly to the work heading as required.

The main maintenance facility for the site will be located on surface and will be used to service all surface and underground equipment. While major repairs and rebuilds will be performed in the surface shop, three satellite repair areas, one on the 250L, a second on the 430L and the third on the 610L, will be provided for minor repairs and preventative maintenance. Each facility will include a service bay and a wash bay. Each service bay will be 8.0 m wide, 6.5 m high by 20 m long.

Refuge stations will be located at various strategic locations throughout the mine. A permanent refuge station will be constructed on each of the three production levels (250L, 430L, 610L) with a 24 person capacity. Between the production levels, three additional commercially supplied portable and totally equipped refuge station units with an eight person capacity will be provided. These stations will be placed just off the ramp on the drill levels and relocated as required as the location of mining activities changes over the life of the mine.

The ramp being the primary egress, the secondary egress from the mine will be provided via a ladderway installed in the fresh air raise. The mine fresh air intake will be downcast so in the case of a fire, it will always be in fresh air and a safe escape route. In the rare case of a fire in or

near the raise, personnel will report to the nearest refuge station on the 250L, 430L or 610L or a portable refuge station if that is closer.

A system to warn workers of fire or other emergency, in addition to the telephones and mobile radios, will be provided. The warning will be given by injecting ethyl mercaptan into the mine ventilation intake airway at the fresh air raise.

16.5.11Development Schedule

The total mine development requirement including ramp, levels, and raises included in the life-of-mine (LOM) schedule is shown in Table 16.12. During the LOM, a total of 32.9 km of development will be excavated of which 5.0 km will be in initial or pre-production capital with the remainder in operating and sustaining capital. This development will produce 0.6M tonnes of kimberlite ore and 1.4M tonnes of waste.

The development is also categorized as capital, operating and ore. Capital development includes all infrastructure development such as, ramps, ventilation raises and main levels. Operating is development that is used to directly access the stopes from the levels and includes stope access, drawpoints, and production drill drifts. Kimberlite (ore) development is identified separately, but is part of operating development.

The pre-production capital development is all the development required for the underground mine to be able to produce 6,000 tpd on a consistent and sustainable basis with all the ore handling facilities constructed and commissioned. This includes the operating development accomplished during that period, since all activities are capitalized until production starts.

The ramp development will be advanced to minimize interference and associated delays. The rate of ramp and level development will increase gradually during 2014 until 2015 and then stabilize in 2016 as the capital development required for 250L production is completed and the development transitions to sustaining work. Development will progress at an average rate of 3525 m/year until 2020, when the maximum development of 4680 meters is scheduled. From 2021 until 2023, the development rate decreases and one development crew remains active after the first quarter of 2021 to complete all lateral development by the beginning of the third quarter of 2023. The strategy to develop the mine is to retain a specialist contractor to develop the ramp down to the 280L with Stornoway supplied equipment. Stornoway crews will then assume the development of the mine levels and the remainder of the ramp development to the 610 L. Stornoway will be responsible for the excavation of all production raises but will retain outside contractors for the raisebore exhaust and fresh air raises. Slot raises for stope blasting will be excavated by Stornoway crews using the Machine Roger ITH raise head.

Table 16.12: Underground Development Requirements and Schedule

Development Type	Capital Initial (m)	Capital Sustaining (m)	Operating (m)	Total (m)	Preproduction Period 2014 (m)	Preproduction Period 2015 (m)	Preproduction Period 2016 (m)	Life of Mine 2016 (m)	2017 (m)	2018 (m)	2019 (m)	2020 (m)	2021 (m)	2022 (m)	2023 (m)
Ramp	2,066	2,774	-	4,840	1,190	876	-	686	701	897	490	-	-	-	-
Production Main Level Access (Outside CCR)	1,660	5,304	-	6,964	-	1,268	392	326	1,075	1,074	1,164	600	1,030	35	-
Production Main Level Access (Inside CCR)	-	277	-	277	-	-	-	-	-	277	-	-	-	-	-
Drill Level Access (Outside CCR)	-	3,499	-	3,499	-	-	-	-	514	501	515	874	-	493	602
Drill Level Access (Inside CCR)	-	570	-	570	-	-	-	-	-	329	98	100	-	43	-
Drawpoint Drift	1,152		7,110	8,262	-	453	699	85	1,239	-	1,223	1,740	1,250	1,573	-
In-Ore Drill Drift	-		6,318	6,318	-	-	-	928	761	1,167	965	1,366	559	81	491
Safety Bays	-	-		-	-	-	-	-	-	-	-	-	-	-	-
Truck Loading/Tip XC/Substation etc	-	125		125	-	-	-	-	100	-	-	-	-	25	-
Raise Ventilation 4m Dia	500	732		1,232	-	125	375	-	183	183	-	366	-	-	-
Raise Ore Pass 3.3 x 3.3m Alimak	30	213		243	30	-	-	-	30	60	-	-	-	-	123
Raise Waste Pass 2.7 x 2.7m Alimak	-	-		-	-	-	-	-	-	-	-	-	-	-	-
Vent Shaft 6.1m dia Raise Bored with Ladder	255	368		623	43	212	-	55	129	32	152	-	-	-	-
Vent Shaft 6.1m dia Raise Bored	-	-		-	-	-	-	-	-	-	-	-	-	-	-
Total Ramp and Lateral Development	4,878	12,549	13,428	30,854	1,190	2,597	1,091	2,025	4,390	4,245	4,455	4,680	2,839	2,250	1,093
Total Shafts and Raises	785	1,313	-	2,098	73	337	375	55	342	275	152	366	-	-	123
Grand Total	5,663	13,862	13,428	32,952	1,263	2,934	1,466	2,080	4,732	4,520	4,607	5,046	2,839	2,250	1,216
Development Tonnages															
Waste	463,590	516,070	819,078	1,798,737	146,882	231,501	85,208	166,341	297,655	294,751	219,834	145,878	78,026	73,181	59,482
Kimberlite	31,814	-	759,125	790,939	-	18,701	13,113	16,871	73,435	78,340	141,662	208,768	121,158	87,200	31,690
Total	**495,405**	**516,070**	**1,578,202**	**2,589,676**	**146,882**	**250,202**	**98,321**	**183,212**	**371,090**	**373,091**	**361,496**	**354,646**	**199,184**	**160,380**	**91,172**

16.5.12 Production Schedule

An annual summary of the underground production schedule by pipe and level is shown on Table 16.13. Initial production blasting in the R2 pipe on the 250L is scheduled to start, at a modest production rate of 550 tpd in October 2016 with the commencement of drawcone blasting. This rate will be maintained until January 2017 when this will increase to 2200 tpd for the next six months, after which it jumps to the full production rate of 6000 tpd in July 2017. The underground production schedule has been designed to coordinate with the open pit production schedule so that production will continue at the full rate of 6000 tpd during the transition period from the completion of the open pit through the build-up to full production from underground.

Underground production will continue from mid-2016 until early 2027 for a mine life of slightly more than ten years. For the R2 pipe, the 250L mining area will be completed in the first quarter of 2019, the 430L area in 2022, and the final 610L area in 2026. The R3 pipe has been scheduled at a rate of approximately 328,000 t/yr from the second quarter of 2018 through 2020. The first production from R4 starts in 2025 and continues to the end of the mine life in early 2027.

Table 16.13: Underground Production and Backfill Schedule

Mining Area		2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027
R2 Pipe														
R2-250	Tonnes (x 1000)	19	59	1,615	1,464	108	-	-	-	-	-	-	-	-
	Grade (cpht)	65	61	67	61	67	-	-	-	-	-	-	-	-
R2-430	Tonnes (x 1000)	-	-	29	183	1,585	1,618	1,250	410	-	-	-	-	-
	Grade (cpht)	-	-	81	88	74	74	72	60	-	-	-	-	-
R2-610	Tonnes (x 1000)	-	-	-	-	14	522	910	1,663	2,128	2,160	1,223	68	-
	Grade (cpht)	-	-	-	-	125	95	102	85	93	75	105	131	-
Subtotal R2	**Tonnes (x 1000)**	**19**	**59**	**1,644**	**1,647**	**1,707**	**2,141**	**2,160**	**2,073**	**2,128**	**2,160**	**1,223**	**68**	**-**
	Grade (cpht)	**65**	**61**	**67**	**64**	**74**	**79**	**85**	**80**	**93**	**75**	**105**	**131**	**-**
R3 Pipe														
R3-250	Tonnes (x 1000)	-	-	10	102	-	-	-	-	-	-	-	-	-
	Grade (cpht)	-	-	83	83	-	-	-	-	-	-	-	-	-
R3-220	Tonnes (x 1000)	-	-	7	287	-	-	-	-	-	-	-	-	-
	Grade (cpht)	-	-	100	90	-	-	-	-	-	-	-	-	-
R3-160	Tonnes (x 1000)	-	-	-	124	453	19	-	-	-	-	-	-	-
	Grade (cpht)	-	-	-	34	95	76	-	-	-	-	-	-	-
Subtotal R3	**Tonnes (x 1000)**	**-**	**-**	**17**	**513**	**453**	**19**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
	Grade (cpht)	**-**	**-**	**90**	**75**	**95**	**76**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
R4 Pipe	**Tonnes (x 1000)**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**87**	**32**	**-**	**937**	**2,092**	**571**
R4-280	**Grade (cpht)**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**42**	**42**	**-**	**41**	**44**	**41**
Total	**Tonnes (x 1000)**	**19**	**59**	**1,661**	**2,160**	**2,160**	**2,160**	**2,160**	**2,160**	**2,160**	**2,160**	**2,160**	**2,160**	**571**
	Grade (cpht)	**65**	**61**	**67**	**67**	**78**	**79**	**85**	**78**	**92**	**75**	**77**	**46**	**41**
	Carats (x1000)	**12**	**36**	**1,116**	**1,440**	**1,692**	**1,711**	**1,826**	**1,690**	**1,997**	**1,616**	**1,665**	**1,000**	**233**
Note: Development ore included in totals														
Backfill	**Tonnes (x 1000)**	**-**	**-**	**866**	**1,283**	**1,336**	**1,096**	**1,379**	**1,320**	**1,426**	**1,426**	**1,426**	**1,426**	**575**

16.5.13Equipment Fleet

The mine has been designed to take advantage of modern drilling and haulage equipment and efficient ore handling systems and mining methods in order to decrease operating costs. This includes the specification of 12 t capacity LHD's to load, haul and transfer stope production to the truck loading stations for direct loading onto 60t trucks. The same equipment will be used for waste development. High productivity down-the-hole (DTH) production drills for the BHS stopes and electric hydraulic top hammer drills for the stope undercut drilling are planned. In addition, a full suite of support equipment will be provided for development and service requirements.

The proposed underground mobile equipment fleet types for the mine are shown in Table 16.14. Some of the development equipment will be required immediately when underground mining activities start in 2014 with one heading active and the remainder in a year when the second development heading will start. However, the production equipment will not be required until mid-2016, when production activities begin.

Table 16.14: Underground Mobile Equipment Fleet

Typical Equipment Type	Supplier/Model	Total Number of Units
Scooptram 5.7 cubic meter	CAT R1700	7
60t Haulage Truck	CAT AD60	12
Two Boom Development Jumbo	AC Boomer 282	3
Scissorlift Truck	Maclean Mine Mate SL3	2
Scissorlift Truck with Cable Boom	Maclean Mine Mate SL3	1
Scissor Rockbolter	Maclean Series 900	3
Explosives Loader	DUX S1 Emulsion Loader	2
Shotcrete Pump and Boom Truck	Putzmeister SPM4210 & MM BT3	1
Boom Truck	Maclean Mine Mate BT3	2
ITH Drill c/w rod handler, booster	Cubex 6200HH, PT50 RH	3
V30 Cubex Reaming Kit	Machine Roger V30	1
Production Longhole Drill	Atlas Copco Simba 1354	2
Blockhole Drill	Maclean Mine Mate BH3	1
High Pressure Water Jet	Maclean Mine Mate WC3	1
Fuel/Lube truck	Maclean Mine Mate FT3	1
Forklift	CAT 906H2	2
Road Compactor	CAT CS56	1
Grader	CAT 12M	1
Backhoe	CAT R420F	2
Mobile compressor	Doosan XP375WJD	2
Underground Emergency Vehicle	Modified Ford F-250	1
Personel/flat deck cassette truck	Maclean Mine Mate CS3	2
Mechanics Service Vehicle	Toyota Landcruiser	2
Electricians Service Vehicle	Toyota Landcruiser	1
Supervisor/Technical Vehicle	Toyota Landcruiser	12
	Total	**45**

16.6 Material Balance Summary

The entire plant feed schedule is the culmination of ore production from the Renard 2/Renard 3 open pit and the Renard 2, Renard 3 and Renard 4 pipes in the underground mine. Figure 16.9, Figure 16.10, Figure 16.11 and Figure 16.12 illustrate the plant feed, diamond production schedules, open pit and underground mining schedules.

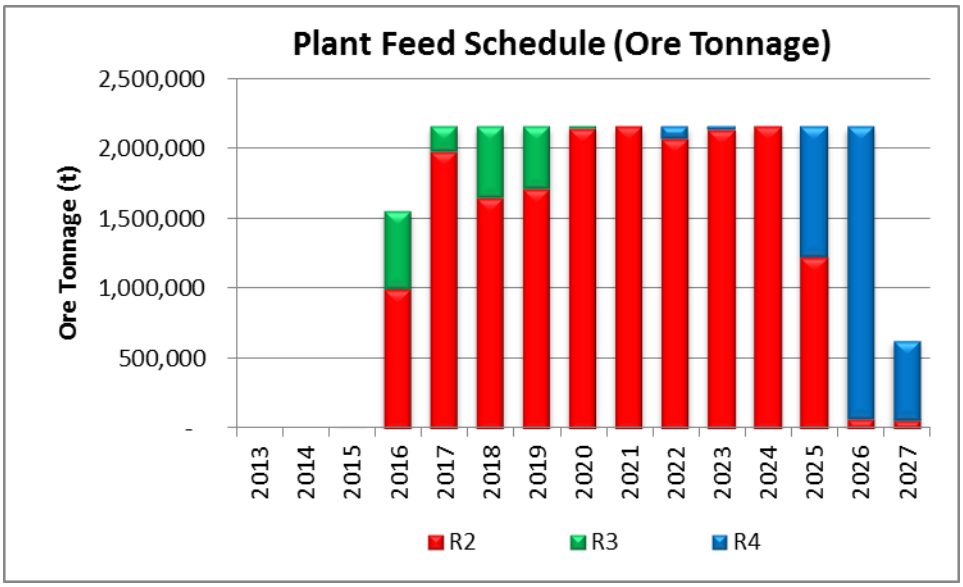


Figure 16.9: Plant Feed Schedule

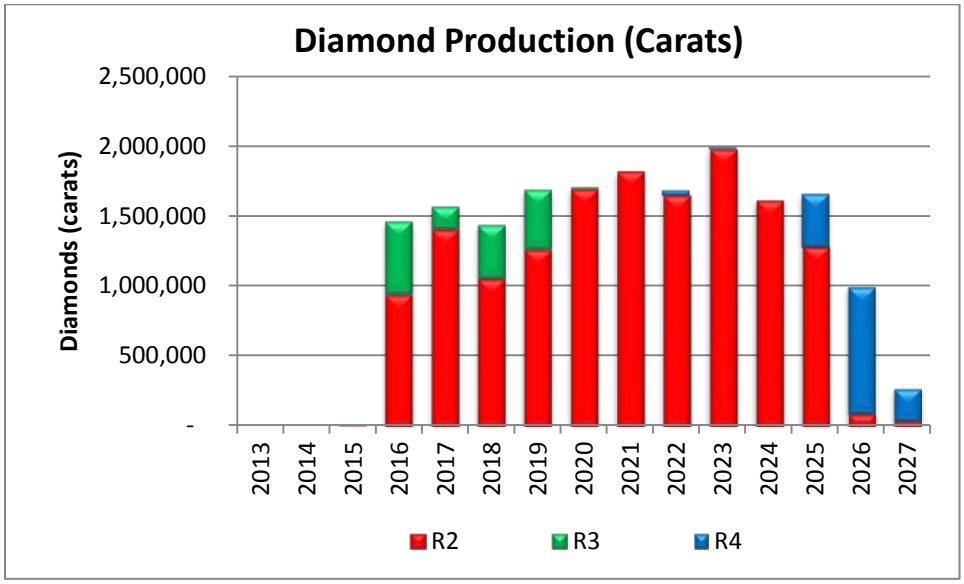


Figure 16.10: Mine Feed Diamond Production (Carats)

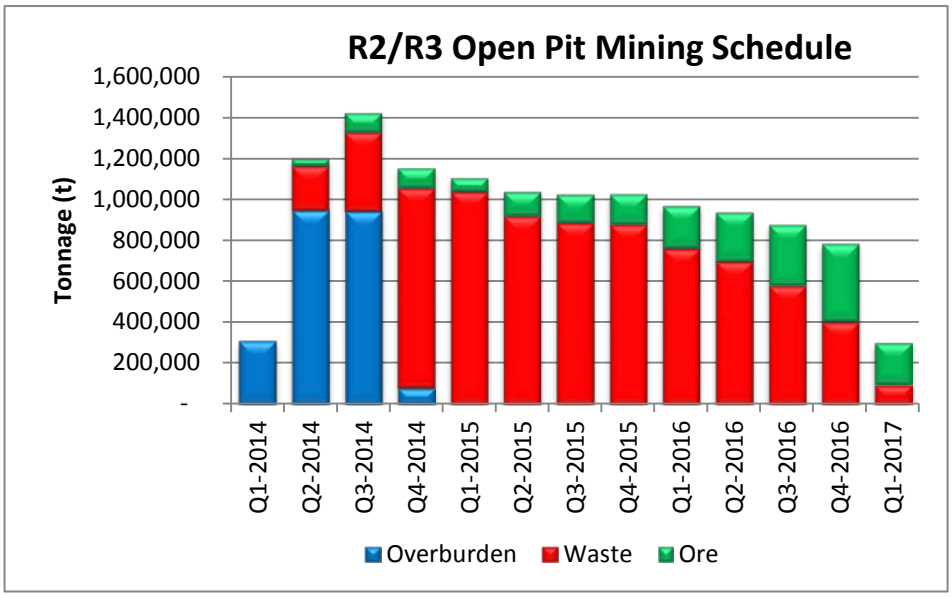


Figure 16.11: Renard 2/Renard 3 Open Pit Mining Schedule

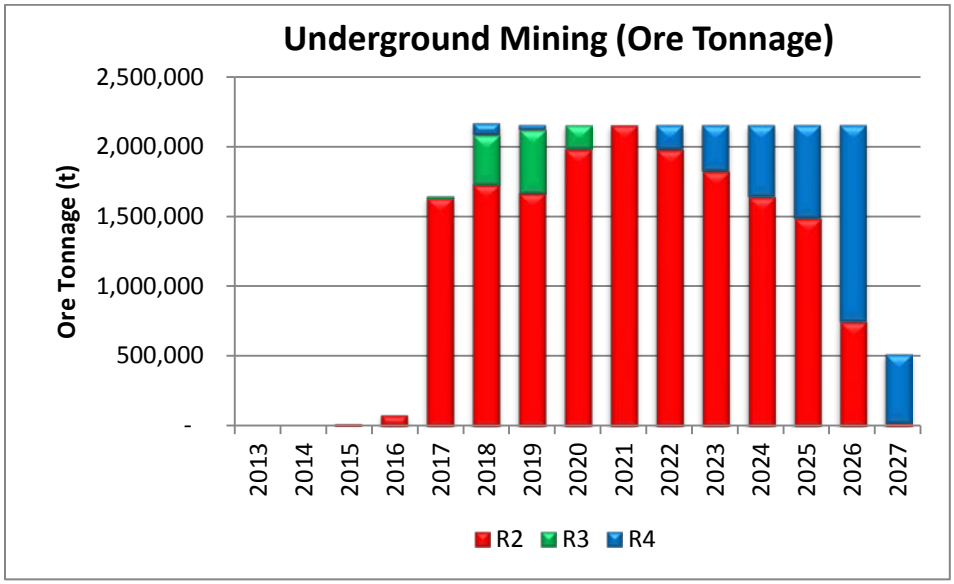


Figure 16.12: Underground Mining Schedule

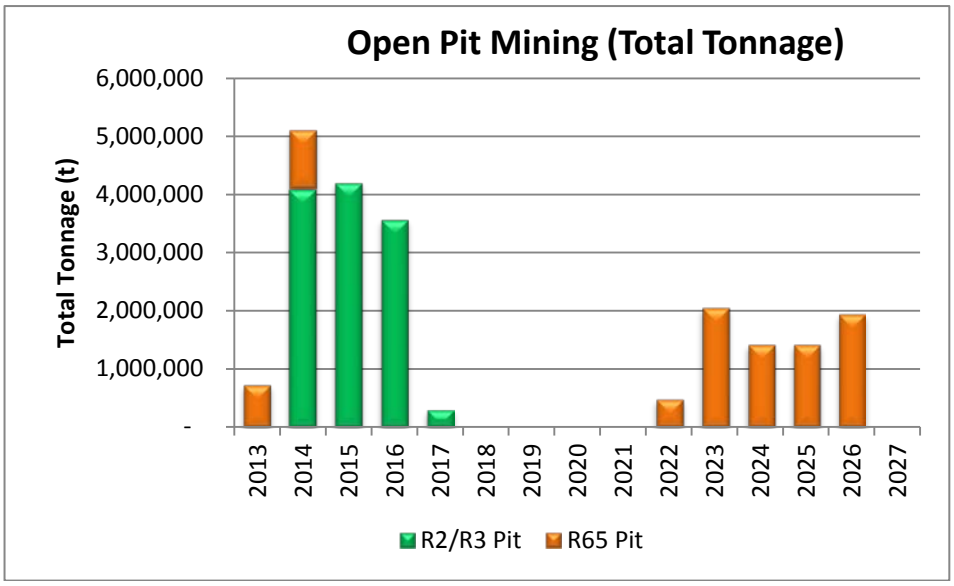


Figure 16.13: Open Pit Mining (Total Tonnage)

17.0 RECOVERY METHODS

17.1 Process Flow Sheet

The process flow sheet in Figure 17.1 was developed based on the following key aspects:

- The material characteristics observed, and data obtained from the treatment of underground samples from Renard 2, 3 and 4 in Stornoway's Lagopede bulk sample plant;
- Specific unit operation test work;
- Conventional diamond processing techniques as successfully employed in Canada;
- Liberation effectiveness;
- Circuit simplicity;
- Cost effectiveness; and
- AMEC Americas Limited's (AMEC) extensive diamond processing experience.

17.2 Plant Design

To minimize footprint and capital costs the diamond process plant design has a single process line for comminution and ore preparation. Capacity considerations dictate that two DMS lines for concentration and three lines for fines dewatering are required. All process equipment including storage bins and materials handling equipment is housed within a heated building, heated transfer towers or heated conveyor galleries.

The process plant design capacity is 2,160,000 tpa (dry solids basis). The plant overall utilization is estimated to be 78%, equivalent to an operating time of 6,833 hours per year or 315 tph and 6,000 tpd. The ore processing system liberates concentrates and recovers diamonds from 1 mm to 30 mm. Once the plant has been ramped-up to nameplate capacity AMEC believes that with a high standard of maintenance, plant overall utilization could be increased by about 5%.

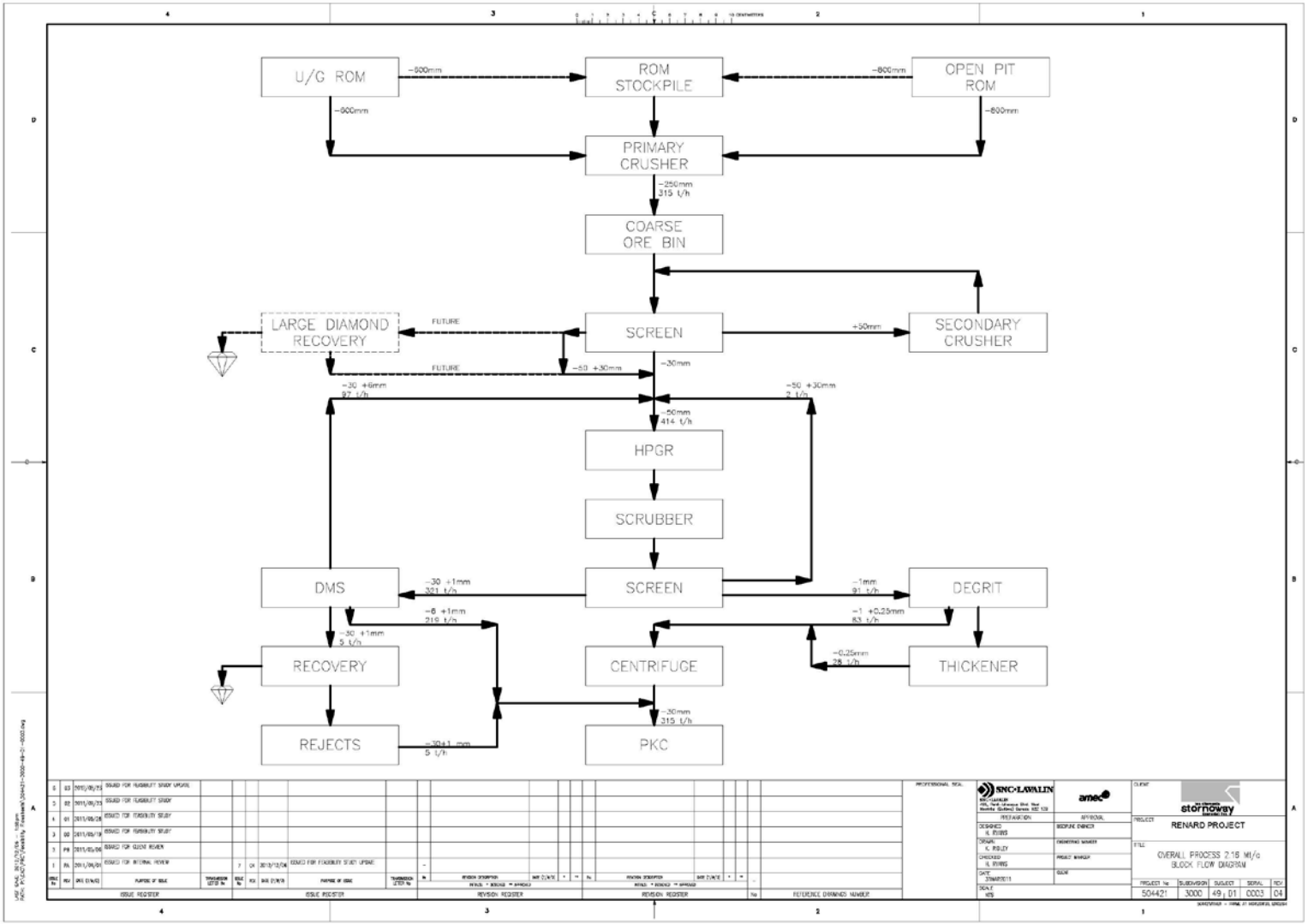


Figure 17.1: Block Process Flowsheet

Liberation design utilizes three stages of crushing where each crusher operates with a large gap to protect against diamond damage. Primary crushing is performed by a jaw crusher, which reduces the size of Run-of-Mine ore to less than 250 mm. Secondary crushing is performed by a cone crusher to reduce ore to less than 50 mm. A HPGR promoting interparticle crushing is used for tertiary crushing and is the principal diamond liberator generating a product size of 80% smaller than 8 mm.

The plant design considers the addition of a future large diamond recovery circuit to recover diamonds between 30 mm and 50 mm with x-ray sorting technology.

Ore preparation design utilizes a rotary scrubber and vibrating screen to wash, de-agglomerate HPGR product and size ore for DMS concentration. The majority of the unwanted fines (-1 mm) are separated from ore in this circuit and then are pumped to the rejects disposal circuit.

A surge bin with a capacity of 400 tonnes decouples the DMS from the ore preparation circuit.

Two 200 t/h DMS circuits concentrate -30 +1 mm feed by separating liberated diamonds and heavy minerals from lighter kimberlite and waste constituent minerals, in a density controlled ferrosilicon medium. The DMS rejects stream is sized into two streams; -30 +6 mm, which is combined with fresh feed reporting to the HPGR and -6 +1 mm, which is mixed with dewatered fines and discarded as dry stackable material to the processed kimberlite containment area.

A concentrate surge bin with a capacity of 80 tonnes decouples the recovery plant from the DMS circuits. The recovery plant is capable of processing at a maximum rate of 6.8 t/h.

Concentrate fed to the recovery plant is sized, then upgraded, using two stages of x-ray sorting per size fraction. X-ray rejects are fed to grease table scavengers to recover diamonds missed by the x-ray sorters and to audit x-ray sorter performance. Grease table rejects are discarded to the processed kimberlite dry stacking area together with plant fines and coarse rejects. X-ray concentrate is dried and further upgraded using magnetic separation and low throughput x-ray sorters for sizes less than 8 mm. Only a low throughput x-ray sorter is used to upgrade sizes greater than 8 mm. Rejects are recycled to the recovery plant feed.

Final diamond recovery is achieved by hand sorting diamonds from waste in secure glove boxes, located within the security controlled and monitored recovery plant. The diamonds will be chemically cleaned before preliminary valuation and export for marketing and sales.

Security is a key element of the process plant design given the high value and small volume of the final diamond product. A security management system to detect, deter and reduce the possibility of diamond theft in conjunction with an automated diamond recovery process is part of the design.

In the rejects disposal circuit, fines rejects (-1 mm) are separated by cyclones into two sizes, a slimes fraction (-0.25 mm) and a grits fraction (-1 +0.25 mm). The slimes fraction is thickened to recover water for reuse within the plant and recombined with the grits fraction to feed dewatering centrifuges to produce dry stackable fines rejects. Fines rejects are mixed with DMS

coarse rejects and recovery plant rejects for disposal by truck to the processed kimberlite containment area.

The following facilities are included in the plant design:

- Main plant modular electrical rooms;
- Main plant offices and central control room;
- Recovery plant modular electrical room;
- Recovery plant offices and control room;
- Security offices, access control and search rooms;
- Metallurgical laboratory;
- Compressor room;
- Maintenance shop;
- Diamond cleaning room;
- Diamond stock room; and
- Diamond valuation rooms.

The level of automation within the process plant is appropriate, to minimize operator intervention and to optimize productivity and safety.

The process plant building has a fire water ring main supplying wall hydrants and fire water hose station distributed throughout the building. Conveyors running in tubular galleries are protected with automatic wet pipe sprinklers.

17.3 Materials Handling

In the first two to three years of mine life ore will be extracted from open pit operations and from underground development activities via a ramp in the Renard 2 and 3 orebodies, and delivered to the surface primary crushing facility. This facility will be located near the Renard 2 and 3 orebodies. Towards the end of open pit operations, ore extraction will transition from open pit to underground operations. Ore will be extracted from underground using the blast hole shrinkage mining method. Underground ore will be loaded to 60 tonne trucks and hauled to surface. Where possible, the trucks will direct dump into the surface plant feed bin, ahead of the primary crusher.

On surface, trucks from the open pit will direct dump into the plant feed bin or dump Run-of-Mine (ROM) ore on a working stockpile located close to the primary crushing station. Ore from underground will also be dumped on this stockpile if the plant is unable to receive ore. Stockpiled ore will be reclaimed by front-end loader and delivered to a 120 t capacity primary crusher feed bin fitted with a static grizzly. A hydraulic rock breaker will be provided to break oversize lumps or rocks unable to pass through the grizzly. Ore will be withdrawn by a variable speed apron feeder to feed a vibrating grizzly screen. Oversized material will discharge to a jaw crusher and crushed product will report to the same conveyor as the undersize from the grizzly

screen. The material from the jaw crusher and undersized material from the grizzly will be introduced into the plant.

Conventional belt conveyors will be used to transport plus 1 mm ore and waste within the plant. Conveyors linking the main building and the transfer tower will be housed in tubular galleries, which are insulated and heated.

Plant rejects will be dewatered and mixed to produce a stackable waste, which will be trucked from a load-out bin to the processed kimberlite containment area.

17.4 Consumption

Kimberlite processing to recover diamonds is essentially a mechanical process, with limited use of chemicals/reagents. The estimated consumption of the chemicals/reagents required for a 2.16 million tonnes per year plant is shown in Table 17.1.

Table 17.1: Consumption of Chemicals/Reagents

Chemical/Reagent	Consumption
Ferrosilicon	432 tonnes per year
Flocculant	7.6 tonnes per year
Caustic Soda	3 tonnes per year
Hydrochloric Acid	1,825 litres per year
Grease	3,500 litres per year
Degreasing Solvent	4,000 litres per year

18.0 PROJECT INFRASTRUCTURE

18.1 Geotechnical Investigations

This section provides a summary of the geotechnical investigations performed at the various project sites to determine the soil and bedrock conditions in order to provide input to infrastructure design and construction. A first set of geotechnical investigations was performed in the fall of 2010 and mainly focused on the process plant, airstrip, landfill and processed kimberlite containment (PKC) facility sites. Bedrock surface profiling in the plant site area was also performed using a drill rig. A second set of geotechnical investigation was completed during the summer of 2012and included the permanent camp complex, process plant and associated infrastructures, domestic waste water treatment site, trench landfill site (TLFS), airstrip, collection ditches, explosive storages site and mine waste water treatment plant.

Visual logging and sampling of soil and bedrock was performed during the advancement of boreholes and test pits. Geotechnical testing of select soil samples and bedrock core was also performed. Groundwater monitoring wells were installed at selected locations. The following is the summary of the geotechnical investigations:

- Process Plant site and camp complex: 72 bedrock profiling boreholes, 23 test pits, and 3 groundwater monitoring wells;
- Airstrip: 6 boreholes, 29 test pits, and 6 groundwater monitoring wells;
- Landfill: 7 boreholes, 8 test pits and 7 groundwater monitoring wells;.
- Domestic water treatment plant: 3 boreholes, 5 test pits and 3 groundwater monitoring wells and;
- Other infrastructures (collection ditches, explosive storage site and mine waste water treatment plant): 5 boreholes, 51 tests pits and 14 groundwater monitoring wells.

The soil and groundwater conditions discussed herein describe those encountered at the testing locations and at the time of testing; conditions may vary substantially from those encountered.

The general stratigraphy of the plant site, airstrip and landfill is topsoil over sand to sand and gravel over till over bedrock. Surficial peat deposits and outcropped bedrock are also present at the sites. The topsoil thickness varied from 0 m to about 1.2 m. The compactness condition of the sand to sand and gravel soils ranged from loose to very dense; very dense for that of the till. The depth to bedrock ranged from 1.0 m to 6.8 m at the plant site, 1.5 m to 8.5 m at the camp site, 16.8 m to 17.3 m at the airstrip, and 2.8 m to 3.8 m at the landfill at the drilling locations The bedrock depth could not be defined in the waste treatment plant site since no invesgation has reached the soil/rock contact.

At the time of the geotechnical investigation, the measured groundwater levels throughout the site was generally shallow (depth below 1 m) but can locally reach depths up to 9 m based on current information. All groundwater monitoring wells are periodically measured by Stornoway.

With provision for suitable groundwater control, no conditions of particular concern were encountered during the investigations.

18.2 On-site Infrastructure

This section describes the on-site infrastructure that will be required for the development of the Renard Project. The section discusses the necessary site preparation, water supply requirements, solid and liquid waste disposal, power requirements, fuel storage, service building requirements, communications requirements, airport facilities and explosive handling and storage. The locations of the plant site infrastructure and the accommodation complex are shown on Figure 18.1, while a more detailed plant site lay-out is shown on Figure 18.2. The airport, located some 10 kilometres south of the Renard Project is not shown on the figures.

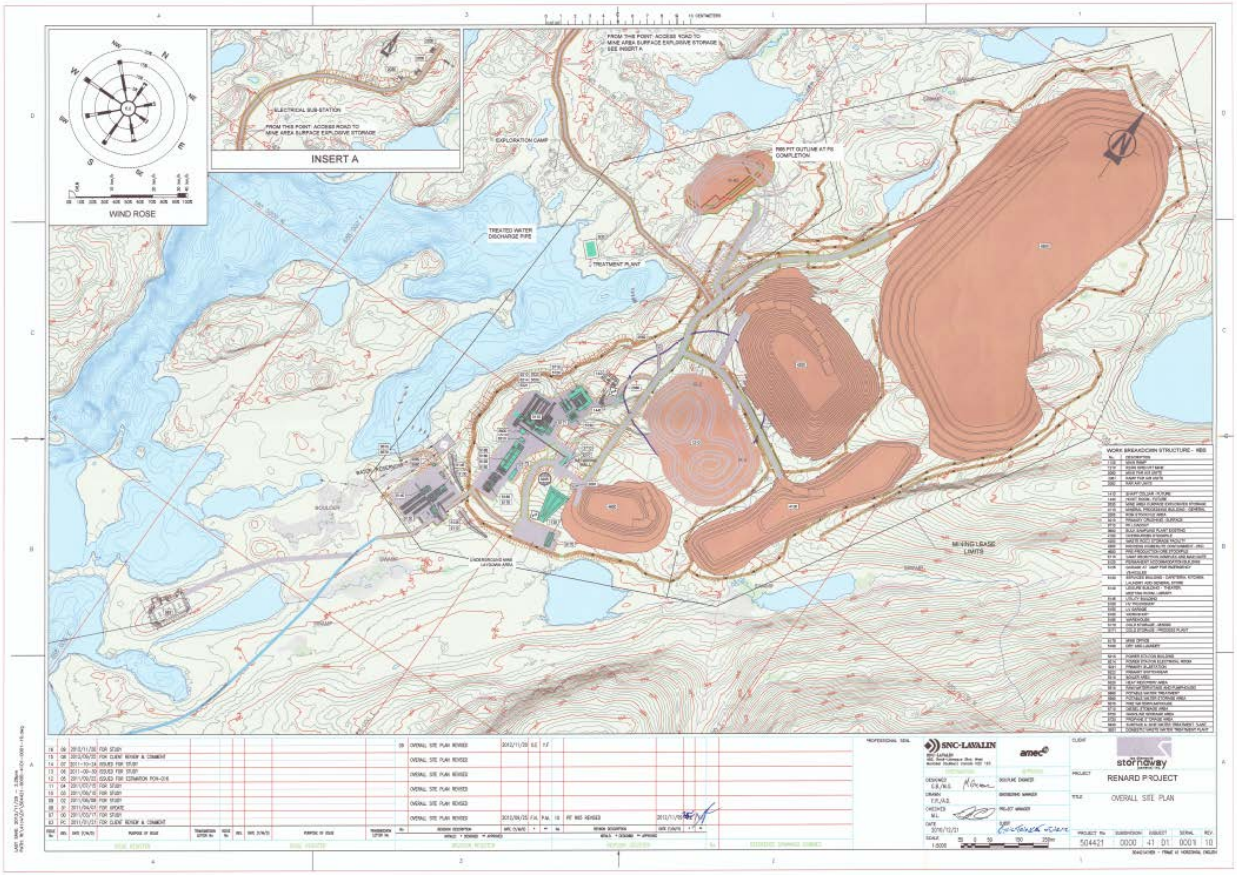


Figure 18.1: Locations of Plant Site Infrastructure and Accommodation Complex

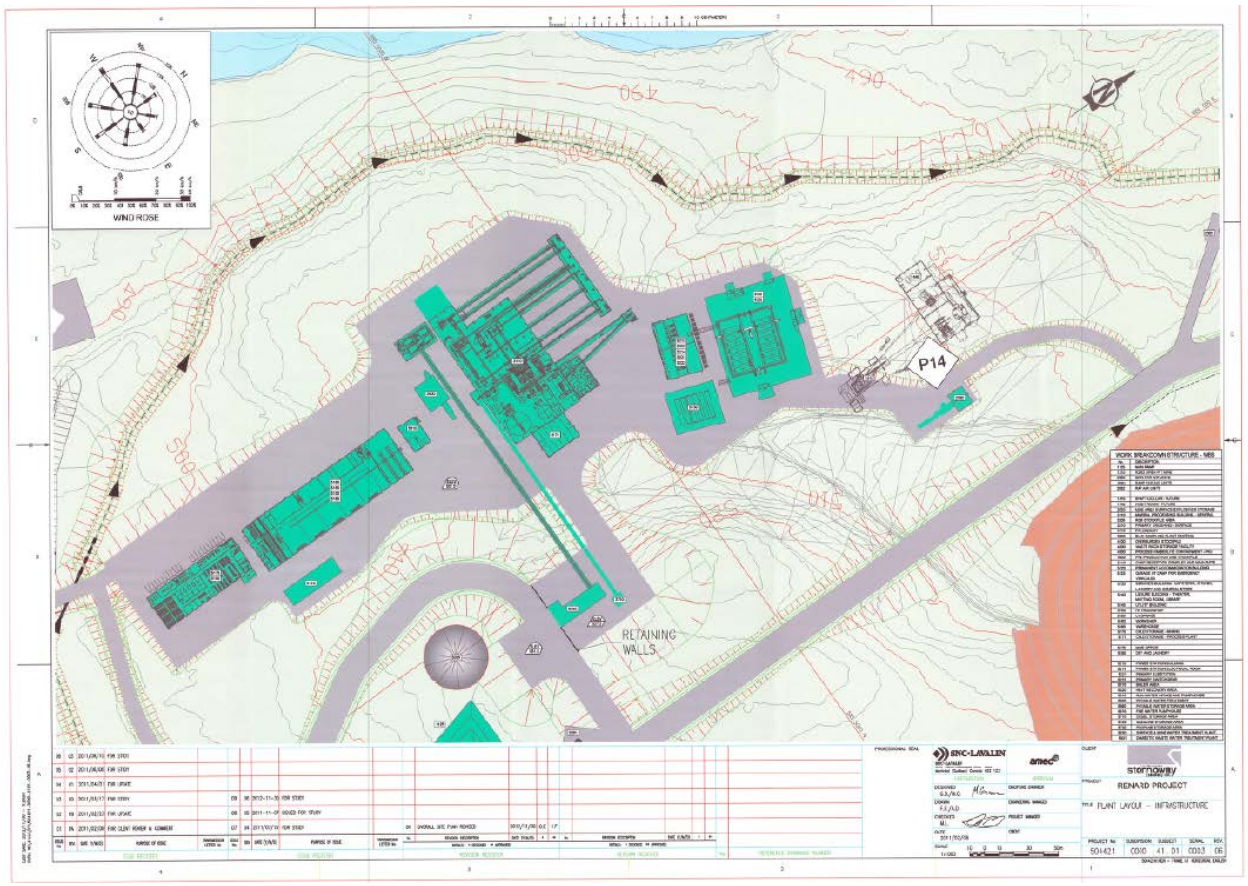


Figure 18.2: Detailed Plant Site Layout

18.2.1 Civil Work

The tree clearing work will be carried out over the total area and includes the areas for the process plant and infrastructure camp, the wastewater treatment site, the trench landfill site and the airport. The work of stump removal and grubbing, top soil removal and rough grading will be done only in areas where the infrastructure work is planned.

Three types of roads will be constructed, namely heavy-load roads (15 m wide), light-vehicle roads (7 m wide) and access roads (6 m wide).

Fences will be installed around the main inhabited areas to restrict unauthorized access and to prevent animals from entering the site. The area that will be fenced is the campsite.

18.2.2 Water Distribution

A raw water intake and pumping station will be installed in Lake Lagopède. It will be located near the accommodation camp, a few hundred metres from the main process area; consequently, the lake will be used as firewater storage. The same pumping station will also supply the potable water treatment plant. Based on field observations the treatment sequence

will comprise the following steps: sand & anthracite filtration, anionic ion exchange, activated carbon filtration, pH correction and UV disinfection followed by chlorination.

18.2.3 Wastewater Treatment

The effluent from the sewage treatment plant will be discharged into Lac Kaakus Kaanipaahaapisk (Lagopède Lake), downstream of the raw water intake.

The effluent standards for the Project will be established according to the characteristics of the receptor area, the loads of the sewage effluent and the uses to protect. The selected treatment system is facultative aerated lagoons, in a series composed of 3 lagoons having an overall retention time of 30 days and a chemical de-phosphating system. A sludge drying bed is planned for the dehydration by freeze-thaw of the sludge which will accumulate in the lagoons.

18.2.4 Solid Waste Management

An outdoor sorting center will be set up for the management of recyclable waste. Containers will be used on the site for recycling. Composting of kitchen waste is not considered for the moment due to the presence of bears in the sector. Separated containers will be used for the storage of hazardous wastes. These containers will be sent to southern Québec for recycling and/or disposal.

A trench landfill system with a volume of 40,000 m^3 is considered for other solid waste management. This volume takes into account the waste which will be produced during the construction, operation and closing phases, plus the volume of soil to be used for regular covering of the waste. Granular materials will be stockpiled in the vicinity of the landfill such that waste will be covered regularly.

18.2.5 Power Plant and Electrical Distribution

The power plant design will include a heat recovery system, fuel day tanks, exhaust systems, storage cells and chargers, 4.16kV synchronization switchgear and a power management controller. The plant will consist of 10 generators with 8 units running, 1 unit undergoing planned maintenance and 1 unit on standby. The generators will operate at 4.16kV, 3 phases, 60Hz, synchronized together on common switchgear. The rated generator power factor will be 0.8. The generators will operate on Arctic-grade diesel fuel..

The main power generation plant will feed the medium voltage distribution network at 4.16 kV. This main switchgear will be housed in a main electrical room located near the power generation shelters. An aerial distribution network at 4.16 kV will feed the buildings on the site, and pole-mounted or pad-mounted step-down transformers, as required by the connected loads, will reduce the voltage to 347/600 V, 3 phases, 60 Hz. Uninterruptible power supply (UPS) units will be provided to the accommodation complex, mine office and airport for telecommunication equipment, fire alarm system, security system and closed circuit television (CCTV).

Emergency generators will be used to support critical loads during power outages at camp buildings.

18.2.6 Heating

Underground mine air will be heated with heat recovered from the cooling systems of the generators in the power plant. Propane burners will supplement mine air heating. Heat recovered from the generators will be used for the generation of heated water/glycol for air intake heating of underground ventilation system. The main buildings will be heated by the propylene glycol network heated by fuel fired boilers.

Two primary glycol networks are needed since the camp complex is far from the power generator facilities. Consequently, the heat recovery from the accommodation camp utility building will be used for the accommodation complex and the power generator facilities will provide for the others.

The other buildings will be heated either by electricity or by forced-air furnaces.

18.2.7 Fuel Storage

A tank farm will be constructed to store arctic-grade diesel, unleaded gasoline for light vehicles and propane tanks will be located near usage points. The entire tank farm will be mostly automated and equipped with a communication system and a warning system.

In addition to fuels stored on-site at the tank farm there will be a jet-fuel and diesel storage facility at the airport. This facility will include a double-walled tank which will be equipped with integrated loading/unloading pumping systems.

18.2.8 Service Buildings

The camp complex and airport terminal will be wooden structural modular buildings. They will be factory built and assembled at site. The cold storage and the primary crusher enclosure will be fabric-covered buildings consisting of a light structural steel frame covered by PVC/polyester membrane. All other buildings will be either self-supporting, pre-engineered steel buildings or conventional pre-engineered steel buildings. In either case the envelope will be prefabricated at the factory and installed on site.

Air conditioning will be installed in the cafeteria, the leisure building and the office building.

First line fire protection will be by portable ABC type extinguishers installed in all buildings. The cafeteria, the truck shop, the office building, the power plant generator building and the process plant and conveyor galleries will also be completely protected by a system of automatic water sprinklers.

An aqueous film forming foam (AFFF) fire suppression system is required for the tank farm.. The confinement dyke under and around the tanks will be compartmentalized to reduce the quantity of foam required. If a fire occurs, the trailer for the production of foam will be

transported to the site of the fire by the fire truck. Two fire hydrants will be located around the tank farm to supply the fire truck and foam trailer with water.

Accommodation Complex

The accommodation complex will house the reception area, which will also serve as the main gate to the site, the service area for the cafeteria, dormitories, one leisure building, a utility building and a garage for emergency vehicles.

The complex will be made of wooden structural modular buildings except for the leisure building, the utility building and the garage for emergency vehicles which will be self-supporting pre-engineered steel buildings.

An access ramp will be provided for physically challenged personnel to access the camp reception area, first floor of dormitories, leisure modular building, and service building.

Camp Reception Complex

This building will be the main entrance to the site. It will serve for security control and as the camp reception area. The security control area will consist of offices, a reception desk for ground traffic, and a security control room including an x-ray scanner for luggage, metal detector for personnel and search rooms for passengers travelling by aircraft.

The camp reception area will consist of offices, a reception desk to register and direct people, and a closed room for lockers to store the belongings of workers in transit.

Dormitories

The dormitories will consist of individual furnished rooms. Each room will be equipped with a bed and night table, a desk and a chair, a closet, a bathroom with shower, a commode and a sink. Each room will also have an IP phone, an internet connection and a TV. Two types of rooms will be provided: those with private bathrooms and those with a shared bathroom between two rooms. The complex will provide 62 private rooms and 168 semi-private rooms.

Emergency Vehicle Garage

This heated building will house the fire truck and ambulance.

Service Building

The service building will provide the area for food service with a seating capacity of 224 people. The service will be cafeteria style with a self-service area. It will be equipped for food preparation, washing, all required storage and a reception area. The cafeteria will be ventilated and air conditioned by a variable volume system. A make-up air system will be provided to compensate for the kitchen exhaust hoods. An insulated and refrigerated room will be provided for garbage storage.

Leisure Building

The leisure building will be fully equipped for physical training activities (weight lifting equipment, stationary bicycles, etc.) and leisure activities (billiards and cards tables, fireplace, television room, cinema, and a veranda).

Utility Building
This building will house the heating and hot water production equipment for the camp complex.

Heavy Vehicle Truck Shop, Light Vehicle Garage and Workshop
This single building will be used for maintenance of heavy and light vehicles (HV and LV). The building will also include a wash bay, welding shop, instrument shop, electrical shop, mechanical maintenance shop. Space over the LV garage and shops will be used as office area which will include the server room and control room for site infrastructure.

Warehouse
A single storey warehouse with racking is joined to the HV truck shop, LV garage and workshops.

Office Building
The office building will consist of two stories. The ground floor will house the mine rescue area, the infirmary, a training room, a laundry room, the women's locker room, the men's locker room, a prevention office, the lamp room and the foremen area.

The second floor will contain the reception area, offices, the training room, the photocopy and paper room, a meeting room, a lunch area, the computer room, two storage rooms, the telecommunication room and heating room.

18.2.9 Technical Facilities
The following buildings will be constructed as technical facilities:

Core Shack
This building will be used to log and analyse core samples. A separate storage area will be available for core storage.

Boiler Area
This building will house the equipment for heating and hot water production for the site except the camp complex which will have independent equipment. The primary glycol heating network of the site comes from this boiler room and includes the two oil fired packaged boilers, pumps, expansion reservoir, and a high pressure glycol unit.

Raw Water Intake
This small building will house the pumps.

Potable Water Treatment
This prefabricated modular building will house the water treatment equipment, pumps and a small office.

Wastewater Treatment
This building will house the equipment needed for the wastewater treatment. A small office with a laboratory is also integrated into this building.

18.2.10 Telecommunications

A single satellite dish antenna with dual low-noise block-down converters (LNB's) block-up converters (BUC's), modems and related equipment will be provided at the mine site for telephone and data transmission. Access to the link will be through separate VLAN's for the administrative and recreational use.

The airport communication systems will be serviced from the mine site by way of a fiber optic cable link. This will afford surveillance of the remote airport site while it is unmanned.

The satellite TV and radio system will be designed to receive and broadcast 12 commercial channels selected by the client. In addition, equipment will be provided to allow local DVD's and CD's to be broadcast.

IP Telephone service will be installed at all site locations where warranted for operation and/or emergency communications needs. The industrial areas of the site (mill, power plant, garages etc.) will be equipped with more robust telephone sets and cabling, where warranted.

The CCTV and access control system will provide coverage of the exterior of the mine complex, the accommodation complex and the airport complex. The Access Control System will include gates and card readers for site access of these three complexes. In addition, since the airport is located at a considerable distance from the main site, CCTV coverage is required to maintain site monitoring at all times.

18.2.11 Access Control

All vehicles accessing the site via the main road have to pass through the main gate located at the accommodation complex. All access checks will be conducted at this location. Once access is permitted, vehicles can move freely (under CCTV monitoring) in both complexes (accommodation and process complexes). The other access gates will only be equipped with a camera and a proximity sensor to allow automatic opening of the portal.

Like the accommodation complex main gate, the airport gate allowing access to the apron will be equipped with a card reader, an intercom and a fixed camera. Security gates will be installed not only to restrict access for vehicles, but also to block the movement of animals, such as bears, that might be present in the area. To do this, a rising fenced barrier will be installed.

The main monitoring station will be installed in the video surveillance office at the camp reception complex building. This station will monitor the main site and the airport.

18.2.12 Airstrip

The airstrip for the Renard Project will be owned and operated by Stornoway with chartered aircraft for its exclusive needs. An agreement will be made with the security services of that airport for the passengers and their baggage to be verified at the departure of flights to the Renard airport. The airstrip will not be certified by Transport Canada. The reference aircraft for the design of the runway is the DASH 8, Series 300 with a capacity of 50 passengers..

A gravel surface is proposed for the runway, taxiway and apron areas.

The airstrip will be equipped to support night operations and will be equipped with visual navigational aids and lighting.

The jet fuel tank will be on the east side of the apron. A 500 gallon reservoir of de-icing liquid will be housed in a shed, and equipped with a heating element.

A pre-fabricated wooden structural modular building will serve as a terminal. It will include an office with suitable visibility for Unicom the airport operator, one (1) caretaker's storage room and one (1) washroom. No personnel and baggage security screening will be installed in the terminal building. The screening and search of personnel will be done at the accommodation complex. Waiting areas will thus be at the accommodation complex, and no waiting room will be provided at the terminal.

A self-supporting pre-engineered steel clad structure building will be the maintenance garage for the storage of materials and the wheeled equipment used for the summer runway maintenance and winter snow clearing activities. All maintenance of equipment will be performed at the mine site truck shop.

A water well will be bored to supply water to the terminal building and maintenance garage. Bottled water will be used for drinking water. Wastewater will be treated by a septic installation, appropriately located with respect to the water supply well.

18.2.13Explosive Storage and Handling Area

Bulk emulsion will be stored on the emulsion site under federal regulations. Packaged explosives will be stored on the explosive site under provincial regulations.

The emulsion ISO container will be built to be heated with electricity when the outside temperature is too low and emulsion risks becoming too viscous. From this container, emulsion will be pumped into an emulsion truck for open-pit blasting or into 1500 kg bins for underground operations.

The wash bay garage is used to park and wash the emulsion truck.

The explosive storage site will consist of one (1) heated and lit explosive magazine and one (1) detonator magazine with a capacity of 100,000 detonators.

18.3 Off-Site Infrastructure

Overview

Land access to the Renard Project will be provided by the extension of provincial Route 167 and the Stornoway mining road. Work on the provincial route will be carried out by the *Ministère des Transport du Québec* (MTQ), while work on the mining road will be conducted by Stornoway.

The original scope of work called for a provincial road extension enabling access to the Renard site; however this plan was modified in November 2012. The new agreement between the Government of Quebec and Stornoway includes the Route 167 extension to kilometre 143 and a Class III mining road (the Stornoway mining road) to be built and operated by Stornoway, to reach the Renard site. Loan agreements are now in place between Stornoway and the Government of Quebec for the capital investment of $77 million for the mining road.

This infrastructure will afford year-round access linking the project to the municipalities of Mistissini and Chibougamau. The provincial road will be constructed to MTQ standards and will be a gravel-topped road, with a design speed limit of 70 km/h, while the mining road will be a one lane gravel topped Class III road with a design speed of 50 km/h.

Provincial Route 167 and Mining Road Layout

Figure 18.3 below illustrates the selected road layout for both the Route 167 and mining road. As per the original MTQ execution plan for the road, lots A and B define the Route 167 extension while lots C and D define the mining road.

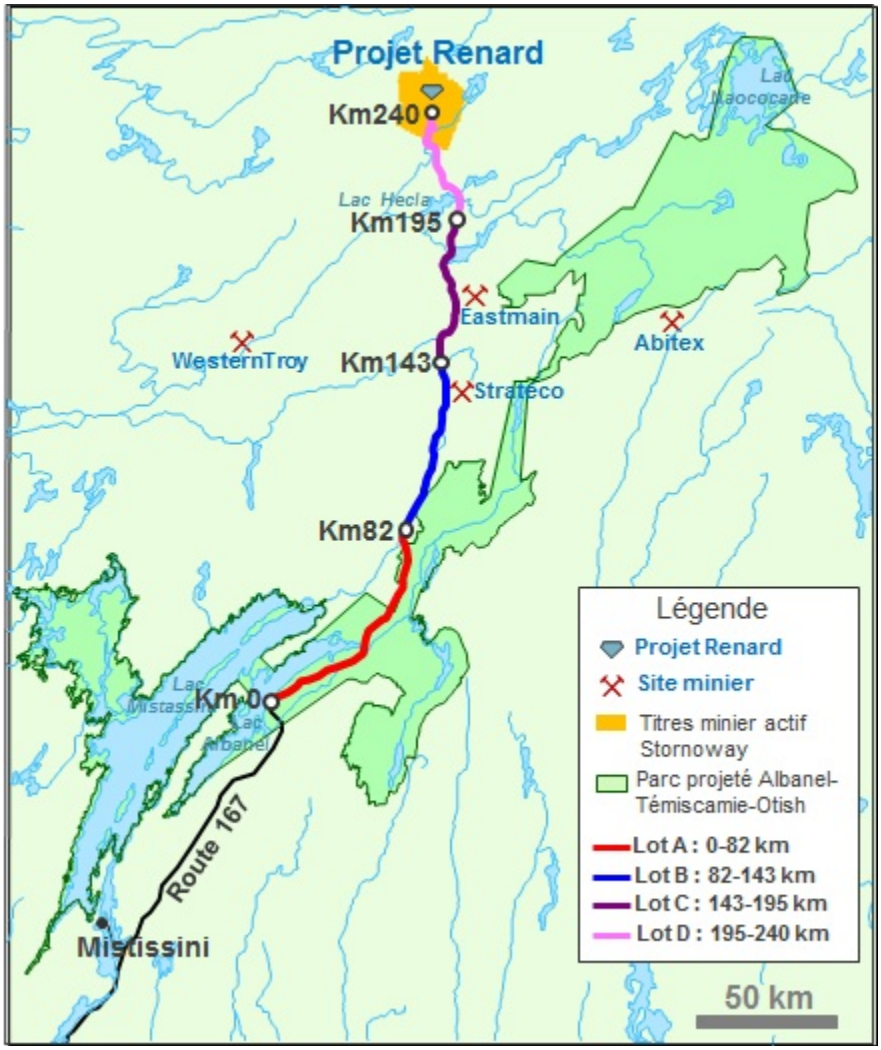


Figure 18.3: Road Location Map

Road Construction Strategy and Schedule

The MTQ's schedule currently includes the following milestones for Provincial road construction:

- Construction started in Q1 2012
- Rough access along lots A and B is planned for Q4 2012

Stornoway's schedule currently includes the following milestones for the mining road construction:

- Construction start is planned to for Q2 2013
- Permanent access to site is planned for Q4 2013

Road 167 Extension Costs

Following the November 2012 agreement signed between the Government of Quebec and Stornoway, all costs related to lots A and B of the Provincial Route 167 extension project are fully assumed by the Government of Quebec.

MTQ is responsible for the construction of a winter road leading from kilometer 143 to the Renard site during the 2012-2013 winter season. This temporary road access to Renard is needed for the mobilisation of mining road construction equipments and has to be completed by March 1st, 2013.

The applicable portion of the Route 167 Extension Certificate of Authorization (CA) is transferred to Stornoway. MTQ also authorizes Stornoway the right of passage on lots A and B, as well as the winter road, prior to and following completion of construction. MTQ will insure the permanent access on lots A and B for the 1st of April 2013. Stornoway and the Government have made a joint commitment to work on the social acceptability regarding the reduced scope of the road project

As part of the agreement, the Ministry of Finance and Economy have agreed to a two part loan (loan A and loan B):

- Loan "A" : unsecured credit facility of 77 M$ (including 15% contingency) to be provided to SWY by the Quebec's Minister of Finance for the construction of the Mining Road
- Loan "B": an additional 10% overrun facility with a right to draw 7.7 M$ @ 6.3% annual interest rate
- Termination of the two pre-existing Agreements dated August 1st 2011

Operating costs for the Provincial Route 167 Extension will be covered by the Quebec Government. Operating cost for the mining road will be entirely assumed by Stornoway.

18.4 Processed Kimberlite Containment Facility

This section provides a summary of the geochemical classification of the kimberlite, processed kimberlite (PK) and waste rock, site selection of the processed kimberlite containment (PKC) facility, and the design of the PKC facility.

18.4.1 Site Selection

A site selection study was performed to determine the location of PKC facility. The PKC facility was sized to provide sufficient storage capacity in a single location for all size fractions of the PK to be generated during the mining operations proposed in this report. Five candidate sites were initially identified for evaluation using economic, social, and economic indicators. The candidate sites are shown on Figure 18.4.

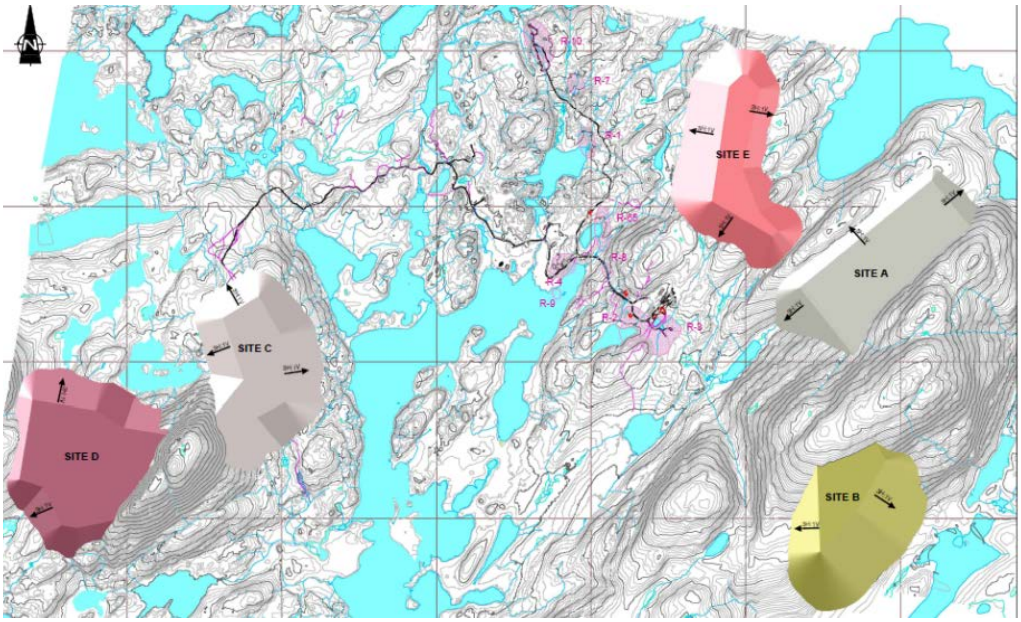


Figure 18.4: Candidate Sites for Processed Kimberlite Containment Facility

Each candidate site was visually inspected by a helicopter fly-over and a walk-through. The observations made during the site visit allowed confirmation that the topographic mapping of the candidate sites generally matched the field conditions. The preliminary facility layout for each candidate site was confirmed to be possible based on the observations made.

The site comparison and the final selection were accomplished with a decision support tool, GoldSET, which evaluated nearly 50 criteria, or indicators, grouped under environmental, social, and economic dimensions in accordance with pre-determined scoring schemes. It is important to note that the potential locations for the PKC facility are compared relative to each other; namely, the selected site is the “best” of those considered in the study.

The scoring scheme of each indicator is either quantitative or qualitative. Quantitative indicators are those with measurable quantities with a determined unit, for example length in metres. The indicator is progressive when an increase of the value corresponds to a favourable situation or regressive when a decrease of the value corresponds to a favourable situation. In order to proceed with the evaluation of the quantitative indicators, all values are normalized on a scale of 0 to 100. The best option receives a score of 100, while the worst option receives 0. The results of intermediate options are calculated in accordance with a linear function. In the case of the qualitative indicators, the scoring scheme is based from 0 to 100.

While the scoring scheme is set in advance and does not change from one project to another, the weighting of each indicator must be uniquely set for each project. The weighting is an evaluation of the relative weight between indicators in a specific dimension (environmental, social and economic) and the value varies between 1 and 3. The weighting is based on the importance of the indicator for the organization, the interest of the stakeholders for that same indicator along with the level of influence of the stakeholders.

Based on the results of the evaluation, Site E was determined to be the optimal location and is shown on Figure 18.5.

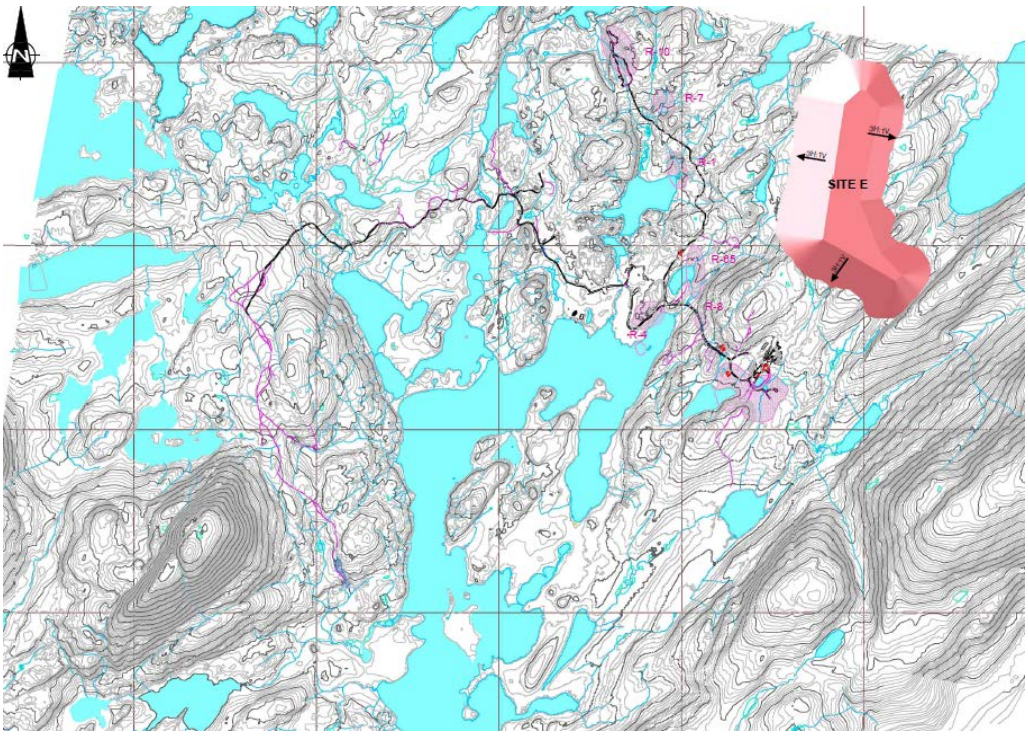


Figure 18.5: Selected Location (Site E) for Processed Kimberlite Containment Facility

18.4.2 Facility Design

The design of the PKC facility is based on the results of the geochemical classification of the PK (refer to Section 20.2) and those of the site selection (refer to Section 18.4.1). The development basis for the design is such that the PKC facility may be readily expanded from the scenario considered in this report, namely, the mining of all mineral reserves identified, to the mining of all mineral resources presented in this report.

Dewatering of the PK at the process plant was evaluated as part of the global assessment of the risk to the Project. The dewatering of the PK at the process plant by means of centrifuging was identified as the preferred option from environmental and economic points-of-view.

The dewatering of the PK at the process plant enables the development of the PKC facility as a dry stack. The three fractions of the PK will be mixed at the process plant and trucked to the PKC facility. This significantly reduces risk relative to a conventional slurry-type facility for facility development, operation, and management as no water is pumped to the facility. Progressive closure is a key element of the facility development and is facilitated by the dry stack basis. Water management of runoff and seepage from the facility is collected by a series of perimeter ditches and sumps for treatment, as required, prior to release to the environment.

The design objective for the PKC facility is that the facility must be capable of receiving all materials that may potentially be produced during mining operations. The meeting of this objective is paramount such that mining and/or processing operations are not shutdown as a result of the PKC facility development. The facility will be developed to promote progressive closure and provide stability over both the short-term and the long-term and without impacting fish habitat or bodies of water. Slope stability (both static and seismic conditions), wind erosion, and water erosion will be provided into the post-closure period through the development of the facility during operations. The PKC facility will be unlined based on the results of the geochemical classification of the PK (refer to Section 20.2) and the estimated seepage rate from the facility. The facility will be developed, for the most part, using the PK generated during operations. A closure cover comprising appropriately-sized rockfill and vegetated overburden will be progressively placed during the development of the facility.

The PKC facility is designed in recognition of several project, site, and environmental objectives. The objectives are that:

- The PKC facility provide adequate storage capacity for the anticipated quantity of PK to be produced;
- The entire life of mine storage is to be contained entirely within the footprint allotted for the PKC facility;
- That the PKC design may be readily expanded to encompass all currently identified mineral resources of the Renard Project;
- The design reduces the potential impact on the local groundwater;
- The design promotes the progressive closure of the facility;
- The design avoids potential impact on local fish habitat; and

- The contact water will be collected for appropriate treatment prior to release into the environment. This will be achieved through a series of perimeter ditches and sumps that will route and collect runoff and seepage from the facility for subsequent treatment, if required.

The PKC facility at the completion of the BFS scenario is shown Figure 18.6.

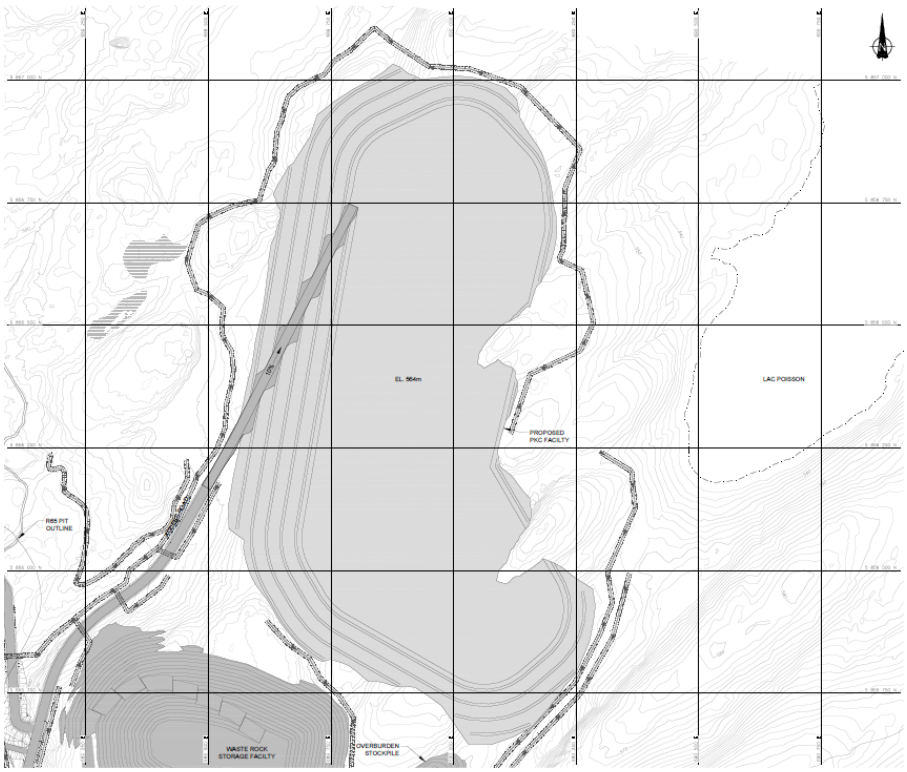


Figure 18.6: Processed Kimberlite Containment Facility - Completed BFS Facility

Construction, development, and progressive closure of the PKC facility, including activities such as foundation preparation, placement of engineered fill, placement of the closure cover and haul road construction being performed during non-freezing periods will be carried out continuously over the life of the facility.

Geotechnical instrumentation, comprising piezometers, thermistors, and survey monuments, will be installed and monitored through operations, closure, and into post-closure periods. The instrumentation will enable monitoring and assessment of the performance of the facility and to verify design assumptions.

18.5 Waste Rock, Ore & Overburden Piles

Overburden, waste rock and kimberlite ore will be stored in three separate piles. The specifications for these piles are presented in Table 18.1 and Figure 18.7.

The overburden pile is sized to include the overburden material mined from the Renard 2/Renard 3 pit and the overburden mined from the Renard 65 borrow pit. The total overburden to be stored is 3.36 Mt, or 2.2 million loose cubic metres since some overburden will be used to surface slopes of the PKC facility. The overburden pile will be constructed in 5 metre lifts and will be 20 metres in height. The slopes of the overburden pile will be armoured with rip-rap material to increase slope stability.

Table 18.1: Pile Specifications

Pile Specifications	Overburden Pile	Waste Rock Dump	Ore Stockpile
Lift height (m)	5	5	5
Lift offset or catch bench (m)	5	8.37	5
Pile face angle (deg)	26.6	37	37
Overall slope angle (deg)	18.4 (3H:1V)	18.4 (3H:1V)	23.3
Pile height (m)	20	52	10
Elevation of pile surface (m)	535	570	530
Footprint surface area (ha)	13.1	20.7	7.6
Volume of pile (Mm^3)	2.2	3.7	0.41

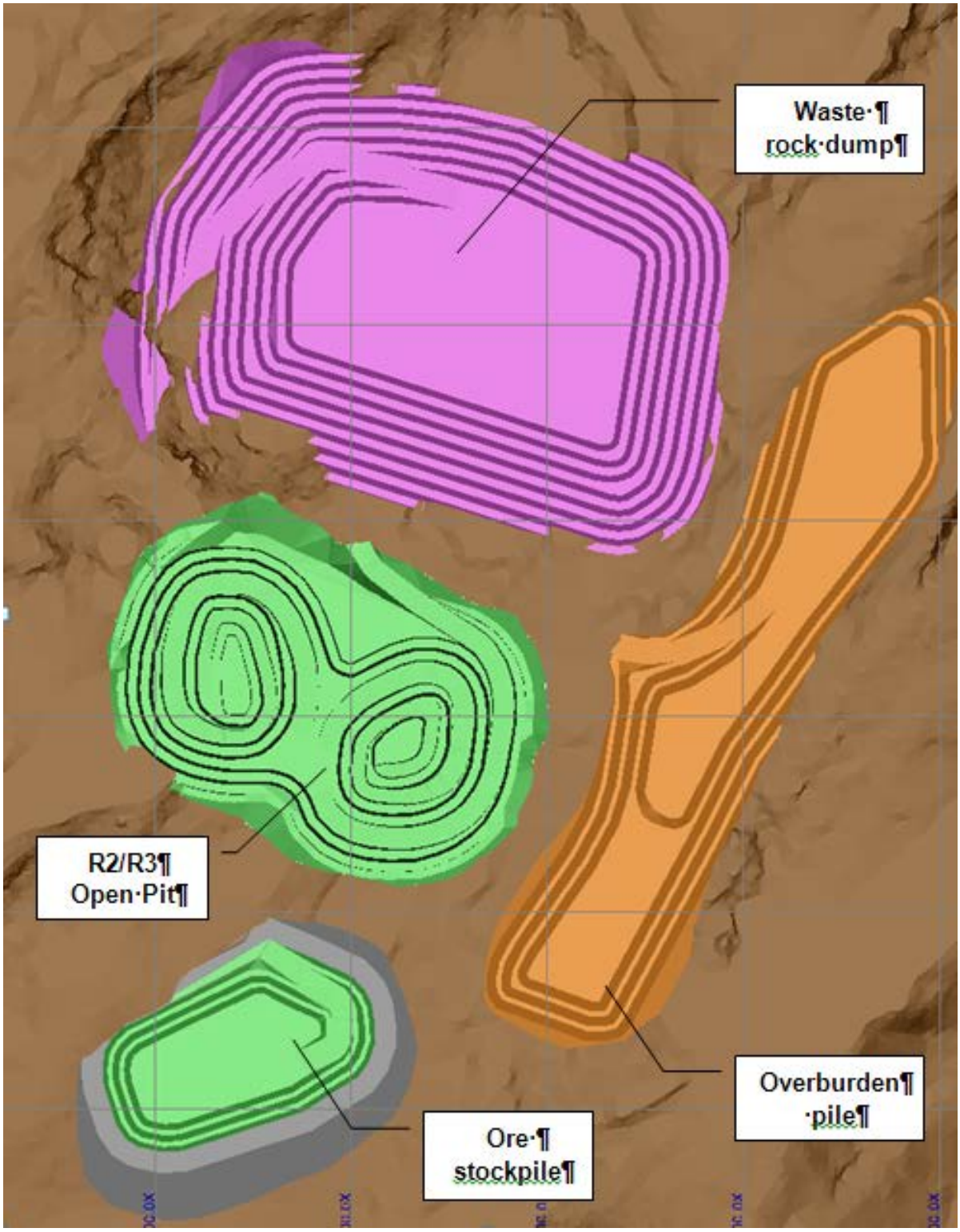


Figure 18.7: General Arrangement of Open Pits and Piles

The waste rock dump is sized to contain waste rock mined from the Renard 2/Renard 3 pit and some waste rock mined from the Renard 65 phase 1 borrow pit. The total waste rock pile at the end of April 2017 is 7.6 Mt. An estimated 3.25 Mt of waste rock will be used for various construction purposes over the Project life.

The waste will be re-handled and used as backfill into the Renard 2/Renard 3 pit, once the underground operation breaks through into the pit bottom, and into the Renard 4 mine through backfill raises. The Renard 65 phase 2 borrow pit will be mined to supply waste rock for backfilling once the waste dump has been completely exhausted.

The ore stockpile base will be built from a combination of waste rock and kimberlite from the Renard 65 phase 1 borrow pit. Building the ore stockpile base from low grade kimberlite material will prevent unnecessary dilution when the ore stockpile is re-handled and sent for processing. The base will be constructed with approximately 580,000 m3 of rock fill and is designed to accommodate approximately one million tonnes of ore.

19.0 MARKET STUDIES AND CONTRACTS

19.1 Diamond Pricing and Market Studies

19.1.1 Diamond Supply Forecasting

In most diamond supply and demand forecasts, future rough diamond supply is assessed on the basis of current and future production plans at the major producing mines. These supply projections are generally robust, as most of the world's major diamond mines are operating on a steady-state basis or transitioning to lower production rates as they age, and it typically takes between 8 and 12 years to find and develop a new diamond mine.

Bain and Company forecast a 27% increase in world diamond production on a volume basis between 2011 (when mine production is expected to have recovered to production cut-backs made during 2008-2009) and 2020 for a Compound Annual Growth rate of 2.7% (Bain, 2012). .

19.1.2 Diamond Demand Forecasting

Diamond demand forecasting (expressed as either demand for rough diamonds in the cutting centers or demand for polished diamonds for diamond jewelry manufacture) is a more complex calculation based on, amongst other things:

- Long term Gross Domestic Pproduct (GDP) growth forecasts in the main consumer markets;
- Diamond jewelry consumer growth trends in developing markets; and
- Short term inventory draw-down or re-stocking trends within the diamond pipeline, often linked to levels of bank debt.

The principal difference between published diamond demand forecasts is whether they are based purely on the first of these elements solely and assume steady state diamond demand linked to GDP growth or whether they also take into account the second and third of these elements, and allow for market changes outside of GDP growth and short term inventory factors.

All current diamond demand forecasts are strongly leveraged to International Monetary Fund-based GDP growth forecasting in India and China. Uncertainty exists on the current size of the Chinese and Indian markets, and their future growth rates. However, should the current growth rates in diamond jewelry consumption in China and India of between 10% and 15% be sustained, by 2020 these markets will exceed the size of the US market, which is expected to grow at a lower rate. Diamond jewelry growth is forecast to increase by 67% in nominal value between 2011 and 2020 (Bain, 2012). Without a commensurate increase in rough diamond supply, this must inevitably lead to an increase in diamond prices and a decrease in the relative quality of diamonds within a piece of diamond jewelry.

19.1.3 May 2011 Diamond Valuation Exercise

During the period of May 9th to May 13th, 2011, parcels of diamonds recovered from the Renard 2, 3 and 4 kimberlite pipes during the 2007 bulk sampling program (the "Valuation Samples") were presented for a valuation in Antwerp, Belgium under the supervision of WWW. The results of the valuation exercise are summarized in Table 19.1.

In addition to performing its own valuation, WWW showed the diamond samples to four other experienced rough diamond companies in order to obtain additional market based valuations (an "open market" valuation). For each Valuation Sample, the average of the five valuations was used by WWW to construct a diamond price model with "High" and "Minimum" sensitivities.

Diamond price models represent the true diamond price that might reasonably be expected for a kimberlite ore body based on standard commercial-scale recoveries of all diamond size classes. They differ from the achieved diamond valuation price principally through a correction which is applied for the absence of large diamonds which are typically under-represented in exploration scale samples.

WWW recommended the adoption of a single diamond price model for the Renard 2 and Renard 3 ore body valuation samples given the similarity of the diamonds in terms of diamond qualities and size distribution. The Renard 2 and Renard 3 base case price model is US$182/carat. A separate diamond price model was adopted for the Renard 4 valuation sample given its apparently finer distribution of diamond sizes. The Renard 4 base case price model is US$112/carat.

Table 19.1: May 2011 Diamond Valuation Results

Body	Valuation Sample (carats)	Achieved Prices[1] for the Valuation Samples					WWW Price Modeling		
		Number of Independent Valuations	Average of Independent Valuations (US$/carat)	Minimum of Independent Valuations (US$/carat)	Maximum of Independent Valuations (US$/carat)	WWW Valuation (US$/carat)	WWW Base Case Model (US$/carat)	WWW "High" Model (US$/carat)	WWW "Minimum" Model (US$/carat)
Renard 2	1,580	5	$173	$143	$195	$195	**$182**	$236	$163
Renard 3	2,753[2]	5	$171	$137	$195	$190	**$182**	$205	$153
Renard 4	2,674	5	$100	$87	$107	$107	**$112**	$185	$105

1 All Price Estimates Based on a +1 DTC sieve size cut-off

2 Includes 105 carats from Renard 3 sample 3105 not previously available for valuation

19.1.4 Review of Diamond Size Frequency Distributions

In previously published NI 43-101 Technical Reports ((Farrow, D. 2010) and (Farrow, D. and Farrow, D.J., 2011) a description has been given of successive studies into the apparent similarities, and dissimilarities, in diamond size frequency distribution exhibited by the Renard diamond bulk samples.

All previous studies and diamond valuation exercises have noted the similarity in size distribution and diamond quality between the Renard 2 and Renard 3 diamond samples, both between each kimberlite pipe as a whole and between each pipe's individual lithological units. In modelling the base case diamond price estimate for the Renard 2 and Renard 3 kimberlite pipes, a single size frequency model, based on the combination of the Renard 2 and Renard 3 Valuation Samples, has been adopted by WWW in the May 2011 valuation exercise.

In contrast to the similar diamond size frequency distributions of Renard 2 and Renard 3, the Valuation Sample collected from Renard 4 has a lower incidence of large diamonds and produces an apparently "finer" diamond size frequency distribution. In the valuation exercise of May 2011, WWW derive a lower base case diamond price model for the Renard 4 sample based primarily, but not exclusively, on a finer size frequency model for Renard 4.

A review of the of the Renard 4 diamond bulk sample process was undertaken in December 2009 (Volume II, "Analysis of Renard 4 Bulk Sample SFD and its implication for Diamond Price Assumptions", Stornoway Diamond Corporation, December 23, 2009) to attempt to explain the anomalous size frequency distribution of the Renard 4 Valuation Sample compared to the other bulk samples from the Project. The review addressed:

- Comparative size frequency distributions of the diamond sample data;
- Comparative diamond breakage analysis of the diamond sample data;
- Analysis of process plant performance for each sample and its potential impact on diamond sample recoveries; and
- Comparative diamond quality assortments of the Renard bulk samples.

The review confirmed that the diamonds in the bulk samples from Renard 2, 3 and 4 show a large degree of similarity in their size frequency distributions, particularly when using a cumulative stone probability or normalized grade per unit analysis. An internal analysis of comparative quality assortment data confirmed little difference in diamond assortment and value per size class between the Renard 2, 3 and 4 bulk samples. Deviations in the cumulative carat probability plot for the Renard 4 sample, from which a valuation model is derived, can be attributed to grade loss through diamond breakage or stone loss in the DTC sieve classes greater than 3GR in size. An independent analysis of diamond breakage in each diamond bulk sample, conducted by Dr. Paddy Lawless, an independent consultant, between August and September 2009, confirmed that there is an anomalous level of diamond breakage in the Renard 4 bulk sample compared to the Renard 2 and Renard 3 bulk samples. The relative proportions of this breakage, when quantified in terms of grade loss, are consistent with that implied by the size frequency distribution analysis. The Renard bulk samples were processed at the 10 tonne per hour dense media separation plant located at the Project site, and process

plant records confirm issues of concern with the processing of several Renard 4 bulk sample rounds that might have led to both diamond loss and diamond breakage.

Consequently, Golder (2010) concluded that the size frequency distributions of Renard 2, Renard 3 and Renard 4 are similar, and that it is appropriate to use a single size frequency distribution to determine value on the basis that there exists a single diamond population within the three kimberlite pipes. This approach was also adopted in the GeoStrat February 3, 2011 NI 43-101 Technical Report when the Mineral Resource statement was further updated.

Consistent with this approach, a base case price model of $164/carat for Renard 4 is derived through the application of the May 2011 WWW valuation price data for the Renard 4 Valuation Sample to the Renard 2-Renard 3 size frequency model.

19.1.5 Diamond Price Model Sensitivities and Diamond Price Assumptions

In establishing their diamond price models, WWW determine High and Minimum sensitivities on the base case diamond price based on alternate interpretations of diamond quality and potential value. The choice of “Minimum” and “High” to describe the sensitivity limits is deliberate: in WWW’s view it is highly unlikely that an actual diamond price achieved for each kimberlite ore body upon production would fall below the “Minimum” sensitivity, but it is possible that the actual diamond price achieved may be higher than the “High” sensitivity, which is not a maximum price.

For the May 2011 exercise, WWW also incorporated two contrasting interpretations of each sample’s potential diamond size frequency distribution in establishing the High and Minimum sensitivities. For each of Renard 2, 3 and 4, the alternate scenarios are:

Each kimberlite pipe has the same diamond size frequency distribution, represented by the sum of the Renard 2 and Renard 3 Valuation Samples (the “Common Size Distribution Scenario” and “Base Case assumption”).

Each kimberlite pipe has a different size frequency distribution represented accurately by each kimberlite pipe’s individual Valuation Sample (the “Individual Size Distribution Scenario”)

The combination of alternate quality and size distribution interpretations for each Valuation Sample give rise to sensitivities of US$163 to US$236/carat for Renard 2, US$153 to US$205/carat for Renard 3 and US$105 to US$185/carat for Renard 4.

As recommended by WWW, Stornoway has adopted a base case diamond price model of US$182/carat for the Renard 2 and 3 kimberlite pipes. Consistent with the prior NI 43-101 Technical Reports, Stornoway has adopted a base case diamond price model of US$164/carat for the Renard 4 kimberlite pipe based on the interpretation of similarity between the Renard 2, Renard 3, and Renard 4, diamond size frequency distributions. The financial analysis includes a sensitivity which estimates the effect of this assumption compared to the alternative interpretation.

19.2 Contracts

SDCI had previously agreed to a $44 million contribution to the cost of the extension of Route 167 payable with interest from July 1, 2015, with the first payment due on December 31, 2015. The Québec Government's original plan was to extend Route 167 to in four segments terminate at the Renard Project site. The Government subsequently decided to terminate the Route 167 extension at the end of the second segment. Under a Framework Agreement signed by SDCI and the *Ministère des Finances et de l'Économie du Québec* in November 2012 and a loan agreement signed in December 2012, SDCI agreed to build a class III forestry grade road (the "Mine Road") from the end of Segment B of the Route 167 Extension to the Renard Project site following completion of a winter road to be constructed over this route by the *Ministère du Transport du Québec*.

The total cost of the Mine Road before escalation is estimated to be $76.7 million (Table 19.2).

Table 19.2: Table: Mine Road

Category	Cost (real k C$)	Cost (nominal k C$)
Direct construction costs	58,696	59,695
Engineering and construction management	6,730	6,845
Subtotal	65,426	66,540
Contingency (15%)	11,251	11,443
Total	76,677	77,983

The cost estimate has been prepared by *Groupe Conseil Forchemex Ltée* on a +/-25% basis.

Escalation has been applied at a rate of 2% per annum from Q4 2012, for an additional $1.31 million.

There are two tranches to the loan agreement; Loan A for $77 million which is intended to fund the base cost of the Mine Road and Loan B, which represents an additional $7.7 million of overrun funding. Interest is payable on Tranche A at a rate of 3.35% per annum, and on Loan B at a rate of 6.3% per annum, compounded monthly in each case.

Loan A and Loan B are repayable in 12 annual instalments commencing at the end of the 48th month after first disbursement, which took place in December 2012. Interest accrues on Loan A and Loan B commencing in January 2016 and is payable annually in arrears commencing December 31, 2016.

Interest and principal payments for Loan A and Loan B, assuming the total escalated cost for the Mine Road is $77.98 million, are shown inTable 19.3

Table 19.3: Mine Road Agreement Payments (k C$, nominal)

	Loan A			Loan B		
	Principal	**Interest**	**Total**	**Principal**	**Interest**	**Total**
31-Dec-16	3,381	2,619	6,000	82	64	146
31-Dec-17	3,496	2,504	6,000	82	58	140
31-Dec-18	3,614	2,386	6,000	82	53	135
31-Dec-19	3,737	2,263	6,000	82	48	130
31-Dec-20	3,865	2,135	6,000	82	42	124
31-Dec-21	3,996	2,004	6,000	82	37	119
31-Dec-22	9,152	1,868	11,020	82	32	114
31-Dec-23	9,152	1,557	10,709	82	27	108
31-Dec-24	9,152	1,245	10,397	82	21	103
31-Dec-25	9,152	934	10,086	82	16	98
31-Dec-26	9,152	623	9,775	82	11	93
31-Dec-27	9,152	311	9,463	82	5	87
Total	**77,000**	**20,450**	**97,450**	**983**	**414**	**1,397**

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

Roche ltée, Groupe-Conseil (Roche) was mandated to carry out an Environmental Baseline Study (EBS) and an Environmental and Social Impact Assessment (ESIA) for the Renard Project. The EBS and ESIA studies were scoped to comply with the directives issued by Ministère du Développement durable, de l'Environnement et des Parcs (MDDEP) and Comité d'évaluation (COMEV).

Roche ltée, Groupe-Conseil (Roche) was awarded the mandate to carry out an Environmental Baseline Study (EBS) and an Environmental and Social Impact Assessment (ESIA) for the Renard Project. The EBS and ESIA studies were scoped to comply with the guidelines issued by Ministère du Développement durable, de l'Environnement et des Parcs (MDDEP) and Comité d'évaluation (COMEV).The Renard ESIA was filed with Québec and federal regulators on December 28th, 2011. It is a comprehensive document that incorporates many years of environmental and social data collection and studies. Its purpose is to allow the permitting regulators to properly assess the impact of the project and to seek inputs from local stakeholders on the proposed development. To this end, Stornoway has made available to the public on its website the full content of the ESIA and its associated EBS as well as the project's Closure and Rehabilitation Plan. During the ESIA Process and after the issuing of the above mentioned documents Stornoway has always maintained a pro-active communication strategy to explain the content of these documents.

The EBS includes information on the physical, biological and human environment of the Project study area (127 km²). This information was collected from various sources including data available in the literature (scientific literature, technical reports, government data, ESIAs and EBS conducted in the region), site specific surveys conducted in the Project study area (2003-2011), traditional ecological knowledge from Cree land users through their participation in the surveys, and information provided as part of the Environmental Exchange Groups and other meetings with land users. The following section provides a summary of the reports while Table 20.1 lists the various baseline studies conducted on the property since 2002.

The ongoing ESIA process is based on the most up-to-date and detailed information available on the physical, biological and human environment and on the project description. Collaboration between the ESIA team and Engineering resulted in a project design that prevented and minimized environmental (mainly fish habitat) and social impacts and risks, and developed specific mitigation and compensation measures to address those impacts and risks. The design also addresses the management of any residual impacts through an environmental management plan that includes environmental monitoring and surveillance. Environmental and social residual risks to the Project schedule or implementation were identified and addressed as part of the Feasibility Study in a risk register. They are discussed in Section 24.4.

To date there are no identified environmental issues that could significantly impact Stornoway's ability to exploit the planned diamond mine on the Foxtrot Property.

Table 20.1: Baseline Studies Completed at Renard Project

Year	Study	Reference
Biological Environment		
Ichthyofauna		
2003	Environmental Baseline Study	Roche. 2003. Environmental Baseline Study – Foxtrot Property. Report prepared for Ashton Mining of Canada Inc.
2005	Environmental Baseline Study	Roche. 2005a. Environmental Baseline Study (2004) – Foxtrot Property. Report prepared for Ashton Mining of Canada Inc. 44 pages + 12 annexes.
2009	Fish habitat assessment	Stantec. 2009. Fish Habitat Assessment - Renard Mine Property. Report prepared for Stornoway Diamonds Canada Inc.
2010	Environmental and social impact assessment	Roche – SNC-Lavalin. 2010. Étude d'impact sur l'environnement et le milieu social du prolongement de la route 167 Nord vers les monts Otish. Préparé pour le Ministère des Transports du Québec. 404 pages + 2 annexes.
2011	Environmental Baseline Study	Roche. 2011. Renard Project. Environmental Baseline Study. Report presented to Stornoway Diamonds Corporation Inc.
Vegetation		
2003	Environmental Baseline Study	Roche. 2003. Environmental Baseline Study – Foxtrot Property. Report prepared for Ashton Mining of Canada Inc.
2005	Environmental Baseline Study	Roche. 2005a. Environmental Baseline Study (2004) – Foxtrot Property. Report prepared for Ashton Mining of Canada Inc. 44 pages + 12 annexes.
2011	Environmental Baseline Study	Roche. 2011. Renard Project. Environmental Baseline Study. Report presented to Stornoway Diamonds Corporation Inc.
Benthos		
2011	Environmental Baseline Study	Roche. 2011. Renard Project. Environmental Baseline Study. Report presented to Stornoway Diamonds Corporation Inc.
Herpetofauna (Amphibians and Reptiles)		
2011	Environmental Baseline Study	Roche. 2011. Renard Project. Environmental Baseline Study. Report presented to Stornoway Diamonds Corporation Inc.
Avifauna		
2005	Environmental Baseline Study	Roche. 2005a. Environmental Baseline Study (2004) – Foxtrot Property. Report prepared for Ashton Mining of Canada Inc. 44 pages + 12 annexes.
2011	Environmental Baseline Study	Roche. 2011. Renard Project. Environmental Baseline Study. Report presented to Stornoway Diamonds Corporation Inc.
Mammals		
2005	Environmental Baseline Study	Roche. 2005a. Environmental Baseline Study (2004) – Foxtrot Property. Préparée pour Ashton Mining of Canada Inc. 44 pages + 12 annexes.
2010	Consultations with Tallymen	Lussier, C. 2010a. Stornoway diamond corporation Inc. Renard Project. Consultations with land users of trapline M11. Preliminary report. Presented to ROCHE ltée. 11 pages + maps.

2010	Environmental and social impact assessment	Roche – SNC-Lavalin. 2010. Étude d'impact sur l'environnement et le milieu social du prolongement de la route 167 Nord vers les monts Otish. Préparé pour le Ministère des Transports du Québec. 404 pages + 2 annexes.
2011	Environmental Baseline Study	Roche. 2011. Renard Project. Environmental Baseline Study. Report presented to Stornoway Diamonds Corporation Inc.
Physical Environment		
Surface deposits		
2002	Desktop study	Amyot, G. 2002. Programme de caractérisation environnementale préliminaire – Projet d'exploration Propriété Foxtrot. SOQUEM Inc. Ashton Mining Canada.
2003	Environmental Baseline Study	Roche. 2003. Environmental Baseline Study– Foxtrot Property. Report prepared for Ashton Mining of Canada Inc. 46 pages + 10 annexes.
2006	Hydrogeological and Geotechnical Study	GENIVAR. 2006. Étude hydrogéologique et géotechnique. Rapport de GENIVAR Québec à GENIVAR Val d'Or. 39 pages et annexes.
2010	Geological Study	Fitzgerald, C., C. Muntener et B. Kupsch. 2010. Geology of the Renard Kimberlites and the Lynx and Hibou Dyke - Foxtrot Property, Québec. Stornoway Diamond Corporation, 10 FEBRUARY 2010. 95 pages.
2011	Environmental Baseline Study	Roche. 2011. Renard Project. Environmental Baseline Study. Report presented to Stornoway Diamonds Corporation Inc.
Hydrology		
2011	Environmental Baseline Study	Roche. 2011. Renard Project. Environmental Baseline Study. Report presented to Stornoway Diamonds Corporation Inc.
2011	Climate and Hydrological Study	Golder 2011a. Renard Project Climate and Hydrological Analysis. Technical Memorandum No. 10-1427-0020/3050. Doc No. 19 Ver. 0. Submitted to Stornoway Diamonds Inc.
2011	Environmental and Social Impact Assessment	Roche. 2011. Renard Project. Environmental and Social Impact Assessment. Report presented to Stornoway Diamonds Corporation Inc
Hydrogeology		
2003	Environmental Baseline Study	Roche. 2003. Environmental Baseline Study – Foxtrot Property. Report prepared for Ashton Mining of Canada Inc.
2006	Hydrogeological and Geotechnical Study	GENIVAR. 2006. Étude hydrogéologique et géotechnique. Rapport de GENIVAR Québec à GENIVAR Val d'Or. 39 pages et annexes.
2011	Geotechnical Study	Golder 2011b. Geotechnical investigation at the processed kimberlite containement (PKC) facility-Renard Project – Québec – Technical Memoramdum, Febuary 4, 2011, 6 pages and appendix.
2011	Hydrogeological Study	Itasca. 2011. Prediction of Inflow Rates, Drawdown, and Particle Migration for the Business Case Mine Plan at Renard Diamond Mine, November 17, 2011, 17 pages and appendix.
2011	Environmental Baseline Study	Roche. 2011. Renard Project. Environmental Baseline Study. Report presented to Stornoway Diamonds Corporation Inc.
2011	Environmental and Social Impact Assessment	Roche. 2011. Renard Project. Environmental and Social Impact Assessment. Report presented to Stornoway Diamonds Corporation Inc
Surface Water and Sediment Quality		
2003	Environmental Baseline Study	Roche. 2003. Environmental Baseline Study – Foxtrot Property. Report prepared for Ashton Mining of Canada Inc.

2005	Environmental Baseline Study	Roche. 2005a. Environmental Baseline Study (2004) – Foxtrot Property. Report prepared for Ashton Mining of Canada Inc. 44 pages + 12 annexes.
2005	Application for an attestation of exemption	Roche. 2005b. Mining Exploration Program 2006-2007 – Application for an attestation of exemption. Prepared for Ashton Mining of Canada Inc. 40 pages + appendices.
2009	Water Quality Study	Stantec. 2009. Programme d'échantillonnage de l'eau brute du lac Kaakus Kaanipaahaapisk – Projet Mine Renard. 8 pages + annexes.
2010	Water Quality Study	Stantec. 2010. Caractérisation de l'eau brute du lac Kaakus Kaanipaahaapisk – Projet Mine Renard. Rapport présenté à Stornorway Diamond Corporation. 9 pages + annexes.
2011	Static and Kinetic Testing Study	Golder. 2011c. Static and Kinetic Testing of Wate Rock, Kimberlite and Processed Kimberlite - Renard Project, Québec. Report No. 10-1427-0020. Doc. No. 069 Rev. 0., 41 pages + appendices
2011	Environmental and Social Impact Assessment	Roche. 2011. Renard Project. Environmental and Social Impact Assessment. Report presented to Stornoway Diamonds Corporation Inc
2011	Environmental Baseline Study	Roche. 2011. Renard Project. Environmental Baseline Study. Report presented to Stornoway Diamonds Corporation Inc.
Soil Quality		
2003	Environmental Baseline Study	Roche. 2003. Environmental Baseline Study – Foxtrot Property. Report prepared for Ashton Mining of Canada Inc.
2010	Environmental Soil Study	Stantec. 2010a. Caractérisation environnementale des sols 2009 – Projet mine de diamant Renard. Rapport présenté à Stornoway Diamond Corporation. 14 pages +annexes.
2011	Static Testing Study	Golder. 2011d. Static Testing of waste Rock, kimberlite and Processed kimberlite Renard Project, Québec, March 4, 2011, 16 pages and appendix.
2011	Environmental Baseline Study	Roche. 2011. Renard Project. Environmental Baseline Study. Report presented to Stornoway Diamonds Corporation Inc.
Human Environment		
Socioeconomics		
2010	Consultations with Tallymen	Lussier, C. 2010a. Stornoway Diamond Corporation Inc. Renard Project. Consultations with land users of trapline M11. Preliminary report. Presented to ROCHE ltée. 11 pages + maps.
2010	Consultations with Tallymen	Lussier, C. 2010b. Stornoway Diamond Corporation Inc. Preliminary consultations with land users of trapline located in the winter road study area. Final report. Presented to ROCHE ltée, Groupe-Conseil. 35 pages + annexes
2011	Environmental Baseline Study	Roche. 2011. Renard Project. Environmental Baseline Study. Report presented to Stornoway Diamonds Corporation Inc.
Land and Natural Resource Use		
2003	Environmental Baseline Study	Roche. 2003. Environmental Baseline Study – Foxtrot Property. Report prepared for Ashton Mining of Canada Inc.
2010	Consultations with Tallymen	Lussier, C. 2010a. Stornoway Diamond Corporation Inc. Renard Project. Consultations with land users of trapline M11. Preliminary report. Presented to ROCHE ltée. 11 pages + maps.
2010	Consultations with Tallymen	Lussier, C. 2010b. Stornoway Diamond Corporation Inc. Preliminary consultations with land users of trapline located in the winter road study area. Final report. Presented to ROCHE ltée, Groupe-Conseil. 35 pages + annexes
2011	Environmental Baseline Study	Roche. 2011. Renard Project. Environmental Baseline Study. Report presented to Stornoway Diamonds Corporation Inc.

Archaeology		
2003	Environmental Baseline Study	Roche. 2003. Environmental Baseline Study – Foxtrot Property. Report prepared for Ashton Mining of Canada Inc.
2004	Review of Archaeological Knowledge	Pintal, J.-Y. 2004. Lac Emmanuel. Bilan des connaissances archéologiques. Rapport inédit remis à Roche ltée, Québec.
2010	Consultations with Tallymen	Lussier, C. 2010a. Stornoway Diamond Corporation Inc. Renard Project. Consultations with land users of trapline M11. Preliminary report. Presented to ROCHE ltée. 11 pages + maps.
2010	Consultations with Tallymen	Lussier, C. 2010b. Stornoway Diamond Corporation Inc. Preliminary consultations with land users of trapline located in the winter road study area. Final report. Presented to ROCHE ltée, Groupe-Conseil. 35 pages + annexes
2011	Environmental Baseline Study	Roche. 2011. Renard Project. Environmental Baseline Study. Report presented to Stornoway Diamonds Corporation Inc.

20.1 Environment

20.1.1 Physical Environment

The Renard Project is located in a region with a subarctic climate. Winters are long and cold, with temperature extremes close to -50°C, while summers are short and relatively warm, with temperature extremes over 30°C. The highest maximum temperatures are recorded in July, with averages varying between 17.9 and 20.4°C, while the lowest temperatures are observed in January, with averages between -29.4 and -27.1°C. Temperatures from November to April average below 0°C. The overall region receives an average of approximately 500 mm of rain and 300 cm of snow annually, with local variations. According to data measurements, snow is expected to occur on the ground from October to May, although it can snow at any time of the year except July (and sometimes August). Lakes are covered with ice from November to May. The maximum amount of rainfall registered in the region over a 24-hour period was 66 mm, which corresponds with a 20 to 100 year return period. The Nitchequon meteorological station (located 96 km from the Foxtrot Property) is the only station in the region that registers wind patterns. Although the dominant winds blow from the west (11.7% of the time), the strongest winds are from the west-northwest with average speeds of 19.4 km/hr (Roche, 2011).

Air quality is generally excellent due to the low intensity of human activities in the region. However, forest fires can generate smoke and fine particulates (PM2.5) that spread over great distances, and this can occur during the summer months.

The Foxtrot Property is located in a region with an elevation between 450 and 550 m above mean sea level (masl). The region's physiography consists of a vast peneplain with a relatively flat topography that is marked by a gently undulating surface. Scattered across this area are numerous lakes and streams and rounded hills rarely reaching 100 m in elevation. The Otish Mountains range, located some 100 km to the south, is the exception to this relatively flat relief, with several summits more than 1,000 m above masl (which is 200-300 m higher than the surrounding area). The hills found on the Foxtrot Property have a southwest-northeast orientation and reach a maximum elevation of 670 m above masl. The main rocky are located one or two kilometres southwest and southeast of Camp Lagopède. The landing strip is located in a sector that is flat with an elevation of approximately 470 m.

Glacial history of the landscape

Québec experienced several successive glacial periods over the last 2.6 million years (Quaternary Period, USGS (2010)). These glaciers eroded the bedrock, modified the hydrographical network and left surface deposits in their wake.

The Wisconsin Glaciation was last glacial period that covered North America and it had a significant impact on the landscape we see today. Few traces of previous glaciations remained following the passage of the last glaciers.

Surface deposits and geomorphology

Due to its elevation and position in the interior of the province, outside of the maximum marine limit, the central portion of Northern Québec was not submerged by either proglacial (in contact with the glacier) or postglacial (following glacial retreat) seas or lakes. The landscape is a direct result of in situ glacial melt.

The landscape is scattered with large glacial forms such as drumlins (glacial material with an elongated shape up to several kilometres long) that have ridges oriented in the direction of dominant glacial flow (northeast-southwest in the area of the Foxtrot Property). Till covers almost all surfaces, with the exception of rocky hills with bare rock. The irregular thickness of deposits has resulted in a system of irregularly shaped lakes and ponds and a dense network of small streams and larger rivers, such as the Eastmain, Misask and Sakami Rivers. Important stretches of Rogen moraines (successive, discontinuous ridges of basal till oriented parallel to the glacial front) are found to the north and south of the Foxtrot Property. This type of till greatly influences the local hydrographical network as well as the distribution of wetlands, which form between the well-drained ridges.

Glacial deposits have a very heterogeneous granulometric composition, often ranging from clay to large metre-sized boulders, and a high proportion of sand (Roche, 2001). On the Foxtrot Property, these deposits range in thickness from 0 to 30 m. Erratic boulders, abundant in some areas, left behind during the glacier's retreat punctuate the landscape. Boulder fields are also found in certain low-lying sectors such as the area of the proposed landing strip. Several small eskers are found on the Foxtrot Property, less than 5 km from Lake Lagopède Lake. A larger esker is located less than 2 km from the proposed landing strip, on the edge of the Misask River.

Aerial photo interpretation and field surveys conducted by Roche (2011) show that the hydrographical network covers approximately 18 % of the 127 km^2 study area. Rocky outcrops and zones with thin till cover account for approximately 14 % of the study area, zones with thicker till account for approximately 60 %, and glaciofluvial sediments (outwash and eskers) cover approximately 6 % of the study area. Wetlands, for their part, cover a little over 2 % of the study area.

Soil

Soil drainage generally varies from good to moderate. There are also zones with poor drainage such as topographical depressions, where wetlands often form, as well as rocky outcrops. Soils at the regional level are mainly composed of humo-ferric podzols and less often of brunisols (Agriculture and Agri-Food Canada, 1998). At several locations on the Foxtrot Property and elsewhere in the region, a thick ferruginous hardpan (called "ortstein") has developed in the soil. In bogs and swamps, organic soils from 30 cm to more than 1 m thick are found.

Permafrost is found in localized patches from Lake Mistassini (~50.40°N) to approximately the latitude of Kuujjuarapik (~55.16°N), covering less than 10 % of the province's landmass between the 50th and 55th parallels (Natural Resources Canada, 2003). However, permafrost zones are rapidly disappearing as a result of climate changes (Thibault and Payette, 2009). The patches of permafrost in the area of the Foxtrot Property will therefore likely become increasingly rare in the future.

Hydrographic network

The Foxtrot Property is almost entirely located within the watershed of the upper Eastmain River. The regional flow pattern runs southwest towards the Eastmain-1 Reservoir. The Renard Project study area (127 km^2) is completely contained within this watershed, in the Misask River sub-watershed (1,510 km^2). This sub-watershed is further sub-divided into over twenty secondary sub-watersheds that for the most part flow into the Lake Lagopède sub-watershed, the largest at 16.1 km^2. A small part of the Foxtrot Property located just north of the Renard

Project is found within the upper Sakami River watershed, which runs northwest towards the La Grande-3 Reservoir.

The study area's waterways have a pluvio-nival hydrological regime (Schetagne et al., 2005), which is characterized by a strong spring freshet in May-June, a rather severe low flow period in summer, an autumn rainfall freshet and a winter low flow period from November to May. Thermal stratification occurs in the deeper lakes during the summer, influencing the physical chemistry of the water at depth. Aside from Lake Lagopède, the study area includes several moderately sized water bodies as well as several small water bodies connected by small streams. Streams within the Renard Project site varied in depth between 0.1 and 0.6 m in July 2010 (May and June were particularly dry that year); according to surveys by Roche (2011), these same streams swelled substantially following rains in August and September, evidence that stream discharge can fluctuate widely within a same season.

An aquifer is present in the rock and in the surface deposits. Groundwater flows mainly towards the different surface water bodies, which suggests that groundwater resurgence occurs in the various lakes whereas aquifer recharge begins along the topographical ridges (Roche, 2011).

20.1.2 Surface Water Environment

Wetlands

Wetlands are important ecosystem components, providing transition zones between terrestrial and aquatic environments where biological productivity and diversity are particularly high (CRRNTBJ, 2010). Wetlands cover over 65% of the surface area around James Bay, gradually diminishing in importance as one move progressively south, east and north. In the northeast sector of the James Bay region, where the Renard Project is located, wetlands cover less than 5% of the surface area. Within the Renard Project study area, wetlands represent only 3% of the plant communities (excluding the hydrographical network). They consist of small ombrotrophic bogs (open, forested or riparian), swamps and patterned fens that cover at the most a few hectares each.

During the construction phase, approximately 37,000 m² of lakes and 4,600 m² of streams will be lost due to lake draining and stream diversion activities, for a total permanent loss of approximately 41,600 m² of fish habitat. To mitigate this impact, lake draining will be done slowly to favour fish concentrations in deep pools and to facilitate their capture and relocation. Stream diversion will be minimized. Habitat losses will be offset through the development of new fish habitats and the restoration, where possible, of hydrological conditions similar to the baseline conditions.

Aquatic environments

The Renard Project study area is located in the Misask River watershed, one of the upper Eastmain River's many tributaries. The hydrographical network consists of lakes of various sizes, ponds, pools, springs and streams, covering approximately 16% of the study area. These waters are generally acidic, well oxygenated, nutrient-poor, of low turbidity and low in mineral concentrations (Roche, 2011).

A total of 36 species of freshwater fish from 11 families are found in the James Bay watershed. Brook Trout, lake Trout, walleye and northern pike are the most sought after game fish in this

area (CRRNTBJ, 2010). The presence of 14 fish species has been confirmed within the Renard Project study area. Among these, 5 species are of interest for sport fishing: northern pike, lake whitefish, brook trout, lake trout and burbot. White sucker, pearl dace and brooktrout are the most abundant species in the area

Several activities undertaken as part of the construction, operation and closure phases of the Project are likely to directly or indirectly affect surface water or sediment quality and alter fish habitat and affect some fish populations.

20.1.3 Terrestrial Environment

The Renard Project study area is located in the spruce-lichen bioclimactic domain of the James Bay region (CRRNTBJ, 2010). This domain occupies the entire taiga subzone, which stretches from the 52nd to the 55th parallel. It distinguishes itself from the spruce-moss domain, located south of the 52nd parallel, by the low density of its forest cover. Black spruce, which must reproduce vegetatively due to the severe climate and low precipitation, punctuates the carpet of lichen, while jack pine and balsam fir reach the northern limit of their distribution range (MRNF, 2003-2011).

20.1.4 Vegetation

The plant community found within the Renard Project study area consists of several forest stands, and is largely dominated by young black spruce-lichen stands (64%), old black spruce-lichen stands (12%) and old black spruce-moss stands (12%) (Roche, 2011). Old black spruce-sphagnum stands and old jack pine-lichen stands represent 4% and 3% of the plant community's surface area, respectively, while the other forest stands each account for less than 1%.

The main Project components that will affect vegetation during construction and operation are site preparation and development activities (clearing, excavation, earthworks, etc.). Through various iterations of the Project design, it was possible to minimize the Project footprint and to limit encroachment on wetlands and on old forest stands.

A closure and rehabilitation plan was developed to stabilize disturbed land conditions and restore them by revegetating surfaces to allow local indigenous species to recolonize the area.

20.1.5 Wildlife

Caribou, moose and black bears are the main large mammals found within the James Bay region. Caribou hunting dominates big game sport hunting activities in the region (CRRNTBJ, 2010). Two ecotypes of the Woodland caribou (Rangifer tarandus caribou) are likely present in the area: forest-dwelling and barren-ground. The forest-dwelling ecotype is designated as threatened in Canada (Thomas and Gray, 2002) and vulnerable in Québec (An Act respecting threatened or vulnerable species, R.S.Q., chapter E-12.01). Nonetheless, the presence of the forest dwelling ecotype has not been confirmed during the 2011 mammal winter survey in an area covering 7621 km^2. The Renard Project study area is located on Trapline M-11, which

covers the entire Foxtrot Property. Subsistence hunting is practiced by Crees in the sector and targets mainly moose and waterfowl.

Close to 25 species of fur-bearing animals are present within the James Bay region, and approximately ten have been trapped in the past by Cree trappers on Trapline M-11 in the sector of the Renard Project. The most commonly captured are American marten, American beaver, mink and muskrat (Roche, 2011). Trapping is a popular traditional activity among native populations.

The main potential impacts of the Project during the construction and operation phases are related to site preparation and development activities (clearing, excavation, earthworks, etc.). These activities will result in habitat loss. Most species in the vicinity of the mine site will be able to find similar habitat at the periphery of the mine and airstrip sites; however, such shifts in habitat may result in additional energy expenditure and competition for habitat and resources. To limit the extent of these impacts, the Project design has been progressively optimized to limit the Project footprint on wildlife habitat, while avoiding encroachment on Trapline M-11 moose hunting areas.

By design, the Project site roads were optimized to limit length and extent. Increased access to the area may lead to additional pressure on wildlife resources (e.g., moose or marten), but hunting and trapping by workers or contractors will be prohibited.

During construction, the draining of two small, shallow lakes will be the main source of impact on waterfowl and aquatic birds. No significant impact is expected on forest birds as a result of site preparation and development activities; however, the relative significance of an impact may vary depending on the species.

20.1.6 Heritage Resources

There are no known archaeological sites within a 50 km radius of the Renard Project site. Nevertheless, based on available ethnological and geomorphological data, a total of 51 zones of archaeological potential have been identified. These are sites were the probability of uncovering traces of human settlement is highest. None of these zones are located within the Renard Project site. However, ten or more of these zones of archaeological potential are located around Lake Lagopède.

20.2 Geochemical Classification

The geochemical classification of kimberlite, processed kimberlite (PK), and waste rock was performed through static and kinetic testing programs on samples of drill core, mineral processing rejects, and overburden collected at the Project site. The geochemical characterisation program followed Québec Provincial guidance of Directive 019 sur l'industrie minière (MDDEP, 2005, revised 2009). The environmental geochemistry of the overburden was characterized by static testing using the same methods as waste rock and PK. The materials expected to be handled through the exploitation of the Renards 2, 3, 4, 9, and 65 ore bodies were tested. The samples subjected to kinetic testing were selected in consideration of the static testing results and focussed on rock types that have the largest representation in future waste rock, kimberlite, and PK.

The objectives of the geochemical classification program were to:

- Classify mine waste according to the intent of Québec Directive 019; and
- Identify chemicals of potential interest in the framework of probable future mine water quality.

Results obtained to date show that all mine wastes are non-acid generating and that leachability is low. Kinetic testing completed to date shows that mine wastes release low concentrations of dissolved constituents; concentrations of chemicals in leachates are below Québec groundwater quality guidelines. As a result, mine waste rock and PK are classified as low risk.

As per the same types of testing, waste rock and country rock is considered to be Category I and is, therefore, suitable for use as construction material without restriction.

Static leaching test methods revealed some exceedances to water quality criteria but the concentration of these constituents met the same criteria under the more representative and longer-term kinetic test methods. The results of the kinetic tests met the water quality criteria. Some initial exceedances of the water quality criteria were noted, but these were limited to only the initial “flushing” period.

Kinetic leaching tests are considered to be more representative than static leaching tests in terms of probably leaching behaviour and leachate concentrations generated from waste material that will be exposed to ambient conditions at the site due to:

- The nature of the contact between the infiltrating water and the waste rock that more closely simulate the infiltration of water through a rock pile or PKC facility as opposed to agitated mixing of rock and water in a short-term leach test;
- The lower solution-to-solid ratio used in the kinetic tests which more closely represents the climate conditions at site;
- The long-term (20 weeks) nature of the testing that allows for the evaluation of transient chemical processes such as mineral dissolution, sulphide oxidation and other weathering reactions; and
- Kinetic weathering test results form the basis of risk evaluation for the mine wastes, together with existing site water quality data, rather than the static leaching test results.

20.3 Water Management and Treatment

20.3.1 Water Management

The strategy for water management on site is to reduce the potential impact of the Renard Project on local and regional water resources while satisfying process water requirements. The two primary objectives of the water management strategy are to manage all contact water in an efficient and effective manner, and to reduce the amount of contact water that will require treatment prior to discharge to the environment.

The approaches that will be used to achieve these objectives include:

- Implementing measures (i.e. clean water diversions) to avoid the contact of clean runoff water with areas affected by the mine or mining activities;
- Collecting, transporting, and treating as necessary, mine water, and runoff water in contact with core project activities;
- Reducing the intake of fresh water from the environment by recycling and reusing water to the greatest extent possible;
- Monitoring quality of discharge; and
- Adjusting management practices if monitoring results indicate discharge quality does not meet discharge criteria.

For the purpose of the water management strategy, surface water has been grouped into two categories: contact water and non-contact water. Contact water is defined as any water that may have been affected by mining activities. Contact water includes:

- Surface runoff from the mining and milling areas;
- Groundwater inflows to mine workings;
- Surface runoff (and shallow drainage) from rock, overburden, and processed kimberlite disposal areas;
- Processed kimberlite transport water;
- Seepage water through the rock, overburden, and processed kimberlite containment areas; and
- Dewatering volumes having high total suspended solids (TSS) concentrations.

All contact water will be intercepted, contained, re-used (to the extent practical), monitored, and if required, treated prior to discharge to the receiving environment. Non-contact water is limited to intercepted and diverted runoff originating from areas undisturbed by mining. As a general rule, 100-year return period design events apply to perimeter infrastructure operating during both the construction and mine operations phases. Ten-year return period events apply to the design of internal infrastructure or to infrastructure operating during the two year construction phase only. The water management infrastructure, comprising ditches and sumps, will be constructed in phases based on the development of the site. The final configuration of the water management infrastructure is shown on Figure 20.1.

The basis of the proposed water management plan is to collect, monitor, and treat (if required) all contact water from the site (with the exception of the camp) at the Renard 65 sump and treatment plant. After the required treatment, water will be discharged to Lagopède Lake. Contact water from the camp will be treated in a separate sedimentation pond (northwest of the camp) prior to discharge into Lagopède Lake.

A ditch network will convey the surface water runoff to the Renard 65 sump by gravity. Once complete, the Renard 2, Renard 3, and Renard 4 underground developments will be inter-connected and drain to a low point of the ramp. Water from the ramp sump will be preferentially

pumped to the mill to satisfy the demand for process water, with any excess pumped to the Renard 65 sump for monitoring and treatment, if required. It is anticipated that the groundwater inflows from the underground workings will fully satisfy process water demand.

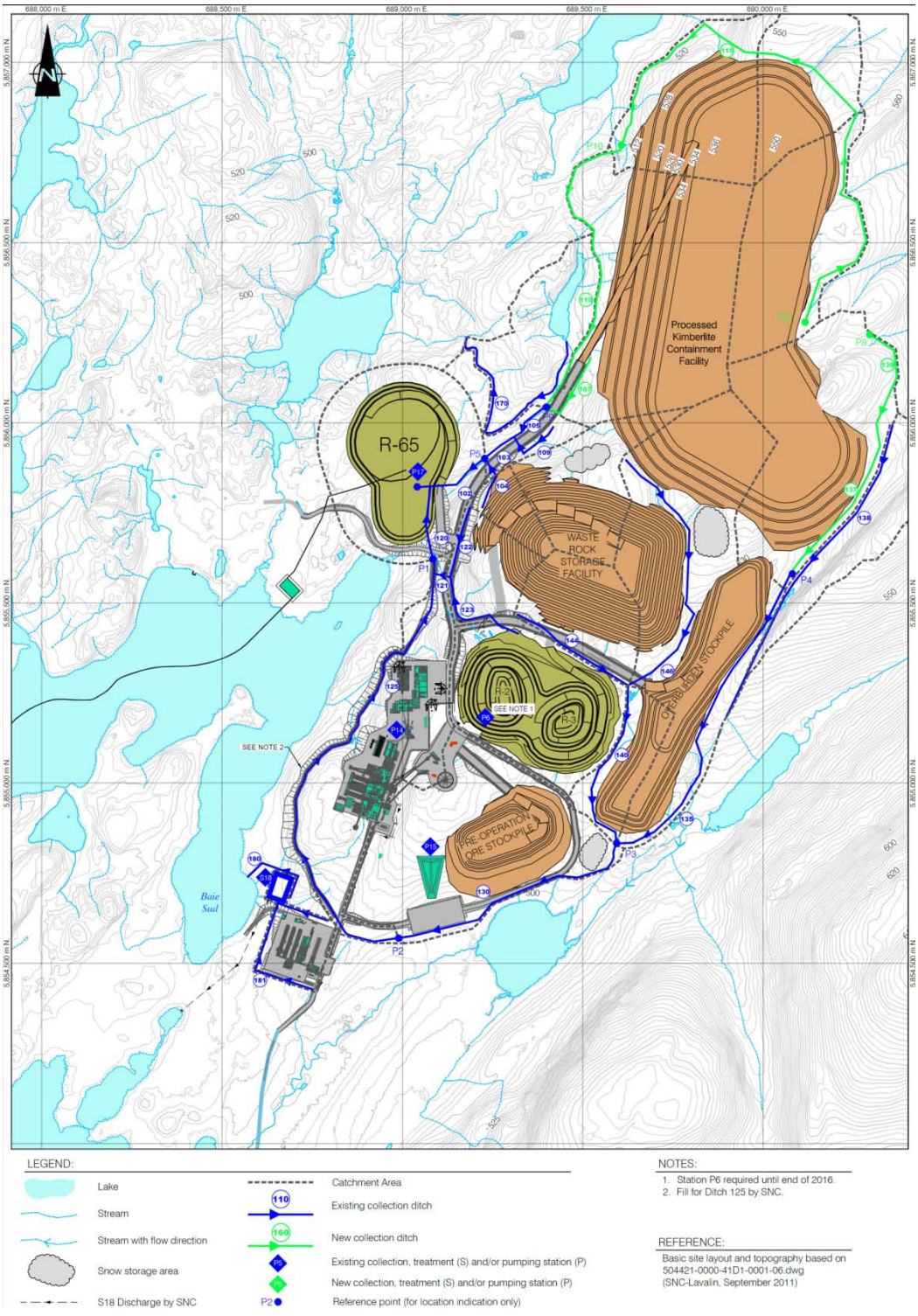


Figure 20.1: Water Management Infrastructure

20.3.2 Water Treatment

The water treatment evaluation focussed on water management during the active mining period; however, part of the system will be in place to manage construction water during the two year construction period. The water treatment system used during the construction period will be expanded prior to the start of mining operations.

The main basis of the proposed water management plan is to collect, monitor and treat (if required) all contact water from the site at the main sump, the Renard 65 sump. After the required treatment, the water will be discharged to Lagopède Lake. Contact water from the camp will be treated in a distinct, dedicated sedimentation pond prior to discharge to Lagopède Lake.

Water will be transferred to the water treatment plant or pumped directly to Lagopède Lake, if water quality permits, from the Renard 65 sump. The treatment option selected consists of ferric sulphate coagulation, followed by flocculation, clarification, and multimedia filtration.

20.4 Permits

The Renard Project must obtain global environmental approval under various legislation, including the *Mining Act* (*Québec*)(R.S.Q., c. M-13.1) (Mining Act), the *Environment Quality Act* (R.S.Q. c. Q-2) (EQA), the *Canadian Environmental Assessment Act* (S.C., c. 37) (CEAA) and the *James Bay and Northern Québec Agreement* (JBNQA).

Following the submission of the Project Notice in February 2010, the MDDEP and COMEV issued guidelines to Stornoway on the scope of the impact assessment to be carried out for the Renard Diamond Project (June 2010). A modification to the Project Notice was submitted to the MDDEP, COMEV and the Canadian Environmental Assessment Agency in February 2011, informing them of some changes made to the Project since February 2010. The ESIA was conducted in 2010-2011 by Roche. The ESIA report was submitted to the Administrator on December 23, 2011.

Section 20.4.1 below provides information about the Québec and Canadian environmental assessment and review process and the legislative framework under which the Project is to be carried out. Section 20.4.2 describes all of the sectoral permits required once the Project has received global environmental approval under the environmental assessment process.

20.4.1 Environmental Assessment and Review Processes

The JBNQA was implemented in 1975 by the Québec government, the Government of Canada, the Grand Council of the Crees (Eeyou Istchee) (GCC-EI) and the Association des Inuits du Nouveau-Québec. It sets out the social and environmental protection regimes for the regions of James Bay and Nunavik.

Three land categories were established in the JBNQA: Category I, II and III. Specific rights are defined for each one. The Foxtrot Property is located on category III land, near the community of Mistissini (the Cree Nation of Mistissini or CNM).

On Category III lands the Crees have certain rights, particularly with regard to hunting, fishing, trapping and outfitting operation. Residents of CNM hunt, fish and trap within the Foxtrot property. The Renard Diamond Project site is located in an area known as yuus kanchiisu saakahiikan (Mild Rock Ptarmigan Lake). More specifically, the Renard kimberlites are located on Trapline M11 which is under the responsibility of Emerson and Sydney Swallow (the "tallymen") and used by their respective families.

According to the JBNQA, the Québec government may authorize the development of Category III lands. However, before approving a proposed development, the Québec government must comply with the social and environmental protection regime which ensures respect of the Cree's hunting, fishing and trapping rights.

Mining and surface rights on Category III land belong to the Québec government and are managed by the legislation and regulations applicable to land use implemented by the regulatory authorities.

Chapter II of the EQA provides specific provisions for environmental assessments applicable to the James Bay territory, in accordance with the provisions of the JBNQA. The environmental assessment procedure specific to this region differs from the provincial procedure, as it involves the active participation of the Aboriginal people who live there.

Provincial Environmental Assessment Process

The Renard Project is subject to the provincial process for the assessment and review of social and environmental impacts under the *JBNQA*, (Chapter 22, Annex 1, Section 1; Chapter 2, Section 2, paragraph 3); and the *EQA,* (Chapter II, Annex A, paragraph (a)).

The provincial environmental assessment process is a five-step process, summarized below, as defined by the MDDEP.

- Proponent's Preliminary Information
- The first step of the process is the presentation of the proponent's preliminary information about the Project. This step begins at the preliminary project planning phase, when the proponent studies the potential options and the technical, environmental and social aspects of the Project before choosing the best options for subsequent studies. This step has been completed in February 2010.
- Assessment
- The JBNQA Administrator then sends the preliminary information to the committee responsible for defining the nature and extent of the impact assessment. In this case, the preliminary information will be sent to the COMEV, because the Project is located south of the 55th parallel.
- When a project is not automatically subject to nor exempt from the process, COMEV makes a recommendation to the Administrator on whether an impact assessment should be conducted.
- When a project is subject to the process, COMEV establishes guidelines that provide details on the scope of the impact assessment to be prepared by the proponent. These

guidelines are submitted to the Administrator, who forwards them to the proponent, with or without changes.

- COMEV recommendation and guidelines were received in June 2010.
- Impact Assessment
- In the third step, the proponent carries out the impact assessment in accordance with the Administrator's guidelines. The *Regulation respecting the environmental and social impact assessment and review procedure applicable to the territory of James Bay and Northern Québec* (Q-2, r. 11) defines the essential elements to be included in an impact assessment. These include a project description and the physical and social environments affected by the project, an assessment of the potential impacts of the project, a description of the project alternatives, and a description and assessment of remedial measures. The ESIA was conducted in 2010-2011 by Roche. The report was pending publication at the time of the NI43-101 report.
- Review
- The project proponent submits its impact assessment to the Administrator, who then sends it to the review committee (COMEX), given that this project is located south of the 55th parallel.
- Aboriginal administrations and the public can make representations to the committee, which may also hold public hearings and any other type of consultation.
- COMEX recommends project approval or rejection, and establishes the applicable conditions, where needed. COMEX is responsible for defining the changes or additional measures it considers appropriate.
- The ESIA will be submitted to the Administrator for review in 2011.
- Decision
- After considering the COMEX recommendation, the Administrator grants or refuses project approval. If the Administrator does not accept the committee's recommendation, he or she must consult COMEX before making a final decision and informing the proponent. The final decision is also shared with the concerned Aboriginal administrations. Moreover, sectoral authorizations (for example, for quarries and sand pits, water and sewer mains, camps, etc.) must be issued pursuant to the various provisions of Chapter I of the EQA.

The MDDEP is coordinating the review of the impact assessment study as a representative of the Québec government. However, other departments, such as the Ministère des Ressources naturelles et de la Faune (MRNF) and the Ministère des Transports (MTQ) may also be involved in the review as experts.

Mining Act

The Québec government is responsible for mining activities in the province. This activity is subject to the Mining Act which defines ownership of the right to mineral substances (claims, mining exploration licenses, mining leases, mining concessions, etc.) and the rights and obligations of the claim holder or other mining right granted by the State.

With regard to the environment, it should be noted that the Mining Act, Chapter IV, Division III, specifies that the holder of mining rights has the responsibility to rehabilitate and restore the lands on which exploration and/or development activities have been carried out. This work must be completed in accordance with the restoration plan pre-approved by the MRNF.

Under the Mining Act, the *Regulation respecting mineral substances other than petroleum, natural gas and brine* (R.R.Q., c. M-13.1, r 2) details certain procedural requirements of the Act, particularly in terms of the information and documents to be provided to the MRNF on restoration measures and locations established to store mine tailings.

Any enterprise engaged in mining activities subject to the Mining Act must submit a restoration plan to the MRNF. After consulting with the MDDEP, the Department approves the plan and implementation schedule.

Enterprises planning to use or using a confinement area must provide a financial guarantee to the MRNF once the restoration plan has been approved. The amount of this guarantee must be 70% of the estimated cost of restoring the confinement areas. The number of yearly payments is determined based on the type of activity (exploration or operation) and the expected duration of the activity (maximum 15 years).

However, where the enterprise carries out progressive restoration work, it can replace the annual payments and significantly reduce the value of the financial security to the MRNF.

Following the 2003 amendment to the Mining Act, (Section 232.7), the Department may require full payment of the guarantee where a proponent's financial situation deteriorates or the expected duration of its mining activities is shortened.

In addition, the MRNF tabled Bill 14 in 2011 and Bill 79 in 2012 to amend the Mining Act in the National Assembly but bills were not ratified. It is expected that the new provincial government will submit another Bill to this effect during spring 2013 parlementary session. Bill 14 and 79 proposed a number of amendments concerning restoration:

- Raise the coverage of the guarantee for restoration work from 70% to 100%;
- Extend the scope of the financial security to include all the mining infrastructures including the mine tailing accumulation areas;
- Review the payment schedule to accelerate payment of the financial security in 3 years; and
- Protection of rehabilitation and restoration work performed in the accumulation areas;

Finally, it should be noted that Directive 019 on the mining industry (MDDEP, April 2005) is the tool currently used by the MDDEP to assess mining projects. This directive sets out the requirements with which enterprises must comply regarding the safe management of tailings.

Federal Environmental Assessment Process

The *Canadian Environmental Assessment Act* (CEAA) is applicable to projects for which the Federal government has decision-making powers under (Section 5 of the CEAA), as a proponent, administrator of federal lands, funding source or regulatory body. For the Renard Project, Fisheries and Oceans Canada (DFO)(Ref 2) must grant authorization to modify the fish habitat under (Subsection 35(2) of the *Fisheries Act*). There is also a possibility of manufacturing explosives on the site, an activity that requires a licence issued by Natural Resources Canada (NRCan) under (paragraph 7(1)(a) of the *Explosives Act*). According to the *Law List Regulation*, (Subsection 35(2) of the *Fisheries Act*), and (paragraph 7(1)(a) of the *Explosives Act*) are environmental assessment triggers. In this context, Stornoway filed a Project Notice in February 2010 with the federal authorities and received a document from DFO and NRCan in July 2010 that sets out the scope of the federal environmental assessment that must be conducted through the CEAA. The environmental and social impact assessment currently underway takes into account the prescribed actions set out in this guidance document.

The Renard Project is subject to a comprehensive federal environmental assessment in accordance with Section 21 of the CEAA. The Renard Project comes under Section 10 of the Comprehensive Study List Regulations "*The proposed construction, decommissioning or abandonment of a facility for the extraction of 200,000 m³/y or more of ground water […]."* In the context of a comprehensive assessment, the project authority is the Canadian Environmental Assessment Agency. The Canadian Environmental Assessment Agency has issued updated guidelines in June 2011 that defines the scope of the comprehensive study to be carried out for the Renard Project.

DFO, NRCan, Transport Canada, Environment Canada and Indian and Northern Affairs Canada will take part as expert authorities. The Canadian Environmental Assessment Agency is also responsible for coordinating the federal and provincial environmental assessment processes and ensures that relevant information is shared with the provincial review committee (COMEX). The Renard Project is encompassed by an agreement with the federal government's Major Project Management Office. The purpose of this agreement is to provide an effective tool to ensure that the expected federal review process is carried out in an efficient manner.

Table 20.2 shows the roles of the key Federal departments involved in the Renard Project environmental assessment procedure.

Table 20.2: Key Roles of Federal Departments involved in the Renard Project Environmental Assessment Procedure

Department	Responsible authority[1]	Expert authority[2]	Study coordination	Key applicable legislation
Fisheries and Oceans Canada		●		*Fisheries Act*
Natural Resources Canada		●		*Explosives Act*
Environment Canada		●		*Canadian Environmental Protection Act*
Indian and Northern Affairs Canada		●		*Cree-Naskapi (of Québec) Act; James Bay* and *Northern Québec Native Claims Settlement Act*
Canadian Environmental Assessment Agency	●		●	*Canadian Environmental Assessment Act* (CEAA)
Major Projects Management Office (MPMO)			●	CEAA

Notes 1 Federal authority with regulatory and statutory responsibilities for a project under the Federal legislation in effect.
2 Federal authorities with specialized information and knowledge or experts concerning the Project.

At the end of the impact assessment process and after receiving the global environmental approval from the Federal government under the CEAA, the Project will finalise sectoral permitting with DFO and NRCan. In addition to the mitigation measures set out in the environmental and social impact assessment, the final project design must comply with the applicable standards on projected infrastructures and equipment. The preparation of plans and specifications must follow the legal framework of the governments of Québec and Canada. In addition to the mitigation measures set out in the environmental and social impact assessment, the final project design must comply with the applicable standards on projected infrastructures and equipment. The preparation of plans and specifications must follow the legal framework of the governments of Québec and Canada.

20.4.2 Sectoral Permits, Approvals and Leases

Provincial Level

Certificate of Authorization: To carry out the Renard Project, several Certificates of Authorization will be required from the MDDEP, under Section 22 of the *EQA*. A CA application includes a form to which are attached the documents and information set out in Sections 7 and 8 of the Regulation respecting the application of the EQA (c. Q-2, r. 3). In addition, because the Renard Project involves discharges in the aquatic environment, it will be necessary to complete the, "*Demande d'objectifs environnementaux de rejet (OER) pour les industries*" [request for environmental quality objectives for effluent discharge] and attach it to the Certificate of Authorization application.

Authorization for Bulk Sampling: According to section 69 of the *Mining Act*, **"***The claim holder shall not extract or dispatch mineral substances except for geological or geochemical sampling nor a quantity in excess of 50 metric tons.*"

If the claim holder shows that it is necessary to extract or dispatch a greater quantity of mineral substances, other than surface mineral substances, the Minister may authorize him to extract or dispatch a fixed quantity of such mineral substances. Within one year after the extraction, the claim holder shall report the quantity of mineral substances extracted and the results of the metallurgical testing to the Minister.

In order to obtain authorization from the MRNF, an application for bulk sampling authorization must be submitted following the instructions set out in Appendix 5 of the Guidelines for Preparing a Mining Site Rehabilitation Plan and General Mining Site Rehabilitation Requirements.

Authorization to Establish Water Supply Intakes and Devices for the Treatment of Drinking Water and Waste Water: Authorization under Section 32 of the EQA is needed to establish water supply intakes and drinking water and wastewater treatment facilities. Two forms must be completed and signed by the project engineer, and the required documents must be attached. The two forms are the *Demande d'autorisation pour réaliser un projet d'aqueduc et d'égouts* (application for authorization to carry out a waterworks or waste water system project) (treatment of drinking water) and *Présentation des demandes d'autorisation pour les systèmes de traitement des eaux usées d'origine domestique* (submission of applications for authorization for domestic wastewater treatment systems).

Authorization to Install an Apparatus or Equipment to Prevent, Reduce or Cause the Cessation of the Issuance of Contaminants into the Atmosphere: Under Section 48 of the EQA, an application for authorization must be submitted for the installation of an apparatus or equipment to prevent, reduce or cause the cessation of the issuance of contaminants into the atmosphere. The *Demande d'autorisation pour un projet industriel* (application for industrial project authorization) must be completed and submitted to the MDDEP regional branch.

Authorization to Perform an Activity that may Alter Fish Habitat: In addition, because some project activities will affect fish habitat, authorization from the MRNF is required, under Section 128.7 of the *Conservation and Development of Wildlife Act* (R.S.Q., C. C-G1.1)(CDWA). In this case, the application for authorization will be made jointly with the application for a certificate of authorization under Section 22 of the EQA. For activities planned in aquatic environments and wetlands, the MRNF has established a single-window service for issuing authorizations required under the EQA and the CDWA.

Authorization for the Location of the Process Concentration Plant and Mine Tailings Confinement Area: Under Section 240 of the Mining Act, "Any person who intends to operate a mill for the preparation of mineral substances, a concentration plant, a refinery or a smelter shall, before commencing operations, have the site approved by the Minister or, where the project is subject to the environmental impact assessment and review procedure provided for in (Division IV.1 of Chapter I of the *EQA* Chapter Q-2), by the Government." Section 241 of the same Act also states, "*Every person responsible for the management of a concentration plant, refinery or smelter shall, before commencing activities, have the site intended as a storage yard for tailings approved by the Minister. The same applies to every holder of a mining right, owner of mineral substances or operator who intends to establish a mine tailings site*." Consequently, a request for approval must be submitted to the MRNF before activities begin at the mine site at the Renard Project.

Depollution attestation: In accordance with Order in Council 515-2002 issued on May 1st, 2002, the Renard Project requires a depollution attestation from the MDDEP. This certificate, which is renewable every 5 years, identifies the environmental conditions that must be met by the industrial establishment when carrying out its activities. The certificate gathers together all of the environmental requirements relating to the operation of an industrial establishment.

Permits and Lease

The following permits and lease will be required for the mining operation:

- Forest management permit for mining activities;
- Occupation licence for works enabling water to be collected or evacuated;
- High-risk petroleum equipment operation permit;
- Explosives permit;
- Mining leases;
- Non-exclusive lease for the mining of surface mineral substances; and
- Lease for the occupation of the domain of the State;

In time, Stornoway will seek all the required authorization, permits and leases necessary for the mining operations.

Federal Level

Authorization to alter fish habitat

Under Section 35 of *the Fisheries Act*, "*No person shall carry on any work or undertaking that results in the harmful alteration, disruption or destruction of fish habitat*" without previous authorization from the Minister.

When a project includes a known risk of affecting fish and fish habitat, the project must be submitted to DFO for review. The general process that must be followed is described on the DFO website. The Proponent's Guide to Information Requirements for Review under the Fish Habitat Protection Provisions of the Fisheries Act, identifies the information requirements for a detailed review by DFO. In order for a project to be reviewed, the proponent must complete the form "*Request for Review under the Fish Habitat Protection Provisions of the Fisheries Act*". The request must be submitted to the local Fish Habitat Management Office.

There are three possible outcomes following a DFO review:

- Mitigation measures (included in the project design or proposed by DFO are sufficient to avoid or mitigate negative impacts to fish and fish habitat – DFO issues a "*Letter of Advice*";
- The residual damage to the fish habitat cannot be avoided, but is considered acceptable – an authorization for a HADD[1] and compensation for fish habitat loss are required; and
- The project will have unacceptable impacts on fish and fish habitat – the project cannot proceed as designed.

In most cases, the Ministry will issue an authorization if the compensation plan results in no net loss of fish habitat. The Renard Project is expected to require an authorization for HADD and compensation for the loss of habitat. Generally speaking, the request for an authorization triggers the environmental assessment process. DFO is reviewing the compensation program proposed as part of the Federal EA process and will issue an authorization that includes the compensation program.

License for explosives factories and magazines

Under Section 7(1)(a) of the *Explosives Act* (R.S.C., 1989 c. G17), licenses issued by the Minister of Natural Resources Canada are required for the operation of explosives factories and magazines in Canada.

The Renard Project only requires licenses for the operation of explosives factories. In fact, according to Section 2 of the same Act, the term "magazine" excludes:

"*a place where an explosive is kept or stored exclusively for use at or in a mine or quarry in a province in which provision is made by the law of that province for efficient inspection and control of explosives*".

[1] The harmful alteration, disruption or destruction (HADD) of fish habitat

In Québec, an Act respecting explosives (R.S.Q., c. E-22) provides for the issuing of permits, the inspection and the control of activities associated with explosives.

Registration
The construction of a landing strip for the Renard Project is not subject to any permit, authorization or approval from Transport Canada, as long as it remains for private use and strictly for the transport of personnel and for charter flights.

Municipal Level

Certificate of compliance with municipal regulations
Under Section 8 of the Regulation respecting the application of the EQA, "*A person who applies for a certificate of authorization shall also submit to the Minister a certificate attesting that the project does not contravene any municipal by-law. The certificate shall be issued by the clerk or the secretary-treasurer of a local municipality or, in the case of an unorganized territory, of a regional county municipality*". Requests for certificates must therefore be submitted to the Municipality of James Bay at Matagami for the Renard Project's activities, except for those activities defined in Section 246 of the *Act Respecting Land use Planning and Development* (R.S.Q., c. A-19.1).

Municipal Permits
Permits are required from the Municipality of James Bay for certain activities that do not require a certificate of authorization under Section 22 of the *EQA*. This will be the case for groundwater catchment facilities (<75 m^3/day) and the landing strip's septic systems, among others (permit for single groundwater catchment works and permit for septic systems).

20.5 Community Agreements

20.5.1 Social and Economic Agreements

On July 2010, Stornoway announced the conclusion of a Pre-Development Agreement (PDA) for the Renard Project with the Grand Council of the Crees (Eeyou Istchee)/Cree Regional Authority, the Cree Nation of Mistissini and DIAQUEM Inc. The PDA was developed on the principle of mutual respect between the parties and the interests of all in seeing the establishment of a successful diamond mining venture at Renard.

On March 2012, Stornoway completed negotiations with the Cree Nation of Mistissini, the Grand Council of the Crees (Eeyou Itschee) and the Cree Regional Authority on an Impacts and Benefits Agreement (IBA) designated the Mecheshoo Agreeement. This agreement replaces the PDA signed by the Parties in 2010. The Mecheshoo Agreement is a binding agreement that will govern the long-term working relationship between Stornoway and the Cree parties during all phases of the Renard Diamond Project. It provides for training, employment and business opportunities for the Crees during project construction, operation and closure, and sets out the principles of social, cultural and environmental respect under which the project will be managed. The Mecheshoo Agreement includes a mechanism by which the Cree parties will benefit financially from the success of the project on a long term basis, consistent with the Mining Industry's best practices for engagement with First Nations communities.

On July 2012 Stornoway executed a Declaration of Partnership (the "Declaration") with the communities of Chibougamau and Chapais in the James Bay Region of Québec. The Declaration is a statement of cooperation between the partners for the responsible development of the Renard Diamond Project based on the principles of environmental protection, social responsibility and economic viability. The Declaration includes provisions to set up a Renard Liaison Committee that will address issues of mutual interest such as communication, employment, and the economic diversification of local communities. In particular, the committee will oversee initiatives to attract and retain new residents to the towns of Chibougamau and Chapais.

20.6 Mine Closure

Stornoway is committed to sound environmental practices in all of its activities through the implementation of its environmental policy. The policy includes a commitment to progressively rehabilitate disturbed areas, to develop a closure plan that can be continuously improved and to incorporate new technologies where practical.

The closure and rehabilitation of the mine site will be carried out progressively throughout the life of mine (LOM). Acknowledging that Stornoway is only a temporary user of land that will be returned to users at the end of the mine life, the Project design has been optimized to reduce the footprint and to facilitate closure and rehabilitation:

- The number of sub-watersheds of Lake Lagopède that will be affected by the development has been reduced;
- Materials and buildings have been designed to facilitate dismantling and/or recycling at the end of the mine life;
- The mine plan has been developed to ensure that all waste rock will be used (construction and backfill) to fill the underground mine or sent back to open-pits;
- The processed kimberlite containment (PKC) facility has been designed to promote progressive revegetation and facilitate maintenance;
- Plans have been developed to progressively revegetate flat surfaces (slope benches) when they are no longer required for the PKC facility operation; and
- Environmental and geotechnical follow-ups have been designed to be carried out for a period of three to five years after closure and dismantling.

The Closure and Rehabilitation Plan has been prepared according to the requirements of the MRNF outlined in the document titled "*Guidelines for Preparing a Mining Site Rehabilitation Plan and General Mining Site Requirements*."

The following activities are part of the planned mine closure and were used to estimate the costs of Rehabilitation of the Renard Project:

- No building will be left in place. Whenever possible, buildings will be sold with the equipment they contain, completely or partially. During dismantling works, a reclamation/recycling program will be established. All non-contaminated waste will be

sent to the project trench landfill. All equipment and machinery will be sent out of the mine site for sale or recycling;

- All hazardous waste will be managed according to applicable laws and regulations and will be sent out of the mine site;
- On-site roads and tracks will be scarified and revegetated;
- Revegetation will include hydroseeding followed by tree planting;
- Flat surfaces of the PKC facility will be revegetated. Revegetation will be carried out on 5 m high benches on eight levels and on the upper surface. Bench revegetation will be carried out progressively during the operation phase. Flat surfaces will be covered with a 0.5 m overburden layer under a 0.3 m to 0.5 m of topsoil (organic matter) layer and revegetated;
- During site construction, topsoil will be stockpiled for revegetation purposes. The overburden stockpile will be constructed in four 5 m lifts. Slopes will be covered with gravel. Flat surfaces will be revegetated. Eighty-five percent of flat surfaces will be located on the final plateau;
- All waste rock will be used (construction and backfill) to fill the underground mine or sent back to open-pits. The area previously occupied by the waste rock pile will be covered with a 0.5 m overburden layer under a 0.3 m to 0.5 m topsoil (organic matter) layer and revegetated;
- The area previously occupied by the temporary ore stockpile will be covered with a 0.5 m overburden layer under a 0.3m to 0.5 m topsoil (organic matter) layer and revegetated;
- The sanitary wastewater treatment plant will be dismantled, and the sludge in the three lagoons will be emptied. Ponds will be either landscaped to develop a wetland or filled and revegetated;
- The outfall will be covered and left in place;
- The industrial wastewater treatment plant will be dismantled. The outfall will be covered and left in place;
- R2-R3 and Renard 65 open-pits will no longer be dewatered and will gradually become “lakes.” Surface hydraulic connectivity will be maintained between Renard 65 lake and Lake Lagopède The small stream derived to prevent surface runoff from entering the pit will be restored to its original route to re-establish the initial drainage;
- A hydraulic link will be constructed between the Renard 2 and Renard 3 pits and the surface water hydrological network to accelerate pit filling. Once filling is complete, a hydraulic link with the surface water hydrological system will be constructed;
- Security bunds will be constructed around the pits;
- A concrete slab will be installed to secure all underground access (shaft, ramp, vent raise and backfill raises);
- Whenever possible, surface water drainage will be re-established to conditions similar to the original hydrological system;
- The airstrip will be scarified and revegetated and its buildings dismantled;
- The explosives storage facilities will be dismantled;

- The sand pit will be covered with topsoil and revegetated;
- The project trench landfill will be covered and revegetated;
- Revegetation will be monitored for three to five years and subsequent revegetation will be undertaken in spots where initial planting has not succeeded;
- Geotechnical monitoring of PKC facility, waste and overburden piles will be conducted for a three-to-five-year period; and
- The main access road, which belongs to Québec's Ministry of Transport, and will be left in place.

Table 20.3 gives the October 2012 estimate for the rehabilitation work of the Renard Project. This estimate is taken from the Closure and Rehabilitation Plan prepared by Roche in 2011 (revised in 2012) and approved by Stornoway.

Table 20.3: October 2012 **Cost Estimate for Rehabilitation of the Renard Project**

Item	Estimated cost
Rehabilitation of the PKC facility	$5,904,438
Revegetation of flat surfaces other than the PKC facility	$2,472,437
Other Rehabilitation activities	$6,091,680
TOTAL	**$14,468,555**

The amount of financial guarantee to the MRNF (70% of the estimated cost of restoring the confinement areas) is thus estimated at $ 4,133,107.The first yearly payment will occur after the fourth year of production and payments will increase annually (Table 20.4).

Table 20.4: Octobre 2012 **Estimate for Yearly Payment of Financial Guarantee**

Production Year	Payment	Production Year	Payment
1	$0	7	$582,768
2	$0	8	$710,894
3	$66,130	9	$839,021
4	$194,256	10	$967,147
5	$322,382	11	$ 0
6	$450,509		

20.7 Renard Mine Road

On December 17, 2012 Stornoway received from the Québec Ministère du Développement durable, de l'Environnement, de la Faune et des Parcs ("MDDEFP") the transfer of the Certificate of Authorization for the construction of the Renard Mine Road. The 97 km long

mining road will be built by Stornoway to provide all season road access to the Renard Diamond Project. The transfer of this Certificate of Authorization is consistent with the schedule contained within the Framework Agreement of November 15th 2012 between Stornoway, the MDDEFP, the Québec Ministère des Transports ("MTQ"), the Ministère des Ressources Naturelles and the Ministère des Finances et de l'Économie. The Renard Mine Road Certificate of Authorization represents the principal regulatory approval required for Stornoway to begin road construction. Given that the single lane road that will be built will have a smaller environmental footprint than the previously tabled Route 167 Extension Project, Stornoway will seek for minor changes to the terms of the authorization. The Renard Mine Road is also subject to other permits and authorizations; most of them were originally granted to the MTQ and some will possibly need some adjustments. The list of specific permits related to the Renard Mine Road is provided in the table below:

Table 20.5: List of authorizations, permits, leases and approvals related to the Renard Mine Road

Description	Department	Regulation/Act/Code
Provincial government		
Authorizations		
Transfert of the certificate of authorization to carry out road project from the MTQ to Stornoway	MDDEFP	Section 22 of the *Environment Quality Act* (R.S.Q., c. Q-2)
Sand and gravel operation (borrow pits)	MDDEFP	Section 22 of the *Environment Quality Act (*R.S.Q., c. Q-2). For activities not subjected to the Regulation respecting standards of forest management for forests in the domain of the State.
Camps (Potable and grey waters, Waste disposal)	MDDEFP	Section 32 of the *Environment Quality Act (*R.S.Q., c. Q-2)
Authorizations Sand and gravel operation (borrow pits)	COMEX	Conditions 5 and 6 of the CA. Chapter II of the *Environnement Quality Act (R.S.Q., c.Q-2).* Section 164.
Authorizations Fish habitat	COMEX	Conditions 10 of the CA. Chapter II of the *Environnement Quality Act* (R.S.Q., c.Q-2). Section 164.
Permits		
Forest management permit for road activities All deforestation activities	MRN-Forests	Chapter I, Divisions I and II of the *Forest Act* (R.S.Q., c. F-4.1) Regulation respecting standards of forest management for forests in the domain of the State (c. F-4.1, r. 7)
Leases		
Non-exclusive lease to mine surface mineral Borrow pits, quarries and sand pit	MRN-Mines	Section 140, Division VII of the *Mining Act* (R.S.Q., c. M-13.1)
Lease to occupy the domain of the state Km 143 to km 240	MRN-Territory	Section 239 of the *Mining Act* (R.S.Q., c. M-13.1). Section 47, Division II of the Act respecting the lands in the domain of the State (R.S.Q., c. T-8.1) Section 35 of the Regulation respecting the sale, lease and granting of immovable rights on lands in the domain of the State (c. T-8.1, r. 7)
Federal government		

Authorizations		
Authorization to modify fish habitat All activities affecting fish habitat	Fisheries and Oceans Canada	Section 35 of the *Fisheries Act* (R.S.C., 1985, c. F-14)
Approval		
Approval of works in, over, under, through, or across any navigable waterway	Transport Canada	Section 5 of the *Navigable Waters Protection Act* (R.S.C., 1985, c. N-22)
Municipal government		
Conformity		
Certificate of conformity to municipal regulations	Municipalité de Baie-James	Various municipal By-laws

21.0 CAPITAL AND OPERATING COSTS

Following a Preliminary Economic Assessment (PEA) study completed by others, SNC-Lavalin Inc. (SLI) was mandated to prepare updated feasibility study cost estimates for the Renard Project. For this Feasibility Study a considerable amount of additional engineering was conducted by SLI, their sub-consultants and Stornoway Diamond Corporation (Stornoway), to ensure that the resulting Capital Cost estimate (Capex) and Operating Cost estimate (Opex) were consistent with a Class III type estimate, per the Association for the Advancement of Cost Engineering (AACE). In addition to the preparation of engineering deliverables, procurement and contract administration documents were prepared to enable the estimating team to contact equipment suppliers and contractors for detailed budget quotations.

In the revised 2012 FS Estimate SLI and its subcontractor AMEC Process are solely responsible for the capital cost estimate (CAPEX) of the infrastructure and process plant direct costs, as well as indirect and other costs related to these direct costs. SLI and its subcontractor AMEC Process are also responsible to provide the operating cost estimate (OPEX) (partial) for the operation of the mineral process facility. SLI is also responsible to provide the load list to Stornoway Diamonds to be used as a basis for power consumption calculations during the mine's operations.

In addition all open pit and underground equipment, development and infrastructure costs are under the responsibility of Stornoway Diamonds and its third party consultants as listed in the Table 21.1 below, namely AMEC Mining, GMining and Golder. R2/R3 open pit development and mobile equipment, overburden stockpile, waste rock stockpile, processed kimberlite containment, water management and pre-production ore stockpile remain excluded from SLI's scope.

The overall site layout is divided into two (2) zones: Zone 1 encompasses all construction performed under the EPCM contractor's responsibility and is regulated by the Québec construction decree[1]; Zone 2 encompasses all work performed under Stornoway's responsibility and is not regulated by the Québec construction decree as it is considered part of the mining activities. Table 21.1 shows the high level responsibilities with respect to sub-areas in terms of engineering development and estimate ownership.

[1] There are no wage schedules in Québec, the wage rates are established under the Québec Construction Decree.

Table 21.1: High Level Responsibility Matrix

Sub Area – Description	Responsible
0000 – General, Multi section – ZONE 1	SLI
0000 – General, Multi section – ZONE 2	Stornoway
1000 – Mine mobile equipment, U/G	Stornoway
1000 – Mine mobile equipment, open pit	Stornoway
1100 – Off shaft underground infrastructures	Stornoway
1200 – R2/R3 open pit pre-production	Stornoway (G-Min)
1200 – R2/R3 Underground ramp and shaft	Stornoway
1300 – R4, Underground	Stornoway (AMM)
1400 – Shaft and off shaft development (including hoists)	Stornoway & SLI
1500 – Pre-production development and construction	N/A
1600 – Renard 65 – Surface water drainage sump	Stornoway (G-Min)
2000 – Mine area surface facilities (explosives and detonator storage)	Stornoway & SLI
3000 – Mineral processing plant	Stornoway & SLI (AMP)
4000 – Overburden, waste rock, pre-production ore stockpile	Stornoway (G-Min)
4000 – PKC facility	Stornoway (Gold)
5000 – On-site utilities and infrastructure	SLI
8000 – Off-site utilities and infrastructure (permanent access road and HQ power line	Stornoway

Note: SLI: SNC-Lavalin
AMM: AMEC Mining
AMP: AMEC Process
G-Min: G-Mining
Gold: Golder

21.1 Capital Costs Estimate

This capital cost estimate qualifies as a Class III type estimate and the estimate accuracy is evaluated at -13% / +17%. The estimate labour costs were based on a 70-hour workweek (10 hrs/day @ 7 days/week), with a rotation of 4 weeks in / 1 week out. There is no allowance for double shifts in the estimate for above ground work. Stornoway's self-performed activities for mine development are based on an 84 hour workweek (i.e. 12 hours per day, 7 days per week) with 2 weeks in / 2 weeks out rotation.

In the 2012 FS Estimate Revision budgetary quotations, estimated costs and allowances were actualized to 4th quarter 2012 Canadian dollars and include escalation beyond the 2nd quarter of 2011 up to and including Pre-Operations Verification scheduled for January, 2016. Escalation is estimated per package based on the schedule and as an additional cost item. Owner's cost was not escalated.

The Direct Costs were based on engineering take offs and scope and quantity reviews were performed with engineering to confirm that the scope of work was entirely covered. Approximately 40% of the estimate is based on budget quotations while the remainder is based on in-house databases or factored historical data. Almost 99% of the estimate is in Canadian dollars.

The Indirect Costs are the estimated costs for the EPCM services and for the construction field indirect costs.

The portion of the Mine development and production costs (Underground Costs and Open Pit Costs) occurring prior to Commercial Operation was summarized and transferred to, and included in the Capital Cost - Mining. The Operating costs of the Process Plant were also calculated as part of the Operating Costs and the portion occurring prior to Commercial Operation was summarized and transferred to, and included in the Capital Cost - Processing Plant. Owner's site operation costs (General and Administrative, G & A, costs) were evaluated in detail in the Operating Costs and the portion of the operating costs occurring prior to Commercial Operation was summarized and transferred to, and included in the Capital Cost - Owner's costs.

Contingencies were evaluated for all packages on the basis of the engineering level of confidence. A summary of the estimated total Capital Cost is shown in Table 21.2.

Table 21.2: Summary of Total Estimated Capital Costs

Major Area / sub area	FS Cost (CAD) (000)
DIRECT COSTS	
0000 –General, Multi-Section	$32,658
1000 – Mining	$141,352
2000 – Mine area surface facilities	$9,863
3000 – Processing plant	$175,410
4000 – Stockpiles, PKC facility and waste rock management	$,0
5000 – Onsite utilities and infrastructures	$114,753
8000 – Offsite utilities and infrastructures	$,0
Sub-Total Direct Costs	**$474,035**
INDIRECT COSTS	
9300 – Owner's Cost	$94,656
9300 –Spares, fills, tools	$7,071
9400 – EPCM services	$47,862
9500 – Field Indirects	$54,693
9600 – Vendor reps construction + assistance during P.O.V.	$3,678
9700 – Freight (incl. Duties)	$5,476
9910 – Contingencies	$64,660
Sub-Total – Indirect costs	**$278,096**
Total – excluding Escalation	**$752,131**
9920 – Escalation	$40,500
9930 – Risk	$29,700
Total – including Escalation	**$822,331**

21.2 Operating Costs Estimate

The annual operating costs (Opex) are subdivided into four areas and were estimated by four different participants during the Feasibility Study - as follows:

- Open Pit Operations — Opex estimated by G-Mining;
- Underground Mine Operations — Opex estimated by AMEC Minin;
- Process Plant and Auxiliaries — Opex estimated by AMEC Process;
- General Administration and Infrastructure — Opex estimated by Stornoway.

The Opex are defined as those costs occurring after the concentrator has achieved commercial production, (which is defined by Stornoway as achieving 60% or more of nameplate capacity for 30 consecutive days). Based on the project schedule and estimated ramp-up production curve, the production date is end of May, 2016 as is illustrated in Figure 21.1

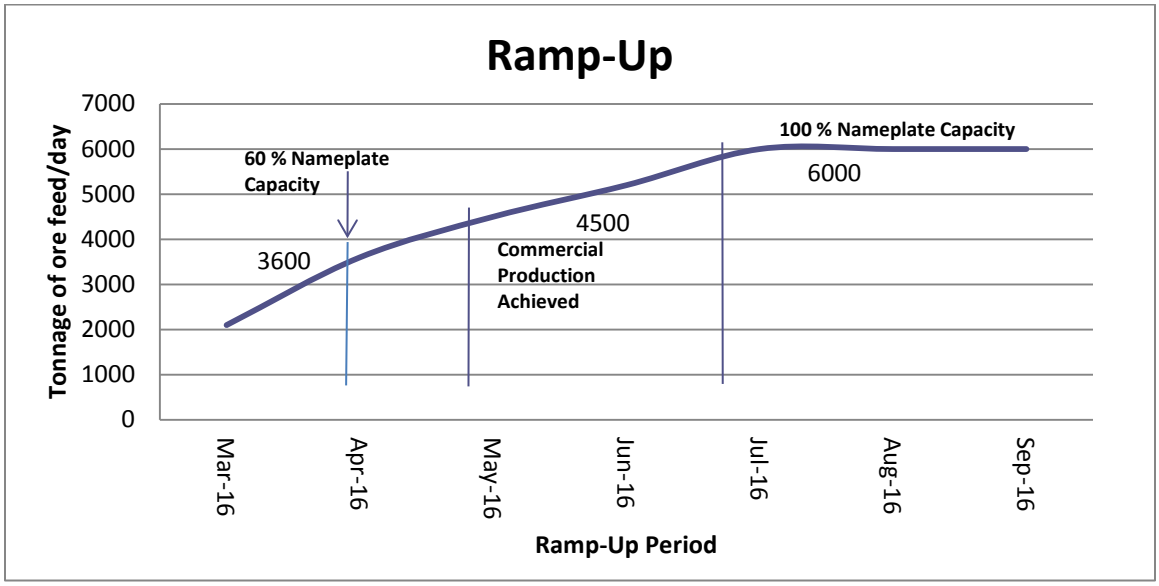


Figure 21.1: Estimated Ramp-Up Production Curve

The Open Pit mining cost estimate, prepared by GMining, was based on the mining plans developed by them. The open pit mine will use standard mine operating procedures. Kimberlite ore will be delivered initially to the ore stockpile or, once concentrator operations start, to the surface primary crusher upstream of the concentrator.

The Underground mining cost estimate is based on the mining plans developed by AMEC Mining. Ore from the underground mine will be transported to the surface using truck hauling via ramp. The operating costs for mining operations were estimated from first principles using the mining plan.

The Process plant costs are based on the quantity of kimberlite ore feed to the process plant. The process plant is expected to operate 365 days per year with an overall utilization efficiency of 78%. The operating costs were estimated using standard manufacturer's cost data for power, fuel, other consumables and hourly maintenance costs, as well as information from the AMEC process internal database.

The cost for the Utilities, Infrastructures and General and Administration were estimated by Stornoway. The Labour costs for all four categories were based on a salary scale and organization chart developed by Stornoway.

A summary of the estimated Opex, on a life of mine basis, for each of the four categories is shown in Table 21.3 and Table 21.4.

Table 21.3: Summary of Estimated Total LOM Opex ($ '000)

Description	Open Pit	Underground	Concentrator	Infrastr. and Admin.
Labour	5,975	263,549	126,275	95,433
Fuel	1,510	45,798	5,432	9,154
Explosives & Supplies	3,1,028	71,700	-	-
Consum. & Reagents	-	-	25,940	-
Equipment	1,435	130,490	-	-
Power	173	40,682	140,926	61,048
Maintenance	-	-	59,223	16,077
Accomodation	-	-	-	58,492
Transportation	-	-	-	37,023
Other	56	2,461	1,019	151,412
Totals	10,177	554,682	358,814	428,638

Total LOM Costs $1,352,310,000

Table 21.4: Summary of Estimated Opex LOM Unit Cost/tonne Ore Processed ($)

Description	Open Pit	Underground	Concentrator	Infrastr. and Admin.
Labour	0.25	11.23	5.38	4.07
Fuel	0.06	0.64	0.23	0.0.39
Explosives & Supplies	0.04	3.06	-	-
Consum. & Reagents	-	-	1.11	-
Equipment	0.06	5.56	-	-
Power	0.01	1.73	6.01	2.60
Maintenance	-	-	2.52	0.69
Accomodation	-	-	-	2.49
Transportation	-	-	-	1.58
Other	0.01	0.10	0.04	6.45
Totals	0.43	23.64	15.29	18.27

Total LOM Unit Costs $57.63/tonne Ore Feed

Deferred Capital Costs

The Deferred Capital category covers capital expenditure during the "ramp-up" period from Commercial Production to achievement of full name-plate capacity (first three months of production in 2016, June to August). Total estimated Deferred Capital Costs from first three months of production are $ 11.8 million.

Sustaining Capital Costs

Sustaining Capital Costs are the costs that will not be capitalized, i.e. periodic addition of capital which is required to sustain operations at existing levels. Estimated Sustaining Capital and Replacement Costs vary for each year and are shown for the planned life of the Project in Table 21.5.

Table 21.5: Estimated Sustaining Capital and Capital Replacement Costs

Description	2016[1]	2017	2018	2019	2020	2021	2022	2023	2024	2025
Open pit										
Underground	23.98	43.22	25.00	27.32	13.12	9.26	8.42	4.54		
Concentrator										
Adm. & Infra.			0.74							
Totals	23.98	43.22	25.00	27.32	13.12	9.26	8.42	4.54		

Note: 1 First three months of production is the Deferred Capital Costs.

Basis and Accuracy Assessment of Estimate

The operating cost estimate was developed for the Project base case, which reflects the following key assumptions:

- Power unit cost was based on an estimate of the fixed and variable costs. Power consumption was based on the average consumption of each motor included in the equipment list.
- Pricing was either based on revised information received in 2012 or information received in the original 2011 FS and actualized to the fourth quarter of 2012 (e.g. manpower salary figures)
- No allowance was made for contingency, escalation and risk. Several activities relating to operating cost were identified during the study risk workshops; no item posed a major risk.
- No allowance was made for final mine closure activities within the Opex estimate, other than the progressive reclamation costs of the PKC facility. All other mine closure costs are considered as part of the Financial Analysis.

The operating cost estimate was based on a Class III estimate methodology as defined by the type and quantity of engineering deliverables produced to support the estimate. The expected order of accuracy is in the range of -15% to +15 %. With the information received and compiled throughout the Opex Study the required accuracy was considered to have been met.

22.0 ECONOMIC ANALYSIS

The following section presents all elements of the financial model including the financial parameter assumptions, diamond production and revenues, royalty payments, operating costs, capital costs, sustaining capital, salvage value, working capital, closure and reclamation costs, financing, taxation and Project cash flows before and after-tax.

The after-tax financial evaluation was undertaken in real dollar terms (or constant dollars) and in nominal dollar terms on a quarterly basis. The base date for NPV and IRR calculations is January 1, 2013.

The nominal dollar financial model introduces escalation factors to estimate nominal dollar cash flows from which taxes are calculated. This methodology better estimates the effective tax shield as a result of tax pools not being escalated over time. However, net cash flow is then de-escalated and presented in real Q1 2013 dollar terms for clarity.

The pre-production period commences in January 2013 and ends in May 2016. Commercial production commences in June 2016 after commissioning, ramp-up for 30 consecutive days at an average processing rate of at least 60% of nameplate capacity.

22.1 Financial Assumptions

The significant financial assumptions affecting the financial analysis of the Project include:

- diamond selling prices;
- crude oil and diesel prices;
- electricity cost;
- exchange rates; and
- escalation rates.

The base case assumptions are explained in the following sub-sections.

22.1.1 Diamond Prices

The diamond selling price utilized in the Feasibility Study is supported by a diamond valuation performed by WWW International Diamond Consultants (WWW) in Antwerp, Belgium which took place between May 9th and 13th, 2011. In addition to WWW, four other diamond companies also provided price estimates, and the average of the five valuations was used by WWW to construct a diamond price model with “High” and “Minimum” sensitivities. The results of the diamond valuation study are presented in Table 22.1 and further discussed in section 19 of this report.

Table 22.1: Diamond Valuation Results, Renard Kimberlite Pipes

Kimberlite Body	Valuation Sample (carats)	Achieved Prices for the Valuation Samples			WWW Price Model		
		Number of independent valuations	Average of Independent Valuations (US$/carat)	WWW Valuation (US$/carat)	WWW Base Case Model (US$/carat)	WWW "High" Model (US$/carat)	WWW "Minimum" Model (US$/carat)
Renard 2	1,580	5	$173	$195	$182	$236	$163
Renard 3	2,753	5	$171	$190	$182	$205	$153
Renard 4	2,674	5	$100	$107	$112	$185	$105

Note: Stornoway has adopted a base case diamond price model of US$164/carat for the Renard 4 kimberlite pipe based on the interpretation of similarity between the Renard 2, Renard 3, and Renard 4, diamond size frequency distributions.
All price estimates based on +1 DTC sieve size cut-off.

Diamond prices are assumed to increase by 2.5% per annum in real terms (with sensitivities ranging from 0% to 5%), from Q3 2011 until the end of the WWW forecast period in 2025, starting with the base case model diamond price estimates established by WWW.

22.1.2 Exchange Rate

The C$/US$ exchange rate is used to convert revenue from diamond sales into Canadian dollars and to calculate a delivered fuel price to site in C$/litre based on a US$ oil price. The base case exchange rate assumption for financial modelling is 1.00 C$/US$. All operating cost estimates are in Canadian dollars.

The capital cost estimate is expressed in Canadian dollars, however some quotes for specialized equipment sourced from South Africa, Europe and the United-States were converted using the rates presented in Table 22.2. The initial capital cost estimate is 98.7%denominated in Canadian dollars according to SNC-Lavalin.

Table 22.2: Initial Capital Cost Exchange Rate Assumptions

Exchange rate	$
1 USD	0.99
1 EUR	1.29
1 ZAR	0.11

22.1.3 Crude Oil and Diesel

The crude oil price assumption is utilized to derive a site delivered diesel fuel price. The base case WTI crude oil price selected for the financial evaluation is US$95/barrel. The delivered diesel price includes transportation costs to site, federal excise tax and provincial fuel tax. Diesel utilized for power generation is exempt from taxes and fuel used for mobile mining equipment is entitled to a refund of the taxes paid (Table 22.3).

Table 22.3: Long-Term Diesel Price Assumptions

Crude WTI (US$/bbl)	85	90	95	100	105
CAD/USD Exchange rate	1.000	1.000	1.000	1.000	1.000
Diesel wholesale price (C$/L)	0.795	0.836	0.876	0.916	0.956
Federal excise tax (C$/L)	0.040	0.040	0.040	0.040	0.040
Provincial fuel tax (C$/L)	0.164	0.164	0.164	0.164	0.164
Delivery cost, admin & profit (C$/L)	0.124	0.124	0.124	0.124	0.124
Gross delivered (C$/L)	1.123	1.163	1.204	1.244	1.284
Less: refundable portion (C$/L)	-0.204	-0.204	-0.204	-0.204	-0.204
Net delivered diesel price (C$/L)	0.919	0.960	1.000	1.040	1.080

Note: Grey shading ipresents base case assumption

22.1.4 Electricity Cost

Power generation is via a diesel power plant located on site. The cost of generating power is therefore dependent on the fuel price assumption. The cost of generating power for various crude oil prices is presented in Table 22.4.

Table 22.4: Power Generation Cost

WTI Crude Oil (US$/bbl)	85	90	95	100	105
CAD/USD Exchange rate	1.000	1.000	1.000	1.000	1.000
Net delivered diesel price (C$/L)	0.919	0.960	1.000	1.040	1.080
Cost of power (C$/kWh)	0.277	0.288	0.299	0.311	0.322

Note: Grey shading presents base case assumption

22.1.5 Escalation Rates

The financial model calculates cash flows in real Q4 2012 dollar terms (constant dollar terms) or in nominal terms utilizing a general inflation factor of 2% per year applied to operating costs, revenues, deferred and sustaining capital expenditures, as well as closure costs and salvage values. The general escalation factor utilized is consistent with the monetary policy adopted by the Bank of Canada and the federal government at keeping total CPI inflation at 2% with a control range of 1 to 3 percent around the target (Bank of Canada Monetary Policy Report, July 2011).

The SNC-Lavalin (SLI) initial capital cost estimate includes a detailed estimate of escalation over the construction period which was used in nominal dollar cash flows. The SLI escalation estimate does not cover Owner's Costs, so in the nominal dollar cash flows, Owner's Costs have been separately escalated at 2% per annum starting in Q4 2012.

22.2 Project Financing

Stornoway must finance the initial construction expenditures, working capital and any debt repayments that commence during the construction period, less pre-production revenue and

Québec Mining Duties refundable credits. For the purposes of the Feasibility Study financial model, 100% equity financing is assumed, with the exception of the Mine Road Loan, which is payable as described in Section 19.2, and a deferred payment under the IBA.

On the above basis, prior to becoming cash flow positive, Stornoway would have to provide funding in the amount of $788 million (nominal).

22.3 Taxation, Duties and Royalty Summary

The Project is subject to three levels of taxation including provincial mining duties, provincial income tax and federal income tax. A royalty is payable as described below.

22.3.1 Existing Tax Pools

The existing tax pools for the project company (Stornoway Diamonds (Canada) Inc.) as of November 30, 2012 were taken into consideration in the tax calculations. The opening tax pool balances are presented in Table 22.5.

Table 22.5: Existing Tax Pools at November 30, 2012 ($ '000)

Federal and Provincial Income Tax	
CCA Class 41(a)	6,002
Canadian Development Expense pool	141,481
Canadian Exploration Expense pool	34,208
Tax loss carry-forwards	26,465
Québec Mining Duties	
Roads, buildings & other	6,002

22.3.2 Québec Mining Duties

Québec Mining Duties

The Québec mining duties regime is based on the notion of mining profit. Mining duties aim to capture part of the profit derived from the value of ore. The mining duty calculation includes a processing allowance to provide for a financial return to the investor on the cost of property used to process the ore.

Under the mining duties regime an operator determines annual profit by subtracting from the gross value all deductions and allowances stipulated under the regime which include depreciation allowance, additional depreciation allowance, additional allowance for a northern mine, the allowance for exploration, mineral deposit evaluation and mine development and the additional exploration allowance.

The Québec mining duties rate is 16% for 2013 and subsequent years.

The depreciation allowance an operator can claim in calculating its annual profit for a fiscal year regarding property class generally corresponds to the lesser of the following amounts:

- A percentage of the capital cost of the property of such class, for such fiscal year;
- The undepreciated capital cost of property of such class, at the end of the fiscal year.

Three property classes are relevant for the Renard Project:

- Roads, buildings and other equipment, depreciable at up to 30% per year (in certain circumstances up to 100% if acquired prior to 31 March 2011);
- Mineral deposit evaluation and pre-production mine development expenses, depreciable at up to 100% in any year; and
- Mineral deposit evaluation and post-production mine development expenses, depreciable at up to 30% in any year.

The mining duties paid to the province are estimated at $387.7 million less refundable credit for losses of $40.6 million for a net amount of $347.1 million (nominal).

22.3.3 Provincial Income Taxes

The Project is subject to Québec provincial income taxes as stipulated in the *Québec Tax Act*. The corporate income tax rate in Québec is 11.9% for 2013 and subsequent years. Taxes paid to the provincial government over the project life are estimated at $183.5 million (nominal).

22.3.4 Federal Income Taxes

The Project is also subject to federal corporate income taxes in Canada under the *Income Tax Act (Canada)* at a rate of 15% for 2013 and subsequent years. Taxes paid to the federal government over the project life are estimated at $214.4 million (nominal).

22.3.5 Royalty Agreement

A royalty agreement was executed on April 1, 2011 between Stornoway and Diaquem Inc. (Diaquem) whereby Stornoway has agreed to pay Diaquem a diamond royalty. The royalty agreement was one element of the transaction to acquire Diaquem's 50% beneficial interest in the Renard JV property.

The terms of the royalty agreement provide that SDCI shall grant and pay to Diaquem a royalty (Diamond Royalty) on all diamonds and other gemstones associated with kimberlite extracted from the property of the Renard Project. The Diamond Royalty shall be calculated and payable in respect of Diamonds sold by or on behalf of SDCI or any affiliate during each fiscal quarter of SDCI. The Diamond Royalty shall equal 2% of the aggregate of (i) the actual gross selling price for arm's-length sales and permitted non-arm's-length sales and (ii) the proceeds of insurance received by or for the benefit of SDCI or any affiliate during the quarter with respect to the loss or destruction of or damage to any Diamonds, less any deductions, which shall be the lesser of the aggregate of diamond selling costs incurred by SDCI and 3% of the aggregate gross selling price in clause (i) above.

The Diamond Royalty is estimated at C$99.9 million over the project life in nominal terms, considering both real terms diamond price escalation and general inflation.

22.3.6 Impacts and Benefits Agreement

SDCI signed an impacts and benefits agreement ("IBA"), the Mecheshoo Agreement, with the Grand Council of the Crees (Eeyou Istchee), the Cree Nation of Mistissini and the Cree Regional Authority in March 2011. The Mecheshoo Agreement provides for employment and business opportunities for the Crees, fosters cultural, environmental and social protection, and provides for the Crees' participation in the Renard Project's long term financial success. The financial model incorporates the principal financial provisions of the Mecheshoo Agreement, which include joint contributions to training and business development established by the Cree Nation of Mistissini, and contributions to a social and cultural fund. Considering the confidentiality of the Mecheshoo Agreement, contact SDCI if more details are needed.

22.4 Financial Analysis Results

22.4.1 Project Cash Flows

Projected revenues, operating costs and cash flow summaries are presented in Table 22.6, Table 22.7 and Table 22.8 respectively.

The gross value from diamond sales over the life of mine is estimated at C$5,136 million in nominal dollar terms or C$4,268 million in real dollar terms based on the base case exchange rate of parity.

The total operating cost estimate is $1,624 million in nominal ($1,352 million in real terms) to process 23.5 million tonnes of ore and produce 17.6 million diamond carats. The average operating cost is $69.21/t of ore ($57.63/t in real) or $92.01/carat ($76.63/carat in real) produced with the average annual profile presented.

The Post-Tax Net Present Value discounted at 7% (NPV 7%) is $391.5 million and $683.5 million Pre-Tax based on de-escalated nominal dollars. The internal rate of return is 16.3% on an after-tax basis and 20.3% on a pre-tax basis based on de-escalated nominal dollar cash flows. The payback period from the end of the pre-production period in May 2016 is 4.8 years on an after-tax basis and 4.7 years on a pre-tax basis.

Table 22.6: Diamond Production and Revenues (Nominal $)

Year	Total	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027
Production Summary (k t)																	
R2/R3 tonnage mined, ore and waste	**12,169**	-	-	4,093	4,203	3,571	302	-	-	-	-	-	-	-	-	-	-
R65 tonnage mined, ore and waste	**9,096**	-	734	1,009	-	-	-	-	-	-	-	485	2,062	1,428	1,426	1,952	-
OP ore mined	**2,035**	-	-	232	476	1,122	206	-	-	-	-	-	-	-	-	-	-
UG ore mined	**21,750**	-	-	-	19	59	1,661	2,160	2,160	2,160	2,160	2,160	2,160	2,160	2,160	2,160	571
Ore processed	**23,785**	-	-	-	6	1,553	2,161	2,160	2,160	2,160	2,160	2,160	2,160	2,160	2,160	2,160	627
Less: Ore processed, pre-production	**(321)**	-	-	-	(6)	(316)	-	-	-	-	-	-	-	-	-	-	-
Ore processed, operations	**23,464**	-	-	-	-	1,237	2,161	2,160	2,160	2,160	2,160	2,160	2,160	2,160	2,160	2,160	627
Diamond Production (k carats)																	
Renard 2	**14,858**	-	-	-	4	942	1,408	1,055	1,264	1,696	1,826	1,654	1,983	1,616	1,284	89	37
Renard 3	**1,514**	-	-	-	1	521	162	385	428	15	-	-	-	-	-	-	1
Renard 4 and Renard 9	**1,575**	-	-	-	-	-	-	-	-	-	-	37	13	-	381	911	233
Renard 65	**-**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total (k karats)	**17,947**	-	-	-	5	1,464	1,570	1,440	1,692	1,711	1,826	1,690	1,997	1,616	1,665	1,000	271
Less: carats produced in preproduction	**(299)**	-	-	-	(5)	(294)	-	-	-	-	-	-	-	-	-	-	-
Total during operations	**17,648**	-	-	-	-	1,170	1,570	1,440	1,692	1,711	1,826	1,690	1,997	1,616	1,665	1,000	271
Diamond Sales (k carats)																	
Renard 2	**14,858**	-	-	-	-	734	1,381	1,175	1,178	1,605	1,828	1,662	1,948	1,676	1,338	282	51
Renard 3	**1,514**	-	-	-	-	403	282	277	483	67	-	-	-	-	-	-	1
Renard 4 and Renard 9	**1,575**	-	-	-	-	-	-	-	-	-	-	35	13	2	319	826	381
Renard 65	**-**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total	**17,947**	-	-	-	-	1,137	1,663	1,452	1,661	1,672	1,828	1,697	1,961	1,678	1,657	1,108	433
Less: carats sold in preproduction	**(116)**	-	-	-	-	(116)	-	-	-	-	-	-	-	-	-	-	-
Total	**17,831**	-	-	-	-	1,021	1,663	1,452	1,661	1,672	1,828	1,697	1,961	1,678	1,657	1,108	433
Prices (US$/carat)																	
Avg. Renard 2	**287**		-	-	-	230	238	248	261	272	284	297	311	325	340	348	355
Avg. Renard 3	**246**		-	-	-	230	236	252	259	268	-	-	-	-	-	-	355
Avg. Renard 4 and Renard 9	**314**		-	-	-	-	-	-	-	-	-	269	281	288	307	316	320
Avg. Renard 65	**-**		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Average	**286**		-	-	-	230	238	249	260	272	284	297	311	325	333	324	324
Revenues (k $)																	
Renard 2 (US$)	**4,269,763**	-	-	-	-	168,698	328,981	291,754	307,448	437,205	520,133	494,112	606,089	544,477	454,470	98,242	18,154
Renard 3	**372,030**	-	-	-	-	92,530	66,424	69,795	124,952	18,027	-	-	-	-	-	-	301
Renard 4 and Renard 9	**494,056**	-	-	-	-	-	-	-	-	-	-	9,470	3,535	620	98,120	260,509	121,804
Renard 65	**-**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Gross Revenue (US$)	**5,135,850**	-	-	-	-	261,228	395,406	361,549	432,400	455,232	520,133	503,582	609,623	545,098	552,589	358,751	140,258
Total Gross Revenue (C$)	**5,135,850**	-	-	-	-	261,228	395,406	361,549	432,400	455,232	520,133	503,582	609,623	545,098	552,589	358,751	140,258
Marketing Fees	**(139,142)**	-	-	-	-	(7,154)	(10,749)	(9,901)	(11,718)	(12,314)	(13,980)	(13,574)	(16,288)	(14,661)	(14,867)	(9,950)	(3,986)
Diaquem Royalty	**(99,934)**	-	-	-	-	(5,081)	(7,693)	(7,033)	(8,414)	(8,858)	(10,123)	(9,800)	(11,867)	(10,609)	(10,754)	(6,976)	(2,725)
Net Revenue before transfer to capital	**4,896,773**	-	-	-	-	248,993	376,963	344,615	412,268	434,060	496,030	480,208	581,469	519,828	526,967	341,825	133,547
Capitalized Pre-Production Net Revenue	**(24,984)**	-	-	-	-	(24,984)	-	-	-	-	-	-	-	-	-	-	-
Net Revenue	**4,871,789**	-	-	-	-	224,008	376,963	344,615	412,268	434,060	496,030	480,208	581,469	519,828	526,967	341,825	133,547

Table 22.7: Operating Costs (real $)

Year	Total	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027
Operating Costs (k$)																
Open Pit																
Labour	**27,146**	-	7,153	10,451	7,898	1,025	61	51	54	49	49	63	63	60	142	27
Fuel	**6,273**	-	1,435	2,414	2,003	226	20	16	16	16	17	20	20	20	36	11
Explosives and accessories	**3,820**	-	699	1,466	1,502	153	-	-	-	-	-	-	-	-	-	-
Equipment	**6,057**	-	1,397	2,343	1,933	205	18	15	15	15	15	18	18	18	37	9
Power	**693**	-	194	231	231	37	-	-	-	-	-	-	-	-	-	-
Other	**246**	-	65	94	71	11	1	0	0	0	0	1	1	1	1	0
Transfer to capital	**(34,059)**	-	(10,942)	(16,997)	(6,119)	-	-	-	-	-	-	-	-	-	-	-
Sub-Total Open Pit	**10,177**	-	-	-	7,519	1,657	101	83	86	81	81	102	103	99	216	48
Underground																
Labour	**290,102**	664	4,811	13,986	10,697	20,624	21,252	24,472	27,019	27,610	28,460	26,849	27,090	25,707	25,361	5,501
Fuel	**45,798**	-	-	-	183	2,695	3,346	3,935	4,326	4,560	5,187	5,557	5,541	4,971	4,527	971
Supplies	**71,700**	-	-	-	910	6,857	6,198	7,731	8,578	8,896	8,269	6,214	6,126	5,725	5,305	890
Equipment	**130,490**	-	-	-	789	8,694	9,983	11,995	13,469	13,848	14,790	14,839	14,834	13,172	11,307	2,770
Power	**40,682**	-	-	-	548	3,306	2,813	4,101	4,969	5,366	5,045	4,462	4,033	3,309	2,160	571
Other	**2,658**	4	33	106	146	229	243	252	251	238	240	233	222	210	209	44
Transfer to capital	**(26,750)**	(668)	(4,844)	(14,092)	(7,146)	-	-	-	-	-	-	-	-	-	-	-
Sub-Total UG	**554,682**	-	-	-	6,126	42,405	43,835	52,486	58,612	60,518	61,990	58,154	57,846	53,095	48,868	10,747
Concentrator																
Labour	**133,994**	-	1,469	1,791	11,724	11,987	11,562	11,544	11,587	11,509	11,456	11,926	11,836	12,078	11,763	1,763
Fuel	**5,978**	-	465	2	434	570	451	446	442	458	472	495	491	507	591	154
Consumables and reagents	**26,269**	-	-	12	1,738	2,472	2,471	2,380	2,339	2,401	2,339	2,360	2,360	2,360	2,360	677
Power	**143,050**	-	-	76	9,516	12,970	12,965	12,965	13,001	12,965	12,965	12,965	13,001	12,965	12,965	3,730
Maintenance	**61,104**	-	772	70	4,362	5,599	5,477	5,473	5,470	5,483	5,484	5,518	5,538	5,551	5,590	717
Other	**1,081**	-	14	13	94	97	93	93	93	92	92	96	96	98	95	15
Transfer to capital	**(12,662)**	-	(2,720)	(1,964)	(7,978)	-	-	-	-	-	-	-	-	-	-	-
Sub-Total Concentrator	**358,814**	-	-	-	19,890	33,695	33,019	32,901	32,931	32,909	32,808	33,361	33,321	33,559	33,364	7,056
G&A																
Labour	**125,623**	4,215	10,484	10,704	11,488	8,890	8,890	8,890	8,890	8,890	8,890	8,498	8,498	7,762	7,621	3,010
Fuel	**11,055**	-	704	845	845	845	845	845	845	845	845	845	845	845	845	211
Maintenance	**18,741**	51	342	1,586	1,547	1,471	1,471	1,471	1,471	1,471	1,471	1,471	1,471	1,471	1,471	500
Accommodation	**68,448**	1,206	2,647	3,974	5,364	5,474	5,492	5,602	5,599	5,408	5,426	5,366	5,213	5,031	4,969	1,675
Transportation	**46,237**	836	4,350	2,628	3,533	3,606	3,587	3,596	3,574	3,476	3,478	3,390	3,293	3,175	3,149	565
Power	**67,636**	-	-	4,243	5,647	5,631	5,631	5,631	5,647	5,631	5,631	5,631	5,647	5,631	5,631	1,404
Other	**185,555**	6,962	13,794	8,456	14,964	14,986	14,410	14,001	13,438	13,336	13,561	12,746	12,652	12,884	11,519	7,846
Transfer to capital	**(94,656)**	(13,270)	(32,321)	(32,437)	(16,628)	-	-	-	-	-	-	-	-	-	-	-
Sub-Total G&A	**428,638**	-	-	-	26,759	40,904	40,327	40,038	39,466	39,058	39,303	37,948	37,619	36,799	35,206	15,211
Cash Operating Costs	**1,352,310**	**-**	**-**	**-**	**60,295**	**118,662**	**117,282**	**125,509**	**131,095**	**132,566**	**134,182**	**129,564**	**128,888**	**123,552**	**117,653**	**33,062**
Less: Other expenses	(2,136)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	(2,136)
Cash Production Costs	**1,350,174**	**-**	**-**	**-**	**60,295**	**118,662**	**117,282**	**125,509**	**131,095**	**132,566**	**134,182**	**129,564**	**128,888**	**123,552**	**117,653**	**30,926**
Cash Cost of Sales	**1,350,174**	**-**	**-**	**-**	**41,591**	**116,114**	**118,525**	**124,032**	**128,578**	**133,689**	**132,982**	**131,183**	**129,204**	**124,257**	**118,878**	**51,143**
Operating Margin	**2,693,448**	**-**	**-**	**-**	**164,927**	**226,356**	**188,231**	**235,741**	**242,705**	**282,380**	**261,961**	**337,529**	**281,775**	**284,080**	**141,033**	**46,730**
	Total	**2013**	**2014**	**2015**	**2016**	**2017**	**2018**	**2019**	**2020**	**2021**	**2022**	**2023**	**2024**	**2025**	**2026**	**2027**
Unit Operating Costs																
Unit Cost ($/tonne processed)																
Open pit	**0.43**	-	-	-	6.08	0.77	0.05	0.04	0.04	0.04	0.04	0.05	0.05	0.05	0.10	0.08
Underground	**23.64**	-	-	-	4.95	19.62	20.29	24.30	27.14	28.02	28.70	26.92	26.78	24.58	22.62	17.15
Concentrator	**15.29**	-	-	-	16.09	15.59	15.29	15.23	15.25	15.24	15.19	15.44	15.43	15.54	15.45	11.26
G&A	**18.27**	-	-	-	21.64	18.93	18.67	18.54	18.27	18.08	18.20	17.57	17.42	17.04	16.30	24.28
Total Unit Cost ($/t ore)	**57.63**	-	-	-	48.76	54.92	54.30	58.11	60.69	61.37	62.12	59.98	59.67	57.20	54.47	52.76
Unit Cost ($/carat)																
Open pit	**0.58**	-	-	-	6.42	1.06	0.07	0.05	0.05	0.04	0.05	0.05	0.06	0.06	0.22	0.18
Underground	**31.43**	-	-	-	5.23	27.01	30.44	31.03	34.26	33.15	36.67	29.12	35.79	31.89	48.86	39.66
Concentrator	**20.33**	-	-	-	16.99	21.46	22.93	19.45	19.25	18.03	19.41	16.71	20.62	20.16	33.36	26.04
G&A	**24.29**	-	-	-	22.86	26.05	28.01	23.67	23.07	21.39	23.25	19.00	23.28	22.10	35.20	56.12
Total Unit Cost ($/carat produced)	**76.63**	-	-	-	51.52	75.57	81.45	74.19	76.63	72.61	79.38	64.89	79.75	74.21	117.63	121.99
Unit Operating Margin ($/carat sold)	**151.05**	-	-	-	161.50	136.12	129.61	141.91	145.15	154.45	154.37	172.14	167.95	171.44	127.32	107.99

Table 22.8: Cash Flow Summary (nominal $)

Year	Total	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033
Net Revenue	**4 871 789**	-	-	-	-	224 008	376 963	344 615	412 268	434 060	496 030	480 208	581 469	519 828	526 967	341 825	133 547	-	-	-	-	-	-
Cash Cost of Sales	**(1 620 982)**	-	-	-	-	(44 893)	(127 177)	(132 362)	(141 326)	(149 420)	(158 427)	(160 759)	(161 723)	(162 489)	(159 387)	(155 516)	(67 502)	-	-	-	-	-	-
Other operating expenses	**(2 854)**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	(2 854)	-	-	-	-	-	-
Operating Margin	**3 247 953**	-	-	-	-	179 115	249 786	212 253	270 942	284 641	337 602	319 449	419 746	357 339	367 581	186 308	63 191	-	-	-	-	-	-
Other costs	**(56 039)**	-	(200)	(200)	(200)	(275)	(350)	(350)	(350)	(7 191)	(10 185)	(5 599)	(9 616)	(6 333)	(8 592)	(2 953)	(3 032)	(232)	(381)	-	-	-	-
Interest Income	**1 367**	-	-	-	-	115	-	0	2	8	18	32	51	77	108	147	809	-	-	-	-	-	-
Foreign exchange losses (gains)	**-**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Working Capital	**(0)**	-	25 409	30 838	(34 223)	(32 197)	(7 808)	1 360	(1 791)	1 901	4 366	(6 665)	5 347	(4 414)	2 583	(5 805)	4 098	17 235	149	(381)	(0)	0	0
Fixed Capital	**(1 039 208)**	(1 000)	(163 591)	(426 614)	(240 921)	(57 154)	(47 526)	(28 918)	(31 286)	(15 286)	(11 037)	(10 244)	(5 630)	-	-	-	-	-	-	-	-	-	-
Ore in stockpile	**-**	-	-	-	-	(4 561)	2 985	34	-	-	-	-	-	-	-	-	1 542	-	-	-	-	-	-
Capital Disposals	**13 313**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	2 618	6 650	4 046	-	-	-	-
Closure Costs	**(19 619)**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	(9 712)	(9 907)	-	-	-	-	-
Mine Closure Guarantee	**-**	-	-	-	-	-	-	(56)	(177)	(305)	(436)	(579)	(730)	(890)	(1 053)	(1 230)	2 672	2 781	-	-	-	-	-
Pre-Debt, Pre-Tax Cash Flow	**2 147 768**	(1 000)	(138 382)	(395 976)	(275 345)	85 043	197 086	184 323	237 341	263 767	320 329	296 394	409 168	345 780	360 627	176 467	62 187	16 527	3 813	(381)	(0)	0	0
Total Taxes	**(745 039)**	-	-	-	-	4 145	17 362	(6 681)	(19 747)	(42 245)	(40 066)	(150 156)	(138 556)	(173 298)	(120 616)	(100 163)	2 004	16 659	6 835	(800)	130	91	64
Pre-Debt, Post-Tax Cash Flow	**1 402 729**	(1 000)	(138 382)	(395 976)	(275 345)	89 188	214 448	177 642	217 593	221 522	280 263	146 238	270 612	172 481	240 012	76 305	64 191	33 186	10 648	(1 181)	130	91	64
Loan Disbursements	**90 700**	-	90 700	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Interest Payments	**(27 071)**	-	(350)	(699)	(699)	(2 693)	(3 235)	(3 061)	(2 881)	(2 693)	(2 498)	(2 295)	(1 956)	(1 569)	(1 178)	(783)	(400)	(79)	-	-	-	-	-
Loan Repayments	**(90 700)**	-	-	-	-	(219)	(4 368)	(4 534)	(4 707)	(4 887)	(5 074)	(5 268)	(10 491)	(10 562)	(10 636)	(10 715)	(10 005)	(9 234)	-	-	-	-	-
Post-Debt, Post-Tax Cash Flow	**1 375 658**	(1 000)	(48 032)	(396 675)	(276 044)	86 276	206 845	170 046	210 006	213 943	272 691	138 674	258 165	160 351	228 197	64 807	53 786	23 873	10 648	(1 181)	130	91	64
De-Esc. Post Debt, Post-Tax Cash Flow	**1 071 548**	(1 020)	(47 473)	(385 473)	(263 752)	79 687	189 937	152 862	185 012	184 629	230 933	114 820	209 906	127 816	178 555	49 849	40 669	17 486	7 751	(841)	91	62	43
De-Esc. Post-Debt, Pre-Tax Cash Flow	**1 674 708**	(1 020)	(47 473)	(385 473)	(263 752)	75 782	173 893	158 829	202 440	221 217	264 910	239 893	322 871	266 372	272 906	126 733	39 000	5 078	2 772	(270)	0	0	0

Capital and operating costs used in the financial analysis are as per Capital Cost Estimate and Operating Cost Estimate prepared by SLI, with the following adjustments:

- *The base case exchange rate was modified from 0.99 C$/US$ to 1.00 C$/US$*
- *Severance costs were included in operating costs in the financial model adding $2.92 million in costs.*
- *The total operating cost in real dollars is $1,347 million in the model prepared by SLI compared to $1,352 million in the financial model which adjusts for severance costs and costs in the closure period that are not otherwise included in closure costs.*

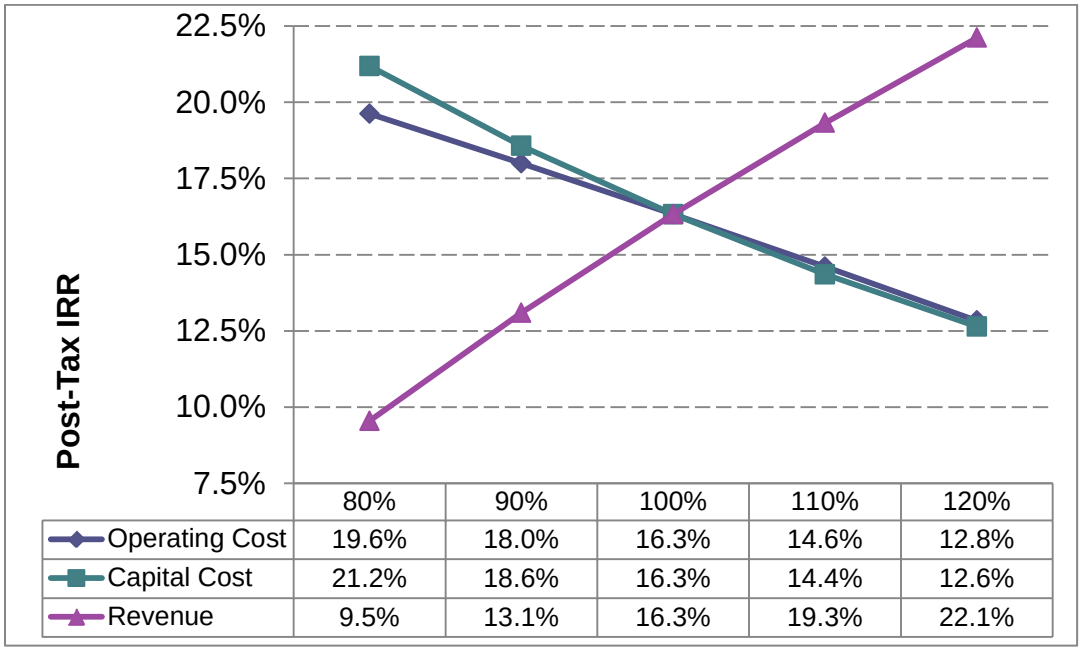
These adjustments were effected to better reflect perceived market conditions at the time at which the financial analysis was undertaken.

22.4.2 Sensitivities

A series of sensitivities are presented for +/- 10% and +/- 20% variations in revenues, operating costs and capital costs (see Figure 22.1, Figure 22.2, Figure 22.3 and Figure 22.4).

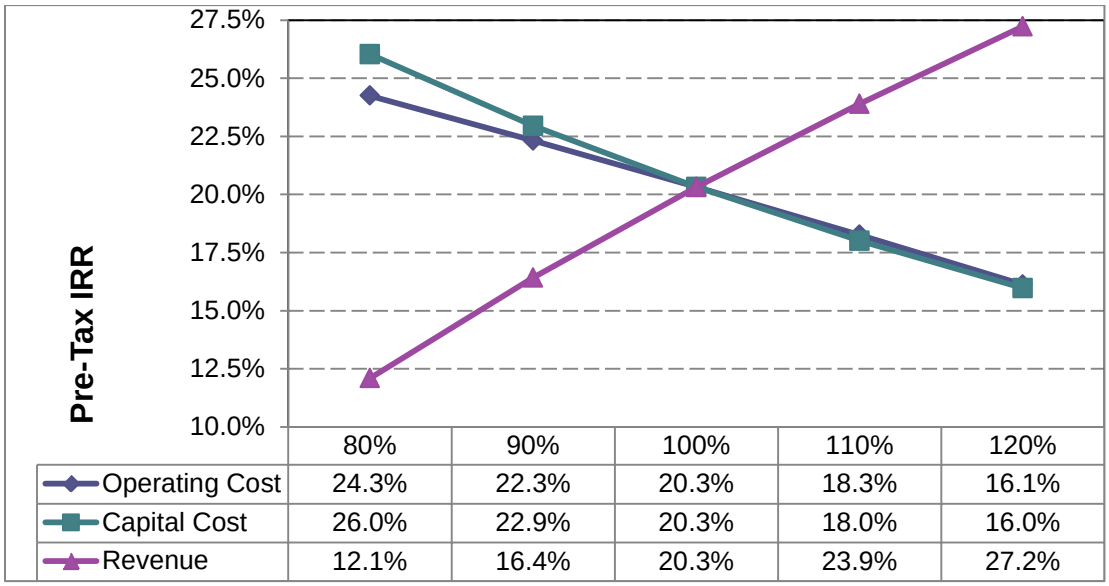
Also, additional sensitivities are presented using different diamond price scenarios, diamond price escalation rates, C$/US$ exchange rates, crude oil price assumptions and recovered diamond grades. The sensitivities presented below in Table 22.9, Table 22.10, Table 22.11, Table 22.12 and Table 22.13 are based on de-escalated nominal dollar cash flow projections using the base case assumption of 2% inflation from Q4 2012.

The project returns are more sensitive to the diamond price assumption, followed by capital costs and finally by operating costs.



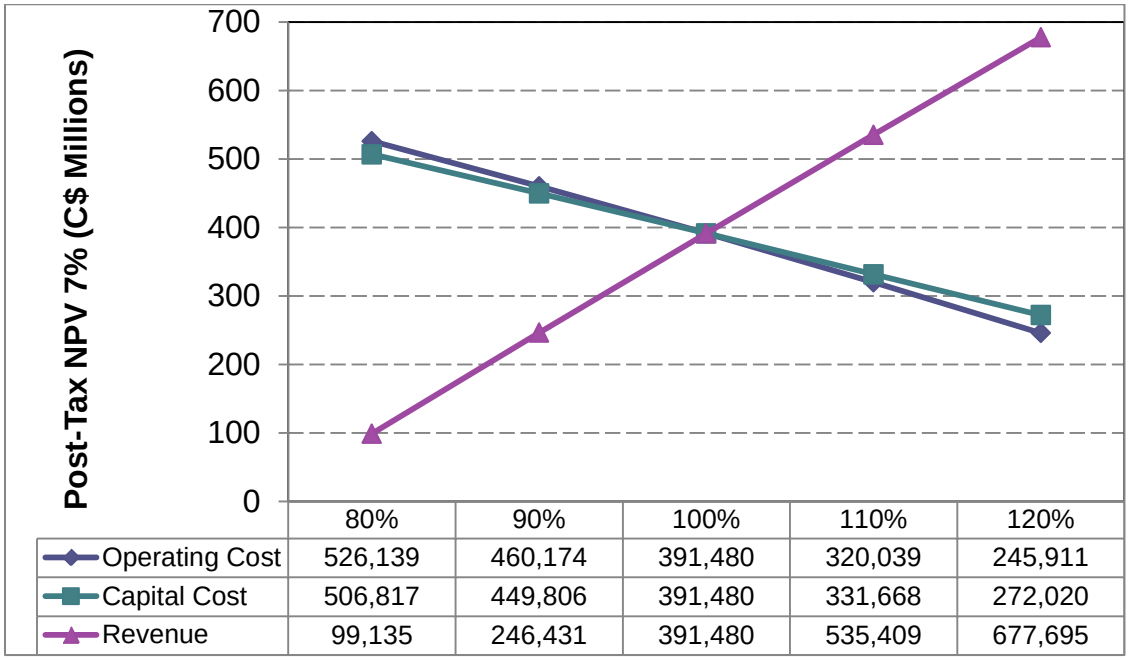
	80%	90%	100%	110%	120%
Operating Cost	19.6%	18.0%	16.3%	14.6%	12.8%
Capital Cost	21.2%	18.6%	16.3%	14.4%	12.6%
Revenue	9.5%	13.1%	16.3%	19.3%	22.1%

Figure 22.1: Post-Tax IRR Sensitivities



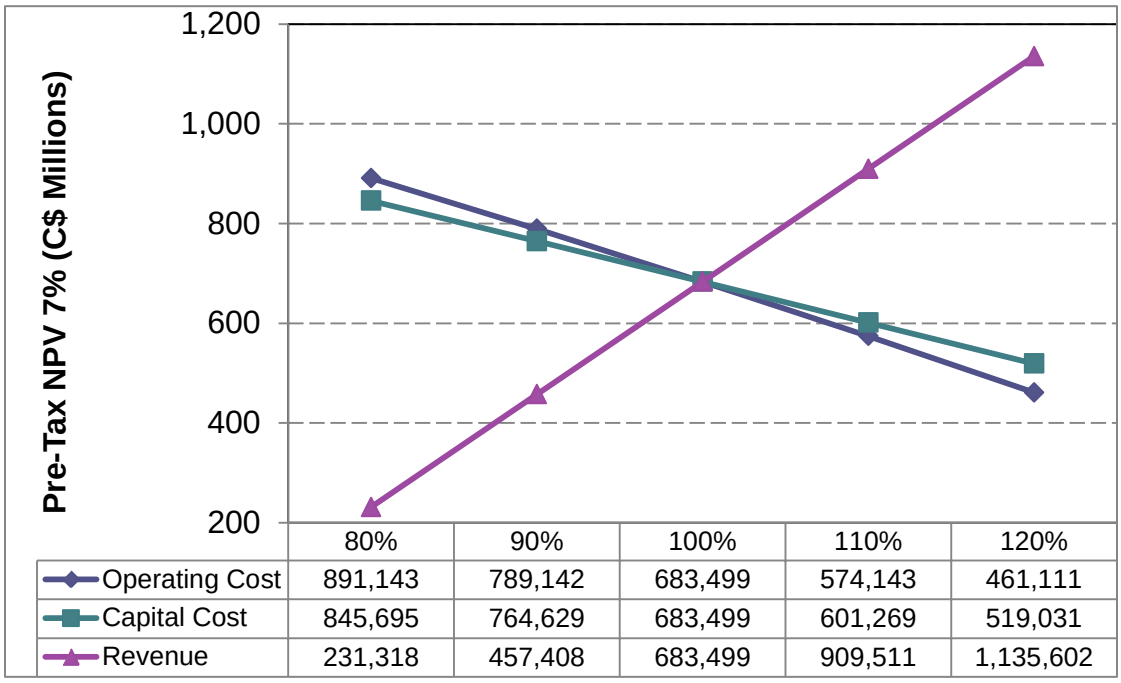
	80%	90%	100%	110%	120%
Operating Cost	24.3%	22.3%	20.3%	18.3%	16.1%
Capital Cost	26.0%	22.9%	20.3%	18.0%	16.0%
Revenue	12.1%	16.4%	20.3%	23.9%	27.2%

Figure 22.2 Pre-Tax IRR Sensitivities



	80%	90%	100%	110%	120%
Operating Cost	526,139	460,174	391,480	320,039	245,911
Capital Cost	506,817	449,806	391,480	331,668	272,020
Revenue	99,135	246,431	391,480	535,409	677,695

Figure 22.3: Post Tax NPV 7% Sensitivities



	80%	90%	100%	110%	120%
Operating Cost	891,143	789,142	683,499	574,143	461,111
Capital Cost	845,695	764,629	683,499	601,269	519,031
Revenue	231,318	457,408	683,499	909,511	1,135,602

Figure 22.4: Pre-Tax NPV 7% Sensitivities

Table 22.9: Diamond Price Scenarios

De-escalated Nominal C$ 000s, Post-Tax									
Diamond Price Scenarios	**NPV 0%**	**NPV 5%**	**NPV 6%**	**NPV 7%**	**NPV 8%**	**NPV 9%**	**NPV 10%**	**Payback Period**	**Equity IRR**
WWW Base	1 003 229	500 684	428 573	363 597	304 991	252 083	204 276	4,82	16,0%
WWW High	1 736 815	981 794	873 213	775 244	686 737	606 678	534 177	3,61	23,9%
WWW Min	746 073	323 686	263 493	209 396	160 739	116 946	77 504	5,60	12,3%
WWW Base + 10%	1 242 300	662 102	578 574	503 212	435 141	373 593	317 885	4,17	19,0%
BFS Base Case	1 072 568	536 607	460 201	391 480	329 604	273 837	223 528	4,82	16,3%
Alternative Model	1 280 323	684 595	598 805	521 399	451 483	388 265	331 047	4,24	19,2%

De-escalated Nominal C$ 000s, Pre-Tax									
Diamond Price Scenarios	**NPV 0%**	**NPV 5%**	**NPV 6%**	**NPV 7%**	**NPV 8%**	**NPV 9%**	**NPV 10%**	**Payback Period**	**Equity IRR**
WWW Base	1 563 192	835 602	731 748	638 315	554 168	478 310	409 859	4,69	19,9%
WWW High	2 757 923	1 605 713	1 440 757	1 292 132	1 158 048	1 036 930	927 393	3,51	29,3%
WWW Min	1 147 266	557 703	474 161	399 200	331 880	271 375	216 952	5,50	15,5%
WWW Base + 10%	1 950 305	1 091 067	967 982	857 099	757 094	666 799	585 185	4,08	23,5%
BFS Base Case	1 675 728	893 849	783 016	683 499	594 041	513 540	441 026	4,69	20,3%
Alternative Model	2 013 804	1 129 132	1 002 353	888 133	785 112	692 092	608 013	4,16	23,8%

Note 1: Grey shading presents Base Case assumption

2: Alternative Model corresponds to “Individual Size Distribution Scenario” presented in Chapter 19 on Marketing

3: The *WWW Base + 10% scenario* is based on the May 2011 Valuation exercise in which WWW’s own achieved diamond price was 10% higher than the average of the five valuations used to establish the Base Case diamond price model

Table 22.10: Diamond Price Escalation Rate Scenarios

De-escalated Nominal C$ 000s, Post-Tax									
Diamond Price Escalation Rate	NPV 0%	NPV 5%	NPV 6%	NPV 7%	NPV 8%	NPV 9%	NPV 10%	Payback Period	Equity IRR
0,00%	513 783	175 708	127 721	84 661	45 999	11 268	(19 945)	6,10	9,3%
1,25%	776 107	345 926	284 693	229 665	180 169	135 613	95 474	5,40	12,9%
2,50%	1 072 568	536 607	460 201	391 480	329 604	273 837	223 528	4,82	16,3%
3,75%	1 408 100	750 573	656 802	572 430	496 425	427 879	365 995	4,23	19,7%
5,00%	1 786 773	989 855	876 266	774 062	681 982	598 922	523 911	3,85	22,9%

De-escalated Nominal C$ 000s, Pre-Tax									
Diamond Price Escalation Rate	NPV 0%	NPV 5%	NPV 6%	NPV 7%	NPV 8%	NPV 9%	NPV 10%	Payback Period	Equity IRR
0,00%	768 258	320 885	257 749	201 187	150 479	104 989	64 158	6,01	11,8%
1,25%	1 193 417	589 802	504 383	427 751	358 934	297 079	241 435	5,27	16,2%
2,50%	1 675 728	893 849	783 016	683 499	594 041	513 540	441 026	4,69	20,3%
3,75%	2 222 279	1 236 732	1 096 932	971 350	858 402	756 698	665 014	4,15	24,2%
5,00%	2 841 491	1 623 292	1 450 492	1 295 243	1 155 580	1 029 782	916 334	3,77	28,1%

Note 1: Grey shading presents base case assumption

Table 22.11: C$/US$ Exchange Rate Scenarios

De-escalated Nominal C$ 000s, Post-Tax									
Exchange Rate C$/US$	NPV 0%	NPV 5%	NPV 6%	NPV 7%	NPV 8%	NPV 9%	NPV 10%	Payback Period	Equity IRR
0,90	838 185	378 772	313 604	255 102	202 535	155 262	112 717	5,45	13,3%
0,95	955 063	457 535	386 770	323 177	265 973	214 467	168 054	5,11	14,8%
1,00	1 072 568	536 607	460 201	391 480	329 604	273 837	223 528	4,82	16,3%
1,05	1 189 950	615 402	533 341	459 480	392 927	332 893	278 686	4,55	17,8%
1,10	1 306 903	693 732	606 015	527 019	455 791	391 496	333 398	4,19	19,1%

De-escalated Nominal C$ 000s, Pre-Tax									
Exchange Rate C$/US$	NPV 0%	NPV 5%	NPV 6%	NPV 7%	NPV 8%	NPV 9%	NPV 10%	Payback Period	Equity IRR
0,90	1 295 726	644 732	552 941	470 681	396 888	330 630	271 086	5,33	16,7%
0,95	1 485 585	769 195	667 890	577 007	495 388	422 013	355 989	5,00	18,5%
1,00	1 675 728	893 849	783 016	683 499	594 041	513 540	441 026	4,69	20,3%
1,05	1 865 852	1 018 464	898 101	789 947	692 650	605 022	526 016	4,39	22,0%
1,10	2 055 979	1 143 094	1 013 203	896 414	791 278	696 523	611 028	4,11	23,7%

Note 1: Grey shading presents base case assumption

Table 22.12: Oil Price Scenarios

De-escalated Nominal C$ 000s, Post-Tax									
Oil Price (US$/bbl)	**NPV 0%**	**NPV 5%**	**NPV 6%**	**NPV 7%**	**NPV 8%**	**NPV 9%**	**NPV 10%**	**Payback Period**	**Equity IRR**
75	1 102 316	557 217	479 463	409 514	346 518	289 727	238 479	4,74	16,7%
85	1 087 535	546 976	469 892	400 553	338 114	281 831	231 050	4,78	16,5%
95	1 072 568	536 607	460 201	391 480	329 604	273 837	223 528	4,82	16,3%
105	1 057 780	526 355	450 618	382 506	321 187	265 929	216 086	4,86	16,1%
115	1 042 807	515 971	440 912	373 418	312 662	257 919	208 548	4,90	15,9%

De-escalated Nominal C$ 000s, Pre-Tax									
Oil Price (US$/bbl)	**NPV 0%**	**NPV 5%**	**NPV 6%**	**NPV 7%**	**NPV 8%**	**NPV 9%**	**NPV 10%**	**Payback Period**	**Equity IRR**
75	1 723 851	926 029	812 871	711 245	619 870	537 624	463 519	4,61	20,8%
85	1 699 939	910 039	798 036	697 458	607 036	525 657	452 342	4,65	20,6%
95	1 675 728	893 849	783 016	683 499	594 041	513 540	441 026	4,69	20,3%
105	1 651 816	877 859	768 181	669 712	581 207	501 573	429 849	4,73	20,1%
115	1 627 605	861 669	753 162	655 754	568 213	489 457	418 534	4,78	19,8%

Note 1: Grey shading presents base case assumption

Table 22.13: Diamond Grade Scenarios

De-escalated Nominal C$ 000s, Post-Tax									
Ore Grade	NPV 0%	NPV 5%	NPV 6%	NPV 7%	NPV 8%	NPV 9%	NPV 10%	Payback Period	Equity IRR
90%	824 015	368 900	304 369	246 448	194 412	147 625	105 527	5,51	13,1%
95%	948 018	452 644	382 197	318 894	261 955	210 693	164 502	5,13	14,7%
100%	1 072 568	536 607	460 201	391 480	329 604	273 837	223 528	4,82	16,3%
105%	1 197 005	620 271	537 887	463 733	396 911	336 633	282 202	4,53	17,9%
110%	1 320 800	703 277	614 921	535 343	463 585	398 806	340 265	4,16	19,3%

De-escalated Nominal C$ 000s, Pre-Tax									
Ore Grade	NPV 0%	NPV 5%	NPV 6%	NPV 7%	NPV 8%	NPV 9%	NPV 10%	Payback Period	Equity IRR
90%	1 272 791	629 416	538 736	457 484	384 608	319 185	260 402	5,38	16,4%
95%	1 474 268	761 640	660 883	570 498	489 331	416 368	350 719	5,02	18,4%
100%	1 675 728	893 849	783 016	683 499	594 041	513 540	441 026	4,69	20,3%
105%	1 877 166	1 026 014	905 102	796 451	698 701	610 661	531 280	4,37	22,1%
110%	2 078 614	1 158 205	1 027 216	909 431	803 391	707 811	621 565	4,07	23,9%

Note 1: Grey shading presents base case assumption

23.0 ADJACENT PROPERTIES

23.1 Diamond Properties

There are no advanced stage diamond exploration properties in proximity to the Foxtrot Property.

In 2004, Majescor Resources Inc. (Majescor) discovered two occurrences of diamond-bearing kimberlite float to the west of the Foxtrot Property claims. A sample of the float processed in 2004 indicated it was diamondiferous with a total of 32 diamonds greater than 0.075 mm recovered from 136 kg of kimberlite (Majescor, 2004). In 2005, Majescor drilled up-ice of the float and discovered the Remick dyke. Additional drilling delineated the dyke for a strike length of 900 m and the kimberlite dyke remains open to the northwest and down dip (Majescor, 2006).

During 2006, Forest Gate Resources, Majescor's joint venture partner at the time, identified a new kimberlite boulder dispersal train located near the Remick dyke. A 54.15 kg coarse kimberlite float sample (U0341-110) collected from this train returned a total of 83 diamonds (Forest Gate Resources, 2006; Majescor Resources Inc., 2007).

Other kimberlite discoveries in the region include:

- Lac Beaver kimberlite, located approximately 90 km south of the Renard kimberlites on claims held by Ditem Explorations Inc (O'Hara, 2004);
- H1, H2, H3 and H4 kimberlites on the Tichegami property held by Ditem Explorations Inc. located approximately 75 km south of the Renard bodies (Robertson, 2003); and
- Hotish 1, 2 and 3 kimberlite dykes, within the Hotish Property held by Dios Exploration Inc., located about 100 km south of the Renard Kimberlites (Vaaldiam Resources Inc., 2006).

23.2 Other Commodities

A number of other commodities have been identified in the general region of the Renard kimberlites, primarily to the south. These include (Houle, 2006):

- Gold and copper mineralization in the Eastmain River greenstone belt (e.g., the Eastmain gold deposit, held by Eastmain Resources Inc. through its wholly-owned subsidiary, Eastmain Mines Inc.);
- Porphyry copper–molybdenum mineralization in the Opatica Subprovince (e.g., the Macleod Lake/Windy 1 deposit, held by Western Troy Capital Resources Inc.); and
- Uranium in the Otish Basin (e.g., the AM-15 zone, held by Strateco Resources Inc.).

Future work at the specific mineral occurrences listed above could have implications to the Renard Project in that they could lead to cost sharing through power line and/or road (winter/all-weather) development activities.

24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 Project Execution Plan

The project execution plan selected is one whereby SWY will directly manage all aspects of all mining activities, including ramp development, and the EPCM contractor will manage the remainder of the construction scope, as described further in this document. Moving forward, the mechanics of the plan will be assessed to ensure the most efficient management approach is adopted.

The scope, responsibility and the overall site will be divided into separate zones as follows:

Zone 1: Construction Zone – EPCM responsibility

The scope of Zone 1 consists of Site preparation, construction of roads, platforms and ditches, the mineral processing plant, the on-site utilities and infrastructures as well as the new airport facilities and runway.

Zone 2: Mining Zone – Stornoway responsibility

The scope of Zone 2 also includes the mining work to be carried out by Stornoway's work force, such as the mine ramp, mine roads, platoforms and ditches, mobile equipment, R2/R3 open pit mining, R2/R3/R4 underground mining, the Renard 65 open-pit mine for the sump, stockpiles, Processed Kimberlite Containment facility and waste rock management.

Figure 24.1 shows the Plant Site and Accommodation Complex and the division of the two separate responsibility zones.

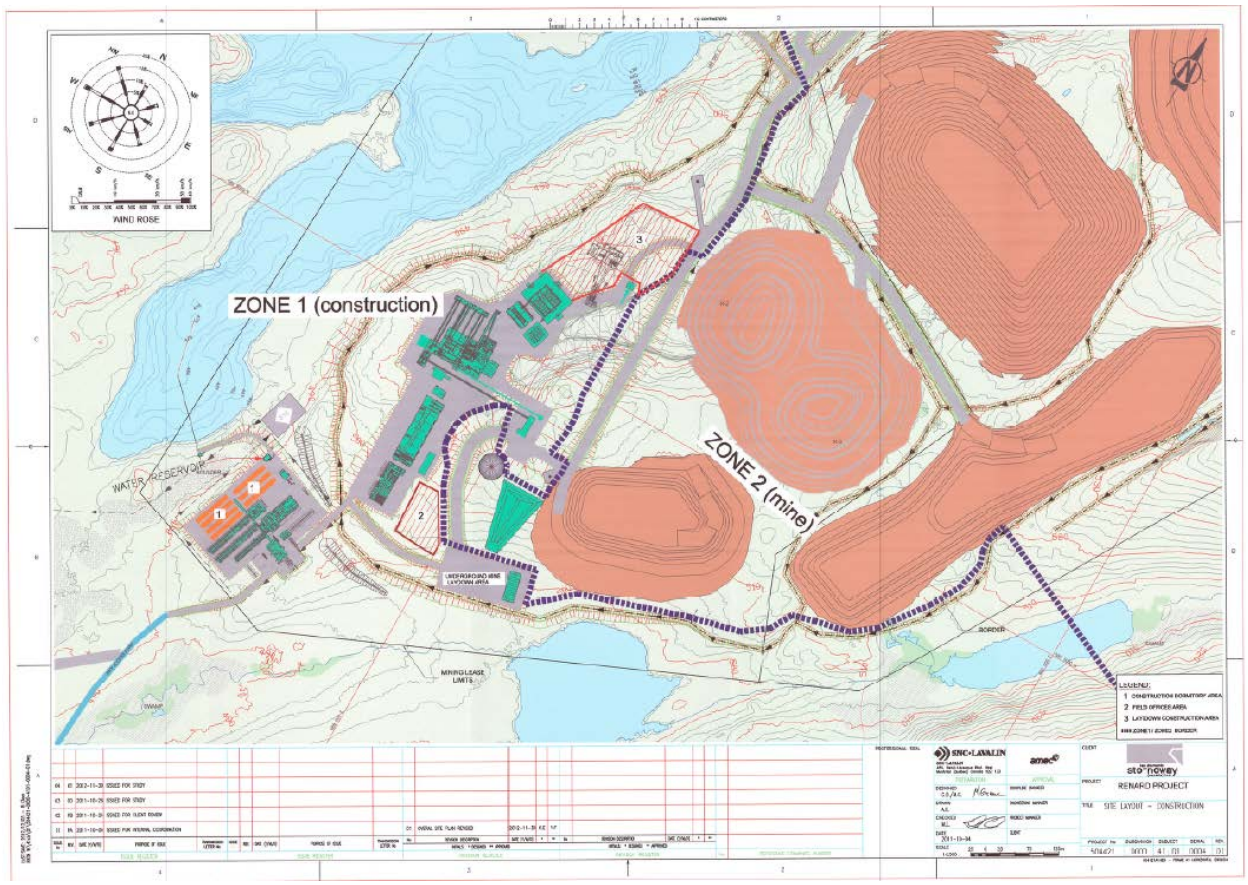


Figure 24.1: Zone 1 and Zone 2 in the Plant site Infrastructure Accommodation Complex

The relationships between the major entities involved in management and execution of the Project are shown below in Figure 24.2.

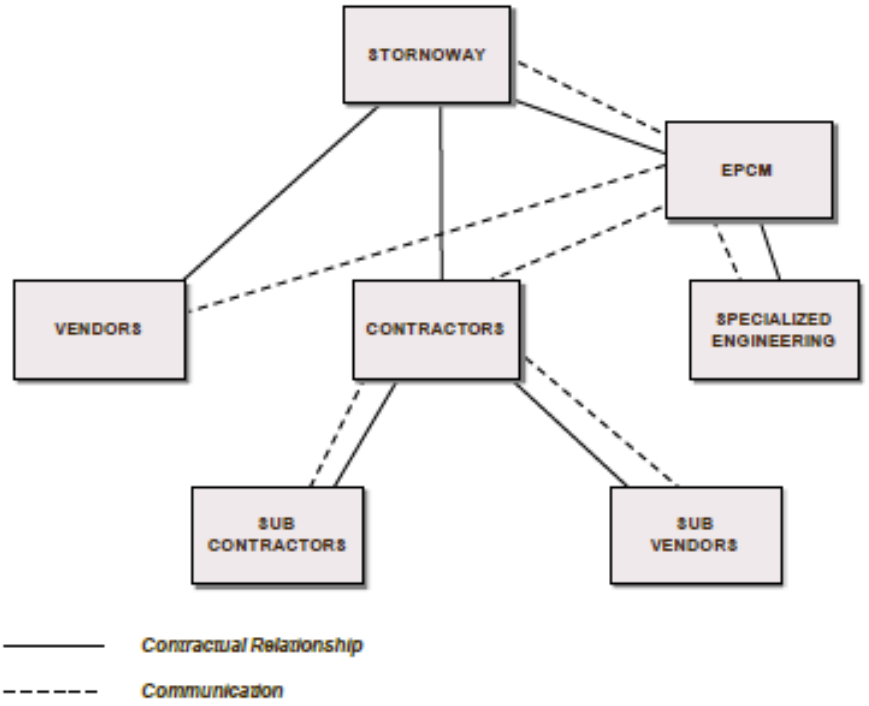


Figure 24.2: Project Structure for Reporting Relationships

Owners Team
The Stornoway project team will be based on the 'best person for the position' philosophy, where certain SWY resources could be seconded within the EPCM Contractor's team.

EPCM Team
The EPCM project organization has been structured into areas created for management purposes. Each area is under the management of an area superintendent who reports to the EPCM construction manager who reports to the EPCM project manager. The two areas are:

- Process Plant
- Infrastructures

Construction Strategy
The overall construction strategy is to adopt a low-risk execution plan ensuring the turnover of the completed facilities in accordance with the project schedule.

The high level construction strategy involves four main contracts in two geographical areas, allowing for the agglomeration of these eight contract elements in the development of the construction contracting philosophies, the two main area being:

- The plant site;
- The airstrip.

The four main contract elements are:

- Process plant;
- Airstrip;
- Infrastructures, such as water supply, hydrocarbon depots, sewage treatment, etc;
- Major Buildings, i.e. Truckshop and warehouse, process plant, and mine office.

Early Construction Activities
In order to perform early works, it is mandatory to access the site from the winter road under the MTQ responsibility. This temporary access to site is forecasted, depending on weather, to be available from March 1st 2013 to the beginning of April 2013. The road will allow the EPCM contractor to mobilize equipment and fuel required on site to execute early works during summer and fall 2013, prior to completion of the first rough access road to site planned in December 2013.

Early works activities are mainly required to do the site preparation of the main roads and the camp area in order to be able to receive the permanent camp in January of 2014. These activities include: tree clearing, draining the lakes at R2 and R65, clearing the overburden on over R65, excavation of the pit at Renard 65 and commencing construction of the drainage ditch network.

Main Mobilization
Once permanent access to site is possible, the priority at the beginning will be the installation of the camp and support suface infrastructure. Stornoway will then mobilize its mining personnel and equipment and will proceed immediately with the development of the ramp and overburden stripping over the R2/R3 Open Pit.

The Civil Contractor will develop the batch and a crushing and screening plant which will be sized to meet scheduled production of crushed and screened rock products. The Civil Contractor will also construct sufficient lay down area for stockpiling each of the required grades.

For the plant, pouring of the concrete foundations will start in the summer of 2014 follwed by structural steel work in the fall. In 2015, the mechanical and electrical contractors will follow the building completion of the process and the infrastructures at the same time.

24.2 Operating Plan

The Operating Plan is designed to provide an effective management scheme with a focus on health and safety, and environmental compliance. The philosophy underlying the Operating Plan is to promote a safe and productive work environment through the implementation of robust health, safety and environmental management systems. Programs of preventative maintenance and continuous improvement will be designed to support the Operating Plan and ensure high standards are established and improved as the project advances. A commitment to training is also an important component of the plan.

The company structure, which includes personnel responsible for site operations, human resources, health and safety, environmental management, community relations, procurement, accounting and diamond sales and marketing, will be designed to accommodate the remote, fly-in/fly-out nature of the Renard Project. All key positions will be supported by adequately trained subordinate personnel who can assume decisional responsibility in the absence of senior management. This will avoid delays and ensure continuous, stable operations by maintaining consistent management. The ability to substitute between positions is a key mitigation factor to address the risk associated with employee absences or departures, particularly those that are unplanned.

A key element of the operating plan will be site security which will include a robust system of access control supported by specific security procedures. The philosophy underlying the security strategy is to limit access and circulation on site and to keep contact with externals to a minimum.

24.3 Schedule

Because the Project is divided into two responsibilities (Mining and Construction) and site access for a large manpower force and equipment will be needed, a precise schedule of activities and camp availabilities has been prepared. Road access to the site is expected by December 2013 therefore major work can commence after that date.

Execution Strategy

- The Project Execution Schedule assumes that the construction site will be closed for 10 days per year during the Christmas breaks.

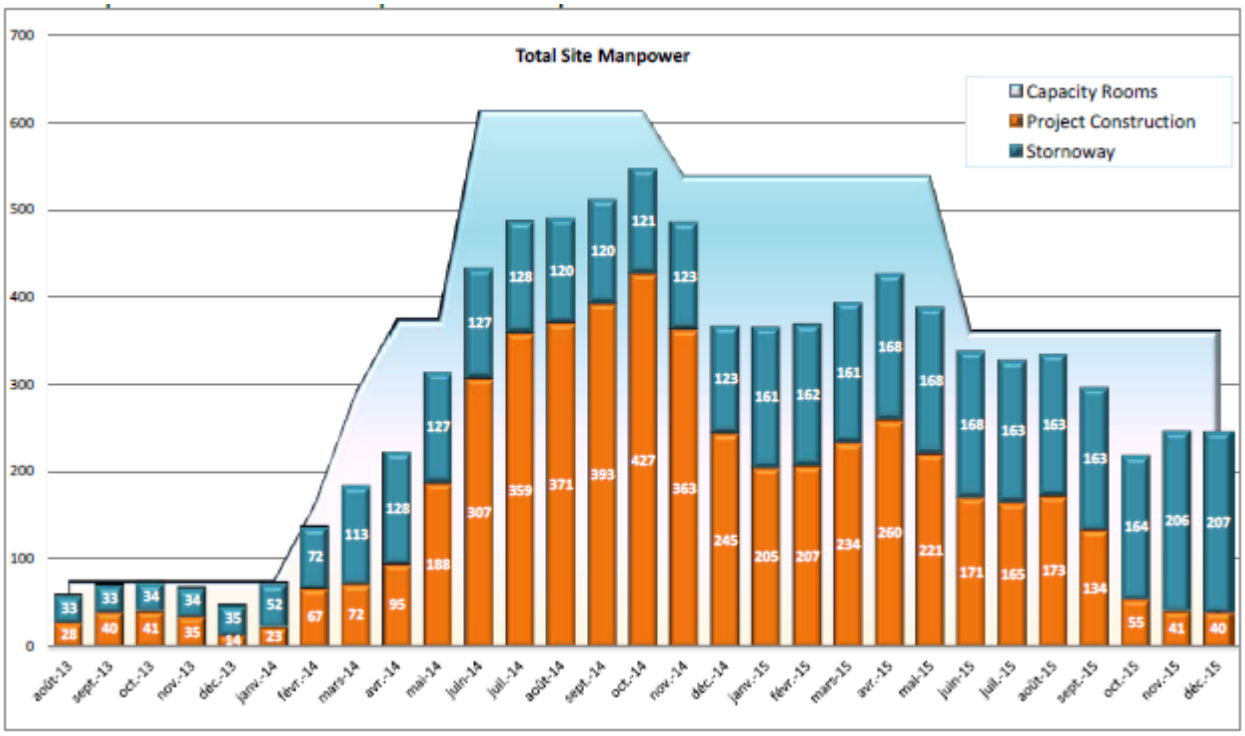
- The standard construction workweek will be 70 hours per week (7 days of 10 hours) with one shift per day. Work will be carried out in accordance with the Québec Construction Decree. The labour rotation schedule will be 4 weeks on and 1 week off.

- The severe winter conditions that can be encountered at the site result in a reduced level of activity in peak winter months; civil and concrete work cannot be scheduled during the winter. Construction hours reflect the expected reduced productivity factors for site work.

- SNC-Lavalin Inc. (SLI) and AMEC Americas Limited (AMEC) have estimated the direct and indirect construction hours for the EPCM responsibility (Zone 1) and the Mining Contractor (Zone 2) respectively and the estimate totals approximately 1.2 million man-hours. The total estimated man-hours for the work carried out by Stornoway personnel (Zone 2) have been estimated by Stornoway and amount to approximately 1.7 million hours

- The manpower estimate shows two peaks: one in late 2014 when rough access to the site is achieved and site development has started, and a second peak forecasted to occur in Q2, 2015 when extensive work will be done in the process plant. The total Site Manpower is shown in Table 24.1.

Table 24.1: Total Estimated Site Manpower



The total estimated manpower on site, shown in Table 24.2, shows the estimated number of personnel on site monthly versus the available accommodation. The mobilization of construction personnel must be carefully managed to support the priorities established by the Project Schedule.

Table 24.2: Stornoway, Project Construction – Estimated Manpower on Site

	MANPOWER		2013					2014											2015											
DISCIPLINE NAME	AVERAGE		Aug	Sept	Oct	Nov	Dec	Jan	Feb	Mar	Avr	May	June	July	Aug	Sept	Oct	Nov	Dec	Jan	Feb	Mar	Avr	May	June	July	Aug	Sept	Oct	Nov	Dec
CIVIL-(SWY)			1	5	0	0	0	0	7	8	17	32	22	23	23	22	23	15	0	0	0	0	0	4	5	5	5	5	0		
CIVIL-(Projet)			21	28	34	28	5	3	11	14	38	74	87	91	79	27	26	19	13	0	0	0	0	0	0	0	0	0	0	
CONCRETE			0	0	0	0	0	5	5	0	0	26	116	135	137	126	83	19	0	0	0	0	0	0	0	0	0	0	0	
STEEL			0	0	0	0	0	0	0	0	0	0	0	4	3	30	37	44	38	42	25	32	23	0	0	0	0	0	0	
ARCHITECURAL			0	0	0	0	0	0	24	22	8	9	19	27	36	52	81	89	38	14	28	40	27	10	4	3	3	2	0	
MECHANICAL			0	0	0	0	0	1	1	0	0	0	0	9	10	19	53	55	60	40	41	44	91	100	83	77	69	40	0	
PIPING			0	0	0	0	0	0	0	0	0	0	0	0	7	17	17	17	12	22	20	24	23	15	15	15	15	12	0	
ELECT. & INSTRUMENTATION			0	0	0	0	0	0	0	0	0	0	1	3	7	28	30	36	29	38	42	41	39	38	19	20	20	19	11	
Sub-Total (Disciplines)	**138.00**		**22**	**33**	**34**	**28**	**5**	**10**	**48**	**44**	**62**	**142**	**244**	**291**	**301**	**320**	**350**	**293**	**190**	**156**	**157**	**180**	**203**	**168**	**125**	**120**	**111**	**78**	**11**	
VPO Direct Labour	-																											16	16	16	16	16
SITE DIRECT MP	**139.83**		**22**	**33**	**34**	**28**	**5**	**10**	**48**	**44**	**62**	**142**	**244**	**291**	**301**	**320**	**350**	**293**	**190**	**156**	**157**	**180**	**203**	**168**	**125**	**120**	**127**	**94**	**27**	**16**	**16**
Const. Supervision (Site Indirect)-**10% Direct**	13.13		2	3	3	3	1	1	5	4	6	14	24	29	30	32	35	29	19	16	16	18	20	17	12	12	13	9	3	2	2
Construction Site Support Labour (Indirect)- **5% Direct**	6.57		1	2	2	1	0	1	2	2	3	7	12	15	15	16	17	15	9	8	8	9	10	8	6	6	6	5	1	1	1
CONTRACTOR SITE INDIRECT MP	**19.70**		**3**	**5**	**5**	**4**	**1**	**2**	**7**	**7**	**9**	**21**	**37**	**44**	**45**	**48**	**52**	**44**	**28**	**23**	**24**	**27**	**30**	**25**	**19**	**18**	**19**	**14**	**4**	**2**	**2**
Construction Management (CM MFL TOTAL)	16.48		2	2	2	2	6	9	10	17	19	20	21	20	20	20	20	21	21	21	21	21	21	22	22	22	21	21	19	18	17
CM Support-Other **25% CM**	4.12		1	1	1	1	2	2	3	4	5	5	5	5	5	5	5	5	5	5	5	5	5	6	6	6	5	5	5	5	4
CONSTRUCTION MANAGMEMENT SITE INDIRECT MP	**20.60**		**3**	**3**	**3**	**3**	**8**	**11**	**13**	**21**	**24**	**25**	**26**	**25**	**25**	**25**	**25**	**26**	**26**	**26**	**26**	**26**	**26**	**28**	**28**	**28**	**26**	**26**	**24**	**23**	**21**
PROJECT CONSTRUCTION TOTAL SITE MP	**184.34**		**28**	**40**	**41**	**35**	**14**	**23**	**67**	**72**	**95**	**188**	**307**	**359**	**371**	**393**	**427**	**363**	**245**	**205**	**207**	**234**	**260**	**221**	**171**	**165**	**173**	**134**	**55**	**41**	**40**
STORNOWAY Site Labour Manning	**123.35**		**33**	**33**	**34**	**34**	**35**	**52**	**72**	**113**	**128**	**127**	**127**	**128**	**120**	**120**	**121**	**123**	**123**	**161**	**162**	**161**	**168**	**168**	**168**	**163**	**163**	**163**	**164**	**206**	**207**
TOTAL ON SITE (STORNOWAY + PROJECT)	**307.69**		**61**	**73**	**75**	**69**	**49**	**75**	**139**	**185**	**223**	**315**	**434**	**488**	**491**	**513**	**548**	**487**	**368**	**367**	**369**	**395**	**427**	**389**	**339**	**328**	**336**	**297**	**219**	**247**	**247**
VACANT ROOMS			**14**	**2**	**0**	**6**	**26**	**0**	**24**	**106**	**152**	**60**	**179**	**125**	**122**	**100**	**65**	**51**	**170**	**171**	**169**	**143**	**111**	**149**	**23**	**34**	**26**	**65**	**143**	**115**	**115**
Temporary (exploration area) Camp Capacity (75)			**75**	**75**	**75**	**75**	**75**	**75**	**75**	**75**	**75**	**75**	**75**	**75**	**75**	**75**	**75**														
Construction Camp Capacity (308)									**88**	**132**	**132**	**132**	**308**	**308**	**308**	**308**	**308**	**308**	**308**	**308**	**308**	**308**	**308**	**308**	**308**	**132**	**132**	**132**	**132**	**132**	**132**
Permanent Accommodation Capacity (230)										**84**	**168**	**168**	**230**	**230**	**230**	**230**	**230**	**230**	**230**	**230**	**230**	**230**	**230**	**230**	**230**	**230**	**230**	**230**	**230**	**230**	**230**
Total Capacity Rooms (618)			**75**	**75**	**75**	**75**	**75**	**75**	**163**	**291**	**375**	**375**	**613**	**613**	**613**	**613**	**613**	**538**	**538**	**538**	**538**	**538**	**538**	**538**	**538**	**362**	**362**	**362**	**362**	**362**	**362**

The Project Schedule has taken the long delivery items, such as X-Ray Sorters (44 weeks), High Pressure Grinding Rolls (54 weeks), Generator Sets (46 weeks first unit; 5 weeks per additional unit), and the conveyor package (55 weeks), into account. During detailed engineering, the project schedule will need to be updated to include all commitment packages.

Basis for the Schedule

The three key milestones for the start of the Project Schedule:

- Ground access to the site;
- EPCM contract award and;
- The start of Detailed Engineering.

Stornoway will obtain and manage the permits.The Global Certificate of Authorization for the project has been received and other specific provincial and federals permits will be obtained through 2013 as required to support the detailed schedule.

Access to the site is a key parameter to the Project Execution Schedule. An agreement has been established by Stornoway with the MTQ to complete a 4-season road access to the site. This work was expected at the time to be carried out in its entirety by the MTQ.

In 2012 Stornoway requested SNC-Lavalin to revise the project schedule based on an access to site date of December 2013 due to delay reported by the MTQ in the construction of the road. Due to cashflow constraints and late road access the permanent airstrip works were delayed and limited airstrip access is now expected by August 2014. The project schedule and its estimate are now based on temporary airstrip access (R2/R3 area) until August 2014.

An existing 75 person exploration camp and a limited amount of equipment will facilitate the initial site mobilization and construction activities. Early works are also dependent on a winter road access during winter 2013. Activities on R65 are planned to commence earlier than in the original 2011 FS schedule. These works are the excavation of waste rock and the use of this rock mine road construction, mine platforms, stockpile preparation and for other construction requirements for waste rocks.

The overall project schedule is an integrated EPCM schedule comprised of all engineering, procurement, construction and Pre-Operational Verification (POV) activities performed by the EPCM contractor and developed by SLI. The underground mining schedule was prepared by AMEC – Mining under Stornoway's supervision.

Early work packages will allow the following works to be performed, between January 2013 and December, 2013.

- Tree clearing;
- Early Civil Works including pads, laydowns, road and ditches;
- Crushing Station;
- Petroleum Product Shipment and Storage;
- Lake Dewatering.

Following the reception of the permanent accommodations, the preparation of all temporary areas and essential facilities, the excavation and foundation will commence for the critical areas. Major concrete works are planned to begin in June 2014.

The Process plant foundation needs to be well advanced in the third quarter of 2014 to allow the start of steel work in September 2014 and continue the principal path for completion in December 2015.

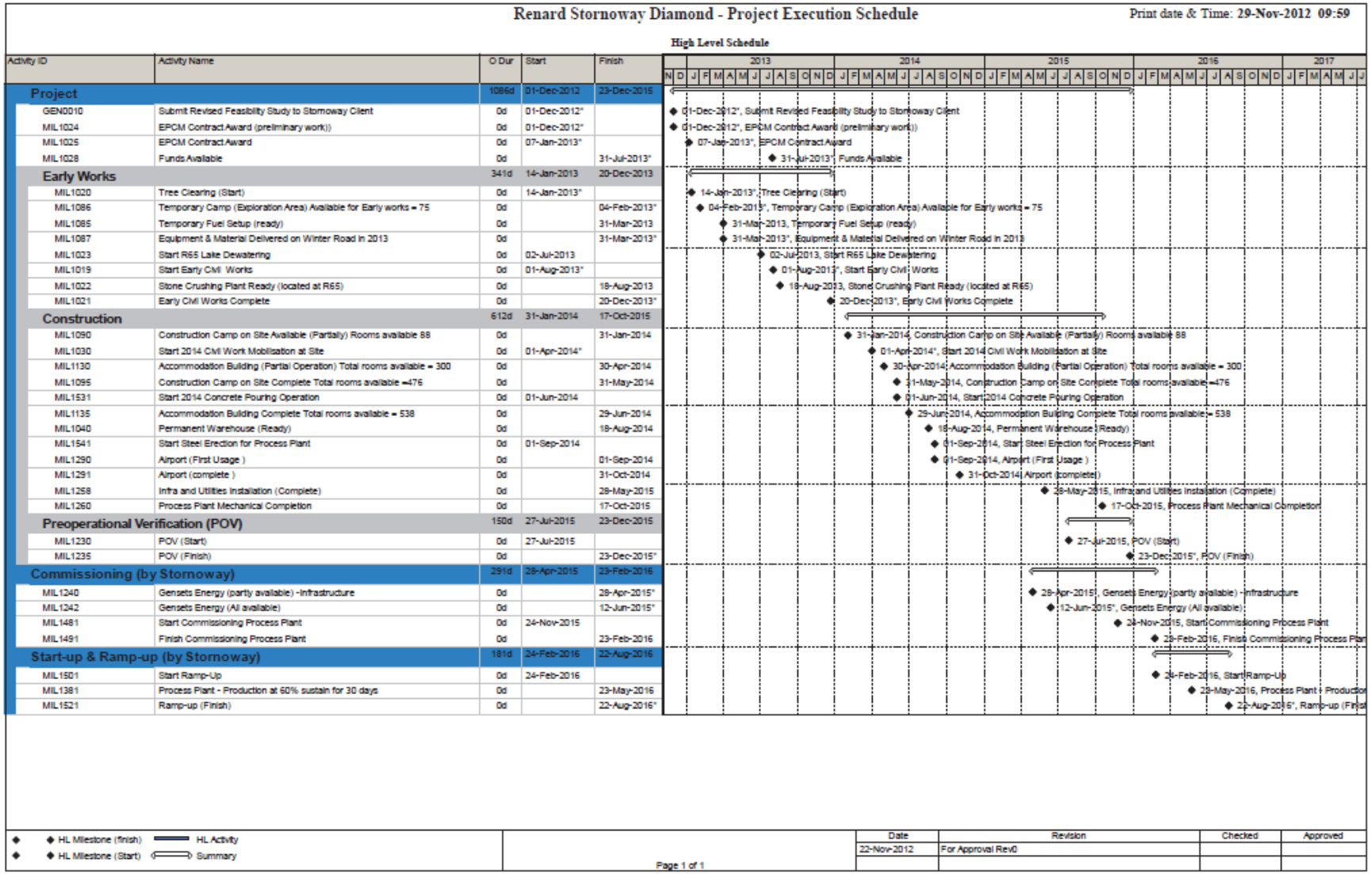
Critical Paths

There is one major critical path in the project execution schedule. The critical path leads to the availability of the process plant start-up to achieve 60 % ramp-up milestone.

In general, the sequence of construction activities in the process plant area starts with the concrete work on the foundations and ends with the electrical and instrumentation installation followed by pre-operational verification (POV - EPCM contractor) and commissioning (Stornoway) activities. These activities are also expected to take place simultaneously (overlapping) in sub-areas inside the process plant area. Schedule activities preceding the process plant construction such as engineering and procurement activities and infrastructure-related activities were also taken into account in the critical path analysis.

An overview of the construction activities is shown in Table 24.3

Table 24.3: Construction Activities



Renard Stornoway Diamond - Project Execution Schedule

Print date & Time: 29-Nov-2012 09:59

High Level Schedule

Activity ID	Activity Name	O Dur	Start	Finish
Project		1086d	01-Dec-2012	23-Dec-2015
GEN0010	Submit Revised Feasibility Study to Stornoway Client	0d	01-Dec-2012*	
MIL1024	EPCM Contract Award (preliminary work))	0d	01-Dec-2012*	
MIL1025	EPCM Contract Award	0d	07-Jan-2013*	
MIL1028	Funds Available	0d		31-Jul-2013*
Early Works		341d	14-Jan-2013	20-Dec-2013
MIL1020	Tree Clearing (Start)	0d	14-Jan-2013*	
MIL1086	Temporary Camp (Exploration Area) Available for Early works = 75	0d		04-Feb-2013*
MIL1085	Temporary Fuel Setup (ready)	0d		31-Mar-2013
MIL1087	Equipment & Material Delivered on Winter Road in 2013	0d		31-Mar-2013*
MIL1023	Start R65 Lake Dewatering	0d	02-Jul-2013	
MIL1019	Start Early Civil Works	0d	01-Aug-2013*	
MIL1022	Stone Crushing Plant Ready (located at R65)	0d		18-Aug-2013
MIL1021	Early Civil Works Complete	0d		20-Dec-2013*
Construction		612d	31-Jan-2014	17-Oct-2015
MIL1090	Construction Camp on Site Available (Partially) Rooms available 88	0d		31-Jan-2014
MIL1030	Start 2014 Civil Work Mobilisation at Site	0d	01-Apr-2014*	
MIL1130	Accommodation Building (Partial Operation) Total rooms available = 300	0d		30-Apr-2014
MIL1095	Construction Camp on Site Complete Total rooms available =476	0d		31-May-2014
MIL1531	Start 2014 Concrete Pouring Operation	0d	01-Jun-2014	
MIL1135	Accommodation Building Complete Total rooms available = 538	0d		29-Jun-2014
MIL1040	Permanent Warehouse (Ready)	0d		18-Aug-2014
MIL1541	Start Steel Erection for Process Plant	0d	01-Sep-2014	
MIL1290	Airport (First Usage)	0d		01-Sep-2014
MIL1291	Airport (complete)	0d		31-Oct-2014
MIL1258	Infra and Utilities Installation (Complete)	0d		28-May-2015
MIL1260	Process Plant Mechanical Completion	0d		17-Oct-2015
Preoperational Verification (POV)		150d	27-Jul-2015	23-Dec-2015
MIL1230	POV (Start)	0d	27-Jul-2015	
MIL1235	POV (Finish)	0d		23-Dec-2015*
Commissioning (by Stornoway)		291d	28-Apr-2015	23-Feb-2016
MIL1240	Gensets Energy (partly available) -Infrastructure	0d		28-Apr-2015*
MIL1242	Gensets Energy (All available)	0d		12-Jun-2015*
MIL1481	Start Commissioning Process Plant	0d	24-Nov-2015	
MIL1491	Finish Commissioning Process Plant	0d		23-Feb-2016
Start-up & Ramp-up (by Stornoway)		181d	24-Feb-2016	22-Aug-2016
MIL1501	Start Ramp-Up	0d	24-Feb-2016	
MIL1381	Process Plant - Production at 60% sustain for 30 days	0d		23-May-2016
MIL1521	Ramp-up (Finish)	0d		22-Aug-2016*

◆ HL Milestone (finish) ═══ HL Activity

◆ HL Milestone (Start) ═══ Summary

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Date	Revision	Checked	Approved
22-Nov-2012	For Approval Rev0		

24.4 Risk Assessment and Management

Risk identification is the process of examining the various project areas and each critical process, to identify and document any associated potential risks. All threats and opportunities, which are identified as having a potential financial impact on the Project, are evaluated and each risk is placed into the Capex or Opex category. All risks are qualitatively assigned a level for consequence to Capex or Opex should the risk materialize, probability of occurrence and manageability of the risk.

Risks review meetings were held on a regular basis during the feasibility study with key project members and with Stornoway in order to encourage this approach, which was supported by a series of procedures and tools used to identify, analyse, evaluate, communicate, mitigate, monitor and eventually treat the project risks.

Based on the results of the risk workshops held throughout the study, the Renard Project is considered by SNC-Lavalin to have a risk profile consistent with the current stage of its development.

The levels of Probable Consequence, Probability of Occurrence and Manageability Matrix are shown in Table 24.4.

Table 24.4: Levels of Probable Consequence, Probability of Occurrence and Manageability Matrix

Level	**Probable Consequence**	
Very High	0.65% and up	
High	0.35 to 0.65%	
Level	**Probability of Occurrence**	**Description**
High	50% to 70%	Might occur under most circumstances
Medium	30% to 50%	Might occur at some time
Level	**Manageability Matrix**	**Description**
High	60%	In most circumstances can be managed
Medium	40%	Can be managed
Low	20%	In most circumstances difficult to be managed

Capex and Opex Threats

For Capex Threats there are eight (8) risks with a “Very High” probable consequence and for Opex threats there are five (5) risks with a “Very High” probable consequence. A summary of these Risks is shown in Table 24.5.

Table 24.5: Summary of Capex and Opex Threats

Capex Threats	Consequence	Probability	Manageability	Status
Long Lead Items	Very High	Medium	Medium	Watch
Delayed Financing	Very High	Medium	Mediium	Active
Site Access - Terrestrial	Very High	Medium	Low	Watch
Late or no Ice Road Construction	Very High	Medium	Low	Watch
Early Thaw of Ice Road	Very High	Medium	High	Watch
Lack of Skilled Labour for Construction	Very High	High	Medium	Active
Process Plant – Diamond Breakage	Very High	High	Medium	Watch
Cash Flow Restriction	Very High	Medium	Medium	Active
Opex Threats	**Consequence**	**Probability**	**Manageability**	**Status**
Mudrush Potential	Very High	Medium	Low	Active
Ore Dilution	Very High	Medium	Medium	Active
Lack of Skilled Mining Labour	Very High	High	Medium	Active
ROM Production Gaps	Very High	Medium	Low	Active
Deep Underground Water Quality	Very High	Medium	Medium	Active

It will be very important to constantly monitor and review these risks during all phases of the Project until they are completely mitigated and retired, transferred and accepted.

Opportunities

Some of the most significant opportunities that were recognized are listed below:

- A large diamond recovery unit was designed during the Feasibility Study, and is carried as a project option. After the start of commercial production, if there are indications that diamonds larger than the top size retained for the design of the process plant are present and being broken, the large diamond recovery unit designed can be implemented to capitalize on this opportunity.
- Possible change from fiber optic cable to satellite link or microwave link or both links (ID-106): This would reduce recurrent cost of telecommunication system between airstrip and the mining complex.
- The possibility to reduce cost by generating power with liquid natural gaz (LNG) instead of diesel. This Opportunity was raised after the last risk session in November 2012.

25.0 INTERPRETATIONS AND CONCLUSIONS

The Updated Feasibility for the Renard Project, which was prepared by experienced and competent independent consultants using accepted engineering methodologies, resulted in the development and cost estimation of a detailed project development plan. The mine design includes open pit mining of Renard 2 and Renard 3, and underground blast hole shrinkage mining of Renard 2, Renard 3 and Renard 4. All elements of the project development plan, including the required infrastructure, mine design, process plant design, waste disposal strategy and cost estimation, were determined to the feasibility level. The resulting information therefore met all of the requirements for conversion of Indicated Mineral Resources to a Mineral Reserve estimate. The Resource estimate was determined in accordance with CIM Definition Standards classification. Considering the risks inherent in all kimberlite deposits, such as sampling for geological continuity, diamond grade and diamond revenue determination as discussed in Section 14, the Indicated portion of the Mineral Resource is considered suitable for feasibility level mine design. The Mineral Reserve estimate of the Renard Project, which was determined in accordance with CIM Definition Standards classification, is presented in Table 25.1.

Table 25.1: Mineral Reserve Summary – Open Pit and Underground

Pipe	Mineral Reserve[(1)] (t x 1000)	Grade (cpht)	Carats (x 1000)
Open Pit			
R2	1,313	94.6	1,242
R3	723	92.8	671
Sub Total	**2,035**	**94.0**	**1,913**
Underground			
R2	17,029	80.0	13,616
R3	1,003	84.1	843
R4	3,718	42.4	1,575
Sub Total	**21,750**	**73.7**	**16,034**
Total[(2)]	**23,785**	**75.5**	**17,947**

Note: 1 All Mineral Reserves are in the Probable category based on Indicated Mineral Resources with an effective date of January 24, 2011
2 Totals may not equal the sum of the individuals due to rounding.

The mine plan comprises the extraction of 17.0 Mt of waste and 23.8 Mt of ore over an 11.5 year period to produce 17.95 Mct of diamonds for net revenues of C$4, 872 million in nominal dollar terms. Ore will be introduced to a processing plant equipped with three stages of crushing, dense media separation, and X-ray/grease diamond recovery circuits. The plant will employ HPGR crushing technology to minimize diamond damage during liberation. The nameplate capacity of the plant will be 2.16 Mt/a. Processed kimberlite from the plant will be trucked to the PKC facility and disposed of using a dry stacking technique that will allow for progressive closure. Appropriate metallurgical testing was completed to support the dry stacking technique. Water management will include the construction of a drainage network to collect surface water and a catch basin that will be used to collect water for testing and treatment, if necessary, before release to the environment. The supporting infrastructure for the mine includes a 18 MW packaged diesel power plant, 1,494 m gravel airstrip, 4,142 m^2 truckshop, mine offices, emulsion plant, fuel tank farm, 230 bed accommodation complex and effluent

treatment plant. A mine closure plan compliant with current Provincial regulations has been developed.

Access to the site will be via an all-weather road (Route 167) and air charter. The all-weather road is being constructed by the Government of Québec and Stornoway. Stornoway will manage construction of the 2 northern sections of the access road and the government is responsible for the 2 southern sections. Stornoway's portion of the road is being financed by a $77 million loan from the Québec Government. The air charter service will be used primarily to transport personnel to and from the site.

Permitting for the development of the Renard Project must be obtained under the James Bay and Northern Québec Agreement (JBNQA), the *Canadian Environmental Assessment Act*, the *Mining Act* and the *Environment Quality Act*. Following submission of the Project Notice in February 2010, and subsequent modification in February 2011, the Ministère du Développement Durable de l'environment et des Parcs (MDDEP) and the evaluating Committee (COMEV) issued guidelines to Stornoway on the scope of the impact assessment to be carried out for the Project. The Environmental and Social Impact Assessment (ESIA) was conducted in 2010-2011 and was filed on December 23, 2011. Community consultations followed and the Global Certificate of Authorization was received at the end of 2012. . Included in the ESIA is a comprehensive Closure and Rehabilitation plan to support the financial guarantee required by the MRNF (currently 70% of the cost estimate).

The Renard Project is located on Category III lands, as defined by the JBNQA, where Crees have certain rights. On July 26th 2010, Stornoway announced the conclusion of a Pre-Development Agreement (PDA) for the Renard Project with the Grand Council of the Crees (Eeyou Istchee)/Cree Regional Authority, the CNM and DIAQUEM Inc. A formal Impact and Benefits Agreement was subsequently signed by all Stornoway, CNM and the Grand Council of the Crees (Eeyou Istchee)/Cree Regional Authority in 2012.

Capital (capex) and operating (opex) cost estimates developed in the Feasibility Study can be classified as Class III type estimates, per the Association for the Advancement of Cost Engineering (AACE). The capex estimate totaled $752M, excluding risk and escalation, and the estimate accuracy was evaluated at -11%/+19% based on a deterministic approach,. The LOM opex was estimated at $57.63/t. The open pit and underground opex estimates are -15%/+15%.

An economic analysis of the Project resulted in a Post-Tax Net Present Value discounted at 7% (NPV 7%) of $391.5 million and a Pre-Tax Net Present Value of $671.9 million, based on de-escalated nominal dollars. The internal rate of return was determined to be 16.3% on an after-tax basis and 20.3% on a pre-tax basis based on de-escalated nominal dollar cash flows. The payback period following the end of pre-production in May 2016 is 4.8 years on an after-tax basis and 4.7 years on a pre-tax basis. Therefore, based on the designs developed and the assumptions inherent in the Feasibility Study, the cash flow analysis presented in this report demonstrates that the Mineral Reserves associated with the Renard Project can be mined and processed for a profit.

26.0 RECOMMENDATIONS

The following section presents recommendations from the Qualified Persons to advance the Project. Costs for the recommended work are included in each section.

26.1 Mine Design

Recommendations for further work in support of the mine design were provided by AMEC Americas Limited (AMEC), G Mining Services Inc. (GMining) and Itasca Consulting Canada Inc. (Itasca). Costs for this work are estimated to be approximately $400,000.

It is recommended that further work be performed to reduce the uncertainties in the geomechanical and design analysis. This would include the following:

- An analysis to represent the effects of jointing on the rockmass and the effects of major faults either through the Ubiquitous Joint Rockmass MDetailed FLAC3D modeling to assess stability of proposed mining sequence;

Kimberlite weathering and mudrush assessment;

- R4 crown pillar stability analysis;
- Rock and overburden pit slope stability analysis;
- Ongoing review of geostructural model;

Fragmentation studies;

- Panel sizing optimization;
- Draw point design optimization;
- Additionnal REBOP flow modeling to refine recovery and dilution factors as design is updated;
- Underground traffic simulation;
- The open pit size could be reviewed at the detailed engineering phase given the more rR2/R3 ultimate pit shell update;

  An opportunity exists to exceed forecast mining recoveries and reduce dilution below estimates for underground mining through optimized drawpoint design and blasted ore drawdown strategy.

As underground experience is gained at this mine site, these analyses should be continually reviewed to determine if they remain valid over time and to ensure that new information is properly considered into the analysis.

26.2 Process Plant Design

The following recommendations for further work in support of the process plant design were provided by AMEC:

- The feasibility study assumes dewatered fine processed kimberlite will be combined with coarse DMS rejects and temporarily stored in a large load-out bin. It is recommended to verify that the flow characteristics of this material will be suitable for the proposed load-out design.

- Further x-ray sorter testing is recommended to confirm vendor selection. Also some dry high intensity magnetic separation tests should be done to verify the mass reduction assumptions made in the feasibility study.

The cost estimate for this work is approximately $50,000.

26.3 PKC Facility Design

The following recommendations for further work in support of the PKC facility design were provided by Golder:

- Additional centrifuge testing to confirm feasibility of dewatering the PK such that it may be trucked and placed in the PKC facility ($20,000 estimate).
- Testing of PK material (post-production) to include, but not necessarily limited to:
    - Grain size analyses;
    - Evaluation of solids content;
    - Atterberg limits;
    - Standard Proctor maximum dry density;
    - Specific gravity;
    - Direct shear;
    - Triaxial testing;
    - Hydraulic conductivity;
    - X-ray diffraction determination of clay mineralogy;
    - Determination of deposited dry density; and
    - Strength testing.

- Additional post-production testing of waste rock, quarried rock, and PK to confirm the geochemical classification.

Costs estimated at $20,000/yr to test the PK material and to confirm the geochemical classification are included in the Opex.

26.4 Permitting, Environment and Socio-Economics

Recommendations and cost estimates from Roche related to the environmental and social aspects of the Project are as follows:

- Incorporate mitigation measures set out in the Environmental and Social Impact Assessment in the final project design and comply with the applicable standards for projected infrastructures and equipment (minimal cost);
- Prepare framework for permitting requests once global environmental approval is received ($50,000);

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