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TECHNICAL REPORT AND PREFEASIBILITY STUDY FOR THE DUPARQUET PROJECT
(according to National Instrument 43-101 and Form 43-101F1)

Project Location

Latitude: 48º30'34"North ; Longitude: 79º12´34"West
Duparquet Township
Province of Québec, Canada

Prepared for

Clifton Star Resources Inc.
1040, avenue Belvédère, Suite 217
Québec, QC
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Prepared by:

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Effective date: March 26, 2014
Signature date: May 23, 2014

CERTIFICATE OF AUTHOR – KARINE BROUSSEAU

I, Karine Brousseau, Eng (OIQ #121871), do hereby certify that:

1. I am Consulting Geologist of: InnovExplo Inc, 560 3^{e} Avenue, Val d'Or, Québec, Canada, J9P 1S4.

2. I graduated with a Bachelor's degree in Geological Engineering (B.Sc.) from Université Laval (Sainte-Foy, Québec) in 1998

3. I am a member of the *Ordre des Ingénieurs du Québec* (OIQ #121871) and Professional Engineers Ontario (PEO #100156917).

4. I have over fourteen (14) years of experience as a geologist in the exploration and mining industry. My exploration expertise has been acquired with SOQUEM, the Department of Energy and Natural Resourcesof Québec (Geology Branch) and Cambior. I have been a consulting geologist for InnovExplo Inc since September 2005.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined 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 responsible for the preparation of sections 4 to 12, 14, 23 and 24, and co-author of sections 1, 2, 3, and 25 to 27, of the technical report titled "Technical Report and Prefeasibility Study for the Duparquet Project (according to National Instrument 43-101 and Form 43-101F1)", effective date of March 26, 2014 and signature date of May 23, 2014 ("the Technical Report").

7. I have completed a Mineral Resource Estimate on the Duparquet Project on behalf of Clifton Star Resources Inc. in 2012 and 2013.

8. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report 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 the Technical Report has been prepared in compliance with that instrument and form.

Effective Date: March 26, 2014
Signature Date: May 23, 2014

(Original signed and sealed)
Karine Brousseau, Eng.
InnovExplo Inc.

CERTIFICATE OF AUTHOR – LAURENT ROY

I, Laurent Roy, Eng. (OIQ #109779) do hereby certify that:

1. I am a Consulting Engineer of: InnovExplo Inc., 560 3e Avenue, Val-d'Or, Québec, Canada, J9P 1S4.

2. I graduated with a Bachelor's degree in mining Engineering from École Polytechnique (Montréal, Québec) in 1992.

3. I am a member of the Ordre des Ingénieurs du Québec (OIQ #109779).

4. I have worked as an engineer for a total of nineteen (19) years since graduating from university. My mining expertise was acquired while working for Talpa Mining Contractor and Richmont Mines at the Francoeur, Beaufor, Doyon and CasaBerardi mines. I have been a consulting engineer for InnovExplo Inc. since September 2012.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined 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 co-author of sections 1, 2, 3, 16, 21.10 and 25 to 27 of the report titled "Technical Report and Prefeasibility Study for the Duparquet Project (according to National Instrument 43-101 and Form 43-101F1)", effective date of March 26, 2014 and signature date of May 23, 2014 ("the Technical Report") I visited the Duparquet Project site on May 27, 2013, accompanied by Marie-Claire Dagenais of InnovExplo and Louis Martin of Clifton Star.

7. I participated in the preparation of the "Technical Report and Preliminary Economic Assessment for the Duparquet Project", effective date of December 31, 2012 and signature date of February 28, 2013. I did not have any other prior involvement with the property that is the subject of the Technical Report.

8. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which would make the Technical Report 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 respecting standards of disclosure for mineral projects and Form 43-101F1, and the sections of the Technical Report for which I was responsible have been prepared in accordance with that regulation and form.

Effective Date: March 26, 2014
Signature Date: May 23, 2014

(Original signed and sealed)
Laurent Roy, Eng.
InnovExplo Inc.

CERTIFICATE OF AUTHOR – CARL PELLETIER

I, Carl Pelletier, P.Geo. (OGQ #384), do hereby certify that:

1. I am Director Consulting Geologist of: InnovExplo Inc., 560 3e Avenue, Val d'Or, Québec, Canada, J9P 1S4.
2. I graduated with a Bachelor's degree in Geology (B.Sc.) from Université du Québec à Montréal (Montréal, Québec) in 1992, and I initiated a Master's degree at the same university for which I completed the course program but not the thesis.
3. I am a member of the Ordre des Géologues du Québec (OGQ #384), the Association of Professional Geoscientists of Ontario (APGO #1713), and of the Canadian Institute of Mines (CIM), Harricana Section.
4. I have worked as a geologist for a total of twenty (20) years since my graduation from university. My mining expertise has been acquired at the Silidor, Géant Dormant, Bousquet II, Sigma-Lamaque and Beaufor mines, and my exploration experience has been acquired with Cambior Inc. and McWatters Mining Inc. I have been a consulting geologist for InnovExplo Inc. since February 2004.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined 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 responsible for supervising the preparation of the Mineral Resource Estimate of the Duparquet Project, as well as supervising the preparation of sections 4 to 12, 14, 23 and 24 of the technical report titled "Technical Report and Prefeasibility Study for the Duparquet Project (according to National Instrument 43-101 and Form 43-101F1)", effective date of March 26, 2014 and signature date of May 23, 2014 ("the Technical Report"). I visited the Duparquet Project on November 16, 2011. I was accompanied by Kenneth Williamson, P.Geo, for a second visit of the Duparquet Project on February 7, 2012.
7. I completed an internal Mineral Resource Estimate on the Beattie Property, which is part of the Duparquet Project, on behalf of Osisko Mining Corporation in 2010. I have completed a Mineral Resource Estimate in 2012 and an NI 43-101 compliant report titled "Technical Report and Preliminary Economic Assessment on the Duparquet Project", signature date of February 28 2013, both on behalf of Clifton Star Resources Inc.
8. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report 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 the sections of the Technical Report, for which I was responsible, have been prepared in accordance with that instrument and form.

Effective Date: March 26, 2014
Signature Date: May 23, 2014

(Original signed and sealed)
Carl Pelletier, P.Geo.
InnovExplo Inc.

CERTIFICATE OF AUTHOR – SYLVIE POIRIER

I, Sylvie Poirier, Eng. (OIQ #112196) do hereby certify that:

1. I am a Consulting Engineer of: InnovExplo Inc., 560 3e Avenue, Val-d'Or, Québec, Canada, J9P 1S4.
2. I graduated with a Bachelor's degree in mining Engineering from École Polytechnique (Montréal, Québec) in 1994.
3. I am a member of the Ordre des Ingénieurs du Québec (OIQ #112196), the Professional Engineers of Ontario (PEO #100156918), and the Canadian Institute of Mines (#145365).
4. I have worked as an engineer for a total of seventeen (17) years since graduating from university. My mining expertise was acquired while working for Lafarge Canada and for Placer Dome and McWatters at the Sigma mine, as well as for Natural Resources Canada on a special research initiative program on narrow-vein mining. I have been a consulting engineer for InnovExplo Inc. since September 2008.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined 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 responsible for the preparation of the report titled "Technical Report and Prefeasibility Study for the Duparquet Project (according to National Instrument 43-101 and Form 43-101F1)", effective date of March 26, 2014 and signature date of May 23, 2014 ("the Technical Report"). I am the author of sections 15, 19, 21.2 to 21.4, 22 and co-author of sections 1, 2, 3, 16, 21.10 and 25 to 27.
7. I participated in the preparation of the "Technical Report and Preliminary Economic Assessment for the Duparquet Project", effective date of December 31, 2012 and signature date of February 28, 2013. I did not have any other prior involvement with the property that is the subject of the Technical Report.
8. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report 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 the sections of the Technical Report for which I was responsible have been prepared in accordance with that instrument and form.

Effective Date: March 26, 2014
Signature Date: May 23, 2014

(Original signed and sealed)
Sylvie Poirier, Eng.
InnovExplo Inc.

CERTIFICATE OF AUTHOR – DAVID DREISINGER

I, David Dreisinger, P.Eng. (APEGBC #15803) do hereby certify that:

1. I am the president of: Dreisinger Consulting Inc. with a business office at 5233 Bentley Crescent, Delta, British Columbia, Canada, V4K 4K2.
2. I am a graduate of Queen's University in Kingston, Canada with a B.Sc. (Metallurgical Engineering, 1980) and a PhD (Metallurgical Engineering, 1984). I am a Fellow of the Canadian Institute of Mining, Metallurgy and Petroleum. I am a Fellow of the Canadian Academy of Engineering.
3. I have practiced my profession continuously since graduation. I have been employed in research and teaching at the University of British Columbia since 1984 and currently hold the title of professor and chairholder, Industrial Research Chair in Hydrometallurgy in the Department of Materials Engineering. I have provided consulting services to the worldwide metallurgical industry since 1987. I have been the President of Dreisinger Consulting Inc since 1998.
4. I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (Registration Number 15803, May 6, 1987).
5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined 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 co-responsible for the preparation of sections 13.2, 13.3 and 17.2 and co-author of sections 1, 2, 3 and 25 to 27 of the technical report titled "Technical Report and Prefeasibility Study for the Duparquet Project (according to National Instrument 43-101 and Form 43-101F1)", effective date of June 26, 2013 and signature date of August 2, 2013 ("the Technical Report").
7. I visited the Duparquet Project site once in September 2011. I also supervised a series of metallurgical test programs at SGS Minerals Laboratory in Lakefield, Ontario, in 2012 and 2013.
8. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which would make the Technical Report misleading.
9. I am NOT independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101. I serve as Vice President of Metallurgy for the issuer.
10. I have read National Instrument 43-101 and Form 43-101F1, and the sections of the Technical Report for which I was responsible have been prepared in accordance with that instrument and form.

Effective Date: March 26, 2014
Signature Date: May 23, 2014

(Original signed and sealed)
David Dreisinger, P.Eng.
Dreisinger Consulting Inc.

CERTIFICATE OF AUTHOR – DAVID SIMS

To accompany the Report entitled “Technical Report and Prefeasibility Study for the Duparquet Project (according to National Instrument 43-101 and Form 43-101F1)”, effective date of March 26, 2014 and signature date of May 23, 2014 (“the Technical Report”).

I, David Sims, do hereby certify that:

1. I am currently employed as a geologist, Mining and Mineral Processing at Roche Ltd Consulting Group, 1015 Wilfrid-Pelletier Avenue, Québec (Québec), G1W 0C4.
2. I graduated from University of Victoria (B.C.) with a B.Sc. in Geology in 1999.
3. I am a Geologist Member of the Ordre des Géologues du Québec (#1431).
4. I have worked as a geologist in the mineral and environmental industries since my graduation from university. My technical expertise includes hydrogeology, geotechnical investigations and stability analysis, mine site water management and tailings storage facilities, preparation of mass water balances, computer modelling and simulation, earthwork dam design, and capital and operating estimates. I have participated in worldwide projects in precious and semiprecious metals, base metals and iron ore. I have been a consulting engineer for Roche Ltd since June 2012.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association (as defined by NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I visited the project property on November 18, 2013.
7. I am author of sections 18.2.1.1 to 18.2.1.7.
8. I am independent of the issuer as described in section 1.5 of National Instrument 43-101.
9. I have no prior involvement with Clifton Star Resources Inc or with the property that is the subject of the Technical Report;
10. I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with this instrument.
11. At the effective date of the Technical Report, to the best of my knowledge, information and belief, the part 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.
12. I have no personal knowledge, as of the date of this certificate, of any material fact or material change which is not reflected in this Technical Report.

Effective Date: March 26, 2014
Signature Date: May 23, 2014

(Original signed and sealed)
David Sims, P.Geo.
OGQ #1431

CERTIFICATE OF AUTHOR – PHILIPPE COTÉ

To accompany the Report entitled "Technical Report and Prefeasibility Study for the Duparquet Project (according to National Instrument 43-101 and Form 43-101F1)", effective date of March 26, 2014 and signature date of May 23, 2014 ("the Technical Report").I, Philippe Cote, do hereby certify that:

1. I am currently employed as Project Manager, Mining and Mineral Processing at Roche Ltd Consulting Group, 1015 Wilfrid-Pelletier Avenue, Québec (Québec), G1W 0C4.
2. I graduated from Laval University (Québec) in 2002 with a B.Eng., Materials and Metallurgical Engineering, Extractive Metallurgy/Ore Processing Option.
3. I am a Metallurgical Engineer Member of the Ordre des Ingenieurs du Québec (#128326).
4. I have worked as a metallurgical engineer in the mineral industry since my graduation from university. My technical expertise includes plant operation, testwork supervision, preparation of mass and metallurgical balances and design criteria, computer modelling and simulation, equipment sizing, and capital and operating estimates. I have participated in worldwide projects in gold, rare earths, base metals and iron ore. I have been a consulting engineer for Roche Ltd since January 2011.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined by NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I have not visited the project propert
7. I am responsible for the preparation of section 13.1, 17.1, 18.1, co-responsible for the preparation of section 13.2, 13.3, 17.2, 18.2, 21.1.8 to 21.2.4 and co-author of section 1, 2, 3, 25 to 27.
8. I am independent of the issuer as described in section 1.5 of National Instrument 43-101.
9. I have no prior involvement with Clifton Star Resources Inc or with the property that is the subject of the Technical Report.
10. I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with this instrument.
11. At the effective date of the Technical Report, to the best of my knowledge, information and belief, the part 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.
12. I have no personal knowledge, as of the date of this certificate, of any material fact or material change which is not reflected in this Technical Report.

Effective Date: March 26, 2014
Signature Date: May 23, 2014

(Original signed and sealed)
Philippe Côté, Eng.
OIQ# 128326

CERTIFICATE OF AUTHOR – MARTIN MAGNAN

To accompany the Report entitled “Technical Report and Prefeasibility Study for the Duparquet Project (according to National Instrument 43-101 and Form 43-101F1)”, effective date of March 26, 2014 and signature date of May 23, 2014 (“the Technical Report”).I, Philippe Cote, do hereby certify that:

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

1. I am currently employed as Project Manager, Environment Division at Roche Ltd, Consulting Group, 1015 Wilfrid-Pelletier Avenue, Québec (Québec), G1W 0C4.
2. I graduated from Laval University (Québec) in 1990 with a B.Sc.A. in Geological Engineering and from Université du Québec à Chicoutimi (Québec) in 1994.with an M.Sc.A in Geology.
3. I am in good standing as a member of the *Ordre des Ingénieurs* du Québec (#126033).
4. I have been a specialist in environmental sciences for 14 years with 10 years previous experience in exploration geology. My expertise includes environmental site assessment studies, environmental impact assessments and rehabilitation plans. I have also been involved in scoping studies and feasibility studies. I have participated in gold, base metals and industrial mineral projects.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association (as defined by NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I have not visited the project property.
7. I am responsible for section 20 and co-author of sections 1, 2, 3, 21.8 and 25 to 27.
8. I am independent of the issuer as described in section 1.5 of National Instrument 43-101.
9. I have no prior involvement with Clifton Star Resources Inc or with the property that is the subject of the Technical Report.
10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
11. At the effective date of the Technical Report, to the best of my knowledge, information and belief, the part 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.
12. I have no personal knowledge, as of the date of this certificate, of any material fact or material change which is not reflected in this Technical Report.

Effective Date: March 26, 2014
Signature Date: May 24, 2014

(Original signed and sealed)
Martin Magnan, Eng.
OIQ# 126033

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1. SUMMARY

On May 14, 2013, InnovExplo Inc. (“InnovExplo”) was retained by Mr. Michel Bouchard, M.Sc., P.Geo., MBA, President and CEO of Clifton Star Resources Inc. (“Clifton Star” or “the issuer”) to produce a Technical Report, an Updated Mineral Resource Estimate and a Prefeasibility Study for the Duparquet Project in accordance with National Instrument 43-101 and Form 43-101F1. The updated Mineral Resource Estimate has been published in an earlier report titled “Technical Report and Mineral Resource Estimate Update for the Duparquet Project (according to Regulation 43-101 and Form 43-101F1)”, dated August 2, 2013 (Williamson et al., 2013b). The Mineral Resource Estimate covers the Duparquet Project, defined here as the amalgamation of the Beattie, Donchester, Central Duparquet and Dumico properties, as well as the historical Beattie mine tailings. The Duparquet Project (“the Project”) is located just north of the Town of Duparquet in the Province of Québec.

This Technical Report (“the Report”) presents the results of the Prefeasibility Study (“the PFS”) on the Duparquet Project using the August 2013 updated resource estimate as its basis.

The PFS was prepared by InnovExplo, Tenova Mining & Minerals–Bateman Engineering Pty Ltd (“Tenova-Bateman”), Roche Ltd Consulting Group (“Roche”), and Dreisinger Consulting Inc. (“Dreisinger Consulting”).

The Report is addressed to Clifton Star Resources Inc., a Canadian mineral exploration company trading publicly on the Toronto Stock Exchange in Canada (TSX-V: CFO). InnovExplo is an independent mining and exploration consulting firm based in Val-d’Or (Québec).

The qualified persons responsible for the preparation of the Report included the principal authors Sylvie Poirier, Eng. (InnovExplo), Karine Brousseau, Eng. (InnovExplo) and Laurent Roy, Eng. (InnovExplo), as well as David Dreisinger, B.Sc., PhD, P.Eng. (Dreisinger Consulting Inc), Philippe Coté, Eng. (Roche Ltd), David Sims P.Geo. (Roche Ltd), and Martin Magnan, Eng. M.Sc.A (Roche Ltd).

The authors believe the information used to prepare the Report and to formulate its conclusions and recommendations is valid and appropriate considering the status of the Project and the purpose for which the Report is prepared. The authors, by virtue of their technical review of the Project’s production potential, affirm that the work program and recommendations presented herein comply with National Instrument 43-101 and CIM technical standards.

Property description and location

The Duparquet Property (“the Property”) is located just north of the Town of Duparquet, partly overlapping the municipal boundary, in the Duparquet Township of Québec, NTS map sheet 32D/11. The coordinates for the approximate centre of the Project are 48º30’34”N, 79º12´34”W (UTM projection: 5374410N, 631517E, NAD 83 Zone 17). The Property covers an area of 1033.6 ha, and consists of the amalgamation of two (2) mining concessions and twenty (20) claims, all in good standing. The Beattie property (MC#292) accounts for approximately 383.6 ha; the Donchester property (MC#384) is about 322.6 ha. The Central Duparquet property

(18 claims) and the two (2) Dumico claims cover the remaining 293.4 ha and 34 ha respectively. Both the Beattie and Donchester properties contain past-producing mines. Historical underground workings and a shaft were developed at the Central Duparquet property but no gold was produced. The Duparquet Project is defined in this report as the Beattie, Donchester, Central Duparquet and Dumico properties, as well as the tailings pond area straddling the southwest limit of the Beattie property.

Geological setting

The Duparquet Property lies within the Archean Superior Province, which forms the core of the North American continent and is surrounded by provinces of Paleoproterozoic age to the west, north and east, and the Grenville Province of Mesoproterozoic age to the southeast. A first-order feature of the Superior Province is its linear subprovinces, or “terranes” of distinctive lithological and structural character, accentuated by subparallel boundary faults.

The Property is located within the Abitibi terrane. The Abitibi terrane hosts some of the richest mineral deposits of the Superior Province, including the giant Kidd Creek massive sulphide deposit (Hannington et al., 1999) and the large gold camps of Ontario and Québec (Robert and Poulsen, 1997; Poulsen et al., 2000).

The local geological setting and property geology is represented by the Kinojevis, Timiskaming, and Blake River groups. The Duparquet Property straddles syenitic plutons and the Kinojevis, Duparquet and Mont-Brun formations. The Property area is characterized by the presence of two syenitic plutons oriented east-west. These syenitic intrusions are bounded by E-W trending major faults, which are interpreted as splays of the main SE-trending Destor-Porcupine-Manneville Fault Zone, which clips the southwest corner of the Property. According to Bevan (2011), the “main” type of gold mineralization in the Duparquet deposit generally occurs within shears or fracture zones along or within the adjacent intrusive syenitic masses, and is associated with finely disseminated pyrite and minor arsenopyrite replacement.

Data verification

New data added to the previously compiled and verified master database comprise drill holes from the last part of the 2012 program and the first part of the 2013 program, as well as new samples collected from previously drilled holes in the Beattie, Donchester and Central Duparquet areas (Williamson et al., 2013b). The new data also include a series of holes drilled on the Dumico property in 2008-2009.

A statistical analysis of the QA/QC data provided by Clifton Star did not highlight any significant analytical issues. InnovExplo is of the opinion that the sample preparation, analysis, QA/QC and security protocols used by Clifton Star for the Duparquet Project follow generally accepted industry standards and that the data is valid and of sufficient quality to be used for mineral resource estimation.

Following the additions, corrections and modifications made to the databases, InnovExplo is of the opinion that the final diamond drill hole (DDH) and channel sample database, as well as the tailings pond database, are adequate to support a mineral resource estimate for the Duparquet Project, including the Beattie, Donchester, Central Duparquet, Dumico properties, and the Beattie Mine tailings.

The cut-off date for the DDH and channel sample database used by InnovExplo for the current Mineral Resource Estimate is May 6, 2013.

Mineral Resource Estimate

An updated Mineral Resource Estimate for the Duparquet Project was prepared in 2013 by InnovExplo and presented in a report titled "Technical Report and Mineral Resource Estimate Update for the Duparquet Project (according to Regulation 43-101 and Form 43-101F1)", dated August 2, 2013 (Williamson et al., 2013b).

The August 2013 Mineral Resource Estimate Update presented herein was performed by Kenneth Williamson, B.Sc., P.Geo. and Karine Brousseau, Eng. under the supervision of Carl Pelletier, B.Sc., P.Geo., using all available results. The main objective was to update the results of InnovExplo's previous Mineral Resource Estimate for the Duparquet Project, dated February 28, 2013 (Williamson et al., 2013a). The updated estimate includes additional new and re-sampled drill holes that were not included in the previous resource estimate. The Dumico area drill holes were also added to this update.

An exception to the above statement is the resource estimate for the tailings pond component, which was not modified during the August 2013 update. The results for this component, also presented herein, are taken directly from InnovExplo's earlier estimate of July 5, 2012 (Brousseau et al., 2012).

Given the density of data, the search ellipsoid criteria, and the specific interpolation parameters, InnovExplo is of the opinion that the current In-pit Mineral Resource Estimate can be classified as measured, indicated and inferred resources, the Underground Mineral Resource Estimate can be classified as indicated and inferred resources whereas the Tailings Mineral Resource Estimate can be classified as measured and indicated resources. All Mineral Resource Estimates are compliant with CIM standards and guidelines for reporting mineral resources and reserves. The overall results of the current Mineral Resource Estimate are presented in the table below.

Global Mineral Resource Estimate results (Measured, Indicated and Inferred Resources) for the Duparquet Project (Table 14.9)

Resources type	Parameters	Area			TOTAL
		Tailings	In-Pit	Underground	
	Cut-off (g/t)	> 0.45	> 0.45	> 2.00	
Measured	Tonnes (t)	19,600	165,100		184,700
	Grade (g/t)	2.06	1.45		1.52
	Au (Oz)	1,295	7,711		9,006
Indicated	Tonnes (t)	4,105,000	53,070,600	3,520,700	60,696,300
	Grade (g/t)	0.93	1.56	2.78	1.59
	Au (Oz)	123,200	2,666,690	314,275	3,104,165
Measured + Indicated	*Tonnes (t)*	*4,124,600*	*53,235,700*	*3,520,700*	*60,881,000*
	*Grade (g/t)**	*0.94*	*1.56*	*2.78*	*1.59*
	Au (Oz)	*124,495*	*2,674,401*	*314,275*	*3,113,171*
Inferred	Tonnes (t)		24,092,300	5,592,400	29,684,700
	Grade (g/t)		1.18	2.96	1.51
	Au (Oz)		910,631	532,059	1,442,689

- The Independent and Qualified Persons for the Mineral Resource Estimate, as defined by NI 43-101, are Kenneth Williamson, M.Sc., P.Geo and Karine Brousseau, Eng. under the supervision of Carl Pelletier, B.Sc., P.Geo. (InnovExplo Inc.), and the effective date of the estimate is May 22, 2012 for the Tailings resource and June 26, 2013 for the In-Pit and Underground mineral resources.
- Mineral Resources which are not Mineral Reserves, do not have demonstrated economic viability.
- Tailings results are presented undiluted and in situ. The estimate includes four (4) tailings ponds.
- In-Pit results are presented undiluted and in situ, within Whittle-optimized pit shells. The estimate includes 60 gold-bearing zones and the envelope containing isolated gold intercepts.
- Underground results are presented undiluted and in situ, outside Whittle-optimized pit shells. The estimate includes 60 gold-bearing zones and the envelope containing isolated gold intercepts.
- Tailings resources were compiled at cut-off grades of 0.35, 0.40, 0.45, 0.50, 0.55, 0.60, 0.65, 0.70, 0.80 and 0.9 g/t Au.
- In-Pit resources were compiled at cut-off grades of 0.35, 0.40, 0.45, 0.50, 0.55, 0.60, 0.65, 0.70, 0.80 and 0.9 g/t Au.
- Underground resources were compiled at cut-off grades of 1.5, 2.0, 2.5, 3.0, 3.5, 4.0 and 5.0 g/t Au.
- Cut-off grades must be re-evaluated in light of prevailing market conditions (gold price, exchange rate and mining cost).
- Tailings: A fixed density of 1.45 g/cm^3 was used in zones and waste.
- In-Pit and Underground: A fixed density of 2.73 g/cm^3 was used in the mineralized zones and in the envelope zone.
- In-Pit and Underground: A minimum true thickness of 3.0 m was applied, using the grade of the adjacent material when assayed, or a value of zero when not assayed.
- Tailings: High grade capping was done on the raw data and established at 13.0 g/t Au for Zone 1, 3.5 g/t Au for Zone 2, 1.7 g/t Au for Zone 3 and 2.2 g/t Au for Zone 4.
- In-Pit and Underground: High grade capping was done on the raw data and established at 25.0 g/t Au for diamond drill hole assays and channel sample assays.
- Tailings: Compositing was done on drill hole sections falling within the mineralized zone solids (composite = 0.5 m).
- In-Pit and Underground: Compositing was done on drill hole and channel sample sections falling within the mineralized zone solids (composite = 1 m).
- Tailings: Resources were evaluated from drill hole and surface channel samples using an ID2 interpolation method in a block model.
- In-Pit and Underground: Resources were evaluated from drill hole and surface channel samples using an ID2 interpolation method in a multi-folder percent block model.
- Tailings: Measured and Indicated categories were defined based on the drill hole spacing (Measured: zones 1 and 2 = 30m x 30m grid; Indicated: zone 3 = 100m x 100m grid and zone 4 = 200m x 200m grid).
- The In-Pit Measured category is defined by blocks having a volume of at least 25% within an envelope built at a distance of 10 m around existing channels.
- The In-Pit and Underground Indicated category is defined by the combination of blocks within a maximum distance of 15m from existing stopes and blocks for which the average distance to drill hole composites is less than 30 m.
- Ounce (troy) = Metric tons x Grade / 31.10348. Calculations used metric units (metres, tonnes and g/t).
- The number of metric tons was rounded to the nearest hundred. Any discrepancies in the totals are due to rounding effects; rounding followed the recommendations in NI 43-101.
- InnovExplo is not aware of any known environmental, permitting, legal, title-related, taxation, socio-political or marketing issues or any other relevant issue that could materially affect the Mineral Resource Estimate.
- Input parameters used for MCoG estimation and Whittle pit design: Gold price: C$ 1,450; Gold selling cost: C$ 5.00; Mining costs: C$ 2.40; Processing cost: C$ 13.46; Transportation cost: C$ 0.25; Administration cost: C$ 4.18; Processing recovery: 93.9%; Mining recovery: 90.9%; Mining dilution: 10.0%; Overall pit slope: 52°.
- Parameters used for UCoG estimation: Gold price: C$ 1,450; Gold selling cost: C$ 5.00; Mining cost: C$ 58.00; Milling cost: C$ 13.46; Processing recovery: 93.9%; Mining dilution: 15.0%.

Overall, InnovExplo estimates that the Duparquet Project has a total **Measured+Indicated Resource** of **60,881,000 metric tonnes** grading **1.59 g/t Au** for a total of **3,113,171 ounces of gold.** This corresponds to an increase of 29% compared to the NI 43-101 compliant report of February 2013. Total **Inferred Resources** are estimated at **29,684,700 metric tonnes** grading **1.51 g/t Au** for a total of **1,442,689 ounces of gold.**

For the In-Pit portion, InnovExplo estimates that the Duparquet Project has a total **Measured+Indicated Resources** of **53,235,700 metric tonnes** grading **1.56 g/t Au** for a total of **2,674,401 ounces of gold** at a cut-off grade of **0.45 g/t Au.** Total **Inferred Resources** are estimated at **24,092,300 metric tonnes** grading **1.18 g/t Au** for a total of **910,631 ounces of gold** at a cut-off grade of **0.45 g/t Au**.

For the Underground portion, InnovExplo estimates that the Duparquet Project has **Indicated Resources** of **3,520,700 metric tonnes** grading **2.78 g/t Au** for a total of **314,275 ounces of gold** at a cut-off grade of **2.00 g/t Au.** Total **Inferred Resources** are estimated at **5,592,400 metric tonnes** grading **2.96 g/t Au** for a total of **532,059 ounces of gold** at a cut-off grade of **2.00 g/t Au**.

For the Tailings portion, InnovExplo estimates that the Duparquet Project has **Measured+Indicated Resources** of **4,124,600 metric tonnes** grading **0.94 g/t Au** for a total of **124,495 ounces of gold** at a cut-off grade of **0.45 g/t Au.** No Inferred Resources have been estimated for the tailings.

Mineral Reserve Estimate

The reserves for the detailed pit design have been calculated in accordance with the definitions and guidelines adopted by the Canadian Institute of Mining, Metallurgy and Petroleum (CIM Standards on Mineral Resources and Reserves) in August 2000.

The effective date of the Mineral Reserve Estimate is March 26, 2014, the date on which the latest processing cost estimation was concluded.

The PFS is based on an earlier report prepared for the issuer titled "Technical Report and Mineral Resource Estimate Update for the Duparquet Project (according to Regulation 43-101 and Form 43-101F1)", dated August 2, 2013.

The mineral resource block model developed by InnovExplo was imported into Whittle™ software from Dassault Systèmes GEOVIA (formerly Gemcom Software International). Design parameters, such as operating costs, mine recovery, dilution and gold price, were used to generate an optimal pit shell.

InnovExplo evaluated dilution and mine recovery by performing an analysis of mineable blocks on typical cross-sections at the mill cut-off grade. After a cross-sectional evaluation, average dilution was estimated at 10% at a grade of 0 g/t, and mine recovery was estimated at 95% assuming good blasting and dilution control practices.

To integrate dilution in Whittle, an initial 5% dilution factor was used when performing reblocking and merging the original small blocks (5x5x5) into grade bands (high and low). This feature defines a maximum number of parcels per reblock blocks. This action regroups the smaller blocks of 5x5x5 into larger parcels, by grade interval. In this evaluation, two parcels were considered. The resulting parcel includes blocks lower than the cut-off grade that are considered internal dilution. If the average grade of the parcel is higher than the cut-off grade, it is considered as ore and sent to the mill. This dilution was estimated at 5%. An additional 5% dilution factor was added to the Whittle parameters for a total dilution factor of 10%.

Iterations were performed to generate a pit design that fits the selected pit shell. To do so, the pit design, including a ramp and catch bench, is made in the selected pitshell. For the Duparquet Project, the initial pit wall angle was too steep to accommodate the ramp. The slope angle was smoothed until a ramp could fit in the Whittle pit shell. Numerous iterations were needed before finding an adequate pit wall angle.

Pit slopes were set at 29° in the overburden, 45° on the north side of the pits, and 48° on the south side of the pit. The ramp was designed on the north side of the pit due to the constraints imposed by the golf course and houses on the south side and due to the location of the mill.

The concurrent prefeasibility work allowed InnovExplo and Roche Ltd Consulting Group ("Roche") to better quantify the operating costs. These costs were used for a final pit optimization in Whittle. These parameters are presented in the following table.

Whittle parameters (Table 15.1)

Input parameters		Value	Provided by
Gold Price		1,417.5 $CAN/oz	InnovExplo
Gold selling cost		5 $/oz	InnovExplo
Dilution		*5 %	InnovExplo
Mining recovery		95 %	InnovExplo
Milling recovery		93.9 %	Roche
Overburden Cost		0.80 $/t	InnovExplo
Mining cost		1.93 $/t	InnovExplo
General & Administration		3.12 $/t	InnovExplo
Processing Cost		16.77 $/t	Roche
Environmental monitoring		0.20 $/t	Roche
Stockpile rehandling cost		0.88 $/t	InnovExplo
Pit slope	North	45 °	InnovExplo
	South	48 °	InnovExplo
	OVB	29 °	InnovExplo

Note: An additional 5% dilution factor is included in the reblocking for a total dilution factor of 10%.

The mining cost of $1.93/t represents the initial cost to mine the rock present at the surface of the open pit. An incremental hauling cost of $0.21 per kilometre of ramp was added, depending on pit depth.

The open pit production is supplemented by 4.1 Mt of available old tailings. The old tailings will be processed during the beginning of the Project at a rate of 750,000 tonnes per year.

The global In-pit and Tailings Proven and Probable Reserves total 39,363,000 tonnes at an average grade of 1.50 g/t (1,895,530 contained ounces of gold). The Whittle pit shell selected for this PFS generates 35,238,429 tonnes of ore, including dilution and losses. Another 4,124,600 tonnes from the old tailings completes the resources for a total of 39,363,029 tonnes of ore from which 0.5% is Proven Reserves and 99.5% is Probable Reserve. A cut-off grade of 0.45 g/t has been considered for the tailings material and 0.51 for the pit. The Mineral Reserve Estimate for the Duparquet Project is presented in the following table.

Mineral Reserve Estimate (Table 15.2)

Reserves type	Parameters	Area		TOTAL
		Tailings	In-Pit 1	
	Cut-off (g/t)	> 0.45	> 0.51	
Proven	Tonnes (t)	19,600	175,100	194,700
	Grade (g/t)	2.06	1.31	1.38
	Au (Oz)	1,295	7,372	8,667
Probable	Tonnes (t)	4,105,000	35,063,400	39,168,400
	Grade (g/t)	0.93	1.56	1.50
	Au (Oz)	123,200	1,763,664	1,886,864
Proven + Probable	*Tonnes (t)*	*4,124,600*	*35,238,400*	39,363,000
	Grade (g/t)	*0.94*	*1.56*	**1.50**
	Au (Oz)	*124,495*	*1,771,035*	**1,895,530**

Mining

Mining of the Duparquet deposit has been designed as an open pit with a planned production of 3,650,000 tonnes per year (3.65M tpy) or 10,000 tonnes per day (tpd), 365 days per year of mill operation and 360 working days operation for the pit. The open pit production is supplemented by 4.1 Mt of available old tailings. The old tailings will be processed during the beginning of the Project at a rate of 750,000 tonnes per year.

The Duparquet Project pit optimization for the present PFS generates 35,238,429 tonnes of ore. Another 4,124,600 tonnes from the old tailings complete the resource for a total of 39,363,029 tonnes of ore. The pit also generates 291,213,881 tonnes of waste and 23,398,085 tonnes of overburden resulting in an LOM strip ratio of 8.26 to 1. Taking into account the overburden, the LOM strip ratio is 8.92 to 1.

Production by year and stripping ratio (Table 16.5)

Year	Stripping		Mineralized material		Total production		Stripping ratio
	Overburden (tpy)	Waste (tpy)	From pit (tpy)	From old tailings (tpy)	Without overburden (tpy)	With overburden (tpy)	
PP1	-	-	-	-	-	-	
PP2	513,834	800,000	-	-	800,000	1,313,834	
PP3	3,467,354	5,398,400	-	-	5,398,400	8,865,754	
PP4	402,075	626,000	894,717	135,450	1,520,717	1,922,792	1.15
1	6,272,956	27,162,421	3,288,695	750,000	30,451,116	36,724,072	10.17
2	-	28,864,822	5,727,596	750,000	34,592,418	34,592,418	5.04
3	1,859,667	29,360,581	3,646,700	750,000	33,007,281	34,866,948	8.56
4	1,260,908	31,612,674	2,236,841	750,000	33,849,515	35,110,423	14.70
5	1,291,105	29,831,261	3,562,544	750,000	33,393,805	34,684,910	8.74
6	4,446,345	28,281,662	2,884,832	239,150	31,166,494	35,612,839	11.34
7	30,288	33,081,261	2,831,059	-	35,912,320	35,942,608	11.70
8	2,954,360	29,211,166	3,649,988	-	32,861,154	35,815,514	8.81
9	743,041	31,605,924	3,650,216	-	35,256,140	35,999,181	8.86
10	156,985	15,377,709	2,865,243	-	18,242,952	18,399,937	5.42
Total	**23,398,919**	**291,213,881**	**35,238,429**	**4,124,600**	**326,452,310**	**349,851,229**	**8.93**

The estimated LOM average grade is 1.50 g/t including the ore from the pit and the old tailings. A total of 1,682,968 ounces of gold would be recovered over the mine life. An average of 173,000 ounces per year would be recovered for the first 5 years, and an average of 158,000 ounces per year over the 11 years of production studied. A stockpile will be used to vary the cut-off grade in order to optimize project economics. The ore, waste and tailings production plan is presented in the table below on a yearly basis.

The Duparquet Pit life of mine (LOM) was based on supplying the mill with 3,650,000 tonnes of ore per year. Initially, 4.1 Mt of tailings would be reserved to supplement the mill when necessary, but it was later decided to process the tailings at a rate of 750,000 tonnes per year right from the start in order to clean the tailings area to make room for the waste stockpile.

The Duparquet Pit LOM will be spread over 11 years, preceded by a 4-year pre-production period. This schedule will yield a yearly production of 3,650,000 tonnes.

Mineralized material processing (Table 16.6)

Year	To the mill from								
	ROM		Stockpile		Old tailings		Total		Total ounces
	Tonnage (t)	Grade (g/t)	Tonnage (t)	Grade (g/t)	Tonnage (t)	Grade (g/t)	Tonnage (t)	Grade (g/t)	Au recovered
PP4	400,000	1.14			135,450	0.93	535,450	1.09	16,608
1	2,900,000	1.90			750,000	0.95	3,650,000	1.70	178,823
2	2,900,000	2.01			750,000	0.93	3,650,000	1.79	187,928
3	2,899,853	1.60	147	2.32	750,000	0.93	3,650,000	1.47	153,598
4	1,913,865	2.22	986,135	1.51	750,000	0.93	3,650,000	1.76	185,127
5	2,897,632	1.68	2,368	1.42	750,000	0.93	3,650,000	1.53	159,869
6	2,543,520	1.44	731,880	0.89	239,150	0.93	3,514,550	1.29	131,004
7	2,827,860	1.44	822,140	0.97			3,650,000	1.33	140,702
8	3,649,988	1.59	12	0.71			3,650,000	1.59	168,114
9	3,650,000	1.42					3,650,000	1.42	149,781
10	2,865,243	1.95	784,757	0.71			3,650,000	1.68	177,808
11			2,463,029	0.70			2,463,029	0.70	50,214
Total	**29,447,961**	**1.69**	**5,790,468**	**0.90**	**4,124,600**	**0.93**	**39,363,029**	**1.50**	**1,682,968**

For the Project, three (3) 6030FS shovels have been selected, as well as one (1) 994H front-end loader, and it has been determined that 785D trucks will be used. A total of 14 trucks will be necessary during the production peak, which covers Years 2 to 7. Sanvik recommended two types of drill for the Project: one DR540 for the pre-shear drilling which will drill 140 mm holes, and four (4) D55SP drills for the production drilling which will drill 215 mm holes.

A total of 339 employees will be needed for the Duparquet Project. This assumes the operation will run 24 hours a day, 7 days a week, 52 weeks per year.

The working schedule for most yearly compensated employees will be a standard 40-hour week at 8 hours per day, 5 days per week, Monday to Friday. Some yearly compensated employees will be working 12-hour shifts, equivalent to 84 hours per two weeks, as part of a two-week repeating schedule: the first week working 4 days followed by 3 days off, the second week working 3 days followed by 4 days off. The hourly workers will be working 12-hour shifts as part of the same two-week repeating schedule. Most activities require 24-hour per day operation, which is split into 4 shifts.

Mineral Processing and Metallurgical Testing

Bench scale and pilot plant metallurgical testwork programs have been carried out for the Duparquet Property of Clifton Star by SGS Canada and Outotec. The preliminary metallurgical testwork was carried out in 2012 by SGS in support of the Preliminary Economic Assessment ("PEA"). In 2013, SGS carried out further flotation, pressure oxidation, cyanidation, rheology, and environmental testwork including a pilot plant for the current PFS.

The testwork and pilot plant test performed in 2013 included a series of grindability tests conducted on 12 t composite bulk sample selected by InnovExplo. The pilot plant sample was characterized as very hard with respect to both resistances to impact (Axb) and abrasion breakage (ta), as well as in terms of the Bond rod mill work index (RWI). The sample was hard with respect to the Bond ball mill work index

(BWI) and High-Pressure Grinding Rolls (HPGR) test, and was also found to be abrasive.

Pilot plant flotation tests have also been conducted on the bulk sample. The pilot plant was operated to generate a bulk sulphide flotation concentrate with 15-18% S for a subsequent pressure oxidation (POX) pilot plant to recover gold, as well as to generate 60-80 kg of a higher grade flotation concentrate assaying greater than 40 g/t Au for direct sale market evaluation.

In the flotation pilot plant, the ore was ground to a P80 of 100 µm and a rougher concentrate was recovered with the addition of collectors. The rougher concentrate was reground in a ball mill, operating in closed circuit with a cyclone. Pilot plant tests PP-01 to PP-06 were conducted with one cleaning stage to generate the POX pilot plant feed. Pilot plant tests PP-07 to PP-09 were conducted with two cleaning stages to generate the high grade concentrate.

With one cleaning stage the recovery of gold was 91.7% in a concentrate which assayed 26.8 g/t Au and 16.1% S. The results of the PP-09 indicated that a concentrate with 47.8 g/t Au could be produced at 86.5% gold recovery.

Flotation tailings were leached with cyanide to recover the gold remaining in this product. The recovery of gold from the PP-05 tailings and the PP-09 tailing were 45.1% and 40.2%, respectively.

The overall gold recovery by flotation and cyanidation of the flotation tailings is shown in the following table.

Overall gold recovery by flotation and cyanidation of the flotation tails (Table 13.18)

Sample	Concentrate Grade		Overall Au Recovery (%)		
	Au (g/t)	S (%)	Flot. Conc.	Tail CN	Total
PP-05 (Conc for POX feed)	26.8	16.1	91.7	3.7	95.4
PP-09 (Conc for direct sale)	47.7	29.8	86.5	5.4	91.9

The sulphur concentrate feed was oxidized efficiently in the autoclave at 210 °C and oxygen overpressure of 700 kPa, with residual sulphide levels averaging 0.13%. A 60 minute retention time appeared sufficient to achieve a high level of oxidation. Recycling a portion of the acidic hot cure thickener overflow allows for neutralization of carbonates present in the feed concentrate. The autoclave discharge was predominately a jarosite product with little hematite formed. The POX liquor contained 47 g/L free acid, 11 g/L Fe (>95% as ferric iron) and ~1 g/L As. No work has been carried out in the pilot plant to optimize the POX operating conditions apart from retention time.

The hot cure was operated at 95 °C with autoclave discharge product having a total retention time of either 4 or 5 hours to promote the conversion of any basic iron sulphate formed during pressure oxidation. There appeared to be little evidence of re-dissolution of basic ferric sulphate in the hot cure circuit with the majority of conversion having taken place within the autoclave flash letdown. The re-dissolution that did occur happened relatively quickly. Overall iron in solution increased from

~11 g/L in the autoclave discharge to ~17 g/L in the first hot cure stage and up to ~20 g/L through the hot cure circuit. Arsenic content remained consistent throughout at an average of ~1.7 g/L. Optimization of the size of the hot cure circuit in conjunction with further pressure oxidation studies is required. The cyanidation testwork program has demonstrated that high gold and silver recoveries are obtainable for cyanidation of pressure oxidation residues. The recovery of gold from the pilot plant hot cure discharge from batch testwork ranged from 94.7% to 96.5% with average residue assay from 0.96 g/t to 1.21 g/t Au.

Lime boil tests on thickened hot cured discharged material were effective in breaking down jarosite and increased silver recovery from 9.2% to 95.5%, and gold recovery from 95.5% to 98.1%. Reducing the lime dosage below 75 kg/t resulted in a reduction in gold and silver extraction, with optimum conditions appear to favour 75 kg/t of lime. Promoting conditions favourable for hematite formation over jarosite in the pressure oxidation would undoubtedly reduce lime consumption in the lime boil step.

A rheometallurgical testwork of the flotation tailings flotation concentrate, hot cure discharge and combined leached tailings that were produced as part of the pilot plant test program was conducted. The preliminary static settling result indicated that samples settled in presence of 20 to 36 g/t of non-ionic flocculant producing a underflow between 36% w/w and 66% w/w with an overflow slightly cloudy after 60 minutes of elapsed settling time. An exception is the hot cure discharge which required 92 g/t of flocculant and producing an underflow at 30% w/w and a cloudy supernatant.

The results of the tests conducted on cyanide flotation tailing and hot cure discharge showed that the cyanide was effectively destroyed with an SO_2 with a copper addition as copper sulphate. Reducing the copper addition by half resulted in an increase in the CN_T (Total Cyanide) although the CN_{WAD} (Weak Acid Dissociable Cyanide) remained similar.

Recovery Methods

Based on the previous testwork results, two processing options were selected for the recovery of the gold from the Duparquet Project; the Pressure Oxidation option ("POX option") and the Concentrate option. While gold doré would be produced on site with the POX Option, the Concentrate option would produce a sulphide concentrate product for sale.

The Concentrate processing option uses a jaw crusher followed, by one 5,670 kW (7,200 hp) SAG Mill and of a secondary grinding stage using a 6,410 kW (8,600 hp) ball mills operating in a closed loop, coupled with a flotation circuit including rougher cells, 1st cleaner cells, 2nd cleaner cells, and a 1.119 kW (1,500 hp) regrinding ball mill of the rougher concentrate, as well as cyanidation of the flotation tails with CIL recovery of the gold. The gold will be recovered from the carbon by elution, followed by electrowinning and doré smelting. The flotation concentrate would be sold for further processing.

The POX option uses the same crushing and grinding circuits as the Concentrate option. The flotation circuit includes a: rougher flotation stage, a rougher concentrate regrind stage and a cleaner flotation stage After thickening, the flotation concentrate will pass through the pressure oxidation circuit (POX) to be oxidized in the autoclave (4.1 m Ø x 21.0 m long) followed by a carbon-in-pulp (CIP) circuit for cyanidation and adsorption for gold recovery. The flotation tailings will be considered as final tailings. In the CIP circuit, gold will be cyanide leached and adsorbed on the carbon and then recovered from the carbon by an elution circuit, followed by electrowinning and doré smelting.

In the POX option the flotation circuit is operated to maximize gold recovery to flotation concentrate instead of maximizing the gold concentrate grade as in the Concentrate option. With much less gold being rejected in the flotation tailings, cyanidation of the tailings is currently not economically feasible.

The plant is designed to operate 24 hrs/day, 365 days/yr, and process 3.65 Mt of mineralized material (dry) annually, at a plant availability of 92%. The nominal daily throughput will be 10,000 tonnes of dry material. The overall gold recovery for the POX option is 90.1% and 92.8% for the Concentrate option. Details are shown in the table below.

Overall gold recovery (Table 1.1)

	POX Option	Concentrate Option
Gold recovery in flotation concentrate	91.7%	86.4%
Gold recovery in POX/Cyanidation of flotation concentrate	98.3%	n/a
Gold recovery in cyanidation of flotation tailings	n/a	6.4%
Overall gold recovery	90.1%	92.8%

<u>Infrastructure</u>

Off-Site Infrastructure

Electric power will be provided by Hydro-Québec through a new 15 km long 120 kV power line which could be available in 36 months. The total power capacity is established at 50 MW and the estimated power demand will amount to approximately 35 MW.
The Duparquet Property partly overlaps the municipal boundary of the Town of Duparquet, located 45 km northwest of Rouyn-Noranda, Abitibi-Temiscamingue, Québec. The future mine site is easily accessible using the already existing provincial highway, Route 393, and no access road other than a connection between the site facilities and Route 393 will be required for project development.

On-Site Infrastructure

Tailings Storage Facility and Water Management

Two options are considered:

1. A Concentrate option where all tailings will result from a CIL circuit; and

2. A POX option where approximately 6% of the tailings will have passed through the CIP process and 94% from the flotation tailings. The two types of tailings will require separate handling and disposal at the tailings storage facility.

Since the CIL and CIP circuit involves cyanidation, the tailings and process water resulting from these circuits will need to be stored in a lined or impermeable tailings cell to prevent seepage into groundwater aquifers.

Concentrate option

The tailings storage facility (TSF) for the Concentrate option is conceived as a staged construction with two separate cells: cell 1 in the southwest and cell 2 in the north. Both cells will be lined. The tailings dams will be constructed in stages to minimize the initial capital construction costs. The TSF was designed to be constructed in 3 phases.

POX option

The POX option involves treating only approximately 6% of the tailings with a CIP circuit. Since only the tailings of the CIP circuit need to be stored in a lined cell, less membrane is required in construction than the Concentrate option, and water management and treatment is simplified. The TSF layout is similar to that of the Concentrate option, except a third smaller lined cell is added to handle tailings from the CIP circuit. The large northern and southwest cells are unlined in this scenario since the early geochemical results indicate that the tailings are not acid generating, nor are they treated by cyanide as was the case for the Concentrate option.

The TSF is conceived as a staged construction with three separate cells: cell 1 in the southwest for low-risk tailings (unlined); the POX cell in the southeast for cyanidation of the process tailings (fully lined); cell 2 in the north for low-risk tailings (unlined). As was the case for the Concentrate option, the tailings dams will be constructed in three stages to minimize the initial capital construction costs.

The current conceptual plan is a centreline dam configuration, predominantly constructed using waste rock from the open pit mining operation.

Site road

Site roads will be used to access the various industrial sites and services. The on-site roads will give access to: the process plant facility and surrounding buildings; the open pits; the ore stockpile; the waste dumps and overburden dump; the explosives storage; pumping stations; and the tailings disposal area. A total of about 13 km of mine roads will be required. All roads will be equipped with the appropriate information, regulatory and warning signs. The work that will be performed at the industrial and service sites include: land clearing and grubbing; topsoil stripping; excavation and backfilling with appropriate material to reach the infrastructure elevation; and construction of work platforms.

Site buildings

Provisions have been made for ancillary buildings and facilities, such as a mine equipment maintenance facility and warehouse, an administrative building, an assay laboratory, a water pumping station, a fuel farm and an explosives plant and storage.

Power distribution will be provided to the administration building, garage, pump houses, laboratory and crushing area through 4160/600V pad mounted transformers with the proper capacity installed near each building. Inside of these buildings, electricity will be distributed through 600/120/208V transformers and panels.

Project Infrastructure

The plan in the present study is to build an overhead polled mounted 4.16 kV power line approximately 5 km long from the main substation to distribute power to the remaining site facilities and infrastructures.

A 120 kV substation will be installed in close proximity to the process plant for the electrical distribution. A single oil-filled transformer will provide power to the process plant's main electrical room. A 5 MW diesel generator will also be installed on-site to provide backup power to critical process equipment's and essential services.

Site telecommunication will be supported by an optical fiber network backbone with an IT network subsystem. A supervisory control system installed in the control room will allow the operator to monitor and control all the process related equipment. VoIP type telephones will be installed.

Environment

The Duparquet Project is subject to the environmental assessment provisions of the provincial Environment Quality Act and the federal Canadian Environmental Assessment Act. The requirements for each of these processes are well understood. The Environmental and Social Impact Assessment that is required pursuant to these Acts will be initiated as soon as a Project Notice and a Project Description are tabled with the provincial and federal authorities, respectively. A schedule for the environmental assessment and permitting has been developed. Environmental and social baseline studies have been and will be conducted and reports either have been or will be prepared. Permitting requirements are also well-defined and have been considered in the project plan.
A tailings and water management strategy has been defined at a prefeasibility design level. A siting study was undertaken and an appropriate area has been determined and located on the site plan, taking into account environmental and social considerations and constraints. Water in the polishing pond will be recycled to the mill, within the constraints of both water availability in the polishing pond, on the one hand, and concentrator water demand on the other. Water in excess of mill requirements will be released to the environment, meeting all regulatory requirements.

Overburden and waste rock stockpiles have also been designed at a prefeasibility level, and locations are defined on the site plan. The identified areas do not contain any significant mineralization and make use of the natural topography. Discharges from the stockpiles will be routed to a series of sedimentation ponds to ensure adequate treatment and to meet required regulatory requirements prior to release to the environment.

A Mine Rehabilitation and Closure Plan will be prepared for the Project. The Plan will describe measures planned to restore the Property as close as reasonably possible to its former use or condition, or to an alternate use or condition that is considered appropriate and acceptable by Québec's Department of Energy and Natural Resources(MERN). The Plan will outline measures to be taken for progressive rehabilitation, closure rehabilitation and post-closure monitoring and treatment. It will also help refine the evaluation of restoration costs completed as part of this Report.

Capital and Operating Cost

The estimate is expressed in Canadian dollars unless specified otherwise.

Capital cost

The pre-production capital costs for the POX option are estimated at \$394M and sustaining capital is estimated at \$118M. The capital costs include various added contingencies depending on the sector. In the base case estimate, contingencies and indirect costs total \$98.7M of the pre-production costs and represent 26% of the costs. Indirect costs (owner's costs; engineering, procurement and construction management (EPCM); and detailed engineering) of 37% have been applied to the process plant and to other surface infrastructure. The average contingency for all environmental items is 20%.

The total capital expenditure of \$512M for the Duparquet Project is broken down into five (5) cost components (see table below): mining; surface installation and equipment; processing facilities; tailings storage facilities; and environmental.
The tailing storage facilities item in the table below includes the reclaim pumping station and pipeline. The remainder of the tailings dam infrastructure is included in the environmental pre-production and sustaining costs.

Breakdown of the capital cost (Table 21.1)

Description	Pre-production ($)	Sustaining ($)	Total cost ($)
Capitalized operating cost	*51,012,141*	-	*51,012,141*
Capitalized revenue	- *21,984,860*	-	- *21,984,860*
Mine production equipment	23,120,924	91,126,227	114,247,151
Surface installation and equipment	58,218,662	10,144,723	68,363,385
Processing Facilities	226,611,220	-	226,611,220
Tailings Storage Facilities	3,374,029	-	3,374,029
Environmental	53,707,038	16,712,074	70,419,112
Total	**394,059,154**	**117,983,024**	**512,042,179**

Operating Cost

The OPEX cost breakdown for the PFS is divided into five (5) main categories: general and administration (G&A); processing; mining; environmental; and overburden removal. The G&A category includes the costs of technical services and administration. Open pit mining costs include drilling, blasting, loading, hauling, auxiliary, and general mine maintenance. The processing category includes manpower, the cost to process ore from the pit, and the cost to process the old tailings. The environmental category includes manpower and departmental costs.

Summary of total operating costs (Table 21.8)

Description	Total cost estimate (production period)		Unit cost	
			($/t ore)	($/oz Au)
General and administration	$	95,457,201	2.46 $/t	56.72 $/t
Processing cost	$	608,136,366	15.66 $/t	361.35 $/t
Mining cost	$	707,899,014	18.23 $/t	420.63 $/t
Environmental monitoring	$	6,900,077	0.18 $/t	4.10 $/t
Overburden removal cost	$	15,966,057	0.41 $/t	9.49 $/t
Total	**$**	**1,434,358,715**	**36.94 $/t**	**852.28 $/t**

Pre-tax and after-tax cash flow projections were generated from the LOM schedule according to the capital and operating cost estimates. It was done in constant 2013 money terms and in Canadian currency unless stated otherwise, with no allowance for inflation or escalation. The net cash flow has been discounted for the purposes of calculating NPV. A base discount rate of 5% per year has been selected as most likely to represent a low capital expense gold project in a mining-friendly environment. Future annual cash flow estimates are based on grade, gold recoveries and cost estimates.

The PFS considered two possible processing scenarios. Cash flow models were created for both options. The POX option generated the highest financial return and, as a result, the POX process is favoured and used as the base case of the present PFS.

Financial Analysis

The undiscounted pre-tax cash flow totals $493.19M over the 11-year mine life and the payback period is 4.3 years.

A summary of the base case cash flow model is given in the table below. LOM totals for undiscounted and discounted cash flows are also provided. The table shows that the pre-tax NPV of the project cash flow at a discount of 5% per year is evaluated at approximately $222M and a pre-tax internal rate of return of 15.11%. The average cash cost of production equates to US$775/oz gold.

Cash flow analysis summary (Table 22.1)

Parameters		Results
Gold Price		1,300 US$/oz
Foreign exchange rate		1.10 : 1.00 (CAN/USD)
Mineable reserves		35,2 Mt @ 1.56g/t Au; 1.89 g/t Ag
Old tailings		4,1 Mt @ 0.93 g/t Au; 2.40 g/t Ag
Recovered Gold	From mine	1.6 Moz
	From old tailings	0.1 Moz
	Total	1.7 Moz
Recovered Silver	From mine	1.9 Moz
	From old tailings	0.3 Moz
	Total	2.2 Moz
Average annual gold production (ounces):		173,000 oz (first 5 years) 158,000 oz (Average 11 year)
Total waste		291 Mt
Total OVB		23.4 Mt
Mine life (excluding 4 years of pre-production)		11 years
Daily mine production		10,000 tpd
Metal recovery Au	Mine	90.10%
	Old tailings	83.90%
Pre-production capital		394M$
Sustaining capital (excluding 24.5M$ for closure cost)		118M$
Average operating cost		36.94 C$/tonne milled
Average total Site Cash Cost (US$/ounce)		775 US$/oz Au
Average total All in Cost, Average (US$/ounce) LOM		1042 US$/oz Au
Net cashflow		493M$
Pre-tax NPV (5%)		222M$
Pre-tax IRR		15.11%
After-tax NPV (5%)		135M$
After-tax IRR		12.06%
Payback period		4.3 years

Risk and opportunities

Risk

- There is a risk that Hydro-Québec might not have the capacity to supply the Duparquet Project, depending on other power consumptions. Further study and discussion should be initiated with Hydro-Québec.

- There is a risk that the MERN will not allow Clifton Star to recover the portion of the Mineral Resources in the tailings that are outside the Property boundary. If this were to happen, the authors wish to point out that this very small portion of the total Mineral Resources would only have a minor impact and would not affect the potential viability of the Project.

- There is a risk that the CAPEX to develop the project will be higher than estimate. Not all recent projects have achieved the expected cost anticipated to reach production. To limit the risk, Clifton Star should put considerable effort into assembling a highly qualified and experienced team for project development.

- The foundations cost could be higher or lower depending on the results of future geotechnical studies.

- If development targets are not achieved during the preproduction and early production periods, the ramp-up schedule to full production may be compromised.

- There is a risk that the Québec Ministry of Transport will not allow heavy equipment to travel across Route 393, or will require a larger buffer zone between the highway and the pit edges.

- The social and economic effects in developing a by-pass for the portion of highway 393 that passes over the mineralized zone should be investigated.

- The risk related to metallurgical issues are that:
  - variations in mineralogy could cause fluctuations in mill recovery;
  - grinding characteristics may differ from test results;
  - thickening and filtration testwork may not be representative of the ore deposit;

- As with all mining projects in the Province of Québec or, more largely, in Eastern Canada and also elsewhere in the world, there is always a risk that the mining company may not be successful in obtaining a “social licence to operate”. In the present case however, the “social acceptability” risk is relatively mitigated in part because the Project is located in an area with a long history of mining development (when compared with other regions). Still, for any mining project located close to urban areas, there is always a risk of developing a project that is not acceptable for the local, regional or provincial communities.

Opportunity

- The project is located in located in the vicinity of the town of Duparquet and close to major center of Rouyn-Noranda and La Sarre. Both skilled and general workforces are readily available. Suppliers, contractors, consulting firms and competent workers are available locally.

- The inferred Mineral Resource blocks as well as all blocks having at least 0.01% of their volume contained within stope solids that fall within the pit limit have been treated as waste and have been assigned a zero grade. If these blocks are incorporated in the mine schedule, it should translate in a tonnage and grade increase.

- The additional 47 drill holes and channel samples that were completed after current resource cut-off date (May 6th, 2013) could be included in a new resource calculation and thus increase the current resource.

- Additional definition drilling could convert the current Inferred Resources into Indicated Resource and Reserves.

- Additional exploration drilling could expand the known mineralized zones along strike and at depth.

- Geotechnical drilling along the proposed pitwalls may allow steeper pitwalls and hence reduce the waste to ore strip ratio.

- The economic effects in developing a by-pass, for the portion of highway 393 that passes over the mineralized zone, that would permit mining a single larger pit, should be investigated.

- Future mining potential of the Duparquet Project, below the current pit shell, by an underground bulk mining method will need to be assessed and could provide an extension of the mine life.

- The province's mining industry is well established. The hydro-electricity is affordable and offers competitive advantage for companies operating in Quebec. Mining companies also enjoy relatively stable mining legislation.

- The historical Beattie Mine site surface area will be cleaned up and the current buildings demolished and removed prior to the development of the open-pit mining. The old tailing area will be reclaimed as part of the mine closure plan.

InnovExplo, Roche and Dresinger Consulting conclude that this PFS allows the Duparquet Project to advance to a feasibility study stage. InnovExplo believes that more advanced engineering work is necessary to support a feasibility study and to reduce the risk to the Duparquet Project.

InnovExplo considers the present PFS to be reliable and thorough, and based on quality data, reasonable hypotheses and parameters compliant with NI 43-101 and CIM standards regarding mineral resource estimations.

Recommendation

Based on the results of the updated Mineral Resource Estimate and the positive outcome of the PFS, InnovExplo, Roche and Dreisinger Consulting recommend advancing the Duparquet Project to the next phase, which would consist of preparing a feasibility study. The recommended work program is summarized in the table below. The estimated cost for the work program would amount to approximately \$8.8M, and would include exploration, definition and condemnation drilling, metallurgical, geotechnical and hydrogeological testwork, and a final feasibility study report. The table also presents the estimated costs for the phases of the recommended program.

Estimated costs for the recommended work program (Table 26.1)

Budget	Cost Estimate
Drilling	
Drilling and compilation of historical diamond drill holes.	$ 2,250,000
Resource update	$ 80,000
Subtotal:	**$ 2,330,000**
Mining	
Geotechnical study	$ 812,500
Subtotal:	**$ 812,500**
Metallurgical testwork budget	
Drilling and sample collection	$ 225,000
Sample preparation	$ 30,000
Variability testwork program	$ 120,000
Grindability testwork	$ 55,000
Testwork to confirm reagent selection and dosage (float)	$ 25,000
Additional testwork to confirm silver recovery	$ 30,000
Mineralogical characterization	$ 70,000
Integrated pilot plant of the whole circuit, operated for an extended period of time	$ 400,000
Investigation of leaching and adsorption kinetics on lime boiled samples	$ 30,000
Subtotal:	**$ 985,000**
Infrastructure work budget	
Condemnation drilling	$ 562,500
Geotechnical work	$ 150,000
Borrow pit assessment	$ 10,000
Subtotal:	**$ 722,500**
Environmental work budget	
Environmental baseline studies (surface water, sediment, fish habitat, air quality, noise level, hydrology, avifauna, wildlife, etc.)	$ 375,000
Hydrogeological study (ground water hydraulics)	$ 225,000
Geochemical characterization (kinetic tests)	$ 35,000
Social studies (economic spin-offs, landscape change analysis, traffic study, land use, risk assessment, etc.)	$ 200,000
Social acceptability program	$ 170,000
Environmental and social Impact assessment	$ 389,750
Mine rehabilitation and closure plan	$ 30,000
Subtotal:	**$ 1,424,750**
Feasibility Study Budget	**$ 2,500,000**
Total:	**$ 8,774,750**

2. INTRODUCTION

On May 14, 2013, InnovExplo Inc. ("InnovExplo") was retained by Mr. Michel Bouchard, M.Sc., P.Geo., MBA, President and CEO of Clifton Star Resources Inc. ("Clifton Star" or "the issuer") to produce a Technical Report, an Updated Mineral Resource Estimate and a Prefeasibility Study for the Duparquet Project in accordance with National Instrument 43-101 and Form 43-101F1. The updated Mineral Resource Estimate has been published in an earlier report titled "Technical Report and Mineral Resource Estimate Update for the Duparquet Project (according to Regulation 43-101 and Form 43-101F1)", dated August 2, 2013 (Williamson et al., 2013b). The Mineral Resource Estimate covers the Duparquet Project, defined here as the amalgamation of the Beattie, Donchester, Central Duparquet and Dumico properties, as well as the historical Beattie mine tailings. The Duparquet Project ("the Project") is located just north of the Town of Duparquet in the Province of Québec.

This Technical Report ("the Report") presents the results of the Prefeasibility Study ("the PFS") on the Duparquet Project using the August 2013 updated resource estimate as its basis.

The PFS was prepared by InnovExplo, Tenova Mining & Minerals–Bateman Engineering Pty Ltd ("Tenova-Bateman"), Roche Ltd Consulting Group ("Roche"), and Dreisinger Consulting Inc. ("Dreisinger Consulting").

The Report is addressed to Clifton Star Resources Inc., a Canadian mineral exploration company trading publicly on the Toronto Stock Exchange in Canada (TSX-V: CFO). InnovExplo is an independent mining and exploration consulting firm based in Val-d'Or (Québec).

2.1 Terms of Reference

The issuer requested an update of the Mineral Resource Estimate to include new and re-sampled drill holes and some additional historical holes. The Mineral Resource Estimate for the tailings pond has not been modified, and the results for this component, as presented herein, are taken directly from InnovExplo's NI 43-101 compliant report released July 5, 2012 (Brousseau et al., 2012).

The global objectives of the PFS, as requested by the issuer, are to:

- Determine the best project design by comparing a concentrate production option and a pressure oxidation (POX) option;
- Optimize a mining plan that takes into account fixed physical constraints in order to minimize any effect on the Town of Duparquet or provincial infrastructures;
- Optimize the potential economic viability of exploiting the Duparquet deposit;
- Estimate a Mineral Reserve for the Duparquet Project; and
- Propose a strategy and preliminary timetable to further develop the Duparquet Project.

The PFS evaluates and/or provides the following items:

- An optimized pit design and schedule;

- Advanced design for most of the facilities, and the infrastructure needed to access, develop and operate the mine;
- A capital cost estimate with an accuracy of ±25%;
- A cashflow model and sensitivity;
- Recommendations for additional work in order to advance the Project to a feasibility stage;
- A technical report.

2.2 Principal Sources of Information

InnovExplo's review of the Duparquet Project was based on published material as well as the data, professional opinions and unpublished material submitted by Clifton Star. The interpretation of the existing tailings pond zones for the purpose of the tailings Mineral Resource Estimate was done by GENIVAR. The following specialists provided information for various portions of the study:

- Bruno Soucy, Hydro-Québec, provided costs from an exploratory study for a 120kV power line.
- Frederic Levesque, Eng., Commercial Lead Q&C Eastern Canada at Orica Canada, provided the budgetary quote for commercial explosives and related services;
- Nicola Fournier, Eng., Vice President Sales & Technical Services at Fournier, provided the costs for stemming materials, aggregates for road maintenance and construction.
- Sébastien Roy, Sales Representative for Sandvik Mining, provided the budgetary quote for the drilling equipment and tools;
- Guy Lefebvre, Sales Location Manager at Xylem, provided the budgetary quote for the pit dewatering system;
- Francine Vallée, Industrial Sales Assistant at "Les Industries Fournier Inc", provided the budgetary quote for the steel tank;
- L.P. Disson, Manager for Continental Blower LLC, provided the budgetary quote for the air blowers;
- François Tellier, President at SAN Compression Inc., provided the budgetary quote for the air compressors;
- Mustapha Bouabdellah, Sales representatives at TYCAN, provided the budgetary quote for the screens;
- Paul de la Durantaye, Senior Sales Manager at Outotec, provided the budgetary quote for the filter press with auxiliary equipment and flotation cells;
- H. Fraser Bringeland, Director of Sales at STT Enviro Corp, provided the budgetary quote for the Lime Slaker System Package;
- Don LaRose, at Heath & Sherwood, provided budgetary quote for the process samplers;
- Julian Hernandez, Project Proposal Engineer at Metso, provided the budgetary quote for the crushers;
- David Komlenic, at Metso, provided the budgetary quote for the Apron feeders;
- Bob Rutkowski, Global Proposals and Projects Manager MPS Vibrating Equipment at Metso, provided the budgetary quote for the SAG Mills screen;

- Brittney A. Eckert, Proposal Engineer at Metso, provided the budgetary proposal for the SAG and Ball Mills;
- Steve Walker, Estimator at Continental conveyor, provided the budgetary quote for the belt conveyors;
- Pete J. Paterson, Director de division at Technologies de procédé WARCO, provided the budgetary quote for the self-cleaning magnet;
- Michel Trussart, at Dynagroup Technologies Inc., provided the budgetary quote for the dust collectors;
- Carl Belair, Industrial Crane Sales at Konecranes Canada Inc., provided budgetary quote for the hoist and overhead crane;
- William Breuer, Sales Director at TENOVA DELKOR CANADA, provided budgetary quote for the trash screen;
- Ravi Shankar, Vice president at Mining Equipment Company (EIMCO K.C.P ltd), provided the budgetary quote for the thickeners and rake mechanism;
- Rick Romney, Manager Mining Products at McLellan Industries, provided the budgetary quote for the SAG Mill Liner Handler;
- Ben Slater, Engineered Product Sales at Hayward Gordon Ltd., Provided the budgetary quote for the agitators and the flocculants preparation and storage system;
- Terry McKague, at BTI Breaker Technology Ltd, provided the budgetary quote for the rock breaker;
- Vladimir Pajio, Application Engineer at Weir Minerals, provided the budgetary quote for the cyclones and the slurry pumps;
- Kelsey McCaslin, Project Coordinator at FLSmidth Summit Valley Technologies, provided the budgetary quote for a Carbon elution, regeneration and electrowinning.
- Ricky Boulanger, Sales vice-president, Québec and East of Canada at outland, provided budgetary quote for the Office and laboratory building;
- Marty Dilworth, Sale representative at MiniBulk, provided the budgetary quote for the concentrate bulk bags;
- Yannick Loiselle , Sale representative at Quadra Chemical, provided budgetary quote for reagents;
- Eric G. Gauthier, Canadian Account Manager at Cyanco Canada Inc., provided budgetary quote for cyanide;
- Isabelle Dumont, Sales Director North East at Chemtrade, provided budgetary quote for SO2;
- Yves Olivier Lamarche, Account Manager at GRAYMONT, provided budgetary quote for Lime and Limestone;

InnovExplo has reviewed the data provided by the issuer and/or by its agents. InnovExplo has also consulted other information sources, such as the Québec government's claims management database, for assessment work and the status of mining titles.

InnovExplo, Roche and Dreisinger Consulting, conducted a review and appraisal of the information used in the preparation of the Report and to formulate its conclusions and recommendations, and believes that such information is valid and appropriate considering the status of the Project and the purpose for which the Report is prepared. The authors have fully researched and documented the conclusions and recommendations made herein.

2.3 Qualified Persons and Inspection on the Property

The qualified persons responsible for the preparation of the Report included the principal authors Sylvie Poirier, Eng. (InnovExplo), Karine Brousseau, Eng. (InnovExplo) and Laurent Roy, Eng. (InnovExplo), as well as David Dreisinger, B.Sc., PhD, P.Eng. (Dreisinger Consulting Inc), Philippe Coté, Eng. (Roche Ltd), David Sims, P.Geo. (Roche Ltd), and Martin Magnan, Eng., M.Sc.A (Roche Ltd).

Technical support from InnovExplo was provided by Marie-Claire Dagenais, Jr Eng., and Serge Morin. In addition, Bruno Turcotte, P.Geo., also of InnovExplo, validated the compliance of the technical report with National Instrument 43-101 (“NI 43 101”) and Form 43-101F1. Technical support from Roche was provided by Simon Thibault, M.Sc. bio., Yves Thomassin, M.Sc.A., Marc Rood, sr. tech., Véronique Simard, Eng., Pierre Côté, sr. tech, Alain Dorval, Eng., Hamidreza Kebriaei, Sr. Metallurgist, Claude Poirier, Eng., Daniel Boucher, Tech., Paul Latreille, Tech., Joé Landry, Eng. Technical support from Tenova-Bateman was provided by Linus Sylwestrzak.

The list below presents the sections for which each Qualified Person was responsible:

- Sylvie Poirier: supervised the preparation of the report, author of Sections 15, 19, 21.2 to 21.4, 22 and co-author of Sections 1, 2, 3, 16, 21.10 and 25 to 27;
- Karine Brousseau: responsible for the preparation of sections 4 to 12, 14, 23 and 24, and co-author of sections 1, 2, 3, and 25 to 27;
- Laurent Roy: co-author of sections 1, 2, 3, 16, 21.10 and 25 to 27;
- Carl Pelletier: supervised the preparation of the Mineral Resource Estimate of the Duparquet Project and supervised the preparation of sections 4 to 12, 14, 23 and 24;
- David Dreisinger: author of Sections 13.2.2 to 13.2.6 and 17.2 and co-author of Sections 1, 2, 3, 13.2, 13.3, 17.2 and 25 to 27;
- Philippe Coté: author of Sections 13.1, 17.1, 18.1, 21.5 to 21.7 and 21.9 and co-author of Sections 1, 2, 3, 13.2, 13.3, 17. 2, 18.2, 21.10 and 25 to 27;
- David Sims: co-author of Section 18.2;
- Martin Magan: author Section 20 and co-author of Sections 1, 2, 3, 21.8 and 25 to 27

For the purpose of the Technical Report mandate, Laurent Roy and Marie-Claire Dagenais, both of InnovExplo, visited the Duparquet Project site on May 27, 2013, accompanied by Louis Martin of Clifton Star. David Sims visited the Property on November 18 and was also accompanied by Louis Martin.

2.4 Units and Currencies

All currency amounts are stated in Canadian Dollars ($ or C$) or US dollars (US$). Quantities are stated in metric units, as per standard Canadian and international practice, including metric tons (tonnes, t) and kilograms (kg) for weight, kilometres (km) or metres (m) for distance, hectares (ha) for area, and grams (g) or grams per metric ton (g/t) for gold grades. Wherever applicable, imperial units have been converted to the International System of Units (SI units) for consistency. A list of abbreviations used in the Report is provided in Appendix I.

3. RELIANCE ON OTHER EXPERTS

The authors, Qualified and Independent Persons as defined by National Instrument 43-101, were contracted by the issuer to study technical documentation relevant to the Report, to perform a prefeasibility study for the Duparquet Project and to recommend a work program if warranted. The authors have reviewed the mining titles and their status, as well as any agreements and technical data supplied by the issuer (or its agents), and any available public sources of relevant technical information.

Some of the geological and technical reports for projects in the vicinity of the Duparquet Project were prepared before the implementation of National Instrument 43-101 in 2001 and Regulation 43-101 in 2005 (hereinafter “NI 43-101”). The authors of such reports appear to have been qualified, and the information prepared according to standards acceptable to the exploration community at the time. In some cases, the data are incomplete and do not fully meet current NI 43-101 requirements. The authors have no known reason to believe that any of the information used to prepare the Report is invalid or contains misrepresentations.

The authors relied on the following reports and opinions for information that is not within the authors’ fields of expertise:

- Information about the mining titles and option agreements described in Section 4.2 was supplied by Clifton Star. InnovExplo is not qualified to express any legal opinion with respect to the property titles or current ownership and possible litigation (source: Clifton Star press releases and agreements on the SEDAR website).
- The hauling and loading fleet recommendation is based on a Fleet Production and Cost (FPC) analysis performed by Yves Laquerre, Eng., and François Tremblay, Mining Account Manager Business Development, both of Hewitt Equipment Ltd. The analysis was performed on the basis of the mine plan data supplied by InnovExplo.
- Based on the testwork, Linus Sylwestrzak, Process Consultant Hydrometallurgy at Tenova Mining & Minerals (Australia) Pty Ltd developed the process design criteria, sized and selected the process equipment of the POX area. He also performed the metallurgical balance for the POX area.
- The preliminary geomechanical assessment was done by Jane Alcott, MASc, Eng., of InnovExplo.
- Pierre-Jean Lafleur, Eng., of P.J. Lafleur Géo-Conseil Inc., participated in the Whittle Pit optimization.
- Lucie Chouinard, CA, M.Fisc., of Samson Bélair Deloitte & Touche, completed the after-tax cash flow estimation.
- Venetia Bodycomb, M.Sc., of Vee Geoservices, provided the linguistic editing on a draft version of the Technical Report.

The authors believe the information used to prepare the Report and to formulate its conclusions and recommendations is valid and appropriate considering the status of the Project and the purpose for which the Report is prepared.

The authors, by virtue of their technical review of the Project's exploration potential, affirm that the work program and recommendations presented in the Report comply with NI 43-101 and CIM technical standards.

4. PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Duparquet Project is located just north of the Town of Duparquet, partly overlapping the municipal boundary, in the Duparquet Township of Québec, NTS map sheet 32D/11. The coordinates for the approximate centre of the Project are 48º30’34”N, 79º12´34”W (UTM projection: 5374410N, 631517E, NAD 83 Zone 17). Figure 4.1 is a location map of the Duparquet Project in the province of Québec.

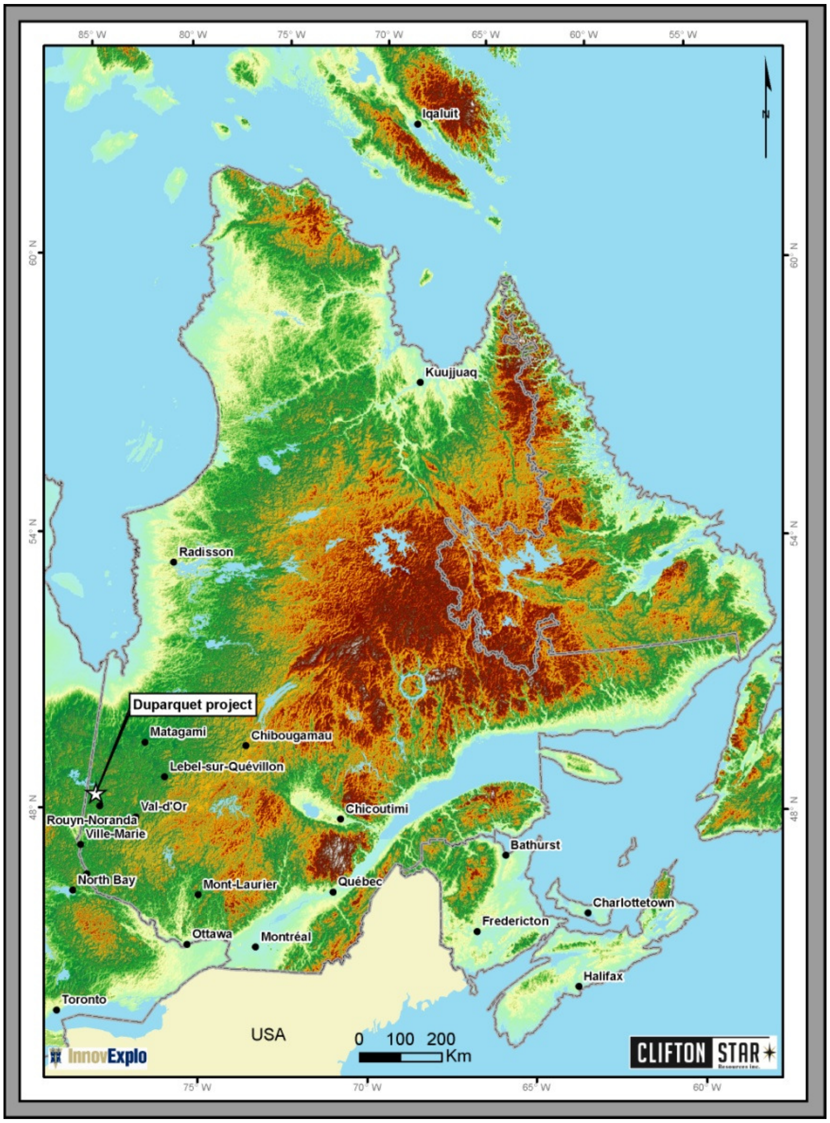

Figure 4.1 – Location map for the Duparquet Property

The Duparquet Property, as defined herein, consists of the amalgamation of four contiguous properties (from west to east): Beattie, Donchester, Central Duparquet and Dumico) (Fig. 4.2).

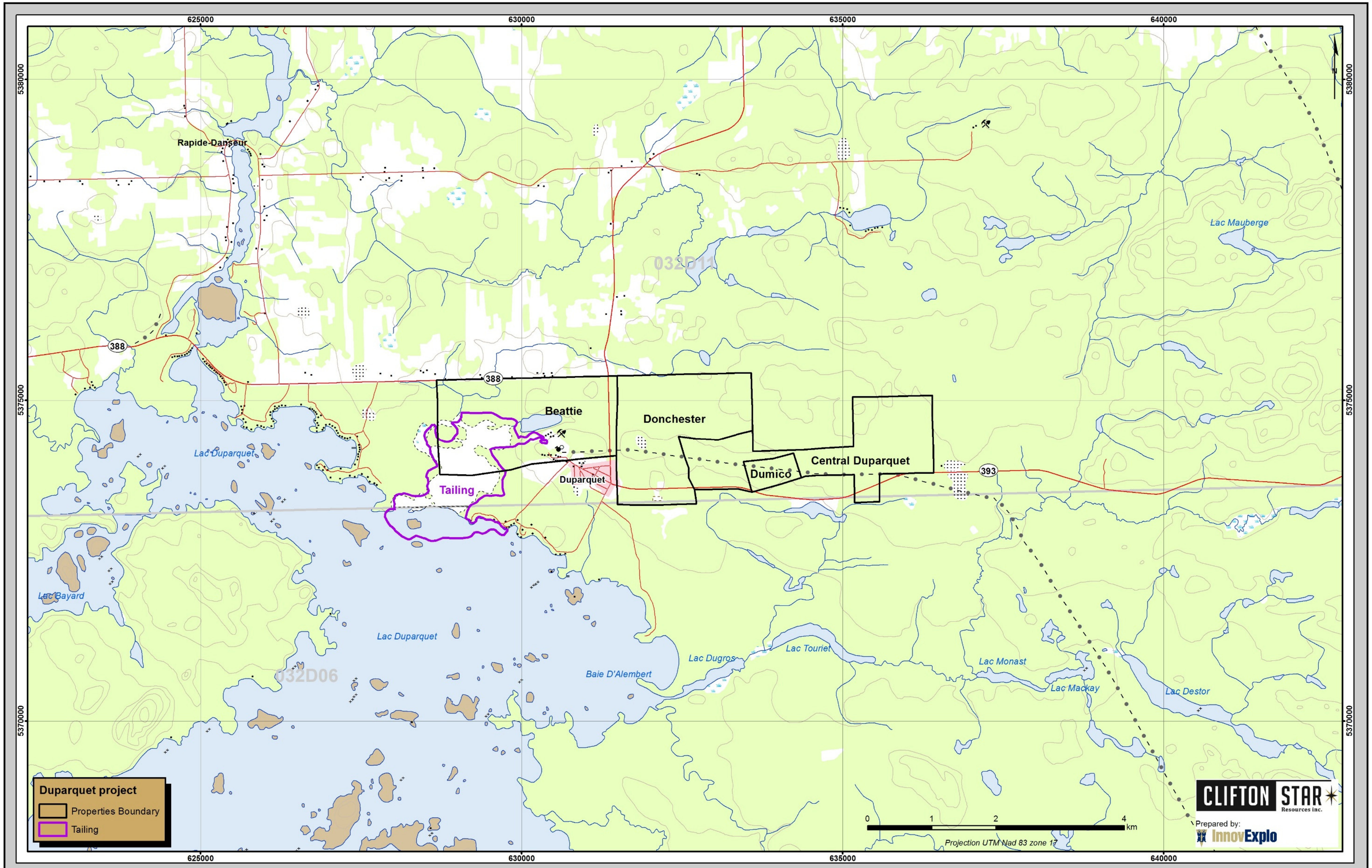

Figure 4.2 – The Duparquet Project includes the Beattie, Donchester, Central Duparquet and Dumico properties. Beattie mine tailings boundaries were outlined by Fillion (2009)

The Duparquet Property covers an area of 1033.6 ha. The Beattie property accounts for approximately 383.6 ha; the Donchester property covers about 322.6 ha; and the Central Duparquet property and Dumico claims account for the remaining 293.4 ha and 34 ha respectively. Both the Beattie and Donchester properties contain past-producing mines. Historical underground workings and a shaft were developed at the Central Duparquet property but no gold was produced. The Duparquet Project is defined in this report as the Beattie, Donchester, Central Duparquet and Dumico properties, as well as the tailings pond area straddling the southwest limit of the Beattie property. Beattie mine tailings boundaries were outlined by Fillion (2009).

4.2 Mining Rights in the Province of Québec

The following discussion on the mining rights in the province of Québec was mostly taken from Guzon (2012) and from an Act to amend the Mining Act (Bill 70) assented on December 10, 2013 (National Assembly, 2013).

In the Province of Québec, mining is principally regulated by the provincial government. The Department of Energy and Natural Resources(MERN) is the provincial agency entrusted with the management of mineral substances in Québec. The ownership and granting of mining titles for mineral substances is primarily governed by the *Mining Act* (the Act) and related regulations. In Québec, land surface rights are distinct property from mining rights. Rights in or over mineral substances in Québec form part of the domain of the State (the public domain), subject to limited exceptions for privately owned mineral substances. Mining titles for mineral substances within the public domain are granted and managed by the MNR. The grant of mining rights in privately owned mineral substances is a matter of private negotiations, although certain aspects of the exploration for and mining of such mineral substances are governed by the Act. This section provides a brief overview of the most common mining rights for mineral substances within the domain of the State.

A claim is the only exploration title for mineral substances (other than surface mineral substances, petroleum, natural gas and brine) currently issued in Québec. A claim gives its holder the exclusive right to explore for such mineral substances on the land subject to the claim but does not entitle its holder to extract mineral substances, except for sampling and in limited quantities. In order to mine mineral substances, the holder of a claim must obtain a mining lease. The electronic map designation is the most common method of acquiring new claims from the MNR whereby an applicant makes an online selection of available pre-mapped claims. In rare territories, claims can be obtained by staking. As of December 10th 2013, claim holders must notify the municipality and the landowner concerned within 60 days after registering a claim of the fact that they have obtained the claim, and must inform the municipality and the landowner at least 30 days before performing work. They must also submit an annual report on all work performed to the Minister of Natural Resources.

A claim has a term of two years, which is renewable for additional periods of two years, subject to performance of minimum exploration work on the claim and compliance with other requirements set forth by the Act. In certain circumstances, if the work carried out in respect of a claim is insufficient or if no work has been carried out at all, it is possible for the claimholder to comply with the minimum work obligations by using work credits for exploration work conducted on adjacent parcels or by making a payment in lieu of the required work. As of December 10th 2013, the holder of a claim where the work has not been performed will have to pay double of the value of the work to renew his claim. Moreover, it is no longer possible to apply all or a part of amounts spent to perform work in respect of a mining lease or mining concession against a claim to be renewed.

Mining leases and mining concessions are extraction (production) mining titles which give their holder the exclusive right to mine mineral substances (other than surface mineral substances, petroleum, natural gas and brine). A mining lease is granted to the holder of one or several claims upon proof of the existence of indicators of the presence of a workable deposit on the area covered by such claims and compliance with other requirements prescribed by the Act. A mining lease has an initial term of 20 years but may be renewed for three additional periods of 10 years each. Under certain conditions, a mining lease may be renewed beyond the three statutory renewal periods.

Mining concessions were issued prior to January 1, 1966. After that date, grants of mining concessions were replaced by grants of mining leases. Although similar in certain respects to mining leases, mining concessions granted broader surface and mining rights and are not limited in time.

As of December 10 2013, a mining lease cannot be granted until a rehabilitation and restoration plan, regarding which the certificate of authorization required under the Environment Quality Act has been issued, and a scoping and market study as regards processing in Québec are submitted to the Minister. Moreover, when granting a lease, the Government may, on reasonable grounds, require that the economic spinoffs within Québec of mining the mineral resources authorized under the lease be maximized and may require the lessee to establish and maintain a monitoring committee to foster the involvement of the local community in the Project as a whole.

The claims, mining leases and concessions, exclusive leases for surface mineral substances and the licences and leases for petroleum, natural gas and underground reservoirs obtained from the MNR may be sold, transferred, hypothecated or otherwise encumbered without the MNR's consent. However, a release from the MNR is required for a vendor or a transferee to be released from its obligations and liabilities owing to the MNR related to the mine rehabilitation and restoration plan associated with the alienated lease or mining concession. Such release can be obtained when a third-party purchaser assumes those obligations as part of a property transfer. For perfection purposes, the transfers of mining titles and grants of hypothecs and other encumbrances in mining rights must be recorded in the register of real and immovable mining rights maintained by the MNR and other applicable registers.

4.3 Mining Titles and Claims Status

The claim list was supplied by Michel Bouchard, president and CEO of Clifton Star Resources Inc. InnovExplo has verified (as of July 1, 2013) the current status of all claims using GESTIM, the Québec government's claim management system available from the MERN via their website at the following address:

http://gestim.mines.gouv.qc.ca. A detailed list of mining titles and claims, ownership, and expiry dates is provided in Table 4.1. The location of these mining titles and claims is presented in Figure 4.3.

Table 4.1 – Duparquet Project mining titles and claims summary (As of July 01, 2013)

Property	Title Number	Approx. Area (ha)	Registration date	Expiration date	Registered Owner
Beattie	292	383	1937-10-07		Beattie Gold Mines Ltd (108) 100% (owner)
Donchester	384	322	1950-10-26		173714 Canada Inc (8153) 100% (owner)
Central Duparquet	3230711	17	1972-04-27	2017-04-07	LISETTE GRENIER (21185) 100% (owner)
	3230712	15	1972-04-27	2017-04-07	LISETTE GRENIER (21185) 100% (owner)
	3230713	16	1972-04-27	2017-04-07	LISETTE GRENIER (21185) 100% (owner)
	3230714	14	1972-04-27	2017-04-07	LISETTE GRENIER (21185) 100% (owner)
	3230715	17	1972-04-27	2017-04-07	LISETTE GRENIER (21185) 100% (owner)
	3230741	18	1972-04-27	2017-04-10	LISETTE GRENIER (21185) 100% (owner)
	3230742	18	1972-04-27	2017-04-10	LISETTE GRENIER (21185) 100% (owner)
	3230744	7	1972-04-27	2017-04-10	LISETTE GRENIER (21185) 100% (owner)
	3806541	17	1979-11-05	2017-10-17	LISETTE GRENIER (21185) 100% (owner)
	3806542	15	1979-11-05	2017-10-17	LISETTE GRENIER (21185) 100% (owner)
	3806543	15	1979-11-05	2017-10-17	LISETTE GRENIER (21185) 100% (owner)
	3806544	17	1979-11-05	2017-10-17	LISETTE GRENIER (21185) 100% (owner)
	3806545	17	1979-11-05	2017-10-17	LISETTE GRENIER (21185) 100% (owner)
	3806551	20	1979-11-05	2017-10-18	LISETTE GRENIER (21185) 100% (owner)
	3806552	17	1979-11-05	2017-10-18	LISETTE GRENIER (21185) 100% (owner)
	3806553	14	1979-11-05	2017-10-18	LISETTE GRENIER (21185) 100% (owner)
	3806554	14	1979-11-05	2017-10-18	LISETTE GRENIER (21185) 100% (owner)
	3806555	16	1979-11-05	2017-10-18	LISETTE GRENIER (21185) 100% (owner)
Dumico	C003231	20	1924-10-18	2017-05-14	173714 Canada Inc (8153) 100% (owner)
	C003232	14	1924-10-18	2017-05-14	173714 Canada Inc (8153) 100% (owner)

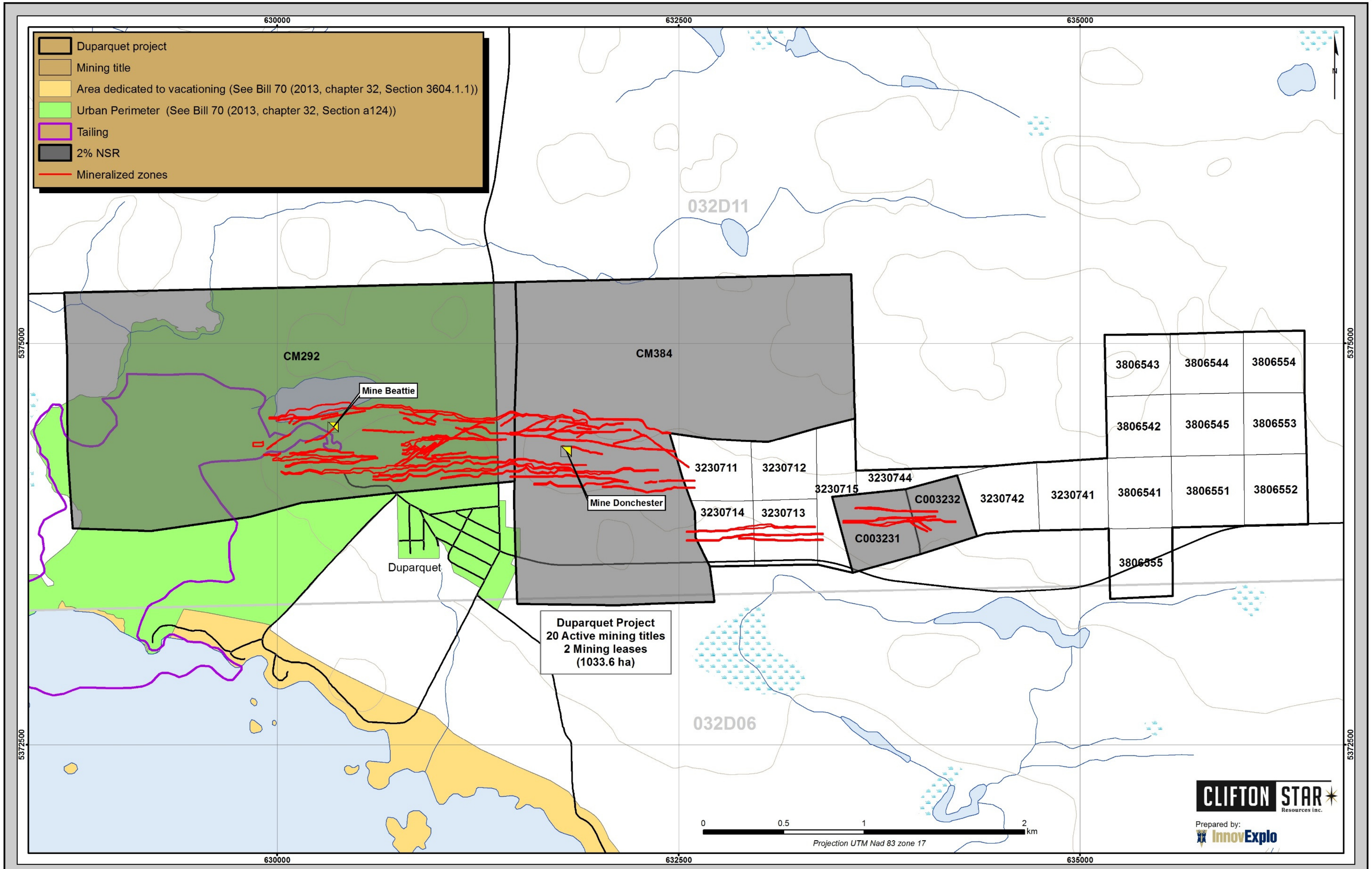

Figure 4.3 – Location map showing mining titles and claims comprising the Duparquet Property

Clifton Star signed mineral property option agreements with the current owners of the Beattie, Donchester and Dumico properties on May 6, 2008. The initial agreement has been amended several times: July 22, 2008; November 24, 2008; and April 8, 2009. In a news release dated September 12, 2012, Clifton Star announced that these agreements had been amended once again (source: CFO – Press Release and agreements on SEDAR website).

Clifton Star signed a mineral property option agreement with the owner of the related Central Duparquet claims in December of 2008 (source: CFO – Press Release and agreements on SEDAR website).

In December of 2009, Clifton Star entered into a mineral property option and joint venture agreement with Osisko Mining Corporation ("Osisko"), which was terminated on June 16, 2011. Clifton Star Resources is under the legal opinion that it is still entitled to a loan of $22.5 M from Osisko, which was to be used to facilitate the payment to the underlying property owners.

4.3.1 Beattie, Donchester and Dumico Properties

The option agreements related to the Beattie, Donchester and Dumico properties involve five (5) other companies.

Beattie Gold Mines Ltd ("Beattie Mines"), having its registered office at 2147 Portage Avenue, Winnipeg, Manitoba, is the legal and beneficial owner of a 100% undivided interest in the mineral mining rights of the Beattie mining concession (MC#292 of the Beattie property).

2588111 Manitoba Ltd ("2588111"), having its registered office at 2147 Portage Avenue, Winnipeg, Manitoba, owns, through its wholly owned subsidiary, 173714 Canada Inc ("173714"), the mineral mining rights of the Donchester mining concession (MC#384 of the Donchester property), the mineral mining and surface rights of the Hunter mining concession (MC#442 of the Hunter property) and mining rights on the Dumico Claims (C003231 and C003232).

2699681 Canada Ltd ("2699681"), having its registered office at 2147 Portage Avenue, Winnipeg, Manitoba, owns, through its wholly owned subsidiary, Eldorado Gold Mines Inc. ("Eldorado"), the surface rights to the Beattie mining concession #292, including the concentrate roasters located thereon, but excepting land that has been sold for the golf course, the houses near the golf course with their accompanying land, and the northeast part of the MC#292. Eldorado also owns part of the surface rights to the Donchester mining concession #384, excepting land that has been deeded to the church for the cemetery, and the northwest part of the MC#384. Eldorado owns the surface rights to the Dumico property. Eldorado also owns the mine tailings that originated from the original Beattie, Donchester, Duquesne and Hunter mines.

On May 1, 2008, Clifton Star signed mineral property option agreements with the three companies presented above, with similar terms. The initial agreements have been amended three times (July 22, 2008, November 24, 2008, and April 8, 2009). On October 26, 2009, Clifton Star signed a Letter of Intent with the optionors. Clifton

Star then renegotiated with the optionors to terminate the aforementioned mineral property option agreements and enter new agreements, which resulted in payments of $600,000 to Beattie Mines, $300,000 to 2699681, and $600,000 to 2588111 under the old option agreements (Clifton Star, MD&A, May 29, 2012).

On September 12, 2012, Clifton Star announced in a new release that new terms had been negotiated with the optionors.

The following terms of the new agreement will earn Clifton Star the remaining 90% of the issued and outstanding shares of the optionors:

- Cash payments of $3,400,000 to Beattie Mines, $1,700,000 to 2699681, and $3,400,000 to 2588111 by the end of the 2011 fiscal year, which will earn Clifton Star 10% of the issued and outstanding shares of the optionors;
- A cash payment of $2,000,000 and 250,000 shares of Clifton to the optionors on December 1, 2012; and
- A cash payment of $10,000,000 to the optionors on December 1, 2014;
- A cash payment of $10,000,000 to the optionors on December 1, 2015;
- A cash payment of $15,000,000 to the optionors on December 1, 2016;
- A cash payment of $15,200,000 to the optionors on December 1, 2017.

In the event of a change of control in Clifton Star or an assignment of the mineral property option agreement prior to the expiry of the aforementioned options, Clifton Star would be obligated to purchase all of the outstanding shares of the optionors for a total payment of $52,200,000 if the event occurred after June 1, 2010, but prior to December 1, 2012, or a total payment of $30,000,000 if the event occurs after December 1, 2012, but prior to December 1, 2017.

As of September 30, 2013, Clifton Star had paid $4,800,000 to Beattie, $2,400,000 to 2699681 and $4,800,000 to 2588111 under the option agreements and now owns 10% of the shares of the optionors. In connection with the September 2012 agreement, Clifton Star has also issued in October 4, 2013, a further 100,000 shares of the Company to Beattie, 50,000 shares to 2699681 and 100,000 shares to 2588111.

These payments will permit Clifton Star to earn the remaining 90% of the issued and outstanding shares of the optionors. There shall be no increase in the share ownership of each of the optionor companies unless all payments for each of the companies, as specified above, are satisfied.

In the event of a change of control in Clifton Star or an assignment of the mineral property option agreements prior to the expiry of the aforementioned options, all the above conditions vest two years in advance.

The optionors have retained a 2% Net Smelter Royalty ("NSR") for the duration of the option period. However, upon the exercise of Clifton Star's option to earn 100%, this 2% NSR will be eliminated.

4.3.2 Central Duparquet Property

The Central Duparquet property is defined as a group of eighteen (18) claims totalling 327.4 ha, registered to Lizette Grenier. On December 15, 2008, Clifton Star signed an option agreement whereby it may acquire a 100% interest in the Central Duparquet property. To earn its 100% interest, Clifton Star paid $400,000 on January 13, 2009.

During the five-year (5-year) period following the date of execution of the agreement, Clifton Star may sell, transfer or otherwise dispose of all or any portion of its interest in the property. A term of this disposition will be a payment to the optionor of shares of any company acquiring an interest in the property at a deemed value or in cash of $1,900,000.

On February 26, 2010, Clifton Star entered into an agreement to acquire the 2% NSR from the Gesmalar, the original owner of the Central Duparquet property. As consideration for the acquisition of the NSR, Clifton Star paid $155,000 and issued 10,000 common shares, valued at $57,400, to Gesmalar.

4.3.3 Mineral Property Option and Joint Venture Agreement with Osisko

On December 10, 2009, Clifton Star entered into a four-year (4-year) mineral property option and joint venture agreement with Osisko regarding a joint venture on the Duparquet Project, comprised of the Central Duparquet, Duquesne, Beattie, Donchester and Dumico properties. However, on June 16, 2011, Osisko notified Clifton Star of its decision to terminate its participation in the Duparquet Project.

Under the terms of this mineral property option and joint venture agreement, Osisko could have earned a 50% interest in the joint venture by contributing, as operator, a total of $70,000,000 to the joint venture over the period of four (4) years. As of June 16, 2011, Osisko had incurred $15,000,000 on the project.

Although the agreement with Osisko has been terminated, Clifton Star still has the right, under the initial terms, to access a loan of $22,500,000 from Osisko to cover the payments to the property vendors that are due on or before December 1, 2012. If Clifton Star were to access the funds, Clifton Star would have the right to retire the loan, at a 5% interest rate, based on the issuance of shares of Clifton Star to Osisko at a set price of $3.12 per share for the principal and at market price for the interest.

At the time this report was being prepared, Clifton Star had requested from Osisko the loan as described above. InnovExplo is not aware of any other information regarding the ongoing discussions between both parties.

4.4 Environment

As far as exploration and mining activities are concerned, the Beattie mining concession (MC#292) is affected by regulations regarding the presence of an “Urban Perimeter” (Fig. 4.3). The restriction, as documented on GESTIM, is one of “Exploration Prohibited” (see Bill 70, 2013, chapter 32, section a124). According to the Bill 70, any mineral substance forming part of the domain of the State and found in an urban perimeter shown on maps kept at the registrar’s office, except mineral substances found in a territory subject to a mining right obtained before December

10, 2013, is withdrawn from prospecting, mining exploration and mining operations as of that date, until the territories provided for in section 304.1.1 of the Mining Act are determined (as of December 10 2013, the Act to amend the Mining Act: Bill 70). According to section 304.1.1, any mineral substance forming part of the domain of the State and found in a parcel of land on which a claim may be obtained and that is included in a mining-incompatible territory delimited in a land use and development plan in accordance with the Act respecting land use planning and development (chapter A-19.1) is withdrawn from prospecting, mining exploration and mining operations from the time the territory is shown on the maps kept at the office of the registrar. A mining-incompatible territory is a territory in which the viability of activities would be compromised by the impacts of mining.

The Beattie mining concession (MC#292) corresponds to a mining right obtained before December 10, 2013 and thus exploration is permitted on this concession.

As of December 10, 2013, claim holders must notify the municipality and the landowner concerned within 60 days after registering a claim of the fact that they have obtained the claim, and must inform the municipality and the landowner at least 30 days before performing work. According to Clifton Star, the company has already advised the municipality of their work program on the mining concession and claims, and the municipality has agreed in writing to the proposed exploration program.

The initial location of the desulphurized tailing and waste rock management is located on the MERN property (verified with Bryan Goulet of the Abitibi-Ouest Regional County Municipality, Feb. 26 2013). Presently, there are clearing rights. Clifton Star would have to purchase the surface rights from the MERN if they want all desulphurized tailings to be recovered. ***There is a risk that the MERN will not allow Clifton Star to recover the portion of the Mineral Resources in the tailings that are outside the property boundary; if this were to happen, the authors wish to point out that this very small portion of the total Mineral Resources will only have a minor impact and will not affect the potential viability of the Project.***

InnovExplo is not aware of any environmental liabilities, permits or municipality social issues with respect to the Duparquet Project. All exploration activities conducted on the Duparquet Project comply with relevant environmental permitting requirements. To InnovExplo's knowledge, Clifton Star has obtained the appropriate permits and authorizations to use the surface rights.

5. ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

The Duparquet Project is located just north of the Town of Duparquet, which is accessible by paved, two-lane, all-season provincial highways from the towns of Rouyn-Noranda (53 km to the south; Route 393 and Route 101) and La Sarre (about 33 km to the north; Route 393) (Fig. 5.1). Access to the Duparquet Property from the Town of Duparquet is by gravel road.

5.2 Climate

The climate is continental with cold dry winters and warm summers. Winter temperatures average -17°C with lows down to -40°C in January, whereas summer temperatures average 17°C with highs up to 35°C in July. Annual precipitation is around 900 mm. Snow falls from mid-November to mid-April.

Mining and drilling operations can be conducted year-round, whereas surface exploration work (mapping, channel sampling) can take place from mid-April to mid-November.

5.3 Local Resources

The Town of Rouyn-Noranda, 53 km by road to the south and easily accessible via provincial highways 393 and 101, is the nearest major city with a specialized workforce and facilities for obtaining mining/exploration repairs and supplies. The Town of Duparquet currently has a population of about 675 residents, most of which are family members and descendants of workers at the past-producing Beattie and Donchester mines. Skilled workers and housing are available in the municipality. Water is available from the town's supply or from surface water on the Property.

Electric power is available at the Beattie mine site. In 2013, Hydro-Québec completed an exploratory study to verify its capacity to supply electric power to the Duparquet Project (Williamson et al., 2013a). It is possible to feed the Project from the Renaud substation, but a new 120 KV line of 14.5 km should be made to connect the Project. Hydro-Québec would require 36 months to complete the power supply.

5.4 Infrastructure

Most of the mine buildings have been demolished and the shafts capped and allowed to naturally flood. The existing glory hole is also flooded. The only remaining buildings are on the Beattie mine site: the roaster building, smoke stack and water tower. Although they are essentially in working order, Québec laws do not allow the roaster to operate.

Parts of the roaster building are currently used as an office, core shack and pulp and reject storage by Clifton Star. The yard holds numerous core racks and now serves as a core, pulp and reject storage area. Access to these facilities is restricted by a locked gate.

5.5 Physiography

The Duparquet Project lies in moderately rolling terrain just south of a flat belt of glacial till deposits and farmland. Outcrop density generally varies from 20% to 50%, but up to 80% in specific areas. The area is covered with an immature to semi-mature forest of poplar with some birch, balsam, spruce and jack pine interspersed on well-drained higher ground. The properties are also characterized by a very dense and thick undergrowth of mainly tag alders, scrub maple and willow. The overburden consists of sandy soil or till, with occasional portions of gravel. Some boulders of granitic composition, up to 3 m in diameter, are scattered here and there.

The mean elevation of the Duparquet Project (preliminary pit position) is approximately 300 metres above sea level (masl).

As described by Fillion (2009), the mine tailings are contained within a well-defined naturally occurring topographic basin that prevents them from spreading, except on the side of Lake Duparquet. The tailings have spread over a distance of 300 m into the lake along a very gentle slope. The tailings overlie lacustrine clay. A cover of organic matter, sometimes containing fire residue, is often present between the tailings and the lacustrine clay. Many intermittent creeks appear and then drain out over the course of the year in the general area of the tailings. One main trail crosses the entire area up to the lake and can be used all year long. Vegetation grows slowly on the tailings surface and consists of several species of trees, including birch, poplar and spruce. A marsh is present in the northwest part of the Property. A permanent waterway isolates the west part of the tailings area. Several beaver dams allow access to the site as well as a number of cross-country ski trails, which enter from the west.

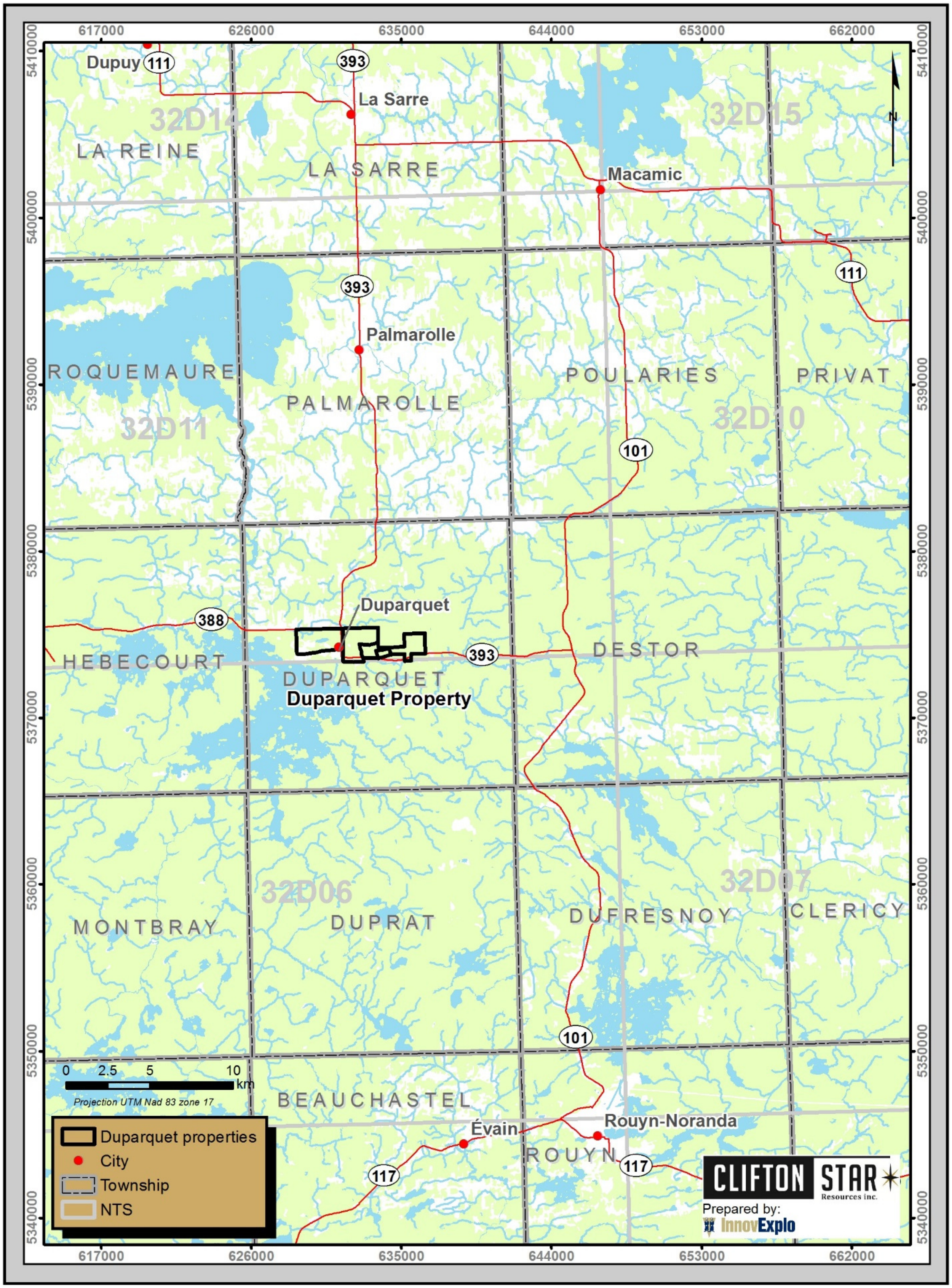

Figure 5.1 – Duparquet Property accessibility

6. HISTORY

The following chronological overview of historical work on the former properties comprising the Duparquet Property was taken mainly from Bevan (2011) and Dupéré et al (2011) and reviewed by InnovExplo. There is a summary of historical work carried out on the Duparquet Property (refer to Table 6.1).

1910 Gold is first discovered in Duparquet Township by John Beattie.

1923 The first claims for the Beattie property, included in Mining Concession 292, are staked by John Beattie, for whom the property is named.

1924 The Victoria Syndicate options the Beattie property and carries out extensive trenching, but results are not encouraging.

1925-1927 Prospecting of the Beattie property claims continues. **Consolidated Mining and Smelting Company of Canada Ltd** options the claims. The company continues exploration by trenching and carries out some diamond drilling.

1930 Consolidated Mining and Smelting drops the option. In the same year, John Beattie discovers the Main (or North) deposit and options the property to Ventures Limited and Nipissing Mining Company Ltd. These two (2) companies advance capital to develop the Beattie mine. The North deposit is diamond drilled and a two-compartment shaft is sunk to 67 m. During the sinking of the shaft, another deposit is encountered, called the "A" mineralized material zone.

1931-1945 Exploration and development is carried out on the Central Duparquet property by Dumico Gold Corporation, then Central Duparquet Mines Limited. Contemporaneous with the start of production at the Beattie mine, further west, Dumico drills 52 surface holes to define the mineralized zone. This work justifies the sinking of an exploration shaft and the driving of five (5) levels.

1932 The operator of the Beattie mine, Beattie Gold Mines Limited, is formed. A six-compartment (6) shaft is sunk to a depth of 442 m and nine (9) levels are established at 46-metre intervals, with the first level at 61 m below the shaft collar.

1933 A 2,000-ton-per-day flotation process plant is erected and production commences, with concentrates being shipped to Asarco's smelter in Tacoma, Washington.

1934-1937 A cyanidation plant is installed in 1935 and, due to the sulphide content in the mineralized material, a roaster is added in 1937 to improve recoveries. Initially, the production rate is 800 tons per day, gradually building up to 1.500 tons per day in 1935 to a maximum of 1,900 tons per day.

1937 A three-compartment (3) winze or internal shaft is sunk from the 5th (244 m) level some 274 m east of the main shaft down to the 9th (427m) level. This winze is later deepened to 625 m with the 13th (610m) level established.

1939 Beattie Gold Mines Limited is re-organized, becoming Beattie Gold Mines (Québec) Ltd.

1938-1940 Mine development is carried out in the Beattie mine toward the Donchester property. The Donchester property is immediately east of the Beattie property. A drift from the third level within the Beattie mine is driven across the boundary with the Donchester property for exploration purposes.

1933-1940 From the start of production in 1933 until the end of 1940, the mill treated 3,921,281 tons of ore and recovered 471,085 ounces of gold and 73,214 ounces of silver, an average of 0.120 oz/t Au and 0.019 oz/t Ag (Dresser and Denis, 1949). This tonnage came from the North and A zones.

1941 Diamond drilling from underground stations at 152-metre intervals outlines several ore-shoots 91 m below surface within the Donchester boundary. Nine (9) holes intersect values varying from 0.15 to 0.40 oz/t Au over widths ranging from 0.3 to 4 m. Beattie Gold Mines acquires the Donchester mine, for stock considerations. On the strength of these results, a drift on the 6th level (330 m below surface) is driven from the Beattie shaft over to the Dumico shaft of the Central Duparquet property, cutting across Donchester ground. Cross-drilling from this drift outlines a vein 792 m in strike length and grading 0.28 oz/t Au over a 2 metre width on average. Peak production at the Beattie mine is reached in 1941 and 1942 at 1,900 tons per day.

1943-1944 A shaft is sunk on Donchester ground to the 6th level and connected to the previously driven drift.

1943 A cave-in at the Beattie mine caused by failure of the main crown pillars results in an inrush of about one million cubic yards of clay, sand and broken rock into the mine workings. Rehabilitation work starts immediately and continues until 1950.

1943-1950 During this period, mining exploration in the original Beattie mine suffers and operations are conducted at a loss. Much of the production slack is taken up by tonnage from the Donchester section which is brought on-stream sooner because of the cave-in at Beattie. Production losses are accentuated during the war years by the shortage of labour and supplies, and after 1946, by rising costs and a fixed price for gold. Only with a government cost aid program (the *Emergency Gold Mining Act*, E.G.M.A.) are gold mines such as Beattie mine able to stay open in the post-war years.

1945 The Donchester shaft is deepened to the 9th level (457 m vertical). Development is carried out above and below the 6th level resulting in seven (7) new levels from the 2nd to the 9th on the “South Zone” deposit. At least four levels are driven across to the “North Zone” deposit: the 4th, 5th, 6th and 8th levels. The 3rd level is also driven across from Beattie to inter-connect with the North Zone.

1946-1956 The company again re-organizes and becomes Consolidated Beattie Mines Ltd. Operations continue on the Beattie property until 1956 when, after 23 years of almost continuous production, the mine closes. During its lifetime, the Beattie mill treated 9,645,000 metric tonnes with an average grade of 4.01 g/t Au and 0.99 g/t Ag from the North zone of the Beattie mine and Donchester mine (Lavergne, 1985).

1956-1987 Except for a small surface exploration program in 1966, the Beattie property remains dormant from 1956 to 1987. The 1966 program consists of line cutting, an electro-magnetic survey and two (2) diamond drill holes totalling 259 m, drilled to test an EM anomaly which is found to be caused by graphite. Unfortunately, some of the old mine records are destroyed by fire and affected by water damage. The latest complete records of reserves and production (stope outlines) are now those on a January 1, 1954, longitudinal section.

1981 The first period of activity on the Central Duparquet property takes place with SOQUEM Inc. SOQUEM performed a drilling program totalling thirteen (13) diamond drill holes.

1987 The second period of activity on the Central Duparquet property takes place, first with SOQUEM Inc followed by Cambior Inc, both companies mainly concentrating their efforts on the western part of the property. Mapping, lithogeochemical and geophysical surveys are carried out as well as eighteen (18) surface trenches and three (3) drilling programs. The aim of this work is to define the mineralized zone more accurately than Dumico Gold Corporation following their discovery of the zone. The first two (2) drilling programs carried out by SOQUEM in 1981 lead to the re-definition of property reserves (see below). The 3rd program is a drilling program conducted by Cambior in an attempt to define a near-surface mineralized zone. Various feasibility studies suggest that the deposit could be mined by open pit methods.

Reserve estimates of the Beattie mine, as carried out by C.W. Archibald Limited (1987) and Derry, Michener, Booth & Wahl (1987), appear to be based on a 1950 longitudinal section (see reference in Bevan, 2011). The following reserves are quoted in a 1987 Prospectus:

449,800 tons averaging 0.095 oz/t Au (A Zone west of the glory hole)
450,000 tons averaging 0.120 oz/t Au (broken muck in the glory hole)
636,000 tons averaging 0.122 oz/t Au (DMBW east of the glory hole)
264,075 tons averaging 0.120 oz/t Au (below 9th level)

These “resources” and/or “reserves” are historical in nature and should not be relied upon. It is unlikely they conform to current NI 43-101 criteria or to CIM Standards and Definitions, and they have not been verified to determine their relevance or reliability. They are included in this section for illustrative purposes only and should not be disclosed out of context.

1988 A diamond drilling program is carried out by Beattie Gold Mines, comprising twelve (12) holes for a total of 1,939.4 m. The holes target the “A”, South and North Zones.

1989-1990 Forbex Mining Resources Inc. re-vamps the reserves from the Central Duparquet property and undertakes an 11-hole drilling program in the eastern part of the property, off the known mineralized zones to the west.

1989 Beattie Gold Mines drills ten (10) holes for a total of 401.7 m. These holes are drilled under a stripped area of the South Zone and to the west of the stripping.

1990 A diamond drilling program is carried out by Beattie Gold Mines, comprising three (3) holes for a total of 713.8 m. The holes target the South Zone.

1991 A diamond drilling program is carried out by Beattie Gold Mines, comprising two (2) holes for a total of 200 m. The holes target the South Zone.

1992 A diamond drilling program is carried out by Beattie Gold Mines, comprising one (1) hole for a total of 185.6 m. This hole targets the South Zone.

1993 A diamond drilling program is carried out by Beattie Gold Mines, comprising four (4) holes for a total of 277.4 m. The holes target the South Zone.

1994 A diamond drilling program is carried out by Beattie Gold Mines, comprising three (3) holes for a total of 298.7 m. The holes target the South Zone.

1994-1995 Fieldex Inc drills six (6) holes in the eastern portion of the Central Duparquet property.

1995 A diamond drilling program is carried out by Beattie Gold Mines, comprising three (3) holes for a total of 284 m. The holes target the South Zone extension to the east and at depth below earlier holes.

1996 A diamond drilling program is carried out by Beattie Gold Mines, comprising seven (7) holes for a total of 625.8 m. The holes target the North Zone–East Extension with three (3) holes on the Beattie property and four (4) holes on the Donchester property. Two (2) holes (total of 62 m) are drilled into the South Zone, targeting the D-Vein on the Beattie property to the east of previous drill holes.

1997 A diamond drilling program is carried out by Beattie Gold Mines, comprising three (3) holes for a total of 477 m. The holes target the North Zone–East Extension with one (1) hole on the Beattie property and two (2) holes on the Donchester property.

1998 A diamond drilling program is carried out by Beattie Gold Mines, comprising three (3) holes for a total of 537 m. The holes target the North Zone–East Extension with one (1) hole on the Beattie property and two (2) holes on the Donchester property.

1999 A diamond drilling program is carried out by Beattie Gold Mines, comprising two (2) holes for a total of 294 m. The holes target the North Zone–East Extension.

2000 A diamond drilling program is carried out by Beattie Gold Mines, comprising one (1) hole for a total of 304 m. The hole targets the North Zone–East Extension.

2001 A diamond drilling program is carried out by Beattie Gold Mines, this was an extension of the hole drilled in 2000 for a total of 203 m.

2002 A diamond drilling program is carried out by Beattie Gold Mines, comprising three (3) holes for a total of 325 m. The holes target the South Zone and intersect the A to D veins.

2003 A diamond drilling program is carried out by Beattie Gold Mines, comprising three (3) holes for a total of 516 m. The holes target the South Zone and intersect the C to E veins.

2004 The owner of the Central Duparquet property, 9085-3615 Québec Inc, embarks on a pilot project with the objective of mining the property. No record has been found documenting this work. A diamond drilling program is carried out by Beattie Gold Mines representing the extension of two (2) holes drilled in 2002 and 2003 for a total of 246 m.

2005 A diamond drilling program is carried out by Beattie Gold Mines, comprising one (1) hole for a total of 313 m. The hole targets the east extension of the South Zone.

2005-2007 Drilling continues on the Central Duparquet property, comprising seven (7) drill holes for a total of 891 m. Golder Associates performs a reserve estimate in 2005 for the Central Duparquet property and reports an open pit reserve of 77,000 tonnes at a grade of 3.37 g/t Au diluted.

These “resources” and/or “reserves” are historical in nature and should not be relied upon. It is unlikely they conform to current NI 43-101 criteria or to CIM Standards and Definitions, and they have not been verified to determine their relevance or reliability. They are included in this section for illustrative purposes only and should not be disclosed out of context.

2006 A diamond drilling program is carried out by Beattie Gold Mines, comprising one (1) hole for a total of 294 m into the South Zone.

2007 A diamond drilling program is carried out by Beattie Gold Mines, comprising five (5) holes for a total of 284 m into the South Zone.

2008-2009 A diamond drilling program is carried out by Clifton Star, comprising 209 holes (58,053 m) on the Beattie property and 99 holes (37,566 m) on the Donchester property.

2009 Genivar is contracted to produce a Mineral Resource Estimate for a portion of the mine tailings area (Fillion, 2009), but the report was never published.

2010 Under the terms of a joint venture agreement with Osisko, the latter becomes the operator of a drilling program comprising 314 drill holes for a total of 102,529 m on the Beattie and Donchester properties. Osisko also carries out a channel sampling program consisting of 220 channels (460 m of cut channels) to complement the drilling program.

2010 Osisko Mining Corporation contracted SGS Mineral Service (Lakefield, Ontario) to conduct testwork on Beattie Duparquet Project samples. The program included comminution testwork and preliminary cyanidation and flotation tests to investigate the recovery of gold.

2011 During December and January 2011, Geophysics GPR International Inc. flew a helicopter-borne magnetic and time-domain electromagnetic geophysical survey for

Osisko Mining Corporation's Duparquet Project. The survey was composed of one (1) single block and covers all Duparquet Project.

2011 Osisko Mining Corporation contracted SGS Canada Inc. (Geostat) to prepare a NI 43-101 compliant Mineral Resource Estimate on the Beattie sector only.

2011 Clifton Star continued drilling the Beattie and Donchester properties and starts drilling the Central Duparquet property. A total of 46 holes and 28 hole extensions are drilled for a total of 17,565 m.

2012 Clifton Star completed surface outcrop stripping on and in the vicinity of the RWRS Zone, South Zone and the North Zone. A total of nineteen (19) outcrops on Beattie, nine (9) on Donchester and one (1) on Central Duparquet properties were mechanically stripped and then sampled using a conventional channelling technique. The primary goal for the stripping and channel sampling was to test and verify the gold mineralization continuity up to surface.

2012 Clifton Star decided to resample fifty (50) of the company's previous holes that had been selectively sampled at the time of drilling. The selective nature of the sampling resulted in untested shoulders adjacent to mineralized zones.

2012 From January to the end of August 2012, Clifton Star drilled a total of thirty-five (35) new holes and eight (8) extensions of older holes. Overall, the drilling program produced 12,471 m of NQ-size core during this period. From September 2012 until January 2013, Clifton Star continued drilling the Duparquet properties. A total of fifty-three (53) drill holes and ten (10) drill hole extensions were completed during this period, for a total of 22,675 m of NQ-size core.

2012 Clifton Star contracted InnovExplo to: (i) prepare a NI 43-101 compliant Mineral Resource Estimate on the Duparquet Project, combining the resources of all three (3) adjacent properties (Beattie, Donchester, Central Duparquet); and (ii) audit Genivar's Mineral Resource Estimate of the tailing ponds in order to incorporate the results into the "global" mineral resource estimate.

2012 Clifton Star contracted InnovExplo to: (i) prepare a NI 43-101 compliant Mineral Resource Estimate (new update) on the Duparquet Project, combining the resources of all three adjacent properties (Beattie, Donchester, Central Duparquet); and (ii) prepare a NI 43-101 compliant Preliminary Economic Assessment (PEA) for the Duparquet Project. The PEA Study was prepared as an open pit mining project relating solely to the mineral resources located on the Duparquet Project.

2012 Clifton Star contracted Tenova Mining & Minerals – Bateman Engineering Pty Ltd ("Tenova-Bateman") to develop preliminary capital and operating costs for the construction and operation of a mineral processing plant to process from the Duparquet Project to produce gold doré. The purpose of the Study was to assess the viability of the Total Pressure Oxidation, Albion Process™ and Biox® Leaching technologies to treat gold rich concentrate from the Duparquet mine deposits, in order to produce gold doré bar and to provide capital and operating cost estimates to a level of accuracy of ±35% for the proposed three flowsheets.

2012 Clifton Star contracted SGS Canada Inc. (Lakefield, Ontario) to conduct testwork on Duparquet Project samples. The program included flotation, pressure oxidation and cyanidation testwork to investigate the recovery of gold from ore and tailing samples. Preliminary comminution and environmental tests were also conducted.

2012 In fall, fifteen (15) of the nineteen (19) holes of the Dumico property were entirely resampled and/or downhole surveyed by Clifton Star. Down-hole orientation surveys were done using a Gyroscope instrument.

2013 Clifton Star sent to SGS Minerals a 12 tonne composite bulk sample of the Duparquet Project mineralized zones, from large diameter drill core, for metallurgical and environmental pilot tests. The planned testwork included a continuous pilot plant test for POX and also for high grade gold concentrates production. Results received in September 2013 confirmed previous recoveries and will be incorporated into the current Prefeasibility Study.

2013 Clifton Star began a surface outcrop stripping program on and in the vicinity of the RWRS Zone, South Zone and the North Zone during spring. A total of five (5) outcrops on Beattie property were mechanically stripped and then sampled using a conventional channelling technique. The primary goal for the stripping and channel sampling was to test and verify the gold mineralization continuity up to surface. The 2013 channel sampling program was not included in the most recent mineral resource estimate.

2013 Clifton decided to resample eight (8) of the company's previous holes that had been selectively sampled at the time of drilling. The selective nature of the sampling resulted in untested shoulders adjacent to mineralized zones. Clifton Star's re-sampling program successfully filled in the gaps in these holes. Of the eight (8) resampled holes, 397 samples have been added to the last updated Mineral Resource Estimate.

2013 Clifton Star contracted InnovExplo to prepare a NI 43-101 compliant Mineral Resource Estimate on the Duparquet Project, combining the resources of all four (4) adjacent properties (Beattie, Donchester, Central Duparquet, Dumico) including Genivar's mineral resource estimate of the tailing ponds in order to incorporate the results into a "global" mineral resource estimate.

Table 6.1 – Summary of historical work carried out on the Duparquet property

Year	Company	Work	Results
1923		• Staking of Beattie property	
1924	Victoria Syndicate	• Trenching	• No significant results
1925-1927	Consolidated Mining and Smelting Company of Canada Ltd	• Optioning of claims • Exploration and prospecting of claims • Trenching • Drilling	• No significant results
1930			• Dropped option
1930	John Beattie Ventures Limited and Nipissing Mining Company Ltd	• Exploration on Beattie property • Drilling • Sinking of shaft to 67 m	• Discovery of "North" mineralized zone • Discovery of "A" mineralized zone
1931-1945	Dumico Gold Corporation	• Central Duparquet: exploration and development • Drilling	• Sinking of exploration shaft • Driving of 5 levels
1932	Beattie Gold Mines Ltd	• Start of production at Beattie mine • 6-compartment shaft sunk to 442m • 9 levels established at 46-m intervals	
1933		• Flotation process set up at Beattie mine	• Production commenced • Concentrate shipped to Tacoma Washington smelter
1934-1937		• Cyanidation plant installed (1935) • Roaster added (1937)	• Initial production at 800 short tons per day • Maximum production 1,900 short tons per day
1937		• Compartment winze sunk from 5th (244m) level down to 13th (610m) level	
1938-1940	Beattie Gold Mines (Québec) Ltd	• Mine development at Beattie mine towards Donchester property	
1941		• Drilling • Donchester mine purchased • Drift on 6th level from Beattie shaft to Dumico shaft of Central Duparquet property	• Intersections in 9 holes ranging from 0.15 to 0.40 oz/t Au • Vein identified with 792-m strike length @ 0.28 oz/t Au over average width of 2 m
1943-1944		• Shaft sunk on Donchester property to 6th level	
1943		• Cave-in at Beattie mine, failed main crown pillars	• Rehabilitation work • Production drops
1943-1950		• Production commenced on Donchester section • Beattie mine stays open during post-war years	• Production loss
1945		• Donchester shaft deepened to 9th (457 m) level • Development of underground levels	• At least 4 levels driven across to the North Zone deposit

Table 6.1 – Summary of historical work carried out on the Duparquet property

Year	Company	Work	Results
1946-1956	Consolidated Beattie Mines Ltd	• Company re-organization; name change (1946) • Production ceased (1956)	• A total of 10,614,421 tons with average grade of 0.12 oz/t Au for Beattie mine • A total of 1,350,000 tons grading 0.14 oz/t Au for Donchester section
1956-1987		• Exploration including line cutting, EM survey and drilling (259 m)	• Property dormant
1981	SOQUEM Inc.	• Diamond drilling	• 13 drill holes on the Central Duparquet property
1987	SOQUEM Inc Cambior Inc	• Central Duparquet • Mapping • Lithogeochemical survey • Geophysical survey	• Re-definition of property reserves • Various feasibility studies – open pit methods suggested
1987		• Reserve estimates • These "resources" and/or "reserves" are historical in nature and should not be relied upon. It is unlikely they conform to current NI 43-101 criteria or to CIM Standards and Definitions, and they have not been verified to determine their relevance or reliability. They are included in this section for illustrative purposes only and should not be disclosed out of context.	• 449,800 tons @ 0.95 oz/t Au (A Zone) • 450,000 tons @ 0.12 oz/t (broken muck in GH) • 636,000 tons @ 0.12 oz/t Au (east of GH) • 264,075 tons @ 0.12 oz/t Au (below 9th level)
1988	Beattie Gold Mines Ltd	• Drilling	• 12 drill holes for total of 1,939 m
1989-1990	Forbex Mining Resources Inc	• Re-vamping of reserves for Central Duparquet property • Drilling	• 11 holes west of known mineralized zones
1989-1994	Beattie Gold Mines Ltd	• Drilling	• 23 holes for total of 2,077 m • Drilling on South Zone
1994-1995	Fieldex Inc.	• Drilling	• 6 holes on eastern portion of Central Duparquet property
1995	Beattie Gold Mines Ltd	• Drilling	• 3 holes for total of 284 m • South Zone and at depth
1996		• Drilling	• 7 holes for total of 626 m • Targets: North Zone–East Extension and South Zone D-vein
1997-2001		• Drilling	• 9 holes with one extension (total of 1,815 m) • Targets: North Zone on Beattie and Donchester properties
2002-2003	Beattie Gold Mines Ltd	• Drilling	• 6 holes for total of 839 m. Targets: South Zone veins A, C, D and E.
2004	9085-3615 Québec Inc. Beattie Gold Mines Ltd	• Pilot project for mining the Central Duparquet property • Drilling (Beattie Gold Mines)	• No significant results • Extension of 2 holes drilled in 2002 and 2003 by Beattie Gold Mines; total of 246 m
2005	Golder Associates Beattie Gold Mines Ltd	Resource estimation for Central Duparquet property These "resources" and/or "reserves" are historical in nature and should not be	• 77,000 t @ 3.37 g/t Au • 1 hole on South Zone for total of 313 m

Table 6.1 – Summary of historical work carried out on the Duparquet property

Year	Company	Work	Results
		relied upon. It is unlikely they conform to current NI 43-101 criteria or to CIM Standards and Definitions, and they have not been verified to determine their relevance or reliability. They are included in this section for illustrative purposes only and should not be disclosed out of context. • Drilling (Beattie Gold Mines)	
2005-2007	Beattie Gold Mines Ltd and 2588111 Manitoba Ltd.	• Donchester property drilling	• No significant results
2006-2007	Beattie Gold Mines Ltd	• Drilling	• 6 holes into South Zone for total of 578 m
2008-2009	Clifton Star	• Drilling	• 209 holes on Beattie property for total of 58,053 m • 99 holes on Donchester property for total of 37,566 m • 19 holes on Dumico property for a total of 4,818 m
2009	Genivar Clifton Star	• Unpublished Mineral Resource Estimate of the tailing ponds	
2010	Osisko and Clifton Star	• Drilling and channel sampling	• 314 holes on Beattie and Donchester for total of 102,529 m • 220 channels for total of 460 m
2010	SGS Mineral Services (Lakefield, Ontario) Osisko and Clifton Star	• Comminution testwork and preliminary cyanidation and flotation tests	• The abrasion index ranged from moderate to high • The maximum recovery of gold was 41.6%
2011	Geophysics GPR International Inc Osisko and Clifton Star	• Helicopter-borne magnetic and time-domain electromagnetic geophysical survey	• Entire Duparquet Project covered by the airborne geophysical survey
2011	SGS Canada Inc. (Geostat) Osisko and Clifton Star	• NI 43-101 compliant Mineral Resource Estimate on the Beattie sector only	• 1,715,888 oz Au (inferred; "in-pit")
2011	Clifton Star	• Drilling	• 46 holes on Beattie, Donchester and Central Duparquet property for a total of 17,565 m
2012	InnovExplo Clifton Star	• NI 43-101 compliant Mineral Resource Estimate for the global Duparquet Project, including addition of tailings resources from Genivar 2009	• 1,714,612 oz Au (measured and indicated; tailings + "in-pit" + underground) • 1,667,909 oz Au (inferred; "in-pit" and underground)
2012	Clifton Star	• Stripping • Channel sampling	• 719 channel samples
2012	Clifton Star	• Re-sampling company's holes	• 4,025 new samples
2012	Clifton Star	• Drilling	• 88 holes on Beattie, Donchester and Central Duparquet property for a total of 35,146 m
2012	InnovExplo	• NI 43-101 compliant Mineral	• 2,404,924 oz Au (measured

Table 6.1 – Summary of historical work carried out on the Duparquet property

Year	Company	Work	Results
	Clifton Star	Resource Estimate for the global Duparquet Project (new update), including addition of tailings resources from Genivar 2009. • NI 43-101 compliant Preliminary Economic Assessment (PEA)	and indicated; tailings + “in-pit” + underground) • 1,477,164 oz Au (inferred; “in-pit” and underground) • PEA study for an open pit mining
2012	Tenova Mining & Minerals – Bateman Engineering Pty Ltd Clifton Star	• Preliminary capital and operating costs for the construction and operation of a mineral processing plant to process from the Duparquet Project to produce gold doré	• The pre-production capital costs and sustaining costs for the Duparquet Project are estimated, respectively, at $370 million and $144 million, excluding $22.6 million for closure costs. The average operating cash cost is estimated at US$726 per ounce of gold.
2012	SGS Canada Inc. (Lakefield, Ontario) Clifton Star	• The program included flotation, pressure oxidation and cyanidation testwork to investigate the recovery of gold from ore and tailing sample.	• The Bond work index of the ore samples varied from 17.2 kWh to 20.2 kWh/t which would classify them as hard to very hard • Preliminary gravity separation tests were performed and the recovery of gold was in the range of 3.7 – 14.9% and averaged 8.6%. • The recovery of gold in a bulk sulphide concentrate by flotation was greater than 90% • The overall recovery of gold ranged from 83.5% to 93.3% for the tailings sample.
2013	SGS Canada Inc. (Lakefield, Ontario) Clifton Star	• 12 tonne sample of the Duparquet Project mineralized zones, from large diameter drill core, for metallurgical and environmental pilot tests. • The pilot plant was operated to generate bulk sulphide flotation concentrate analysing 15-18% S for a subsequent pressure oxidation pilot plant to recover gold.	• With one cleaning stage the recovery of gold was 91.7% in a concentrate which assayed 26.8 g/t Au and 16.1% S. • The overall recovery of gold was 95.4% with one cleaning stage and 91.9% with two cleaning stages.
2013	Clifton Star	• Stripping • Channel sampling • Re-sampling the company’s holes	• 397 new samples
2013	InnovExplo Clifton Star	• NI 43-101 compliant Mineral Resource Estimate for the global Duparquet Project (new update), including tailings resources from Genivar 2009. • The Dumico area drill holes were also added to this update.	• 3,113,171 oz Au (measured and indicated; tailings + “in-pit” + underground) • 1,442,689 oz Au (inferred; “in-pit” and underground)

7. GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geological Setting

7.1.1 Archean Superior Province

The Archean Superior Province (Fig. 7.1) forms the core of the North American continent and is surrounded by provinces of Paleoproterozoic age to the west, north and east, and the Grenville Province of Mesoproterozoic age to the southeast. Tectonic stability has prevailed since approximately 2.6 Ga in large parts of the Superior Province. Proterozoic and younger activity is limited to rifting of the margins, emplacement of numerous mafic dyke swarms (Buchan and Ernst, 2004), compressional reactivation, large-scale rotation at approximately 1.9 Ga, and failed rifting at approximately 1.1 Ga. With the exception of the northwest and northeast Superior margins that were pervasively deformed and metamorphosed at 1.9 to 1.8 Ga, the craton has escaped ductile deformation.

A first-order feature of the Superior Province is its linear subprovinces, or “terranes”, of distinctive lithological and structural character, accentuated by subparallel boundary faults (e.g., Card and Ciesielski, 1986). Trends are generally east-west in the south, west-northwest in the northwest, and northwest in the northeast. In Figure 7.1, the term “terrane” is used in the sense of a geological domain with a distinct geological history prior to its amalgamation into the Superior Province during the 2.72 Ga to 2.68 Ga assembly events, and a “superterrane” shows evidence for internal amalgamation of terranes prior to the Neoarchean assembly. “Domains” are defined as distinct regions within a terrane or superterrane.

The Duparquet Property is located within the Abitibi terrane. The Abitibi terrane hosts some of the richest mineral deposits of the Superior Province (Fig. 7.1), including the giant Kidd Creek massive sulphide deposit (Hannington et al., 1999) and the large gold camps of Ontario and Québec (Robert and Poulsen, 1997; Poulsen et al., 2000).

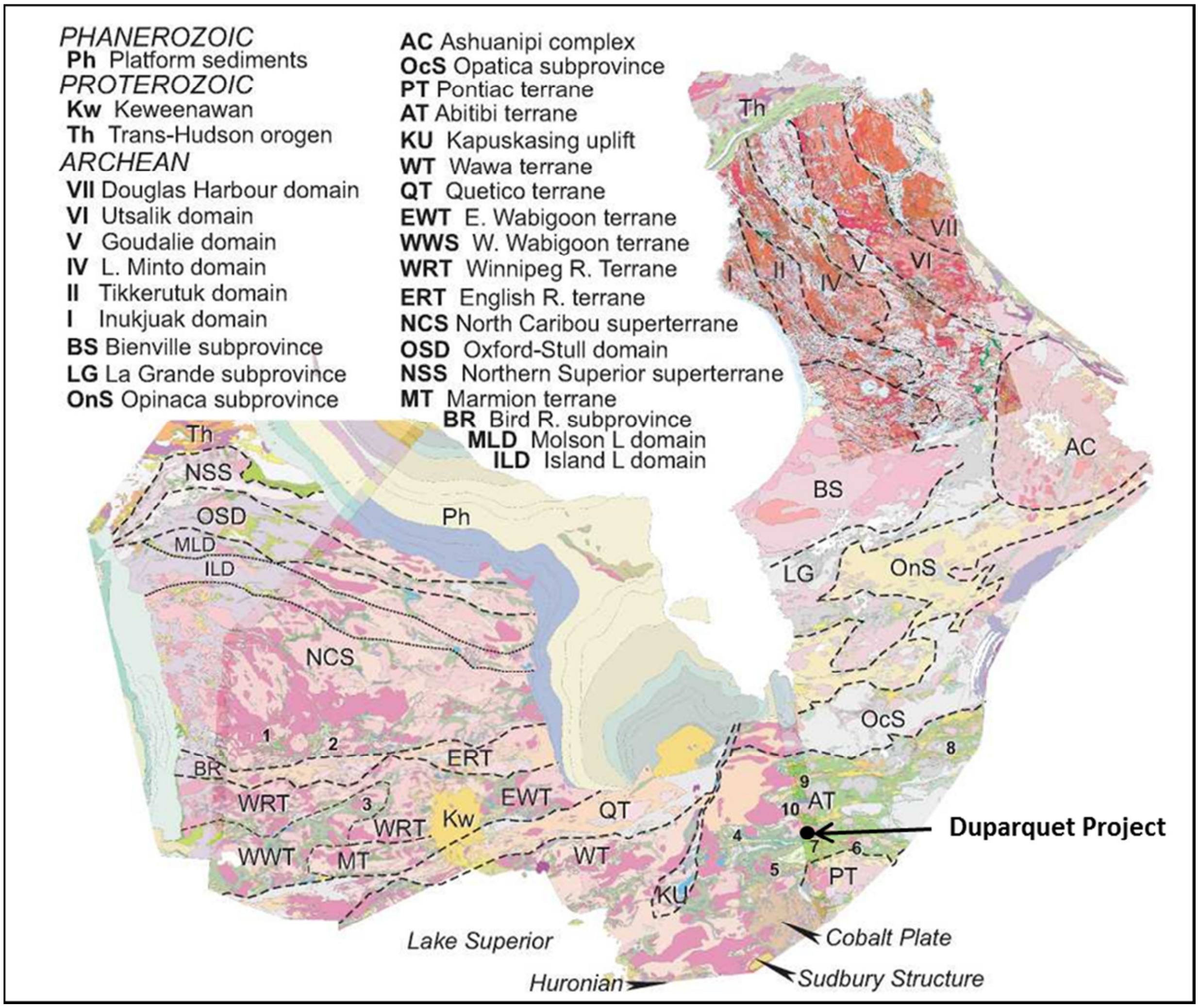

Figure 7.1 – Mosaic map of the Superior Province showing major tectonic elements, from Percival (2007). Data sources: Manitoba (1965), Ontario (1992), Thériault (2002), Leclair (2005). **Major mineral districts: 1 = Red Lake; 2 = Confederation Lake; 3 = Sturgeon Lake; 4 = Timmins; 5 = Kirkland Lake; 6 = Cadillac; 7 = Noranda; 8 = Chibougamau; 9 = Casa Berardi; 10 = Normétal.**

7.1.2 The Abitibi Terrane (Abitibi Subprovince)

The Abitibi Subprovince (Abitibi Greenstone Belt) is located in the southern portion of the Superior Province (Fig. 7.1). The Abitibi Subprovince is divided into the Southern and Northern volcanic zones (**SVZ** and **NVZ**; Chown et al. 1992) representing a collage of two (2) arcs delineated by the Destor-Porcupine-Manneville Fault Zone (**DPMFZ**; Mueller et al. 1996). The SVZ is separated from the Pontiac Terrane sedimentary rocks, an accretionary prism (Calvert and Ludden 1999) to the south, by the Cadillac–Larder Lake Fault Zone (**CLLFZ**). The fault zones are terrane “zippers” that display the change from thrusting to transcurrent motion as documented in the turbiditic flysch basins unconformably overlain by, or in structural contact with, coarse clastic deposits in strike-slip basins (Mueller et al. 1991, 1994, 1996; Daigneault et al. 2002). A further subdivision of the NVZ into external and internal segments is warranted, based on distinct structural patterns with the intra-arc Chicobi sedimentary sequence representing the line of demarcation. Dimroth et al.

(1982, 1983a) recognized this difference and used it to define internal and external zones (Fig. 7.2) of the Abitibi greenstone belt. Subsequently, numerous alternative Abitibi divisions were proposed (see Chown et al., 1992), but all models revolved around a plate tectonic theme. The identification of a remnant Archean north-dipping subduction zone by Calvert et al. (1995) corroborated these early studies.

The 2735-2705 Ma NVZ is ten (10) times larger than the 2715-2697 Ma SVZ, and both granitoid bodies and layered complexes are abundant in the former. In contrast, plume-generated komatiites, a distinct feature of the SVZ, are only a minor component of the NVZ, observed only in the Cartwright Hills and Lake Abitibi area (Daigneault et al. 2004). Komatiites rarely constitute more than 5% of greenstone sequences and the Abitibi is no exception (Sproule et al. 2002). The linear sedimentary basins are significant in the history because they link arcs and best chronicle the structural evolution and tempo of Archean accretionary processes. The NVZ is composed of volcanics cycles 1 and 2, which are synchronous with sedimentary cycles 1 and 2, whereas the SVZ exhibits volcanic cycles 2 and 3, with sedimentary cycles 3 and 4 (Mueller et al. 1989; Chown et al. 1992; Mueller and Donaldson 1992; Mueller et al. 1996).

The Abitibi Subprovince displays a prominent E-W structural trend resulting from regional E-trending folds with an axial-planar schistosity that is characteristic of the Abitibi belt (Daigneault et al. 2002). The schistosity displays local variations in strike and dip, which are attributed to either oblique faults cross-cutting the regional trend, or deformation aureoles around resistant plutonic suites. Although dominant steeply-dipping fabrics are prevalent in Abitibi Subprovince, shallow-dipping fabrics are recorded in the Pontiac Subprovince and at the SVZ-NVZ interface in the Preissac-Lacorne area.

The metamorphic grade in the Abitibi Subprovince displays greenschist to sub-greenschist facies (Joly, 1978; Powell et al., 1993; Dimroth et al., 1983b; Benn et al., 1994) except around plutons where amphibolite grade prevails (Joly, 1978). In contrast, two (2) extensive high-grade zones coincide with areas of shallow-dipping fabrics. They are as follows: (1) the turbiditic sandstone and mudstone of the Pontiac Subprovince at the SVZ contact, which exhibit a staurolite-garnet-hornblende-biotite assemblage (Joly, 1978; Benn et al., 1994); and (2) the Lac Caste Formation turbidites at the SVZ-NVZ interface (Malartic segment) with sandstone and mudstone metamorphosed to biotite schist with garnet and staurolite. Feng and Kerrich (1992) suggested that the juxtaposition of greenschist and amphibolite grade domains indicates uplift occurred during the compressional stage of collisional tectonics.

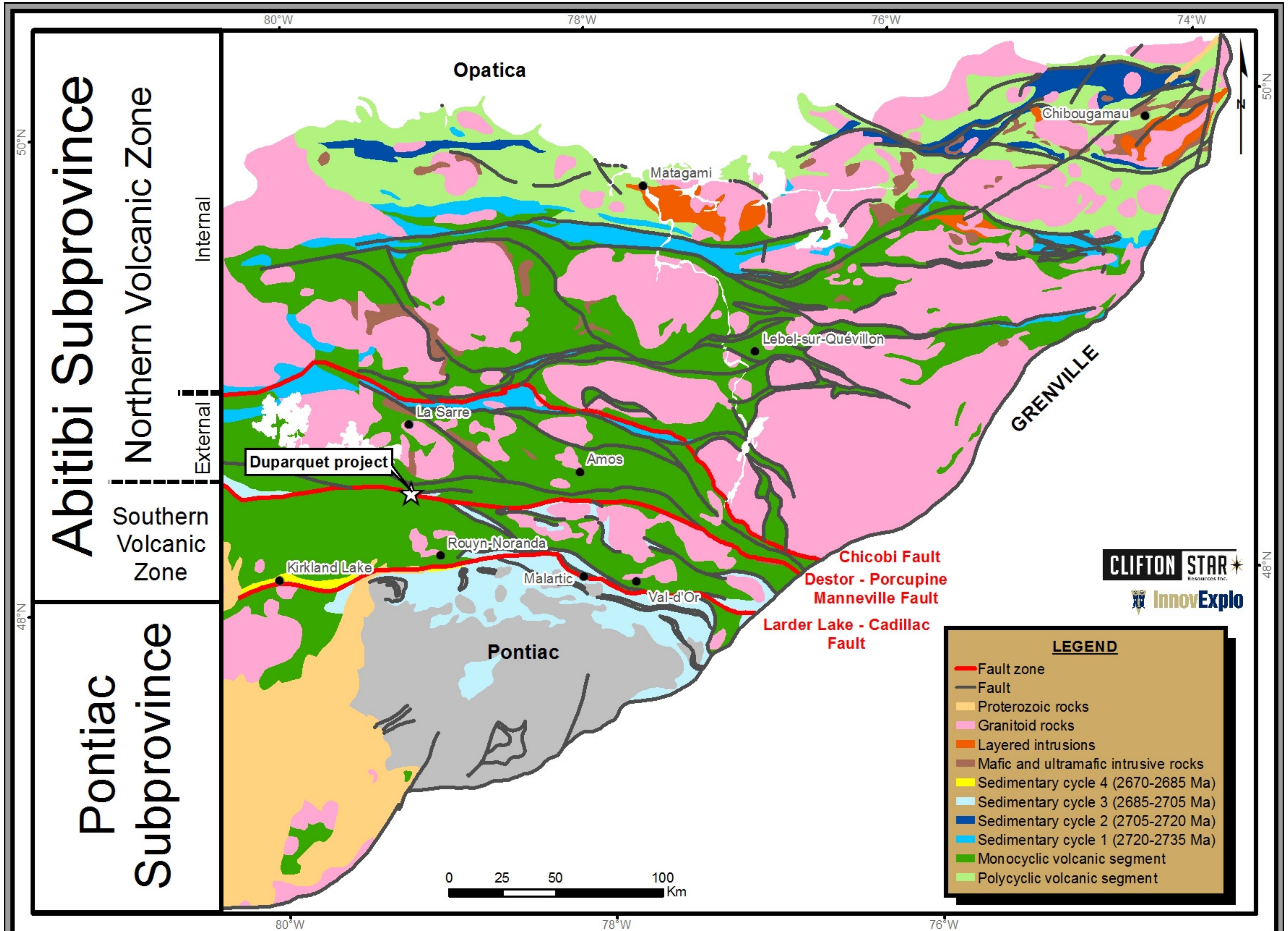

Figure 7.2 – Divisions of the Abitibi greenstone belt into southern (SVZ) and northern volcanic zones (NVZ) with external and internal segments in the NVZ. Modified from Chown et al. (1992), Daigneault et al. (2002, 2004), and Mueller et al. (2009).

7.2 Local Geological Setting

The local geological setting and property geology is represented by the Kinojevis, Timiskaming, and Blake River groups (Fig. 7.3). The Blake River Group is located to the south of the Destor-Porcupine fault. This group is characterized by a 2706-2696 Ma volcanic sequence 4 to 7 km thick belonging to the southern volcanic zone (SVZ). The Blake River Group is composed of mafic volcanic rocks with several felsic volcanic centres (Pearson and Daigneault, 2009), and is interpreted as a mega-caldera complex representing a multi-stage collapse structure occupying most of the present Blake River Group surface area (Pearson and Daigneault, 2009). Several arguments support a mega-caldera complex, including: (1) the mafic-intermediate swarm pattern; (2) the overall geometry; (3) the fault pattern; (4) the distribution of volcaniclastic rocks; and (5) the distribution pattern of carbonate-rich hydrothermal alteration (Pearson and Daigneault, 2009).

The Kinojevis Group is located north of the DPMFZ and is subdivided into two (2) units: the Deguisier Formation, composed of massive to pillowed tholeiitic basalt associated with some amounts of andesite, felsic pyroclastic rocks and gabbro, and the Lanaudière Formation, consisting of basalts, andesites, rhyolites, komatiites and multiple mafic to ultramafic intrusions (Goutier and Lacroix, 1992).

The Duparquet Formation, part of the Timiskaming group, is a sedimentary unit that occurs within a small structurally controlled basin, herein named the Duparquet Basin (Mueller et al., 1991). Development of the Duparquet basin at a late orogenic stage (of the Kenoran orogeny) classifies it as a successor basin (pull-apart basin). The wedge-shaped basin, delineated by bounding faults, may be compared to divergent fault-wedge basins in the Cenozoic basins of southern California (Crowell, 1974). Mueller et al. (1991) suggested a late Archean dextral strike-slip movement within a regime dominated by north-south compression.

Detailed stratigraphic mapping in the Duparquet Formation led to the recognition of three (3) mutually transitional facies associations (Mueller et al., 1991). Based on predominant lithology, grain size and sedimentary structures, the facies end members are as follows: (1) conglomerate-sandstone facies association (CSFA); (2) sandstone-argillite facies association (SAFA); and (3) argillite-sandstone facies association (ASFA).The CFSA, up to 100 m thick, predominates near faulted basin margins and can be divided into subfacies in which the porphyry clast component is either dominant or negligible. The SAFA is characterized by a set of coarse-grained to very coarse-grained trough cross-beds 5 to 100 cm thick; planar to wavy, medium- to coarse-grained beds 5 to 20 cm thick; and argillite beds 1 to 30 cm thick. The ASFA is distinguished by laterally continuous, well-laminated argillite interstratified with fine- to coarse-grained sandstone beds 2 to 10 cm thick.

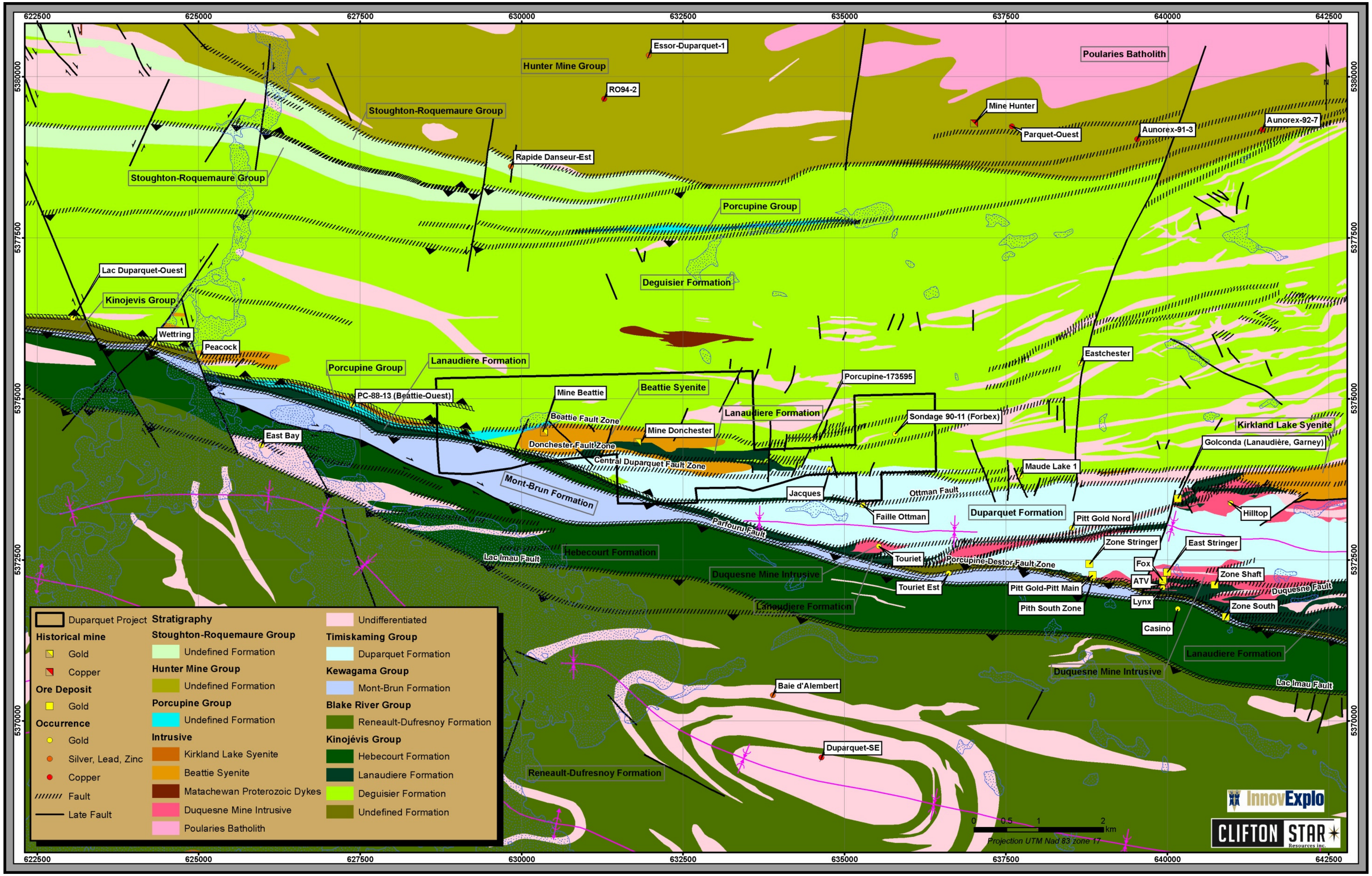

Figure 7.3 – Stratigraphy of the Duparquet Property

7.3 Property Geological Setting

This section is a slightly modified version of the property geology description provided in the technical report by Bevan (2011). The authors have reviewed and compared Bevan's geological description to other such accounts in publicly available documents and consider it accurate to the best of their knowledge.

The Duparquet Property straddles syenitic plutons and the Kinojevis, Duparquet and Mont-Brun formations (Fig. 7.4). The Duparquet Project area is characterized by the presence of two (2) syenitic plutons oriented east-west. These syenitic intrusions are bounded by E-W trending major faults, which are interpreted as splays of the main SE-trending DPMFZ, which clips the southwest corner of the Property. The geological formations strike generally E-W and dip steeply (80°-85°) to the north. The grade of metamorphism is low (greenschist facies) with local alteration represented by chloritization, silicification and sericitization. Most of the known mineralization appears to be related to the late intrusions of syenite and feldspar porphyry in the Keewatin mafic flows and tuffs, along zones of weakness adjacent to or coincident with the E-W trending major faults.

7.3.1 Stratigraphy

The rocks underlying the Duparquet Property are generally made up of intercalated felsic (rhyolitic to dacitic) and mafic (basaltic to andesitic) metavolcanic flows, with felsic flows being the oldest. Metasedimentary layers are also present, consisting of arkosic sandstones, greywackes, argillites, crystal tuffs and conglomerates, and are generally more prevalent on the south side of the DPMFZ. All the units have been intruded by syenite porphyry units, which appear to be concordant with the location of the major fault zones. Quartz feldspar and lath porphyries have been injected along minor faults affecting the syenite intrusions.

7.3.2 Structure

The predominant structures on the Property are the E-W-trending splays of the DPMFZ. The Beattie Fault Zone (BFZ) is located along the north contact of the main syenite body, whereas the Donchester Fault Zone (DFZ) is located along the south contact. The Central Duparquet Fault Zone (CDFZ) is located along the south contact of the second smaller syenite intrusive to the east. The syenite porphyry generally plunges to the east.

The Beattie and Donchester fault zones dip steeply to the south and north respectively, thus suggesting that the contacts of the syenite porphyry converge within the central portion of the complex at depth. The Central Duparquet fault has an orientation subparallel to the Donchester fault. Strike-slip offset along these major structures is not significant. Neither the downdip component of movement nor the slip vector could be determined.

Late crosscutting faults interpreted in plan view (Fig. 7.4) are likely to have some impact on displacing the gold-bearing zones, although InnovExplo was unable to verify their existence while modelling the mineralized material solids. It is assumed that displacement along such crosscutting faults is minimal and without significant consequences at the scale of InnovExplo's interpretation. Minimal degree of horizontal displacement can be seen from the drift mapping and drilling of the mine level plans.

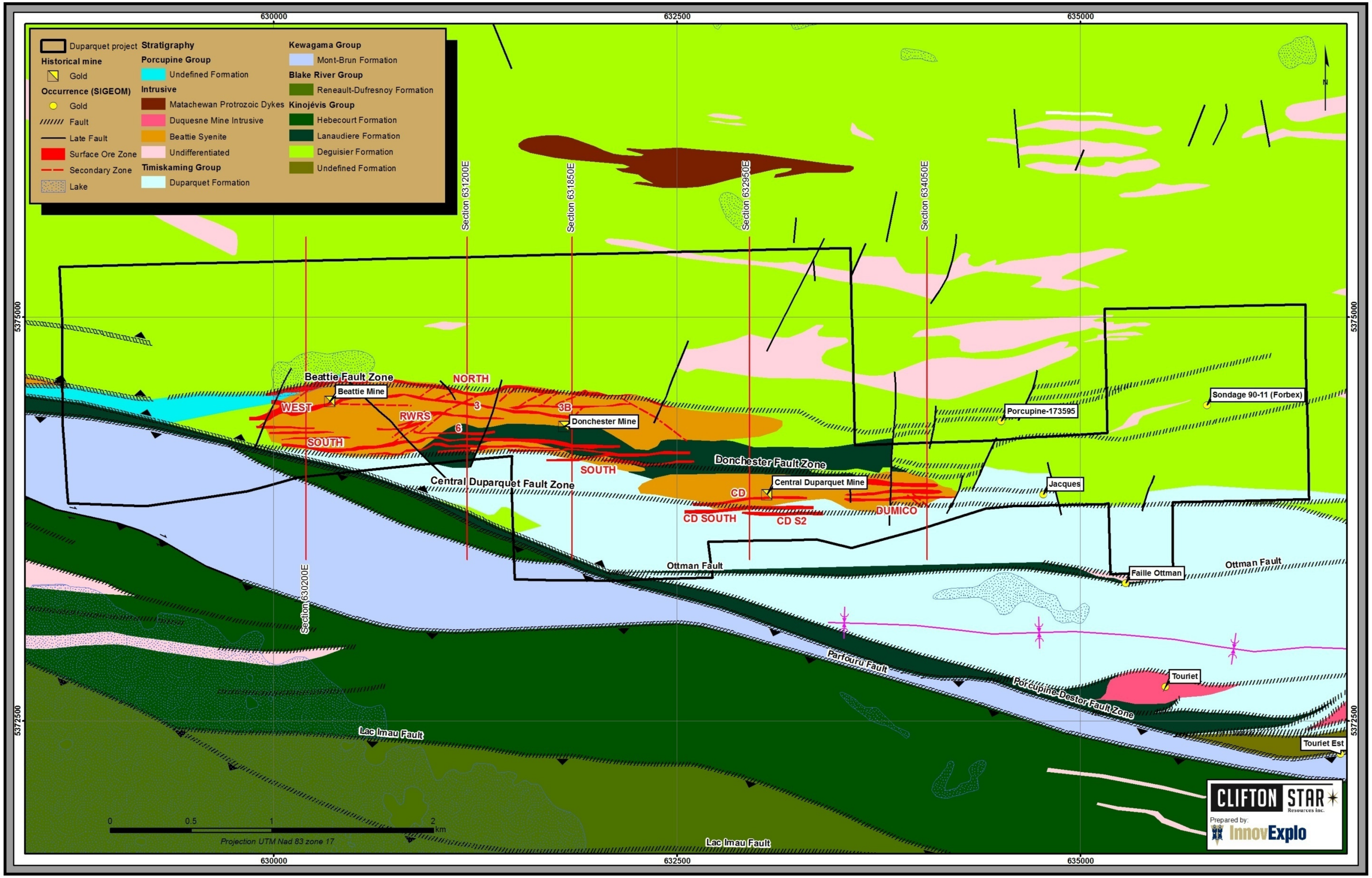

Figure 7.4 – Geology of the Duparquet Property. Structural and geometric details are represented by surface projections of the relevant structural elements from the deposit-scale 3D litho-structural model

7.3.3 Alteration

Gold-bearing quartz veins within the DPMFZ have historically been associated with sericite-carbonate-ankerite-chlorite alteration haloes and late-stage quartz-carbonate veins with ankerite haloes.

Gold-bearing mineralization within the Duparquet deposit (including the Beattie and Donchester mines, as well as the Central Duparquet deposit) is associated with carbonate, chlorite, fuchsite, and sericite alteration as a product of hydrothermal fluid injection within sheared and brecciated sections of the syenite porphyries. Silicification of fractures and chert-calcite-rich accumulations along fractures accompanied by pyrite-arsenopyrite host rock replacement have been observed within the mineralized zones. The chert is dark grey due to its potassic-rich composition and the hematite and tourmaline content of the hydrothermal fluids.

7.3.4 Mineralization

Gold mineralization at the Beattie mine was historically associated with silicified and brecciated zones containing a low percentage of very fine-grained pyrite and arsenopyrite (Goutier and Lacroix, 1992). According to Bevan (2011), the “main” type of gold mineralization in the Duparquet deposit generally occurs within shears or fracture zones along or within the adjacent intrusive syenitic masses, and is associated with finely disseminated pyrite and minor arsenopyrite replacement. Sulphide content is generally low (0.5 to 4%), although it can be up to 10% in cases. Higher gold grades appear to be related to the finer grained sulphides (Bevan, 2011). Historically, gold production at the Beattie mine was accompanied by extraction of arsenic trioxide and silver as by-products. The “breccia” type of mineralized material is found within the metavolcanic rocks (lavas and tuffs) and consists of well-mineralized, siliceous, brecciated, grey-coloured and bleached zones. The porphyry mineralized material type consists of fine-grained and strongly silicified mineralized zones hosted by the porphyry intrusives. The latter generally has lower gold grades than other types of mineralized zones within the deposit (Bevan, 2011).

The typical mineral assemblage found within mineralized material zones of all types is characterized by the presence of feldspar, quartz, sulphides (pyrite and arsenopyrite), sericite, chlorite and other secondary minerals. Mill tests suggest that some 35% of the gold is in a free state, with the remainder associated with sulphides. According to Bevan (2011), three (3) phases of gold enrichment or remobilization can be interpreted from cross-cutting relationships of gold-bearing veins. Bevan (2011) also states that higher gold contents are found along cross-cutting faults, along the nose of folds and within the lath porphyry dyke intrusion as a consequence of remobilization processes.

At the Beattie mine, the main mineralized lens is hosted by a shear zone (BFZ) at the north contact of the syenite intrusion (Fig. 7.4). In this report, the main zone is referred to as the North Zone. A second gold-bearing lens, also hosted by a shear zone (DFZ) but occurring at the south contact of the syenite body, is also known at the Beattie mine and is referred to as the South Zone herein. Gold mineralization at the Donchester mine was higher grade and associated with an E-W shear zone cutting across some volcanic rocks and syenitic dykes (Goutier and Lacroix, 1992). This zone has been interpreted herein as the east extension of the South Zone. At both the Beattie and Donchester mines, the South Zone can be subdivided into several mineralized “lenses”, modelled herein as ten (10) individual sub-zones. Six (6) other major striking mineralized zones occurring within the Beattie-Donchester area have been interpreted by InnovExplo for the purpose of the current Mineral Resource Estimate.

Mineralization at the Central Duparquet is hosted by the CDFZ, and is of a similar nature as the South and North zones (Bevan, 2011). At Central Duparquet, three (3) mineralized zones have been interpreted by InnovExplo.

The Dumico property corresponds to the eastern extension of the Central Duparquet property. At Dumico, five (5) mineralized zones have been interpreted by InnovExplo. Three (3) of these mineralized zones are striking E-W and are interpreted as the “natural” extension of the CD Zones found at Central Duparquet. The two (2) other zones are striking NW-SE occur on the eastern portion of the Dumico property. Based on the current interpretation, the two (2) latter zones are considered to be associated to a subsidiary structure sub-parallel to the regional DPMFZ.

A total of thirty-four (34) secondary mineralized zones have been interpreted within the previously defined “inter-zone” mineralized envelope. The interpretation of these secondary mineralized zones, which are in majority striking SW-NE, is based on field observations as well as grade continuity throughout the samples point set. These SW-NE striking mineralized zones are interpreted to be hosted by subsidiary structures associated with the BFZ and DFZ.

The geometry, size and structural context of these zones are presented on Figures 7.4 to 7.9. The interpreted zones show continuous mineralization occurring within a 4.5 km strike-length corridor measuring 1 km wide and extending to about 1 km below surface. This mineralized corridor contains a relatively complex system of E-W and SW-NE striking structures hosting mineralized lenses, mainly confined between the BFZ and DFZ. Details on InnovExplo’s interpretation of the mineralized zones at the Duparquet Project can be found in Section 14.1.1.2 – Interpretation of Mineralized Zones.

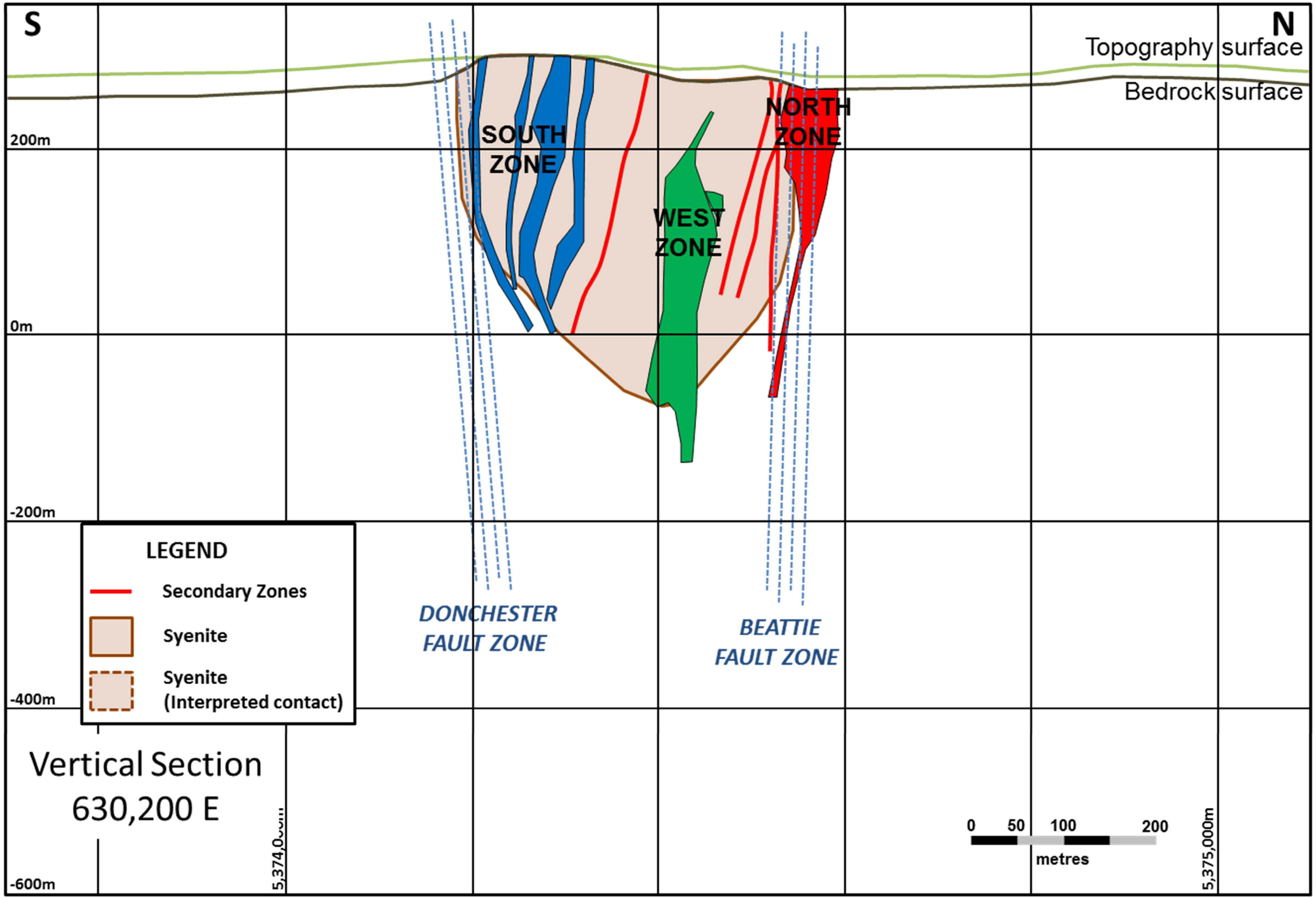

Figure 7.5 – Vertical section 630,200E, looking west, showing the structural and geometrical details of the relevant structural elements from the deposit-scale 3D litho-structural model (location on Figure 7.4)

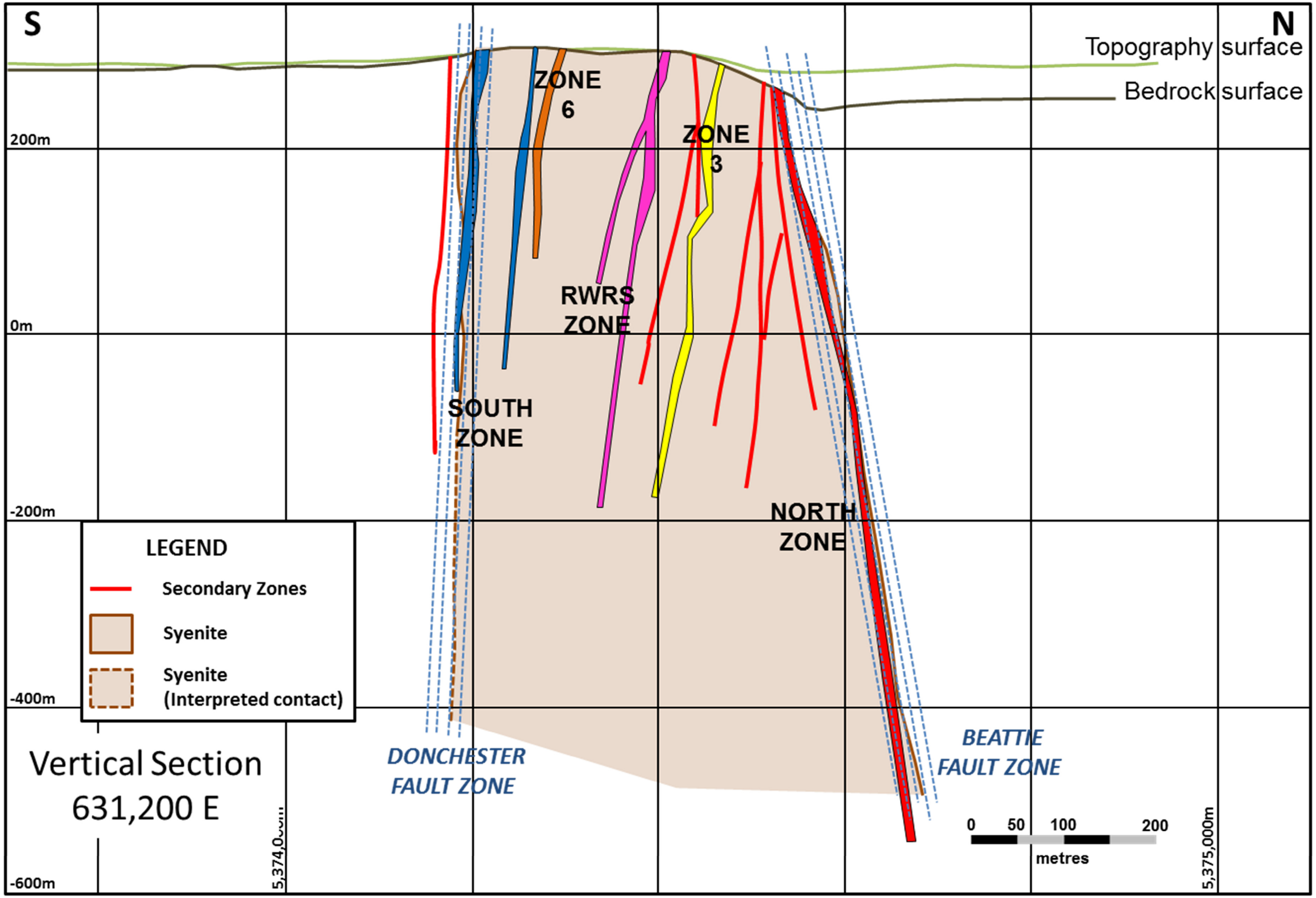

Figure 7.6 – Vertical section 631,200E, looking west, showing the structural and geometric details of the relevant structural elements from the deposit-scale 3D litho-structural model (location on Figure 7.4)

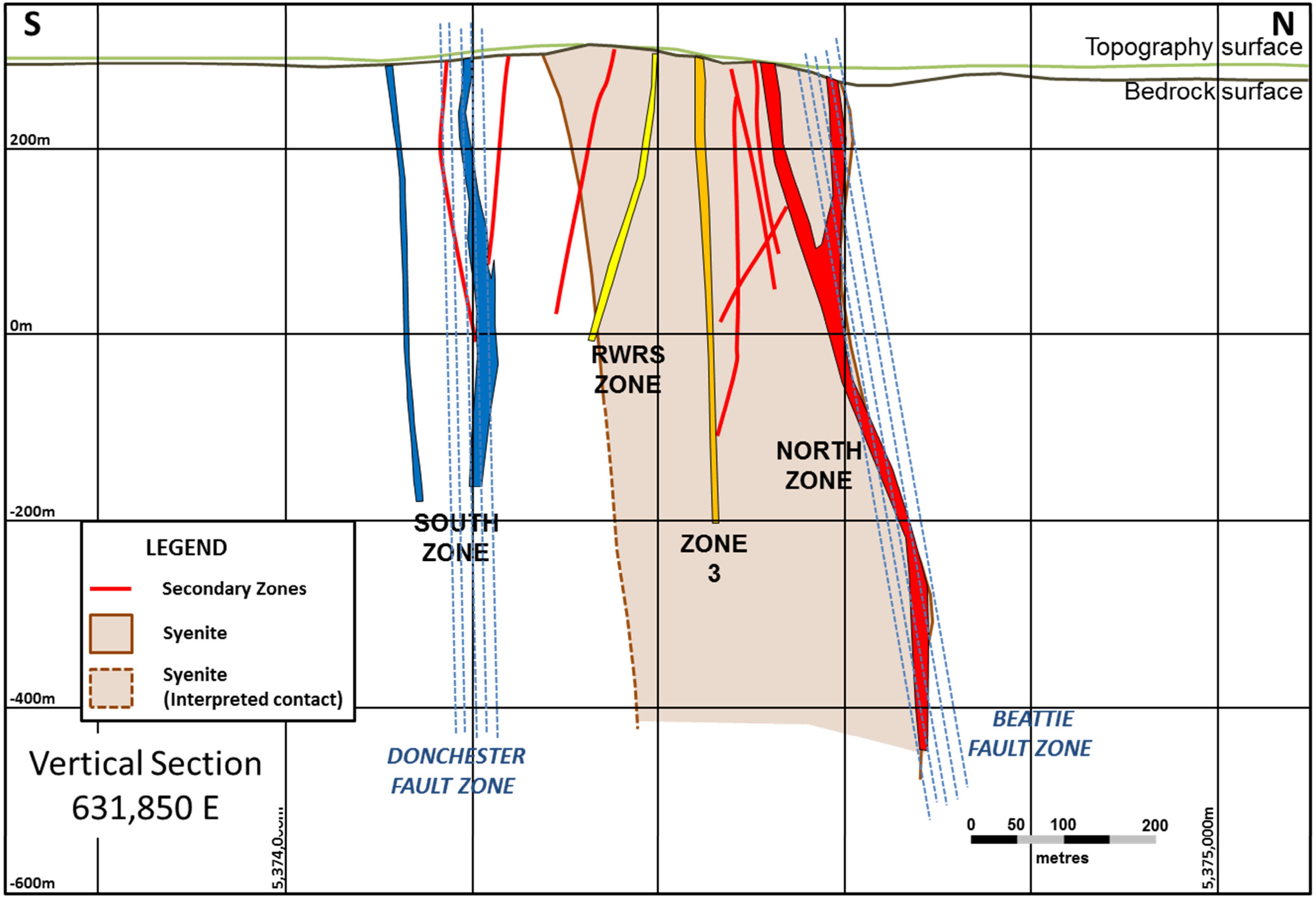

Figure 7.7 – Vertical section 631,850E, looking west, showing the structural and geometric details of the relevant structural elements from the deposit-scale 3D litho-structural model (location on Figure 7.4)

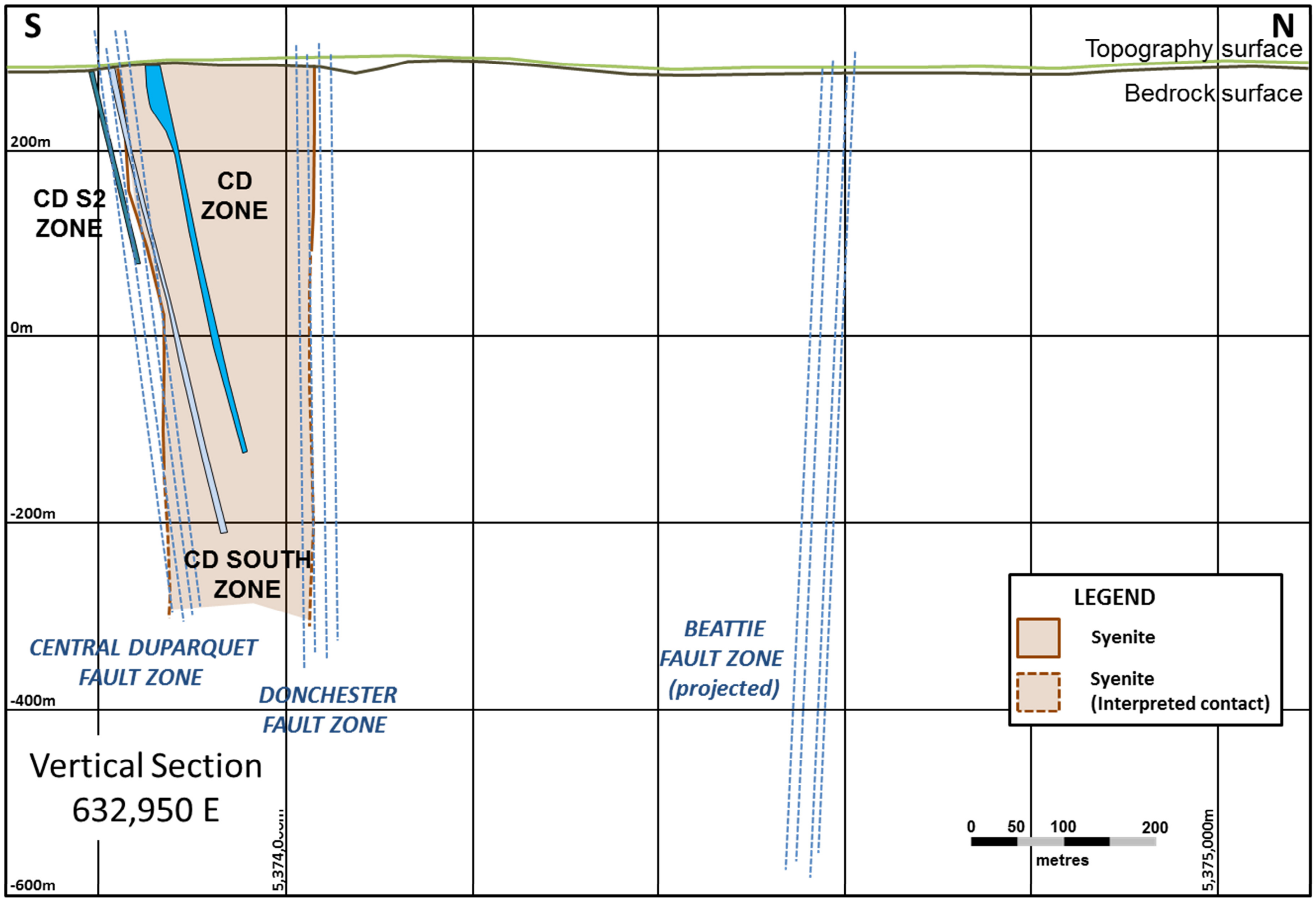

Figure 7.8 – Vertical section 632,950E, looking west, showing the structural and geometric details of the relevant structural elements from the deposit-scale 3D litho-structural model (location on Figure 7.4)

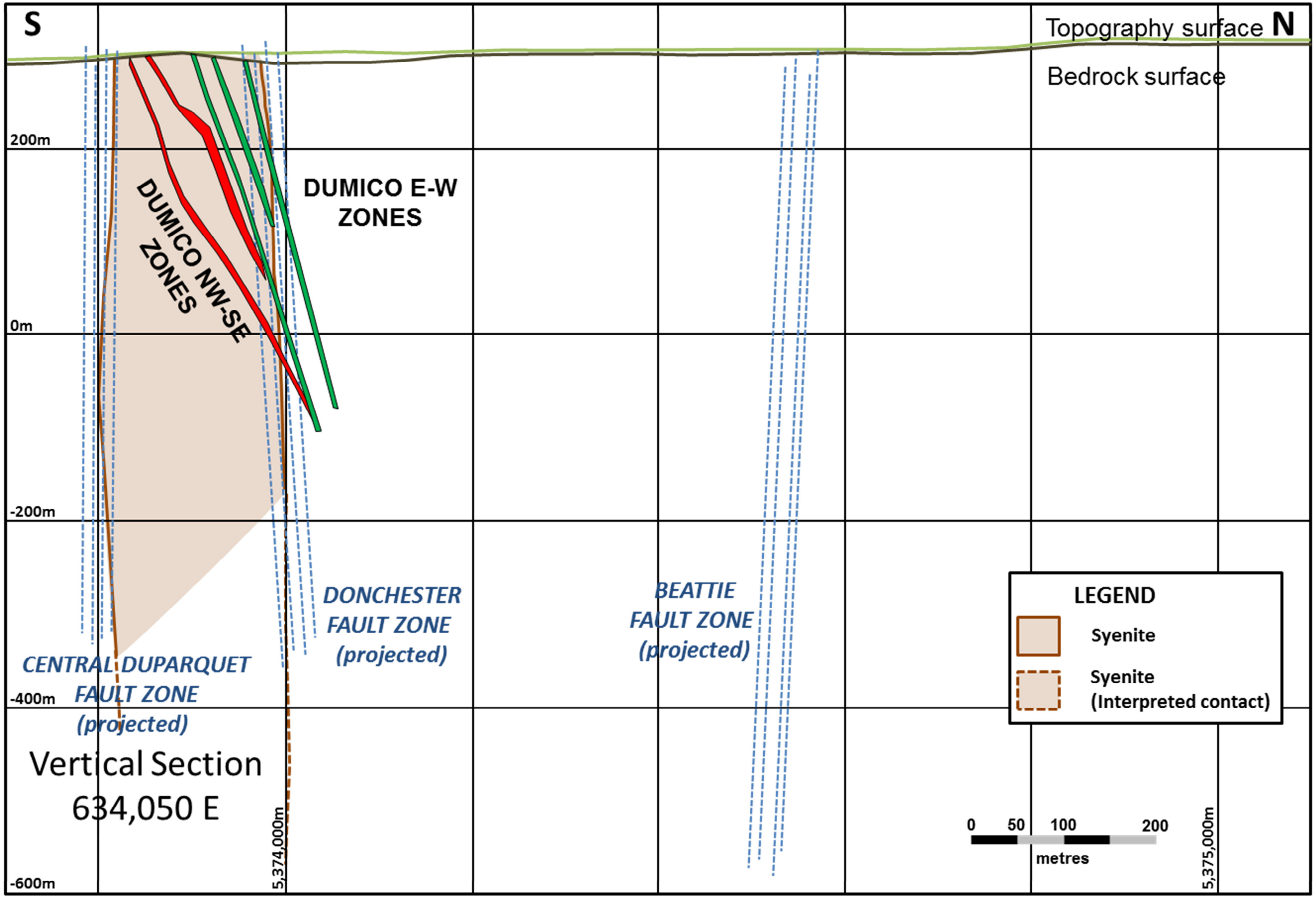

Figure 7.9 – Vertical section 634,050E, looking west, showing the structural and geometric details of the relevant structural elements from the deposit-scale 3D litho-structural model (location on Figure 7.4)

8. DEPOSIT TYPES

The standard orogenic gold model characterizes the majority of gold deposits within the Abitibi belt. However, several examples of late mineralization are disseminated and associated with alkaline intrusions (Robert, 2001), thus differing from the standard orogenic gold model. Syenite-associated disseminated gold deposits in the Abitibi greenstone belt consist of disseminated sulphide zones with variably developed quartz stockworks, which are intimately associated with Timiskaming-age, monzonitic to syenitic porphyry intrusions (Robert, 2001). Like quartz-carbonate vein deposits, all known syenite-associated disseminated gold deposits in the southern Abitibi belt occur along a major fault. As a result of their distribution along major faults, these deposits commonly occur at or near boundaries between contrasting lithological domains. Examples of these deposits are Young-Davidson, Matachewan Consolidated, Ross, Holt-McDermott, and Lightening in Ontario; and Beattie, Douay, Canadian Malartic, East Malartic, and Barnat-Sladen in Québec.

The Timiskaming-type sedimentary rocks occur along restricted segments of a major fault zones, where they are preserved as synclinal keels (Muller et al., 1991). The syenite intrusions form small stocks, commonly elongated subparallel to the overall structural trend and generally surrounded by numerous satellite dykes (Robert, 2001). Although some intrusive phases are equigranular, most are porphyritic, with K-feldspar phenocrysts in a fine-grained to aphanitic groundmass. Both Timiskaming-type sedimentary rocks and syenitic intrusions have been overprinted by at least one generation of structural fabrics and folds (Corfu et al., 1991). The sedimentary rocks are folded into tight synclinal structures, which probably accounts for their preservation, subparallel to the trace of the major faults along which they occur (Muller et al., 1991). A related penetrative foliation, best developed in sedimentary rocks, parallels the regional penetrative east-west, subvertical S_2 foliation. This foliation is absent to only weakly developed in the larger syenitic intrusions, except where they have been sericitized. Overprinted fault and shear zones are also common in these deposits (Robert, 2001). They range from relatively ductile shear zones to narrow brittle faults. The older faults are parallel to the regional foliation overprinting the Timiskaming-type sedimentary rocks.

Syenite-associated disseminated gold deposits consist of zones of disseminated sulphides with variably developed stockworks in intensely altered wallrocks (Robert, 2001). They have sharp diffuse limits, defined by a decrease in sulphide content, gold grades, and intensity of stockwork fracturing. Owing to the abundance of microveinlet stockworking and fracturing, many orebodies take on a breccia appearance. The morphology of the deposits ranges from overall tabular to pipe-like, although many have rather irregular outlines. Most mineralized zones are steeply dipping or steeply plunging, but examples of moderately to shallowly dipping orebodies, discordant to lithological units, are also known (Robert, 2001).

The total sulphide mineral content of the orebodies is typically less than 10% by volume, and commonly a few percent (Robert, 2001). Disseminated sulphides are fine- to very fine-grained and consist dominantly of pyrite, with significant arsenopyrite in a few deposits. Associated stockworks consist of millimeter- to centimeter-thick veinlets of grey to cherty quartz, with subordinate amounts of carbonate (Fe-dolomite and calcite), albite, and pyrite. In addition to pyrite and arsenopyrite, ore-related metallic minerals include minor to trace amounts of chalcopyrite and hematite. Telluride minerals, molybdenite, and magnetite are common associates of this type of mineralization, whereas galena, tennantite, and bismuthinite occur at few deposits. Accordingly, orebodies are generally enriched in Cu, As, Te, with common, but variable, enrichments in Pb, o, W, Zn, and locally Sb. The gold:silver ratios of the ores generally range from about 1:1 to 5:1.

Zones of hydrothermal alteration are spatially coincident with zones of disseminated sulphide minerals and veinlet stockworks, and most intense alteration corresponds in a general way to economic mineralization (Robert, 2001). Carbonatization and albitization are significant alteration types at nearly all deposits; K-feldspar alteration and sericitization are also present in several deposits.

The disseminated gold orebodies occur in a variety of positions relative to syenite intrusions, ranging from proximal to distal (Robert, 2001). A conceptual model is proposed by Robert (2001) for occurrence of disseminated gold mineralization as part of large, syenite-centred hydrothermal systems (Fig. 8.1). This system is shown in its interpreted primary configuration, prior to post-Timiskaming folding and faulting. A composite syenite stock is shown intruding along a pre-Timiskaming fault zone juxtaposing contrasted lithological domains. The early phase of intrusion is truncated by the unconformity; whereas the younger phase intrudes sedimentary rocks.

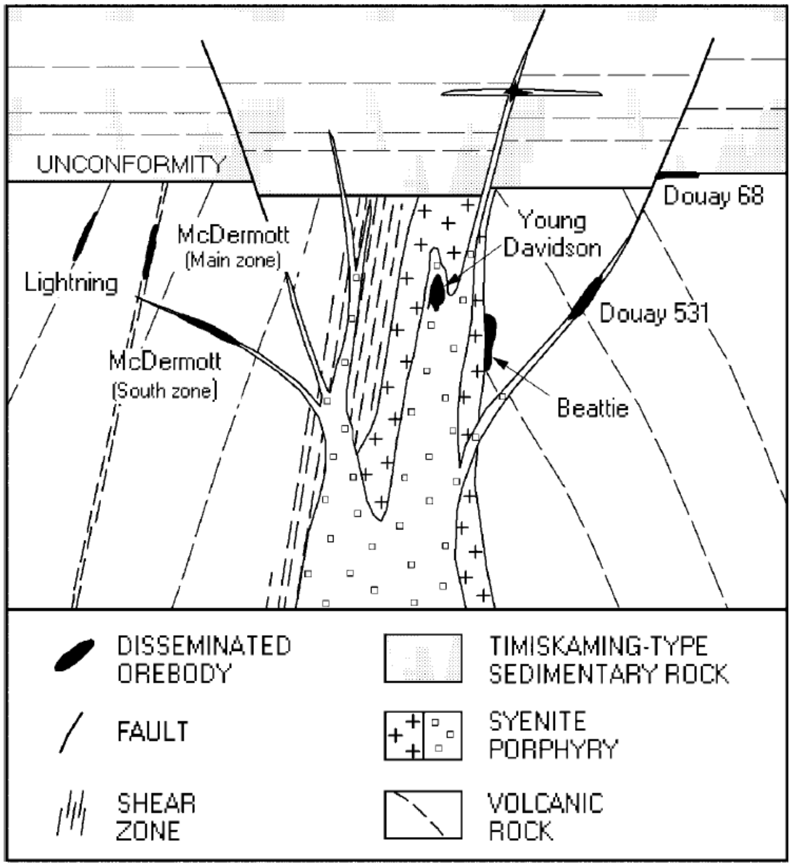

Figure 8.1 – Schematic geological model showing the distribution of disseminated-stockwork orebodies relative to a composite syenite stock intruded along a major fault zone, near the base of the Timiskaming unconformity (from Robert, 2001).

The Beattie Syenite is an Archean porphyry intrusion emplaced along the major Porcupine-Destor-Manneville Fault Zone (Fig. 7.4). The syenite is aligned along an east-west axis; it is hosted by mafic and intermediate volcanic rocks and is penecontemporaneous with the Timiskaming sedimentary rocks deposition. The main gold mineralization is associated with a network of east-west dextral strike-slip faults dipping steeply to the north. Syenitic intrusions indirectly control the location of gold deposits by their effect on the development of potentially mineralized structures during diapirism. The current structures hosting the mineralization are shear zones and folds. The Duparquet Formation is characterized by folding (Goutier and Lacroix, 1992). Gold deposits in the Duparquet area are genetically related to these intrusive rocks. The gold deposits tend to occur in concentric, normal and radial strike-slip faults and shears. Lateral ballooning and forceful intrusion may also generate concentric compressional structures (reverse and strike-slip faults and compressional fabrics). Gold deposits in the Duparquet area tend to occur close to the syenite-sedimentary and/or volcanic rock contacts. This is attributed to the competency contrasts between the syenitic rocks and the Duparquet Formation lithologies during deformation, resulting in favourable structural traps for gold mineralization.

In the Duparquet area, the main alteration observed is silicification. Sericitization, ankeritization and chloritization are other types of alteration associated with gold mineralization (Goutier and Lacroix, 1992). The main sulphide is pyrite (<10%) accompanied by arsenopyrite (Davidson and Banfield, 1944). Gold is hosted in arsenopyrite and arsenian pyrite (Bigot and Jébrack, 2012). Gold grains are less than micron-sized. The association with arsenian minerals and the very small size suggests that gold was incorporated into the crystalline structure of arsenopyrite and arsenian pyrite in solid solution or as nanoparticles (Bigot and Jébrack, 2012).

The metallic assemblage in Beattie syenite is polyphased (Bigot and Jébrack, 2012): (1) a primary phase enriched in iron-titanium appears to have produced martite in a more oxidizing environment; (2) several subsequent sulphidation phases were marked by the presence of pyrite and arsenopyrite, some rich in gold, suggesting crystallization under more reducing conditions and at lower temperatures. During the sulphidation phases, three generations of pyrite are identified; the first generation is arsenian and gold-bearing, whereas the second and third are arsenic-poor and gold free. (3) A late silica-enriched hydrothermal phase remobilized the gold and is marked by cataclasis. Gold migrated into the fractures developed in the cataclastic pyrite, where it crystallized with silver in the form of electrum.

According to Bigot and Jébrack (2012), several petrological characteristics in the Beattie gold deposit, including gold appearances, metallic mineralogy, type of alterations, and ore control, suggest a shallow magmatic deposit.

9. EXPLORATION

Since the most recent resource cut-off date (May 6, 2013), Clifton Star has completed 47 diamond drill holes totalling 7,422.5 metres of drilling. The 2013 diamond drill program was complete as of August 14th, 2013.

The 2013 outcrop stripping and surface channel sampling program, which was not included in this report, or the most recent resource calculation, was completed in November of 2013. During this surface sampling program, a total of 41 outcrops were mechanically stripped and channel sampled, which resulted in the collection of 1,001 samples.

10. DRILLING

Information in this section was obtained from the Clifton Star exploration team and combined with InnovExplo's database compilation work.

Clifton Star contracted Foramex to operate two (2) diamond drill rigs capable of recovering NQ size core on the Duparquet Project. The drilling was done using NQ rods. Clifton Star has been positioning its drill hole collars using a hand-held GPS unit. Once the holes are drilled, a more accurate DGPS survey is used on the collar to measure its location (x,y,z) as well as the collar azimuth and dip. Down-hole orientation surveys are done using a Reflex-Shot instrument. Rock on the Duparquet Project is of a really good quality and the recovery is high. The samples taken from that core is then representative and of a good quality. Clifton Star uses the core shack facility at the Beattie mine site for core handling, core logging and storage. Once delivered to the core shack, the core boxes were opened. The core was measured for recovery and RQD as well as photographed. RQD measurements indicated excellent recoveries. InnovExplo is of the opinion that the core samples from those drilling programs are valid and of sufficient quality to be used for mineral resource estimation.

The vast majority of the holes are drilled from the hanging wall to the footwall of the mineralized zones, mainly perpendicular in azimuth to the east-west trending of the mineralized zones but with variable angles in dip. Downhole lengths of mineralized zones do not represent true thicknesses of the given zones.

Drill core samples were taken along the entire length of the holes. The sample lengths varied from typically 1.0 metre length in the mineralized sections to 1.5 m length in the more barren sections. Limit of individual samples respected the contacts of the geological units and the visible mineralization. The true thicknesses of the intersections in the mineralized zones are typically 60% of the core lengths, but can be as low as 30% due to structural variations in the dip and strike. The mineralized zones are typically sub-vertical with the holes drilled at -50 degrees to the horizon.

Drill holes are divided into two (2) groups: those that were completed prior to the last mineral resource cut-off date of May 6, 2013 performed by Williamson et al., (2013b), and thus included in this update ("Phase 1"), and those that were drilled after the resource database cut-off date of May 6, 2013 ("Phase 2").

<u>Phase 1: 2012-2013 drilling program – included in the last resource estimate update</u>
From September 2012 to May 2013, Clifton Star drilled a total of ninety eight (98) new holes. Overall, the 2012-2013 drilling program produced 25,253 m of NQ-size core during this period.

Table 10.1 provides details on drilling carried out from September 2012 to May 2013 on the Beattie, Donchester and Central Duparquet properties. To summarize, a total of 25,253 m of new core has been produced across the three (3) properties: 7,758 m on Beattie; 11,974 m on Donchester; and 5,521 m on Central Duparquet.

Table 10.1 – Duparquet 2012-2013 drilling program summary for holes drilled prior to and included in the last resource estimate update

Program		PROPERTY			TOTAL
		Beattie	Donchester	Central Duparquet	
2012	DDH	14	24	15	53
	length (m)	3,880	8,389	3,632	15,901
2013	DDH	20	14	11	45
	length (m)	3,877	3,585	1,889	9,351
TOTAL	DDH	34	38	26	98
	length (m)	7,758	11,974	5,521	25,253

Figure 10.1 shows the locations of the drill holes drilled on the Duparquet properties. This figure illustrates the "in-filling" drilling strategy (2012-2013 validated DDH) used by Clifton Star when planning the drilling programs. The holes cover sections for which there was inadequate information.

<u>Phase 2: 2013 drilling program – NOT included in the last resource estimate update</u>

From May 6, 2013 until June 2013, Clifton Star continued drilling the Duparquet properties. A total of forty-one (41) drill holes were completed during this period, for a total of 6,086 m of NQ-size core. Figure 10.1 shows the location of the drill holes (2013 not validated DDH) not included in the last updated mineral resource estimate by Williamson et al., (2013b). Table 10.2 presents details of this drilling program. Table 10.3 presents the significant results coming from this second phase of the 2013 drilling program (source: Clifton Star news release of June 19, 2013; August 21, 2013; September 4, 2013).

Table 10.2 – Duparquet 2013 drilling program summary for holes drilled after the resource database cut-off date and thus NOT included in the current resource estimate update

Program		PROPERTY			TOTAL
		Beattie	Donchester	Central Duparquet	
2013	DDH	19	22	6	47
	length (m)	2,957	3,648	817	7,422

Table 10.3 – Significant results for Phase 2 of the 2013 drilling program: holes not included in last mineral resource estimate

Hole ID	From (m)	To (m)	Width (m)	Grade (Au g/t)	Section (Easting)
BD13-22	62.0	68.0	6.0	1.07	630350
BD13-22	137.0	171.0	34.0	5.64	630350
Incl.	*141.8*	*153.5*	*11.7*	*8.36*	
And	*162.0*	*169.5*	*7.5*	*7.21*	
BD13-23	296.0	314.5	18.5	2.46	630,340
Incl.	*296.0*	*302.3*	*6.3*	*2.88*	

Hole ID	From (m)	To (m)	Width (m)	Grade (Au g/t)	Section (Easting)
And	*309.6*	*314.5*	*4.9*	*5.26*	
BD13-24	55.6	58.7	3.1	1.04	631,250
BD13-24	115.0	117.0	2.0	1.18	631,250
BD13-25	51.0	68.0	17.0	2.5	630,175
Incl.	*56.0*	*64.0*	*8.0*	*3.81*	
BD13-26	36.0	43.5	7.5	2.18	629,950
BD13-26	73.5	75.0	1.5	2.16	629,950
BD13-27	6.0	7.5	1.5	2.59	630,175
BD13-27	61.5	133.5	72.0	1.14	630,175
Incl.	*63.0*	*70.5*	*7.5*	*1.79*	
And	*86.4*	*95.0*	*8.6*	*2.24*	
And	*106.5*	*112.0*	*5.5*	*1.91*	
And	*123.0*	*127.5*	*4.5*	*2.37*	
BD13-28	65.3	66.3	1.0	1.21	630,050
BD13-29	182.0	201.0	19.0	1.21	630,050
BD13-30	28.5	30.0	1.5	1.08	630,625
BD13-31	76.5	78.0	1.5	0.8	630,575
BD13-34	42.0	45.0	3.0	1.31	630,725
BD13-34	50.0	61.0	11.0	1.9	630,725
Incl.	*50.0*	*53.0*	*3.0*	*4.68*	
BD13-34	85.0	99.0	15.0	2.27	630,725
Incl.	*93.0*	*99.0*	*6.0*	*4.03*	
BD13-35	4.6	10.0	5.4	2.51	630,650
BD13-35	302.4	303.3	0.9	4.89	630,650
BD13-36	46.3	47.4	1.1	0.64	630,560
BD13-37	74.0	108.0	34.0	2.25	629900
Incl.	*74.0*	*89.0*	*15.0*	*3.48*	
And	*103.0*	*108.0*	*5.0*	*4.86*	
BD13-38	47.0	86.0	39.0	1.51	630250
Incl.	*63.5*	*86.0*	*22.5*	*2.19*	
BD13-39	25.5	37.5	12.0	1.04	630365
CD13-12	78.0	84.0	6.0	2.16	632815
CD13-12	167.0	169.0	2.0	1.51	632815
CD13-14	6.0	9.0	3.0	0.97	633,000
CD13-15	54.0	57.0	3.0	0.63	633,150
CD13-17	128.5	129.6	1.1	1.25	632,550
D13-15	5.0	6.0	1.0	1.02	632450
D13-15	161.1	167.0	5.9	1.66	632450
D13-16	66.0	67.0	1.0	0.85	632350
D13-17	88.5	90.0	1.5	1.00	632050
D13-17	99.0	102.0	3.0	1.17	632050
D13-17	127.5	129.0	1.5	1.07	632050

Hole ID	From (m)	To (m)	Width (m)	Grade (Au g/t)	Section (Easting)
D13-18	145.0	150.0	5.0	5.58	632050
Incl.	*147.5*	*150.0*	*2.5*	*9.45*	
D13-19	72.5	77.5	5.0	1.7	631,850
D13-19	93.0	94.0	1.0	2.34	631,850
D13-20	15.0	19.5	4.5	1.14	631,625
D13-20	39.0	46.5	7.5	1.51	631,625
D13-20	81.2	103.1	21.9	1.55	631,625
Incl.	*82.9*	*88.1*	*5.2*	*2.87*	
D13-21	84.5	100.1	15.6	2.22	631,740
Incl.	*91.5*	*98.5*	*7.0*	*3.48*	
D13-22	47.0	48.0	1.0	2	631,650
D13-23	94.1	95.1	1.0	1.97	631,750
D13-24	38.6	43.0	4.4	2.29	632,000
D13-25	24.0	27.0	3.0	1.49	632,150
D13-25	33.0	39.0	6.0	1.84	632,150
D13-26	30.0	31.5	1.5	2.75	632,125
D13-27	24.0	67.5	43.5	1.31	632,100
D13-27	135.7	137.1	1.4	2.59	632,100
D13-28	9.0	10.5	1.5	0.64	632,300
D13-29	30.0	31.5	1.5	0.88	632,350
D13-30	18.0	22.5	4.5	1.98	632,250
D13-32	157.5	159.0	1.5	9.64	632,450
D13-33	13.5	15.5	2.0	0.95	632,350
D13-33	30.0	31.5	1.5	2.21	632,350
D13-33	109.5	114.0	4.5	1.03	632,350
D13-33	178.5	180.0	1.5	2.02	632,350
D13-33	202.5	204.0	1.5	2.61	632,350
D13-34	99.0	103.5	4.5	4.98	632,250
D13-34	154.5	156.7	2.2	3.21	632,250
D13-35	57.0	60.0	3.0	3.69	632,140
D13-35	198.0	202.5	4.5	1.29	632,140
D13-35	208.5	214.5	6.0	1.37	632,140
D13-36	114.0	117.0	3.0	1.95	631,950

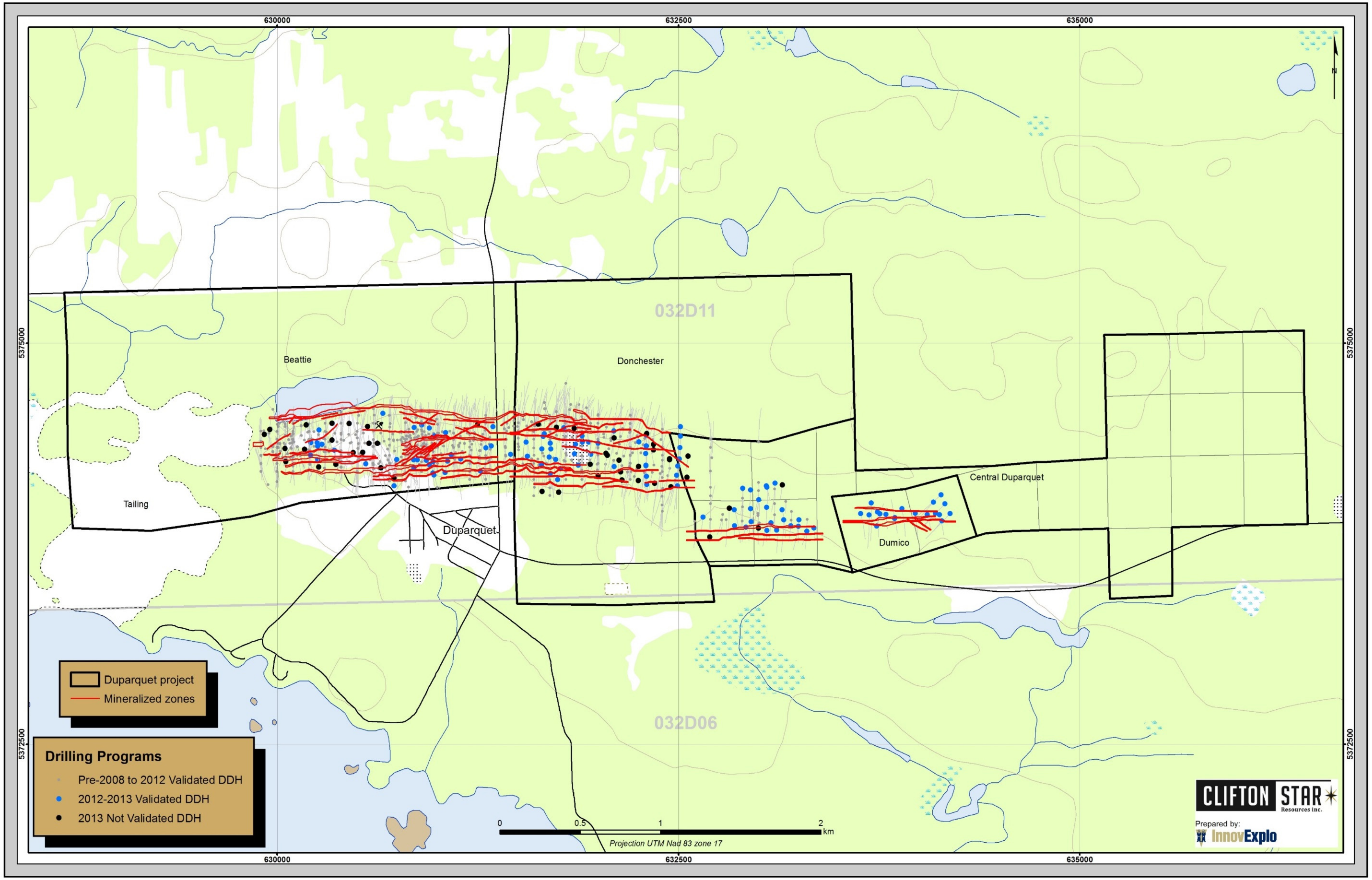

Figure 10.1 – Drill hole distribution on the Duparquet Project

11. SAMPLE PREPARATION, ANALYSES, AND SECURITY

The following paragraphs describe Clifton Star's sample preparation, analyses, and security procedures during the diamond drill hole (DDH) program carried out between September 15, 2012 and May 6, 2013 on the Duparquet Project (Williamson et al., 2013b).

Techni-Lab S.G.B. Abitibi Inc. ("Techni-Lab"), an ISO 17025 accredited facility in Sainte-Germaine-Boulé, Québec, was used for assaying samples from the DDH program performed between September 15, 2012 and May 6, 2013 on the Beattie, Donchester and Central Duparquet properties. Techni-Lab is a commercial laboratory independent of Clifton Star and has no interest in the Duparquet Project.

The 2012 drill cores were shipped to the laboratory in batches containing variable numbers of samples. Regardless of the number of samples per batch, one (1) blank and one (1) certified material reference (standard) were inserted for every twenty (20) samples. At the request of Clifton Star, the laboratory assayed one (1) pulp duplicates every twenty (20) samples.

For each 100 samples sent to Techni-Lab S.G.B. Abitibi Inc., numbers ending in the following digits were selected for the QA/QC samples:

- 15, 35, 55, 75, or 95 = standard;
- 17, 37, 57, 77, or 97 = pulp duplicate of preceding sample;
- 20, 40, 60, 80, or 00 = blank.

11.1 Sample Preparation (Techni-Lab S.G.B. Abitibi Inc.)

The drill core was boxed, covered and sealed at the drill rigs, and transported by Foramex drilling employees to the core logging facility at the Beattie mine where Clifton Star personnel would take over the core handling. The core was logged and sampled by, or under the supervision of, Clifton Star geologists, all members in good standing of the OGQ (Québec Order of Geologists). Each sample was tagged with a unique number.

Drill core samples were sawed by technicians and then bagged and sealed before being grouped in batches. The sample batches, and the shipping manifest listing all samples included in the batch, were shipped directly to Techni-Lab by Clifton employees where they were prepared according to the laboratory's sample preparation protocol for the given analytical procedure.

At Techni-Lab, each sample was crushed in its entirety using either an oscillating jaw crusher or a roll crusher, with the specification that more than 85% of the crushed material must pass a 2.4 mm (8 mesh) screen. A 250- to 300-gram fraction derived from the crushing process was then pulverized using a ring mill to 90% passing 106 µm (150 mesh).

11.2 Sample Preparation (Accurassay)

Furthermore, 10% of samples having a content of gold superior to 0.3 g/t Au are randomly selected and sent to another laboratory. First, selected pulp samples were sent to Accurassay Laboratory, an ISO 17025 accredited facility in Thunder Bay in Ontario.

The sample preparation was the same as Techni-Lab S.G.B. Abitibi Inc. If no pulp was available for the second assaying, the reject of the sample was used for assaying. In the case where no pulp and no reject was available, a quarter-split of the core was sent to the laboratory.

11.3 Gold Analysis (Techni-Lab S.G.B. Abitibi Inc.)

For the drill core, gold was analyzed by fire assay with AAS (atomic absorption spectroscopy) finish using a 50-g nominal sample weight. For the fusion process, only twenty-four (24) samples can be assayed together. Twenty (20) samples of Clifton Star and four (4) quality control samples added by Techni-Lab S.G.B. Abitibi Inc form a single batch of twenty-four (24) samples. For each batch, Clifton Star thus ensures that each of their quality control samples (1 analytical blank, 1 certified reference materials and 1 pulp duplicate) are present during the fusion process. To these batches, the laboratory randomly added four (4) additional quality control samples that correspond to one (1) analytical blank, two (2) certified reference materials and one (1) pulp duplicate).

For grades over 5.0 g/t Au obtained by fire assay, samples were re-assayed with a gravimetric finish. If the assay result from gravimetric finish is over 10 g/t Au, the sample is again assayed by metallic sieve method.

11.4 Gold Analysis (Accurassay)

The analytical method was the same as Techni-Lab S.G.B. Abitibi Inc. For grades over 5.0 g/t Au obtained by fire assay, samples were re-assayed with a gravimetric finish. If the assay result from gravimetric finish is over 10 g/t Au, the sample is again assayed by metallic sieve method.

11.5 Quality Assurance and Quality Control Procedure

Blanks

The field blank used for the drilling (DDH) program performed between September 15, 2012 and May 6, 2013 was a crushed sample of gold-barren marble. One (1) field blank was inserted for every twenty (20) field samples.

Clifton Star’s quality control protocol stipulates that if any blank yields a gold value above 0.03 g/t Au, all samples that precede the failed blank up to the previous passed blank and all samples that follow the failed blank up to the next passed blank should be re-analyzed.

A total of 1226 blanks were assayed during the period between September 15, 2012 and May 6, 2013. Only one blank did not pass the quality control procedure of Clifton Star, which represents less than 0.08%.

11.5.1 Pulp Duplicates

At the request of Clifton Star, the laboratory assayed one (1) pulp duplicates every twenty (20) samples. The precision of the pulp duplicates can be used to determine the incremental loss of precision for the pulp pulverizing stage of the process, thereby establishing whether a given pulp size taken after pulverization is adequate to ensure representative fusing and analysis.

Clifton Star used a graph generated by GeoticLog software to validate the data from pulp duplicate. On graph, the gold values were plotted with a confident interval of 90%. All pulp duplicate, with each sample reference, located outside of the confident interval were re-analyzed.

A total of 1111 pulp duplicates was assayed during the period between September 15, 2012 and May 6, 2013. Only thirty-two (32) pulp duplicates did not pass the quality control procedure of Clifton Star, which represents less than 2.88%.

11.5.2 Certified Reference Materials (standards)

For the drilling (DDH) program performed between September 15, 2012 and May 6, 2013, one (1) certified reference material (CRM) standard was inserted for every twenty (20) samples. The assigned grades for the nine (9) CRM standards used for the drilling program ranged from 0.263 g/t Au to 2.360 g/t Au. Table 11.1 presents details on the CRMs used by Clifton Star for the 2012-2013 drilling program.

Table 11.1 – CRMs used by Clifton Star for the drilling program between September 15, 2012 and May 6, 2013

CRM NAME	Provider	Theoretical Au grade (g/t)	Standard deviation (g/t)
CDN-GS-1J	CDN Resource Laboratories Ltd	0.946	0.051
CDN-GS-1P5F		1.400	0.060
CDN-PGMS-23		0.496	0.029
CDN-GS-2J		2.360	0.100
CDN-GS-2K		1.970	0.090
CDN-GS-2L		2.34	0.120
CDN-GS-P3B		0.409	0.021
CDN-GS-P3C		0.263	0.010
CDN-GS-P7E		0.766	0.043

The accuracy of the result (as a percentage) is measured as the difference between the average of the standard's samples and the value assigned for the standard. For a laboratory, a good accuracy constitutes of the ability to give results as near as possible to the expected value. The precision of the result (as a percentage) is represented by the dispersion of the standard's samples versus their average. For a laboratory, a good precision constitutes the ability to repeat results with the smallest standard deviation possible. The difference between accuracy and precision is illustrated by Figure 11.1.

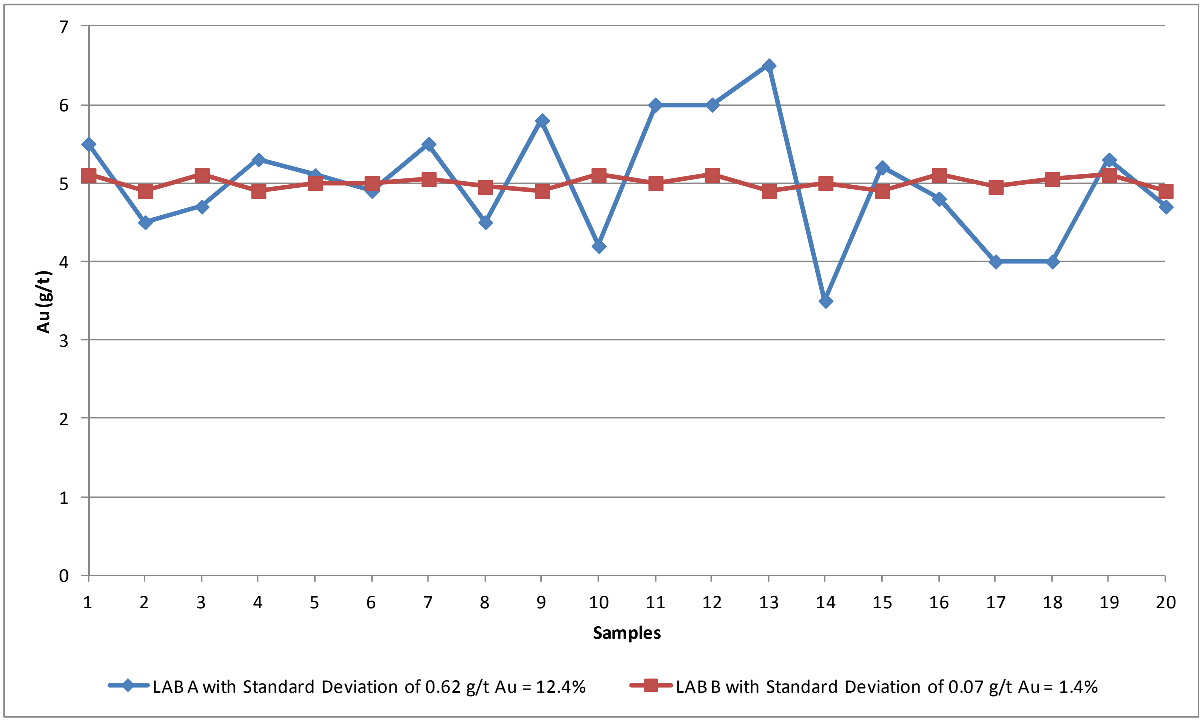

Figure 11.1 – Two laboratories (LAB A and LAB B) have analyzed the same standard grading 5.0 g/t Au using the same number of samples (n=20) to produce the same final average (5.0 g/t Au). Accuracy is perfect (0%) for both, but the precision of LAB B is better (1.4%) than the precision of LAB A (12.4%).

Clifton Star's quality control protocol stipulates that if any standard yields a gold value above three standard deviations (Table 11.1), the standard is failed. But, if the gold values obtained for two consecutive standards are between two and three positive standard deviation or between two and three negative standard deviation, these two standards are failed. All samples that precede the failed standard up to the previous passed standard and all samples that follow the failed standard up to the next passed standard should be re-analyzed.

A total of 1241 standards were assayed during the period between September 15, 2012 and May 6, 2013. Fifty-two (52) standards did not pass the quality control procedure of Clifton Star, which represents less than 4.2%.

11.5.3 Duplicate Analyses by Two Different Laboratories

Ten percent of samples having a content of gold superior to 0.3 g/t Au are randomly selected and sent to another laboratory. A total of 164 samples were sent to the Accurassay laboratory in Thunder Bay, Ontario. Of these, four (4) are considered outliers.

Figure 11.2 plots the duplicate gold analyses for 164 samples from the two different laboratories. The green circles represent gold results with a field of relative difference of about ±20%. Two green lines illustrate this interval of relative difference. Both laboratories produced generally similar gold results with relatively slight scatter (low random error), as indicated by the abundance (majority) of points falling between the two green lines. The linear regression slope corresponds to 0.969 with a correlation coefficient of 99.31%. The correlation coefficient (%) is given by square root of R² and represents the degree scatter of data around the linear regression slope. The results obtained indicate an excellent reproducibility of gold values.

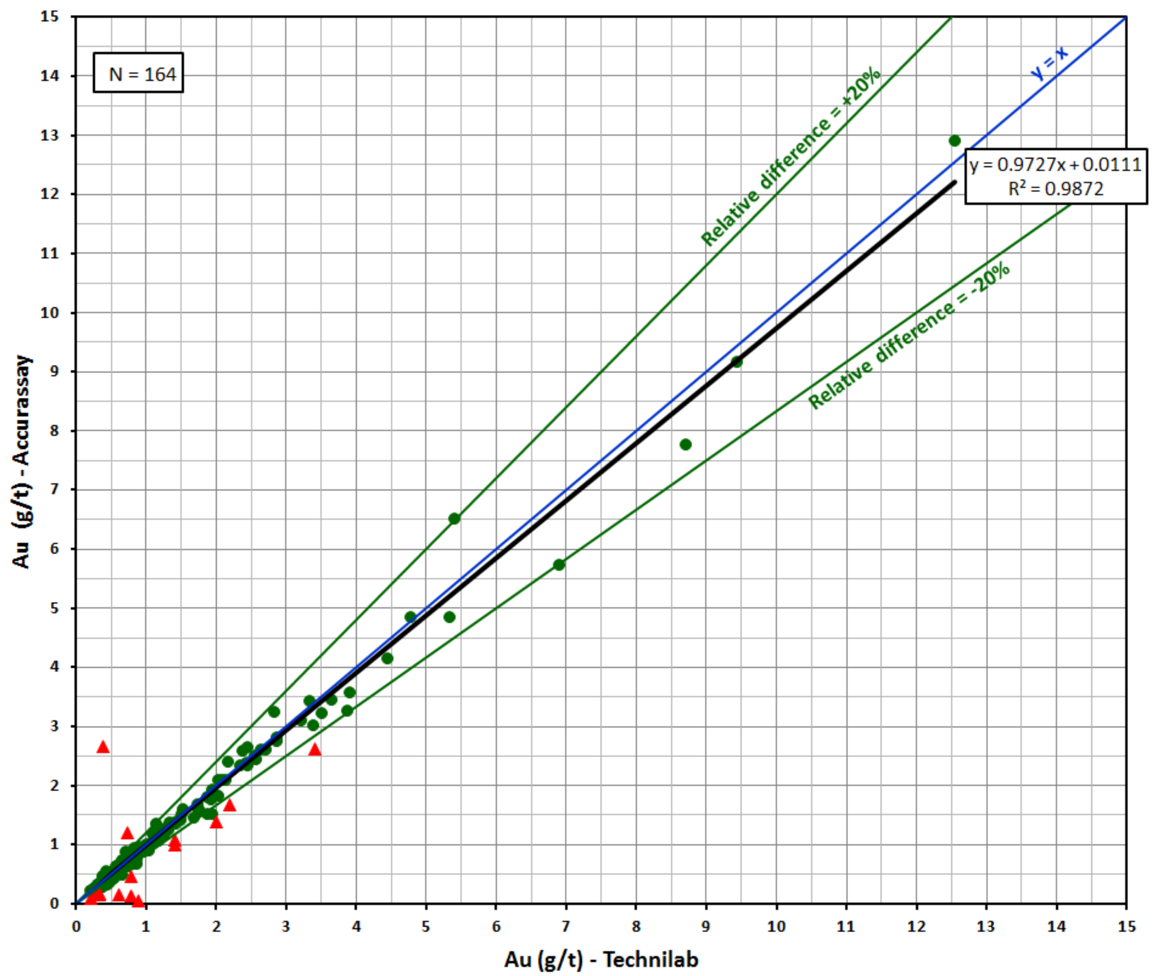

Figure 11.2 – Plots duplicate analyses from two different laboratories. Green circles represent results with a field of relative difference of about ±20%. Red triangles are outliers

For the current drilling program, only fifteen (15) results among 164 plotted outside the ±20% field of relative difference, representing 9.1% of the total population. These outliers are represented by red triangles on Figure 11.2. The observation of some outliers on Figure 11.2 suggests the presence of a nugget effect in some of the Duparquet Project gold results.

InnovExplo is of the opinion that results obtained for duplicate analyses by two different laboratories are reliable and valid.

11.6 Conclusions

A statistical analysis of the QA/QC data provided by Clifton Star did not outline any significant analytical issues.

InnovExplo is of the opinion that the sample preparation, analysis, QA/QC and security protocols of Clifton Star for the drilling (DDH) program carried out between September 15, 2012 and May 6, 2013 on the Duparquet Project generally follow accepted industry standards.

Based on these results, InnovExplo is of the opinion that following the addition of new drilling data, the compiled resource database is valid and of sufficient quality to be used for mineral resource estimation.

11.7 Bulk Sampling Drilling

Clifton Star hired SGS Lakefield (“SGS”), in Ontario, to carry out the metallurgical testwork. SGS requires a minimum of 10 tonnes of rock and Clifton Star selected the representative sample from the resource area defined inside the pit-shell referred to in the 2013 Technical Report by Williamson et al. (2013b). The sample comprises material from six (6) mineralized zones. Clifton Star decided to sample the entire drill core of mineralized intervals using large-diameter diamond drill holes (HQ-diameter) (“HQ-holes”). The HQ-holes were twins (i.e., in close proximity) of the reference diamond drill holes, labelled master holes. The master holes were of NQ-diameter. Clifton Star decided to extract 12 tonnes of material in anticipation of possible loss of material during drilling activities.

Forty-six (46) large HQ-holes were completed on the Duparquet Project for a total of 3,894.9 m, including 1,467.7 m of sampled mineralized zones for a total weight of 11,991 kg (Table 11.2). One to two drill rigs were operating during the program from January 23 to March 6, 2013. Drilling was carried out by Foramex Diamond Drilling Inc.

The collar positions of HQ-holes (Fig. 11.3) were determined from the casings of master holes and/or from HQ-holes already drilled with an initial spacing of 0.5 m in the N-S direction and of 1.0 m in the E-W direction. Note that the selected master holes are oriented in a general North-South direction except for master hole BD11-339 (Table 11.3), which is oriented 71° north.

InnovExplo and Clifton Star were both responsible for defining the sampling protocol described below (Servelle and Pelletier, 2013). Daily follow-up of bulk sample drilling was realized using Clifton Star’s reports of information on vertical cross sections.

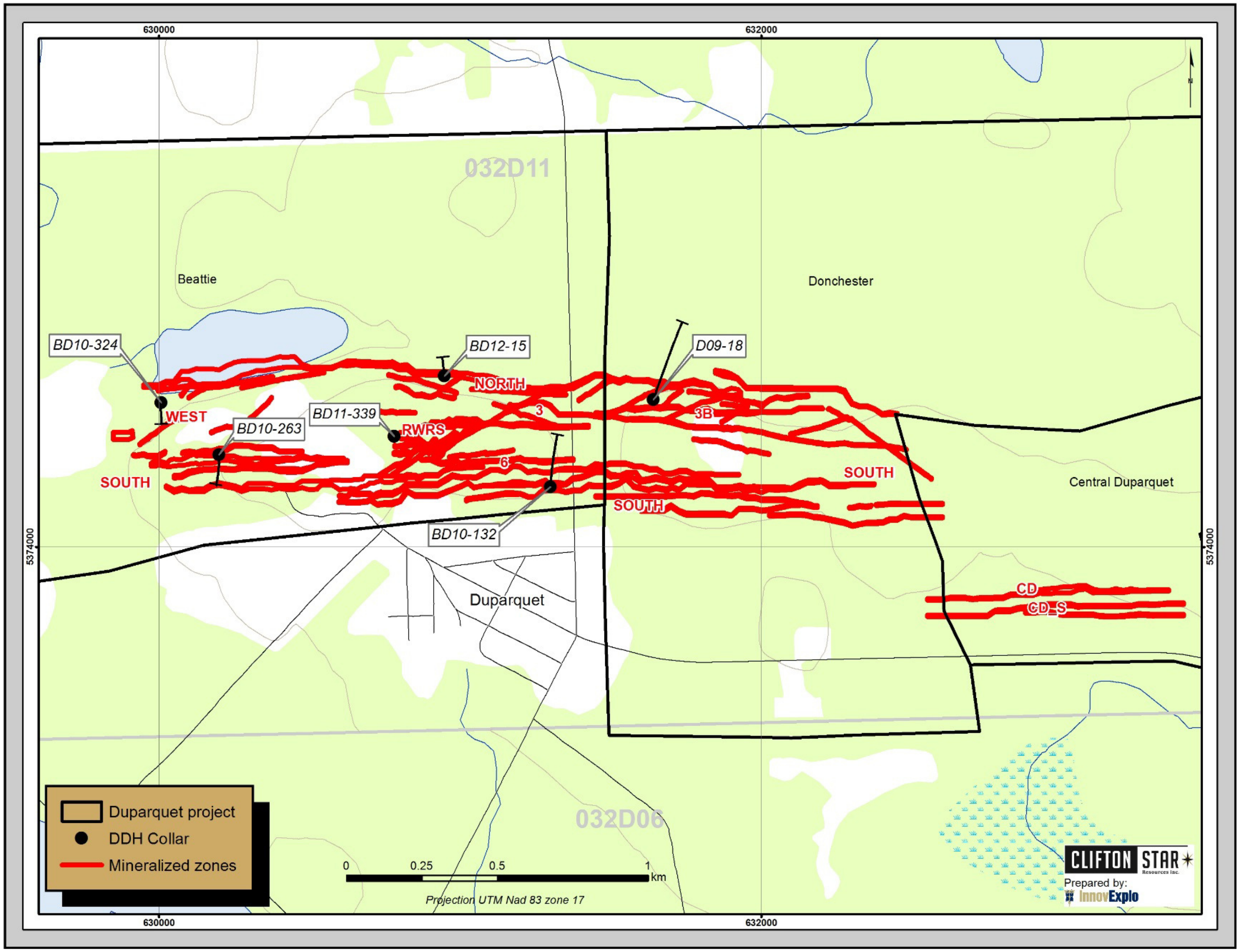

Figure 11.3 – Collar positions of HQ-holes determined by the locations of master hole casings

Table 11.2 – Summary of the 2013 bulk sample drilling

Mineralized Zone	Master Hole	Zone-From	Zone-To	Zone Width	HQ Core Sampled (m)	HQ Holes Realized	Total Drilling (m)	Sampled Weight (kg)
North Zone - Beattie	BD12-15	7.1	36.7	29.6	175.1	5	288.0	1,474
North Zone - Donchester	D09-18	63.0	88.0	25.0	332.8	16	1,524.1	2,653
South Zone - Beattie	BD10-263	9.0	109.0	100.0	270.2	3	329.1	2,123
South Zone - Donchester	BD10-132	17.5	40.0	22.5	192.6	9	438.4	1,606
West Zone	BD10-324	63.0	103.0	40.0	365.8	11	1,111.3	3,016
RS/RW	BD11-339	5.0	82.0	77.0	131.2	2	204.0	1,119
TOTAL					**1,467.7**	**46**	**3,894.9**	**11,991**

Note: Zone Width is expressed according to core length

Table 11.3 – Coordinates of master holes

Mineralized Zone	Master Hole	Easting	Northing	Elevation	Azimuth	Dip	Length (m)
North Zone - Beattie	BD12-15	630947.9	5374567.8	285.3	353	-50	102.0
North Zone - Donchester	D09-18	631641.7	5374489.5	303.6	17	-44	372.0
South Zone - Beattie	BD10-263	630199.8	5374306.2	299.5	179	-69	302.0
South Zone - Donchester	BD10-132	631299.7	5374200.2	299.2	359	-68	458.0
West Zone	BD10-324	630008.1	5374479.6	281.7	188	-45	108.0
RS/RW	BD11-339	630781.6	5374367.4	311.7	71	-44	90.0

The drill core was boxed, covered and sealed at the drill rig and then moved by Foramex's drilling staff to the Clifton Star logging and sample preparation facilities in Duparquet. The area was exclusively dedicated to the bulk sample program. Core was immediately checked by Clifton Star's geologists to validate drill progress and lithologies. Geologists had access to the complete master hole as reference for the associated HQ-holes.

Drill core measurements were validated by field workers from Clifton Star and consisted of correcting any significant discrepancies in the down hole distance measurements (wooden block depth markers placed every 3 m along the core) as well as checking core recovery. Rock quality designation (RQD) using a reference spacing of 3 m and discounting core pieces less than 10 centimetres long was measured. The core was systematically photographed by the field workers.

Logging and quick descriptions of the drill core of HQ-holes were completed by qualified professionals from Clifton Star who are members, in good standing, of the OGQ (Québec Order of Geologists) or the OIQ (Québec Order of Engineers). Core logging and data entry was done using a laptop with Geotic Log® software. Core logging protocols required the following to be documented and described:

- Principal lithologies with colours, textures and contacts;
- Secondary lithologies were described according to the same parameters;
- Alteration style and intensity;
- Mineralization, generally determined by sulphide type and sulphide concentration in total core volume;
- Vein type, density and orientation;

- Structural parameters, such as fractures, fault angles, hydrothermal breccias, folds, kink bands, etc.

The master hole was used as reference for core logging and for the selection of zones to be sampled. Zone selection was carried out by Clifton Star's geologists under the supervision of Louis Martin, P.Geo. Identification of zones was done by direct comparison between the master hole and the HQ-holes. Independent verification of each zone to be sampled was carried out by Guilhem Servelle, MSc, P.Geo., of InnovExplo Inc. The verification was carried out with the collaboration of Clifton Star's geologists.

After being validated by InnovExplo, whole drill core zones were sampled and put in large white rice bags (Fig. 11.4) according to the following protocol.

Figure 11.4 – Rice bags completely filled and sealed

The full core of zones selected in the twin drill holes was inserted in rice bags specifically manufactured for bulk sampling. The rice bags were 1.0m x 1.0m x 1.0m and were rated to handle 1000 kilograms of material. Made by Manyan Inc., they had never been used. To optimize integrity of loaded rice bags, they were put on wooden pallets and re-enforced with a plastic tarp lining. The rice bags were filled up to 900 or 950 kilograms of core, numbered and stored in a secure area. Each filled rice bag is sealed by Clifton Star personnel with a numbered tie-wrap in the presence of InnovExplo (Fig. 11.5).

For each sampled zone, before insertion in rice bags, a weight check was conducted to identify hypothetical bias with the initially expected weight. For this purpose, whole drill core zone was subdivided in intervals of equal length of approximately 6.0 m. Weight measurement for intervals of whole drill core was determined with a mechanical balance. A secondary check with an electronic balance returned the same weight, with <2% variation. Weight checks were continuously followed on site by InnovExplo. Whole drill core recuperation is documented in Servelle and Pelletier (2013), and no critical loss of material was noted. During the program, no additional drilling was required due to recuperation discrepancy.

Figure 11.5 – Numbered tie-wrap used to seal filled rice bags

All rice bags were filled on site in the presence of InnovExplo. For rice bags partially filled between site visits, a temporary seal was affixed for security purposes. The seal was checked before commencing and broken by InnovExplo. Complete and sealed rice bags were stored at the roaster facility of the Beattie Mine in Duparquet pending final shipment to SGS. Before shipping to SGS, the rice bags were wrapped in plastic cling wrap with their respective weight chart prepared by InnovExplo enclosed.

The rice bags left Duparquet on March 11, 2013 and were delivered to SGS on March 14, 2013 by Transport Manitoulin Inc. All drill core not sampled from twin holes is stored and categorized for future reference at the Clifton Star core storage facility. The core is currently kept in good condition in roofed outdoor core racks near the roaster facility of the Beattie Mine in Duparquet. All remaining core boxes are labelled and properly stored.

According to Servelle and Pelletier (2013), there is no indication of anything in the drilling, core handling, selection of samples and sampling procedures, methods and approach that could have had a negative impact on the reliability of the integrity of material destined to SGS.

12. DATA VERIFICATION

This section presents the verification of all new data added to the previously validated database used by InnovExplo to prepare an update of the Mineral Resource Estimate dated December 31, 2012 and presented in the "Technical Report and Preliminary Economic Assessment for the Duparquet Project (according to Regulation 43-101 and Form 43-101F1)", released on February 28, 2013 (Williamson et al., 2013a).

The new data includes drill holes from the last portion of the 2012 and the first portion of the 2013 drilling programs (Williamson et al., 2013b), as well as new samples collected along previously drilled holes, for Beattie, Donchester and Central Duparquet areas. New data also includes a series of holes drilled on the Dumico property in 2008-2009. Thirteen (13) of the nineteen (19) Dumico holes have been entirely resampled by Clifton Star in 2012. InnovExplo created the final database in Geotic format and transferred it into GEMS software for the purposes of the resource estimation.

The database cut-off date for the last updated resource estimate is May 6, 2013. Holes that were incomplete at the cut-off date and recent channel samples were not included in the compilation.

For the last updated Mineral Resource Estimate, InnovExplo has verified a representative portion of the new data provided by Clifton Star for the Beattie, Donchester and Central Duparquet portion of the dataset. The Dumico related holes have been submitted to full data verification where all collar locations and assays certificates have been checked and validated prior to the import in the final database.

The following are the details of the data verification conducted by InnovExplo and presented in the "Technical Report and Mineral Resource Estimate Update for the Duparquet Project (according to Regulation 43-101 and Form 43-101F1)", dated August 2, 2013 (Williamson et al., 2013b).

12.1 Drill Hole Data

A total of 117 new holes have been added to the previously compiled and verified master database (Williamson et al., 2013b). The addition comprises thirty-four (34) holes from the Beattie property, thirty-eight (38) holes from Donchester, and twenty-six (26) holes from Central Duparquet. Nineteen (19) holes from Dumico properties were also added to the master database. Eight (8) pre-existing holes have been re-sampled by Clifton Star and were updated within the database. A total of seven (7) holes, drilled for the purpose of metallurgical testing in 2011, have been rejected and recent metallurgical twin holes (see Section 11.7) were not taken into consideration for the purpose of the Mineral Resource Estimate. The updated master database used for the last Mineral Resource Estimate thus contains a total of 849 diamond drill holes totalling 260,948.4 m and 168,555 sampled intervals (Williamson et al., 2013b).

12.1.1 Drill Hole Collar Locations and Downhole Surveys

Collar position coordinates and azimuths are presented using the UTM system (NAD 83, Zone 17). All new drill holes, including the Dumico holes, incorporated into the master database have been surveyed by Patrick Descarreaux, and the data used herein is as at May 6, 2013.

In a small number of cases, there was a difference between the collar azimuth and the first valid downhole survey. In these cases, InnovExplo systematically used the first valid downhole survey azimuth for the collar azimuth.

Downhole surveys' azimuths obtained by any other methods than Gyro are giving a bearing relative to geographical north. These azimuths were corrected by -1.3334 degrees in order to obtain UTM azimuths.

12.1.2 Assay Verification

For the Beattie, Donchester and Central Duparquet drill holes, InnovExplo verified the assay certificates for about 5% of the new drill holes received, corresponding to a full verification of five (5) holes. The holes were selected based on their representativeness, both in terms of the drilling program they were part of (i.e. 2012 and 2013) and their geographical position with respect to the mineralized zones interpretation. Table 12.1 presents the details of the drill holes selected by InnovExplo.

Table 12.1 – Drill holes selected for the purpose of assay data verification for the Beattie, Donchester and Central Duparquet properties.

HOLE-ID	DRILLING PROGRAM	PROPERTY	SAMPLES VERIFIED	CORRECTIONS APPLIED	% OF FAILURE
BD12-19	2012	Beattie	279	0	0
D12-28	2012	Donchester	396	0	0
CD12-13	2012	Central Duparquet	116	0	0
BD13-02	2013	Beattie	225	0	0
D13-02	2013	Donchester	273	0	0

For these selected drill holes, 100% of the assays were reviewed by comparing the data provided by Clifton Star and the raw data obtained directly by the assay laboratory (Techni-Lab in Sainte-Germaine-de-Boulé). In all five (5) cases, no correspondence errors were found between the raw assays table provided by Clifton Star and the assay certificates obtained from the laboratory.

For the nineteen (19) Dumico drill holes, 100% of the assays have been verified against the assay certificates.

Table 12.2 presents the details of the assay verification done by InnovExplo for the Dumico drill holes. Minor correspondence errors, less than the 5% threshold used by InnovExplo as a failure criteria, have been corrected in three (3) drill holes. Ninety-three (93) missing assay results have been entered from the assay certificate by InnovExplo for hole DUM09-08.

Table 12.2 – Dumico drill hole assay data verification

HOLE-ID	SAMPLING PROGRAM	PROPERTY	SAMPLES VERIFIED	CORRECTIONS APPLIED	% OF FAILURE
DUM08-01	2013	Dumico	157	0	0
DUM08-02	2013	Dumico	114	0	0
DUM08-03	2013	Dumico	236	0	0
DUM08-04	2013	Dumico	300	0	0
DUM08-05	2013	Dumico	371	0	0
DUM08-06	2013	Dumico	437	0	0
DUM08-07	2013	Dumico	135	0	0
DUM09-01	2013	Dumico	87	0	0
DUM09-02	2013	Dumico	93	4	4.30
DUM09-03	2013	Dumico	128	0	0
DUM09-04	2013	Dumico	105	0	0
DUM09-05	2013	Dumico	86	0	0
DUM09-06	2013	Dumico	183	0	0
DUM09-07	2013	Dumico	180	0	0
DUM09-08	2013	Dumico	235	93*	N/A
DUM09-09	2013	Dumico	245	0	0
DUM09-10	2013	Dumico	166	1	0.60
DUM09-11	2013	Dumico	131	0	0
DUM09-12	2013	Dumico	115	2	1.74

**:correction applied corresponds to the incorporation of missing assay results from the assay certificate; thus not considered as a failure by InnovExplo.*

InnovExplo also randomly verified approximately 5% of the laboratory QA/QC checks and did not uncover any significant failures in the dataset.

12.2 Channel Sample Data

A total of 2,371 samples coming from 892 channels (for a total of 1,827 m in length) had already been entered and validated in the master database (Williamson et al., 2013b). As no new channel samples have been provided by Clifton Star before the effective date of May 6, 2013, since then, no further verification of the channel sample data was deemed required by InnovExplo. The 2013 channel sampling program carried out by Clifton Star was not included in the last mineral resource estimate.

12.3 Field Visits

Kenneth Williamson of InnovExplo visited the Duparquet Project on July 4, 2013.

This field visit provided an opportunity to verify the new data integrated into the master database. In particular, key intervals of a few of the new drill holes and re-sampled drill holes were reviewed to ensure the drill hole logs accurately corresponded to sample positions. Special emphasis was placed on the Dumico property as this series of drill holes had never been integrated into the master database by InnovExplo prior to the current Mineral Resource Estimate.

Several outcrops and drill sites were visited and manual checks, using a hand-held GPS unit, were performed to verify the location of some selected drill hole collars.

Visits to outcrops provided an opportunity to confirm the interpretation of several new zones added to the current Mineral Resource Estimate by performing a preliminary structural analysis of areas where the new zones crop out. This analysis showed that structures related to these new zones are present on the outcrops, strengthening the geometrical interpretation of the zones.

The tailings area was revisited to ensure that the site had remained intact since the last visit, and most of the sampling sites were still in good condition.

12.4 Database Integrity

Following the database additions, corrections and modifications, InnovExplo considers the results to be adequate for calculating mean gold values for intervals containing multiple assays, regardless of the sample source or assay method used. InnovExplo is of the opinion that the final in-pit and underground drill hole database is adequate to support a Mineral Resource Estimate for the Duparquet Project, including the Beattie, Donchester, Central Duparquet and Dumico properties.

13. MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

Bench scale and pilot plant metallurgical testwork programs have been carried out for the Duparquet Project. The preliminary metallurgical testwork was carried out in 2012 by SGS to support the Preliminary Economic Assessment ("PEA"; Willamson et al., 2013a). In 2013, SGS carried out further flotation, pressure oxidation, cyanidation, rheology and environmental testwork including a pilot plant for the current PFS. Outotec was also mandated in 2013 to carry out filtration testwork. The following is a list of reports from these test programs:

SGS Canada

- An investigation into the recovery of gold from Duparquet Project Samples, Project 13054-001 Final Report, May 24, 2012
- An investigation into the recovery of gold from Duparquet Project Samples, Project 13054-002 Report 1, September 27, 2012
- An investigation into environmental characterization of minerals processing residues from the Duparquet Project, Project 13054-002 Report 2, November 29, 2012
- An investigation into cleaner flotation for the recovery of gold from Duparquet Project Samples, Project 13054-003 Final Report, February 5, 2013
- An investigation into the grindability characteristics of a pilot plant feed sample from the Duparquet deposit, Project 13054-004 Grindability Report, May 21, 2013
- A flotation pilot plant investigation into the recovery of gold from Duparquet samples, Project 13054-004 Report 1, June 20, 2013
- A pilot plant investigation into pressure oxidation followed by hot curing of Duparquet flotation concentrate prepared for Clifton Star Resources Inc. Project 13054-004 Report #2, August 26, 2013
- An investigation into the rheometallurgical response of Duparquet metallurgical samples, Project 13054-001 – Draft - Report #3, August 27, 2013
- An investigation into the design of a carbon adsorption circuit and downstream testing for the Duparquet deposit, Project 13054-004 Report #4, September 12, 2013
- An investigation into the grindability characteristics of a pilot plant feed sample from the Duparquet deposit, Project 13054-004 Grindability Report – Rev 1, September 20, 2013

Outotec

- Filtration test report, Project 108096T 1, August 15, 2013

The sections that follow review the data presented in each of these reports and how the data have been used to establish the process flowsheets, design criteria, and process plant design for the PFS.

The samples used for the various testwork programs were selected under the supervision of InnovExplo and Clifton Star as described in Section 11.7.

13.2 Previous testwork

13.2.1 SGS Project Reference No. 13054-001 Final Report, May 24, 2012

SGS Project Reference No. 13054-001 Final Report includes the results of the flotation, pressure oxidation, and cyanidation testwork performed in order to investigate the recovery of gold from ore and tailings samples. Preliminary comminution tests and environmental tests were also conducted during this program.

The testwork was conducted on six ore samples and two in-situ tailing samples. The gold grade of the ore samples ranged from 1.3 to 3.5 g/t Au. The sulphide content ranged from 0.5% to 2.0% S.

The Bond Work Index of the ore samples varied from 17.2 to 20.2 kWh/t, classifying them as hard to very hard ore.

The preliminary gravity separation testwork on ground ore samples indicated low gold recovery ranging from 3.7% to 14.9% and averaging 8.6%. Hence, gravity separation was not pursued.

As shown in Table 13.1 the recovery of gold to concentrate by flotation was greater than 90% for most samples. The Central Duparquet Main sample is the exception with 84.6% gold recovery, increasing to 87.9% with finer grinding. Gold recovery by pressure oxidation and cyanidation treatment of the flotation concentrate was also investigated and was consistently high.

Samples of the flotation tailings were leached under conventional cyanidation conditions. Standard bottle roll tests were conducted at 40% solids and pH 10.5 with 0.5 g/L NaCN for 48 hours. These tests gave poor gold extractions varying from 26.2% to 56.3%, confirming the refractory nature of the ore.

The overall gold results from flotation concentrate, pressure oxidation (POX) and CIL circuit as well as flotation tail cyanidation for the ore are summarized in Table 13.1. The overall recovery of gold was ranging from 91.9% to 95.4%

Table 13.1 - Summary of overall results for the ore samples

Sample	Flotation Concentrate Test No.	Flotation Concentrate Wt %	Flotation Concentrate Assay Au, g/t	Flotation Concentrate Rec'y, % Au	POX Time min	Concentrate POX-Cyanidation Test No.	Concentrate POX-Cyanidation Add'n, kg/t of ore NaCN	Concentrate POX-Cyanidation Add'n, kg/t of ore CaO	Concentrate POX-Cyanidation Cons., kg/t of ore NaCN	Concentrate POX-Cyanidation Cons., kg/t of ore CaO	Concentrate POX-Cyanidation Extr'n Au	Concentrate POX-Cyanidation Residue Au, g/t	Tailing Cyanidation Test No.	Tailing Cyanidation Add'n, kg/t of ore NaCN	Tailing Cyanidation Add'n, kg/t of ore CaO	Tailing Cyanidation Cons., kg/t of ore NaCN	Tailing Cyanidation Cons., kg/t of ore CaO	Tailing Cyanidation Extr'n Au	Tailing Cyanidation Residue Au, g/t	Overall Au Recovery, % Conc	Overall Au Recovery, % Tail	Overall Au Recovery, % Comb. Rec'y	Comb Tail Au, g/t	Head (calc) Au, g/t	Head (direct) Au, g/t
A Zone	F-9	15.2	18.0	94.6	60	CIL2	0.16	0.95	0.03	0.94	79.8	3.72	CN-7	0.76	0.42	0.09	0.42	26.7	0.22	75.5	1.4	76.9	0.75	2.89	3.54
	F-9	15.2	18.0	94.6	90	CIL12	0.19	6.33	0.14	6.33	98.5	0.25	CN-7	0.76	0.42	0.09	0.42	26.7	0.22	93.2	1.4	94.6	0.22		
	F-9	15.2	18.0	94.6	120	CIL8	0.20	9.27	0.05	9.27	97.6	0.31	CN-7	0.76	0.42	0.09	0.42	26.7	0.22	92.3	1.4	93.8	0.23		
	F-9	15.2	18.0	94.6	60**	CIL10	0.19	0.67	0.02	0.65	91.7	1.46	CN-7	0.76	0.42	0.09	0.42	26.7	0.22	86.7	1.4	88.2	0.41		
South Zone	F-10	12.7	9.76	91.8	60	CIL3	0.12	0.56	0.03	0.48	97.6	0.23	CN-8	0.71	0.37	0.10	0.37	56.3	0.06	89.6	4.6	94.2	0.08	1.35	1.55
	F-10	12.7	9.76	91.8	90	CIL13	0.13	0.83	0.02	0.83	98.9	0.11	CN-8	0.71	0.37	0.10	0.37	56.3	0.06	90.8	4.6	95.4	0.07		
RW Zone	F-12	19.9	15.8	94.4	60	CIL4	0.19	0.42	0.05	0.41	76.3	3.95	CN-10	0.57	0.69	0.02	0.62	38.6	0.13	72.0	2.2	74.2	0.89	3.33	3.32
	F-12	19.9	15.8	94.4	90	CIL15	0.23	8.88	0.18	8.88	97.0	0.46	CN-10	0.57	0.69	0.02	0.62	38.6	0.13	91.6	2.2	93.7	0.20		
	F-12	19.9	15.8	94.4	120	CIL9	0.17	5.97	0.03	5.97	97.5	0.31	CN-10	0.57	0.69	0.02	0.62	38.6	0.13	92.0	2.2	94.2	0.17		
	F-12	19.9	15.8	94.4	60**	CIL11	0.19	0.66	0.03	0.64	78.4	3.49	CN-10	0.57	0.69	0.02	0.62	38.6	0.13	74.0	2.2	76.2	0.80		
Donchester	F-11	18.2	10.1	91.3	60	CIL5	0.21	4.36	0.14	4.36	98.1	0.17	CN-9	0.69	0.55	0.18	0.55	43.3	0.11	89.6	3.8	93.3	0.12	2.01	2.25
North	F-11	18.2	10.1	91.3	90	CIL14	0.15	5.68	0.04	5.68	97.4	0.25	CN-9	0.69	0.55	0.18	0.55	43.3	0.11	88.9	3.8	92.7	0.14		
Donchester	F-13	18.5	7.22	92.7	60	CIL6	0.26	5.28	0.08	5.28	94.8	0.16	CN-11	0.75	0.44	0.24	0.44	26.2	0.09	87.9	1.9	89.8	0.10	1.44	1.29
South	F-13	18.5	7.22	92.7	90	CIL16	0.17	4.40	0.02	4.40	97.1	0.20	CN-11	0.75	0.44	0.24	0.44	26.2	0.09	90.0	1.9	91.9	0.11		
Central	F-14	22.3	8.92	84.7	60	CIL7	0.28	1.98	0.11	1.98	94.9	0.43	CN-12	0.55	0.51	0.04	0.48	50.0	0.28	80.4	7.7	88.0	0.31	2.33	2.33
Duparquet	F-14	22.3	8.92	84.7	90	CIL17	0.19	1.83	0.02	1.82	95.8	0.36	CN-12	0.55	0.51	0.04	0.48	50.0	0.28	81.1	7.7	88.8	0.30		
Main	F-17	23.9	9.27	87.9	60	CIL18	0.37	2.53	0.11	2.53	98.2	0.16	CN-12*	0.54	0.50	0.04	0.47	50.0	0.28	86.3	6.1	92.4	0.25		

**flot conc reground before POX

*CN12 from F-14 (not F-17)

Testwork was also performed on two samples of existing tailings on the Duparquet Property. The overall results for the flotation-POX/CIL flowsheet for both tailing samples are summarized in Table 13.2. The overall recovery of gold was 83.5% for sample T-1 and 93.3% for sample T-2.

Table 13.2 - Summary of overall results for the tailing samples

Sample	Flotation Concentrate				POX	Conc POX-Cyanidation			Tailing Cyanidation			Overall Recovery, %			Comb	Head	Head
	Test No.	Wt %	Assay Au, g/t	Rec'y %, Au	Time min	Test No.	Extr'n Au	Residue Au, g/t	Test No.	Extr'n Au	Residue Au, g/t	Conc Au	Tail Au	Comb. Rec'y	Tail Au, g/t	(calc) Au, g/t	(direct) Au, g/t
T-1 Tailing	F-18	24.7	3.59	72.1	90	CIL19	92.9	0.29	CN-19	59.1	0.22	67.0	16.5	83.5	0.24	1.23	1.18
T-2 Tailing	F-16	47.4	12.5	94.9	90	CIL1	97.3	0.30	CN-6	19.7	0.48	92.3	1.0	93.3	0.39	6.25	6.40

13.2.2 SGS Project Reference No. 13054-002 – Report 1 – September 27, 2012

SGS Project Reference No. 13054-002 – Report 1 includes the metallurgical testwork that was conducted on six flotation concentrate samples produced in the previous 13054 – 001 test program.

The first objective of this program was to attempt to reduce costs of the pressure oxidation (POX) and carbon-in-leach (CIL) process by optimizing conditions and reducing reagent requirements. Previous investigation showed high lime consumption in CIL which was attributed to the slow breakdown of basic iron sulphates produced during pressure oxidation. To address this problem, the POX products were kept at 95°C for 4 hours in what is known as a hot cure process allowing the precipitated basic iron sulphate to solubilise back into solution. By this approach, the lime consumption in the CIL circuit was reduced by up to 95%.

Acid additions in the pre-acidulation stage before pressure oxidation were reduced by approximately half (by 60 kg/t H_2SO_4) from the initial tests while maintaining the high degree of sulphide oxidation and high gold recoveries. In addition, the hot cure product solution was successfully used as the source of acid in the pre-acidulation step, eliminating the fresh acid requirement and the cost of neutralizing the POX product. Furthermore, the testwork showed similar gold recovery values in the range of 96-99% at a lower POX temperature of 210°C compared to 225°C applied in previous testwork. This would result in additional savings in pressure oxidation costs.

The second objective was to generate final products for environmental studies. This included the flotation tailings, detoxified CIL pulp and hot cure neutralization sludge with each sample to be evaluated separately as well as a combined tailing product which included all three tailing streams.

The results of the POX and CIL tests in terms of sulphide oxidation and gold recovery as well as reagent consumption are presented in Table 13.3.

Table 13.3 - Summary of pressure oxidation – CIL testwork

Sample	Conc S^{2-}%	POX Test No.	Pre-acidulation H_2SO_4 Add'n, kg/t fresh	Pre-acidulation H_2SO_4 Add'n, kg/t recycled	S^{2-} oxid'n in POX %	Hot Cure h	Carbon-in-leach Results: Test No.	Reag. Cons., kg/t NaCN	Reag. Cons., kg/t CaO	% Extr'n Au	Residue Au. g/t	Head (calc) Au, g/t
A Zone	6.54	POX21	121.2		99.3	0	CIL21	0.32	53.2	98.1	0.3	17.4
		POX27	107.8		99.4	4	CIL27	0.29	2.7	98.3	0.31	16.3
		POX33	58.8		99.4	4	CIL33	0.24	19.5	98.1	0.41	17.9
		POX36		58.8	99.4	4	CIL36	0.2	5.5	98.1	0.38	17.8
South Zone	4.15	POX22	99.1		99.0	0	CIL22	0.22	4.1	98.9	0.12	10.4
		POX28	94.8		99.3	4	CIL28	0.26	4.5	98.5	0.14	9.7
		POX38		47.4	99.3	4	CIL38	0.14	1.5	97.8	0.22	9.4
RW Zone	8.75	POX24	98.1		99.4	0	CIL24	0.46	53	97.9	0.3	15.3
		POX30	96.7		98.9	4	CIL30	0.26	3.6	97.1	0.44	14.3
		POX39		48.4	99.3	4	CIL39	0.14	2	97.8	0.38	14.5
Donchester N	6.38	POX23	167.1		98.8	0	CIL23	0.35	21.9	96.2	0.35	9.1
		POX29	157.4		99.1	4	CIL29	0.19	3.1	97.8	0.22	9.6
		POX37		100	98.8	4	CIL37	0.16	6.7	98.1	0.18	9.2
Donchester S	6.19	POX25	129.7		99.1	0	CIL25	0.29	18.3	97.9	0.16	7.5
		POX31	119.8		98.8	4	CIL31	0.26	4.4	97.5	0.18	6.6
		POX40		59.6	99.1	4	CIL40	0.13	1.7	97.7	0.16	6.5
Duparquet Central	4.8	POX26	38.6		99.0	0	CIL26	0.22	8	94.5	0.43	7.5
		POX32	34.7		97.2	4	CIL32	0.28	3.1	95.5	0.38	8.5
		POX41		17.4	99.1	4	CIL41	0.1	1.4	96.4	0.34	8.8

13.2.3 Pressure oxidation optimization and cyanidation testwork

The pressure oxidation tests were carried out in a 2 L Parr autoclave. For initial testing, the concentrates were pre-acidulated at pH 1.8 with sulphuric acid for 120 minutes as in the previous work. The POX temperature was lowered to 210°C from 225°C. The oxygen overpressure was 100 psi during the 90 minute residence time. As per established practice, the pulp density used for each sample was calculated based on its sulphide content. The autoclave discharge was filtered and washed. The residue was repulped to 40% solids and cyanide leached for 24 hours maintaining 0.5 g/L NaCN and pH 10.5 with lime in the presence of 10 g/L activated carbon. The results are summarized in Table 13.4..

Table 13.4 - Effect of oxidation temperature

Sample	Conc S^{2-} %	POX Test No.	Pre-acidulation H_2SO_4 Add'n, kg/t		S^{2-} oxid'n in POX	Hot Cure	Carbon-in-leach Results					
			fresh	recycled	%	h	Test No.	Reag. Cons., kg/t NaCN	CaO	% Extr'n Au	Residue Au. g/t	Head (calc) Au, g/t
A Zone	6.54	POX21	121.2		99.3	0	CIL21	0.32	53.2	98.1	0.3	17.4
		POX27	107.8		99.4	4	CIL27	0.29	2.7	98.3	0.31	16.3
		POX33	58.8		99.4	4	CIL33	0.24	19.5	98.1	0.41	17.9
		POX36		58.8	99.4	4	CIL36	0.2	5.5	98.1	0.38	17.8
South Zone	4.15	POX22	99.1		99.0	0	CIL22	0.22	4.1	98.9	0.12	10.4
		POX28	94.8		99.3	4	CIL28	0.26	4.5	98.5	0.14	9.7
		POX38		47.4	99.3	4	CIL38	0.14	1.5	97.8	0.22	9.4
RW Zone	8.75	POX24	98.1		99.4	0	CIL24	0.46	53	97.9	0.3	15.3
		POX30	96.7		98.9	4	CIL30	0.26	3.6	97.1	0.44	14.3
		POX39		48.4	99.3	4	CIL39	0.14	2	97.8	0.38	14.5
Donchester N	6.38	POX23	167.1		98.8	0	CIL23	0.35	21.9	96.2	0.35	9.1
		POX29	157.4		99.1	4	CIL29	0.19	3.1	97.8	0.22	9.6
		POX37		100	98.8	4	CIL37	0.16	6.7	98.1	0.18	9.2
Donchester S	6.19	POX25	129.7		99.1	0	CIL25	0.29	18.3	97.9	0.16	7.5
		POX31	119.8		98.8	4	CIL31	0.26	4.4	97.5	0.18	6.6
		POX40		59.6	99.1	4	CIL40	0.13	1.7	97.7	0.16	6.5
Duparquet Central	4.8	POX26	38.6		99.0	0	CIL26	0.22	8	94.5	0.43	7.5
		POX32	34.7		97.2	4	CIL32	0.28	3.1	95.5	0.38	8.5
		POX41		17.4	99.1	4	CIL41	0.1	1.4	96.4	0.34	8.8

The results showed that lowering the POX temperature to 210°C did not significantly affect gold recovery. All further pressure oxidation testwork was performed at this reduced temperature.

Subsequent testing was conducted to reduce reagent costs for the POX-CIL flowsheet. The high lime consumption for most of the samples was indicative of the formation of basic iron sulphate (BFS) during the oxidation process. BFS slowly breaks down at the high pH during CIL, generating acid which requires neutralization. In order to overcome this, a hot cure stage was added. In the hot cure, the autoclave discharge was maintained at 95°C to allow the basic iron sulphate to re-dissolve at a much faster rate. After 4 hours, the pulp was filtered and washed and the solids were repulped for CIL as before. A test was conducted on each sample with the addition of the hot cure stage.

Additional tests were performed to reduce the acid requirement in the pre-acidulation stage. In tests POX33 and POX34 conducted on the A Zone concentrate, the fresh acid addition was reduced. Then the effect of recycling acid from the hot cure solution to replace the fresh acid requirement was investigated. Based on the results of POX-CIL33, the addition of acid was reduced by 50% up to a maximum of 60 kg/t and the required acid addition was provided by recycling hot cure solution from a previous test on the same sample. The results are summarized in Table 13.5 and Table 13.6.

The addition of the hot cure stage was effective in reducing the lime requirement in CIL. Lime consumption was decreased by as much as 95% (A Zone) after hot curing. The recovery of gold remained essentially the same and ranged from 95.5% to 98.5% (based on concentrate).

Sulphide oxidation was high in all tests except POX34. Reducing the acid addition by 50% up to 60 kg/t did not significantly affect gold recovery. Using the hot cure solution as the source of acid for pre-acidulation was effective for all samples eliminating the need for fresh acid. Reducing acid addition further to 75% in test POX34 conducted on the A Zone concentrate did not result in adequate destruction of carbonates during pre-acidulation so that oxidation of the sulphides was not achieved under the conditions applied.

The recovery of gold from the pilot plant hot cure discharge ranged from 94.7% to 96.5% with gold recovery slightly higher for tests conducted after 90 minute POX time (96.2%, Au in residue 0.96 g/t) than those tests conducted after the 60 minute POX time (95%, 1.21 g/t Au in residue). No testwork was conducted to determine the nature of the gold losses in the pilot plant residue samples.

Lime boil was conducted at 95°C for 4 hours with the addition of 120 kg/t CaO for two tests (CN15, 24) and at 75 kg/t CaO in a third test (CN25) to break down the jarosite and render silver recoverable by cyanidation. The addition of the lime boil resulted in an increase in the silver recovery from 9.2% to 95.5% as well as an increase in gold recovery from 95.5% to 98.1%. Reducing the lime dosage below 75 kg/t resulted in a reduction in gold and silver extraction. Optimum dosage appeared to be 75 kg/t of lime.

Cyanidation of batch pressure oxidation tests (e.g., CN5-6) yielded better gold recovery of ~98% and sulphur oxidation >99% than the pilot plant operation. Visual observations indicated that the pilot plant product contained more basic iron sulphate and jarosite and less hematite than the lab tests. The reason for the slightly lower gold recovery in the tests conducted on the pilot plant product may be due to differences in the precipitate produced in the pilot plant compared to the batch pressure oxidation tests. The acidity of the solution and the concentration of cations such as Na^+, and K^+ will impact whether hydrolysis reactions favour haematite or jarosite formation.

There is insufficient data to determine the relationship between sulphide oxidation and gold recovery and whether full sulphide oxidation is required to recover the majority of the gold. Earlier batch testwork showed a direct relationship between sulphide oxidation and gold extraction. Hence, a high overall sulphide oxidation target was maintained.

Table 13.5 - Summary of pressure oxidation and hot cure results

Sample	Conc	POX	Pulp	Pre-acidulation			S^{2-} Oxid'n	POX Solution Assays					POX Residue Assays			Hot	HC Solution		HC Residue	
	S^{2-}	Test	Density	pH	H_2SO_4 Add'n, kg/t		in POX	emf	Free acid	As	Fe	Fe^{2+}	As	S^{2-}	SO_4^{2-}	Cure	Free acid	Fe	SO_4^{2-}	Fe
	%	No.	% solids		fresh	recycled	%	mV	g/L	mg/L	mg/L	mg/L	%	%	%	h	g/L	mg/L	%	%
A Zone	6.54	POX21	34	1.8	121		99.3	552	48	296	7900	53	0.23	0.05	15.0	0				
		POX27	35	1.8	108		99.4	551	45	443	15000	195	0.22	<0.05	16.2	4	31	29900	14.0	3.2
		POX33	35	3.8	58.8		99.4	526	42	369	13200	268	0.26	<0.05	12.0	0				
		POX34	35	7.2	29.4		35.6	212	0	<1	<1	<1	0.20	4.39	8.0	4	0	<1	7.1	7.6
		POX36	35	3.8		58.8	99.4	539	48	377	15900	176	0.26	0.05	10.0	4	34	30100	9.8	6.0
South	4.15	POX22	41	1.8	99.1		99.0	550	48	517	5330	33	0.23	0.08	8.0	0				
		POX28	42	1.8	94.8		99.3	564	44	439	6730	43	0.25	<0.05	10.2	4	38	12200	10.4	3.2
		POX38	42	2.7		47.4	99.3	530	37	123	5110	119	0.37	<0.05	7.0	4	29	10200	8.7	6.6
RW Zone	8.75	POX 24	30	1.8	98.1		99.4	531	48	246	7790	946	0.29	0.06	16.0	0				
		POX 30	30	1.8	96.7		98.9	542	44	352	11400	82	0.27	0.12	13.0	4	36	21100	11.0	7.3
		POX 39	30	2.7		48.4	99.3	527	45	342	14900	369	0.34	0.08	10.0	4	35	21100	10.0	9.8
Donchester N	6.38	POX 23	41	1.8	167		98.8	447	52	285	17000	465	0.09	0.09	17.0	0				
		POX 29	42	1.8	157		99.1	510	46	244	16100	332	0.11	0.07	15.0	4	34	30100	15.0	3.4
		POX 37	42	3.6		100	98.8	517	43	174	18700	351	0.17	0.09	13.0	4	28	25900	14.0	7.3
Donchester S	6.19	POX 25	38	1.8	130		99.1	490	49	681	12600	117	0.45	0.07	11.0	0				
		POX 31	39	1.8	120		98.8	512	40	1030	17900	388	0.40	0.09	9.9	4	34	24700	9.0	5.6
		POX 40	39	2.6		59.6	99.1	506	38	432	11900	214	0.59	0.07	NA	4	29	18000	3.8	8.6
Duparquet	4.80	POX 26	38	1.8	38.6		99.0	500	42	164	5300	56	0.08	0.06	4.7	0				
		POX 32	40	1.8	34.7		97.2	539	39	283	8470	78	0.08	0.15	4.9	4	36	11300	4.2	4.63
		POX 41	40	2.7		17.4	99.1	527	39	140	6860	49	0.10	0.05	NA	4	38	9020	3.3	5.38

Table 13.6 - Summary of cyanidation tests after pressure oxidation

Sample	POX Test No.	Pre-acidulation			S^{2-} Oxid'n in POX	Hot Cure	Carbon-in-leach Results							
		pH	H_2SO_4 Add'n, kg/t				Test	Reagent Add'n., kg/t		Reagent Cons., kg/t		% Extr'n	Residue	Head (calc)
			fresh	recycled	%	h	No.	NaCN	CaO	NaCN	CaO	Au	Au, g/t	Au, g/t
A Zone	POX21	1.8	121		99.3	0	CIL21	1.21	53.21	0.32	53.2	98.1	0.30	17.4
	POX27	1.8	108		99.4	4	CIL27	0.98	2.77	0.29	2.7	98.3	0.31	16.3
	POX33	3.8	58.8		99.4	0	CIL33	0.99	19.57	0.24	19.5	98.1	0.41	17.9
	POX34	7.2	29.4		35.6	4	CIL34	0.97	0.88	0.23	0.8	31.0	13.0	18.8
	POX36	3.8		58.8	99.4	4	CIL36	0.84	5.53	0.20	5.5	98.1	0.38	17.8
South	POX22	1.8	99.1		99.0	0	CIL22	0.91	4.10	0.22	4.1	98.9	0.12	10.4
	POX28	1.8	94.8		99.3	4	CIL28	1.00	4.55	0.26	4.5	98.5	0.14	9.7
	POX38	2.7		47.4	99.3	4	CIL38	0.87	1.58	0.14	1.5	97.8	0.22	9.4
RW Zone	POX 24	1.8	98.1		99.4	0	CIL 24	1.33	53.05	0.46	53.0	97.9	0.30	15.3
	POX 30	1.8	96.7		98.9	4	CIL 30	0.99	3.68	0.26	3.6	97.1	0.44	14.3
	POX 39	2.7		48.4	99.3	4	CIL 39	1.00	2.02	0.14	2.0	97.8	0.38	14.5
Donchester N	POX 23	1.8	167		98.8	0	CIL 23	0.99	21.91	0.35	21.9	96.2	0.35	9.1
	POX 29	1.8	157		99.1	4	CIL 29	0.96	3.16	0.19	3.1	97.8	0.22	9.6
	POX 37	3.6		100	98.8	4	CIL 37	0.80	6.74	0.16	6.7	98.1	0.18	9.2
Donchester S	POX 25	1.8	130		99.1	0	CIL 25	1.01	18.34	0.29	18.3	97.9	0.16	7.5
	POX 31	1.8	120		98.8	4	CIL 31	1.00	4.40	0.26	4.4	97.5	0.18	6.6
	POX 40	2.6		59.6	99.1	4	CIL 40	0.97	1.76	0.13	1.7	97.7	0.16	6.5
Duparquet	POX 26	1.8	38.6		99.0	0	CIL 26	0.98	8.00	0.22	8.0	94.5	0.43	7.5
	POX 32	1.8	34.7		97.2	4	CIL 32	0.99	3.15	0.28	3.1	95.5	0.38	8.5
	POX 41	2.7		17.4	99.1	4	CIL 41	0.88	1.43	0.10	1.4	96.4	0.34	8.8

Table 13.7 - Summary of batch cyanidation tests on hot cure discharge

Test No	Feed		POX residence time (Minutes)	Lime boil (Min)	Pulp density (% solids)	Reagent Add'n NaCN (kg/t*)	Reagent Add'n CaO (kg/t*)	Reagent Concs NaCN (kg/t*)	Reagent Concs CaO (kg/t*)	Extraction Au (%)	Extraction Ag (%)	Residue assays Au (g/t)	Residue assays Ag (g/t)	Residue assays S (%)	Calculated head Au (g/t)	Calculated head Ag (g/t)
CN5	Hot cured autoclave residue	Lab	90	0	36	1.37	4.55	0.24	4.53	98.9	…	0.37	…	0.12	27.1	…
CN6		Lab	90	0	36	1.19	5.44	0.2	5.39	98.4	…	0.5	…	0.08	26.2	…
CN9		Pilot	90	0	35	1.08	7.86	0.17	7.84	96.5	…	0.84	…	0.38	24.3	…
CN10		Pilot	90	0	40	1.01	4.41	0.20	4.34	96.2	…	0.97	…	0.14	25.8	…
CN11		Pilot	90	0	44	0.72	4.08	0.13	4.01	96.0	…	1.01	…	0.43	25.1	…
CN12		Pilot	90	0	39	0.89	3.83	0.10	3.74	96.0	…	1.02	…	0.22	25.8	…
CN13		Pilot	60	0	45	0.96	4.91	0.08	4.83	95.5	…	1.17	…	0.72	26.4	…
CN14		Pilot	60	0	33	1.10	4.57	0.13	4.47	95.5	9.2	1.27	32.8	0.43	28.5	36.3
CN15		Pilot	60	120	25	1.47	119	0.16	119.0	98.1	95.5	0.52	1.6	0.08	31.8	41.2
CN16	Thk u/f shutdown	Pilot	60	0	35	0.85	3.97	0.05	3.92	96.0	…	1.22	…	0.11	30.5	…
CN17	HC4 pump tank out	Pilot	60	0	39	0.84	3.24	0.35	3.19	94.7	…	1.19	…	0.27	22.5	…
CN23	PP HC Thk u/f			0	30	1.24	4.1	0.23	4.0	96.4	9.3	1.06	32.4		28.6	35.4
CN24	PP HC Thk u/f			120	30	1.41	110.2	0.25	110.1	98.6	95.3	0.36	1.5		28.4	34.6
CN25	PP HC Thk u/f			75	30	1.35	78.6	0.42	78.6	98.3	89.9	0.47	3.4		28.1	35.7

13.2.3.1 Gold leaching and carbon adsorption testwork

Leach kinetic tests were carried out to determine the rate of gold leaching on a washed hot cure thickener underflow composite. The leach testwork was conducted by bottle roll tests. The leach was conducted at 35% solids density as directed by Clifton Star. The NaCN concentration was maintained at 0.5 g/L and the pH at 10.5 with lime. Each test was carried out for 48 hours with kinetic subsampling at 1, 2, 4, 8, 12 and 24 hours.

To obtain absorption loading kinetics and capacity data the cyanided leach pulps that were generated during the leach kinetic testing were contacted with predetermined amounts of activated carbon in batch tests and bottle rolled for 72 hours. Solution samples were taken for gold analysis at specific time intervals. The data were modelled using an SGS in house program which uses semi-empirical models developed by Mintek in the 1980s to predict operation and plant performance. Equilibrium isotherms for gold cyanide loading on activated carbon were generated and fitted to a Freundlich isotherm non-linear regression.

Leaching kinetics were fast with gold leaching largely complete within the first two hours as illustrated in Figure 13.1.

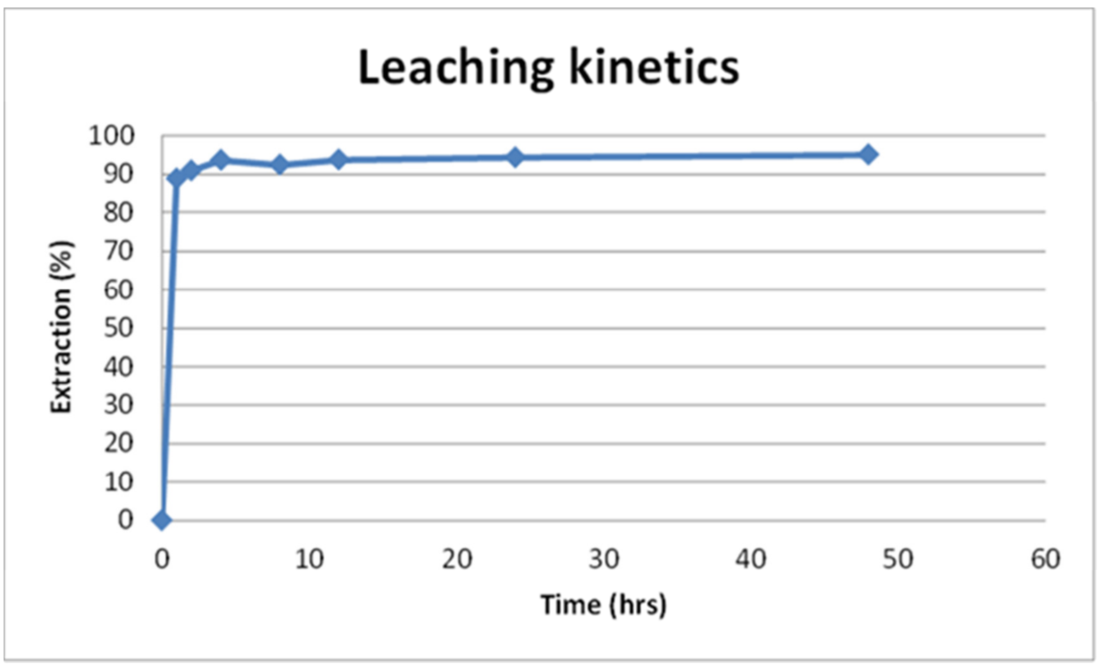

Figure 13.1 - Cyanidation Bottle Roll test results showing rapid gold leaching kinetics

The gold adsorption isotherm plot with the predicted curve fit is shown in Figure 13.2. The adsorption equilibrium constant (Freundlich constant ‘a’) determined for the Duparquet washed solids hot cure thickener underflow was high at 13,513 and is attributed to the high gold grade of the residue and purity of the water.

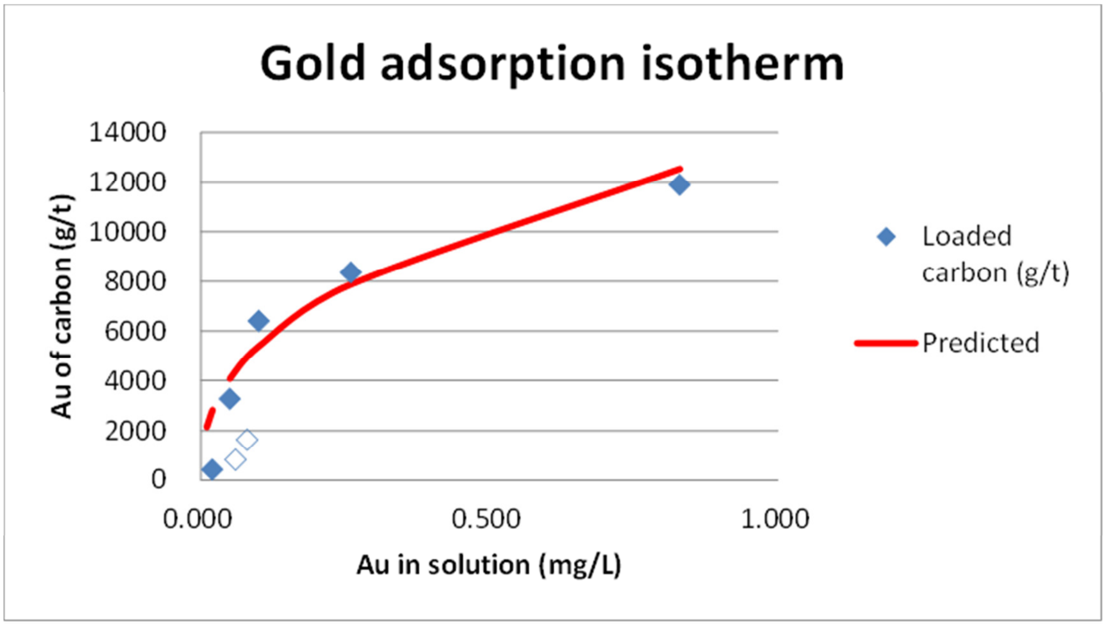

Figure 13.2 - Gold adsorption isotherm

SGS modelled the leaching and adsorption kinetics for gold and proposed a leaching and carbon in pulp configuration for recovering the gold based on the data from the testwork sample, but no data were provided for silver leaching or adsorption kinetics. Based on the test results listed in Table 13.7 above, extractable gold and silver yields are expected to increase following lime boil from 95% to 98% for gold and from 9% to ≥89% for silver.

The increased extraction of silver will impact the precious metal loading rates and capacity on carbon. Silver loads onto carbon more slowly than gold but elutes and electrowins more rapidly. Hence it will also impact carbon in pulp, elution and electrowinning design.

The leaching and carbon in pulp circuit were remodelled by Tenova Mining & Minerals–Bateman Engineering Pty Ltd ("Tenova-Bateman") using the SIMCIL program developed in the AMIRA P420 project to take into account silver. The throughput and pulp density have also been adjusted in the SIMCIL model to reflect expected conditions in the commercial plant. Both SGS and SIMCIL model conditions are presented in Table 13.8.

Gold and silver leach rates were assumed to be similar at 0.8 t/h.g, and an adsorption rate of 0.010/hour. Silver leaching rates are normally slower than gold but as the silver is liberated from jarosite following lime boil it may be assumed that leaching will take place at a similar rate to gold. The Freundlich exponent constant 'a' of 13,513 determined by SGS was used and the 'b' value set at 1.0 for silver modelling. The SIMCIL modelled output is shown in Table 13.9. The configuration of two leach tanks followed by six stages of carbon contacting was used in the SIMCIL model, similar to that utilised in the SGS model. The increased precious metal loadings on carbon necessitated almost doubling the carbon transfer rate from 1.5 tpd to 2.5 tpd.

Further work is required to investigate leaching and adsorption kinetics on lime boiled samples to determine the impact of liberated silver on precious metal leaching and adsorption kinetics to ensure cyanidation extraction and recovery circuit configurations are correctly designed.

Table 13.8 - Parameters modelling cyanide leach and adsorption circuit

Inputs	SGS optimized regime	Modified by Tenova-Bateman for throughput and silver extraction
Slurry feed rate (m^3/h)	45.9	50.5
Solids (t/h)	21.5	32
Solution (m^3/h)	39.9	38.7
Consider leach after carbon addition	N	N
Ore head grade Au (g/t)	26.5	17.2
Ore head grade Ag (g/t)		28.8
Gold on stripped carbon (g/t)	50	50
Silver on stripped carbon (g/t)		250
Adsorption tank(s) size	120	154
Carbon frequency advance (% in 24hours)	50%	73%
Leaching		
Au leached	94.6%	98.0%
Ag leached		89.0%
Leach time before carbon addition (h)	24	24
Leach only total tankage (m^3)	1,102	1,290
Number of leaching tanks	2	2
Volume of leaching tanks (m^3)	551	645
CIP		
Number of stages	6	6
Total CIP/CIL volume (m^3)	720	924
Slurry residence time in each adsorption tank (h)	2.6	3.1
Gold grade in residue (g/t)	1.45	0.29
Silver grade in residue (g/t)		2.91
Gold in final barren solution (mg/L)	0.004	0.005
Silver in final barren solution (mg/L)		0.064
Gold in loaded carbon (g/t)	8,838	4,630
Silver in loaded carbon (g/t)		7,273
Carbon residence time/stage (h)	48	33
Carbon Concentration (g/L Solution)	25	25
Equivalent transferred carbon unit flowrate (kg/h)	63	117
Daily carbon transfer / batch elution capacity (kg/day)	1,500	2,800
Carbon Inventory per stage (kg)	3,000	3,850
Carbon inventory all stages (tons)	18	23.1
Gold on barren carbon (g/t)	34	50
Silver on barren carbon (g/t)		260
CIP gold recovery per day (g/day)	13,183	12,964
CIP silver recovery per day (g/day)		20,364
Au&Ag in loaded carbon / Au&Ag in feed	327	259

Table 13.9 - Output of modelling leaching and adsorption circuit with SIMCIL program

Stage	Gold				Silver			
	Solution Au (g/t)	Carbon Au (g/t)	Ore Au (g/t)	Gold ext'n (%)	Solution Ag (g/t)	Carbon Ag (g/t)	Ore Ag (g/t)	Silver ext'n (%)
Feed	0	-	17.15	-	0	-	28.8	-
1	13.112	-	1.52	-	20.431	-	4.45	-
2	13.892	-	0.59	-	21.434	-	3.25	-
3	1.653	4,630	0.49	86	7.038	7,273	3.13	60
4	0.212	665	0.44	72	2.308	2,698	3.07	49
5	0.033	144	0.41	36	0.736	1,049	3.03	33
6	0.01	72	0.38	11	0.250	495	3.01	15
7	0.006	59	0.37	5	0.107	320	2.99	6
8	0.005	53	0.36	3	0.064	266	2.98	2

13.2.4 Neutralization testwork

Five neutralization tests were performed to determine the quantity of flotation tailings, limestone ($CaCO_3$) and lime (CaO) required to neutralize the POX & hot cure solution. The results of testwork investigating limestone/ lime and flotation tails/lime combinations are shown in Table 13.10.

Table 13.10 - Results of neutralization testwork on CCD overflow liquors

Test No	Reagent addition								Solution analyses mg/L		% solids w/w
	Limestone addition			Rougher tails to pH 4			CaO to pH 8				
	pH	g/L	kg/t	pH	g/L	kg/t	g/L	kg/t	As	Fe	
NT-1	4.9	137	579	-	-	-	7.8	33	0.1	0.07	18.7
NT-2	-	-		4.1	727	3,072	37.4	158	0.11	<0.05	43.2
NT-3	5.1	133	561	-	-	-	8.13	34.3	0.1	0.4	33
NT-5	-	-	-	4.4	832	3,516	27	114	0.18	<0.2	44.1
NT-6	5.6	110	465	-	-	-	8.17	34.5	0.08	0.08	15
NT-7	5.1	106	446**	-	-	-	7.4	31.3	0.08	0.08	19

** kg per tonne of hot cure discharge solids (4.225 L of HC solution per kg of HCD solids)*
*** 2/3 mine site limestone (38% CO_3) + 1/3 reagent grade limestone (97.6% $CaCO_3$)*

A combination of limestone and lime successfully reduced the arsenic and iron level in solution to below 0.1 mg/L. The limestone used in this study was classified as a mine site limestone with a grade of 38% carbonate content. Assuming all the carbonate is available for dissolution this is equivalent to between 176.7 and 220 kg of pure calcium carbonate per t. of hot cure discharge solids. For the final test (NT-7) where a combination of mine site limestone and reagent grade limestone was used, a higher consumption was observed and was equivalent to 258 kg of pure calcium carbonate per t. of hot cure discharge solids.

A combination of flotation tailing and lime additions (NT 2 and NT5) was also investigated but it was found to be an ineffective method for neutralizing the CCD overflow liquor due to the large quantity of flotation tailing required to reduce the pH to 4.0. In addition, slightly elevated concentrations of As, Mn and Sr were observed in the final effluent neutralized with flotation tailings and lime.

More testwork is required and should be carried out on limestone sourced for the Project. In particular, tests should be performed to determine optimum particle size of the limestone, reaction kinetics and required dosage rates.

After filtering the hot cure product, the solutions for each sample were neutralized using limestone and lime. The purpose was to prepare the neutralized pulp for subsequent environmental studies. $CaCO_3$ was initially added to increase the pH to 4.5. The slurry was conditioned for 60 minutes. Hydrated lime was then added to increase the pH to 8. The test data are shown in Table 13.11.

Table 13.11 - Neutralization test results

Sample	Test No.	Reagent Addition				Solution Analysis*		Product	
		$CaC0_3$ to pH 4.6		CaO to pH8		mg/L		precipitate	density
		g/L	kg/t	g/L	kg/t	As	Fe	kg/t	% solids
A Zone	Neut 36	135	241	12	21	0.02	<0.03	591	25
South RW	Neut 38	65	85	10	13	0.02	<0.03	241	15
Zone	Neut 39	89	169	8	15	0.04	<0.03	476	17
Donchester N	Neut 37	106	161	14	22	0.02	<0.03	391	24
Donchester S	Neut 40	81	127	11	16	0.05	<0.03	327	17
Duparquet	Neut 41	84	126	7	11	0.03	<0.03	275	16

**solution filtered on 0.45 μm filter paper*

The sequential neutralization testwork with limestone and lime successfully demonstrated that the arsenic and iron levels in thickener overflow solutions can be reduced to below 0.1 mg/L. The Fe^{3+}/As ratio in the hot cure discharge liquor is suitably high to favour the effective stabilisation of arsenic as an arsenate upon neutralization.

However the dosage and source of limestone need to be re-evaluated. The neutralization capacity of flotation tailings is insufficient to be used as a first step neutralization agent and hence was not included as a neutralizing agent in the process design.

13.2.5 Cyanide destruction testwork

The SO_2/air method was used to destroy the cyanide in the CIL tailings. Batch tests were conducted at pH 8.5 to lower the weak acid dissociable cyanide (CN_{WAD}) level in the pulp to approximately 1 mg/L. The required amount of copper as copper sulphate was added to catalyse the cyanide oxidation reaction at the start of the test. SO_2 was added continuously as a sodium metabisulphite solution. After 1 hour, the barren solution was sampled to determine the CN_{WAD} level by picric acid. If the analysis was >1 mg/L, the test was continued. The results are presented in Table 13.12.

Table 13.12 - Summary of cyanide destruction tests

Sample	Test No.	Retention Time h	Reagent Addition g/g CN_{WAD} SO_2	CaO	Cu	g/L Feed Pulp SO_2	CaO	Cu	CND Barren Solution CN_T mg/L	CN_{WAD} mg/L	Cu mg/L	Fe mg/L
A Zone	CND36	60	4.75	0	0.04	0.50	0	0.004	6.45	1.09	0.17	2.08
South RW	CND38	180	18.58	2.2	0	2.07	0.25	0	1.70	0.13	<0.05	0.69
Zone	CND39	60	6.33	0.8	0.04	0.65	0.08	0.004	0.77	0.77	0.08	0.34
Donchester N	CND37	90	11.88	1.3	0.04	1.04	0.11	0.004	8.25	1.32	0.05	2.70
Donchester S	CND40	120	12.67	1.5	0.03	1.49	0.18	0.004	0.79	0.62	0.26	0.28
Duparquet	CND41	120	12.67	0.3	0.03	1.58	0.04	0.004	0.29	0.22	3.57	0.12

Although the CN_{WAD} level was reduced to<1 mg/L, the total cyanide (CN_T) level was significantly higher for the A Zone and Donchester N samples due to the presence of ferrocyanide. It should be noted that batch tests generally require higher reagent additions than required in continuous operation. Continuous testing is required to optimize the cyanide destruction conditions.

13.2.6 Preparation of samples for environmental characterization

Four samples were prepared for environmental characterization. Three of the samples represented the individual tailing streams namely the flotation tailing, the cyanide destruction barren pulp and the POX neutralized liquor and sludge. The fourth sample represented the combined tailing. It was prepared by combining the individual streams in the appropriate relative amounts based on the metallurgical test results. The details of sample preparation and the results of the environmental tests will be presented and discussed in a separate report and are presented below.

13.2.7 SGS Project Reference No. 13054-002 – Report 2 – November 29, 2012

SGS Project Reference No. 13054-002 – Report 2 includes the environmental characterization of the flotation rougher tailings, cyanide destruction barren pulp, neutralization sludge, and the combined tailings products from the six ore samples. Environmental tests were conducted on the twenty-four pulp samples from metallurgical testwork programs (SGS Project Reference No. 13054 – 002 Report 1).

Test Methods

The following sections provide a brief overview of the test methods included in the environmental characterization program.

X-ray Fluorescence (XRF) Whole Rock Analyses

Whole rock analyses were completed on the samples using XRF in order to determine the elemental concentrations of the major rock forming constituents. This method quantifies major elements present and reports them as oxides to permit a mass balance assessment against the component of a sample that is amenable to oxidization (loss on ignition).

ICP-OES/MS Strong Acid Digest Elemental Analyses

The samples were digested using an acid mixture of HNO_3, HF, $HClO_4$, and HCl to obtain a near total dissolution of the elements being analyzed. ICP-OES/MS trace metal scans were performed to provide quantitative analyses of the elemental components of the sample material. Analyses requested included: Ag, Al, As, B, Ba, Be, Bi, Ca, Cd, Co, Cr, Cu, Fe, Hg, K, Li, Mg, Mn, Mo, Na, Ni, P, Pb, Sb, Se, Si, Sn, Sr, Th, Ti, Tl, U, V, W, Y, and Zn. Mercury analyses were completed by cold vapour atomic absorption spectroscopy (CVAAS).

Toxicity Characteristic Leaching Procedure (TCLP- US EPA Method 1311)

The TCLP was used to determine the mobility of inorganic contaminants present in the waste materials which would be chemically stable and insoluble under the aggressive pH environment imposed by the method.

A brief assessment of the sample's neutralization capacity determines the leachant used in the test. In cases where the sample has limited acid neutralization capacity, TCLP extraction fluid #1 is used. Extraction fluid #1 is a combination of glacial acetic acid and deionised water buffered by sodium hydroxide (resulting pH 4.93 ± 0.05). In cases where the residues have sufficient acid neutralization capacity, TCLP extraction fluid #2 is used. This extraction fluid provides 2 eq/kg of acid through addition of glacial acetic acid to deionised water (resulting pH 2.88 ± 0.05). The leachant is added to the sample at a 20:1 liquid-to-solids ratio and the sample container is rotated end over end at 29 ± 2 rpm for 18 hours. The resultant slurry is then filtered on 0.7µm filter, the extract pH is determined and the extract is analyzed for total metals (including Ag, Al, As, B, Ba, Be, Bi, Ca, Cd, Co, Cr, Cu, Fe, Hg, K, Li, Mg, Mn, Mo, Na, Ni, P, Pb, Sb, Se, Si, Sn, Sr, Th, Ti, Tl, U, V, W, Y, and Zn).

Modified Acid Base Accounting

The modified ABA test provided quantification of the total sulphur, sulphide sulphur, and sulphate concentrations present and the potential acid generation (AP) related to the oxidation of the sulphide sulphur concentration. The test method determined the neutralization potential (NP) of the samples by initiating a reaction with excess acid and then identified the quantity of acid neutralized by the samples, NP, by back-titrating to pH 8.3 with NaOH. The balance between the AP and NP assists in defining the potential of the sample to generate acid drainage. In addition, quantification of the carbonate mineral content permitted calculation of the theoretical carbonate NP.

Net Acid Generation Testing

The net acid generation (NAG) test facilitates a reaction between the sample and concentrated hydrogen peroxide in order to force complete oxidation and reaction of the acidity produced with the neutralizing minerals present within the sample. This test provided confirmation of data obtained in the ABA test. It also indicated the quantities of acid produced, and thus the amount of additional alkalinity, if any, needed to neutralize this acidity.

Decant Solution Analysis

The supernatants were decanted from the settled pulp slurries and analyzed to quantify contaminant concentrations that may report to surface or ground water systems in a tailings pond setting. The solutions were collected and submitted for chemical analysis of the following parameters: pH, alkalinity, acidity, conductivity, Eh, TDS, TSS, F^-, ammonia (NH_3+NH_4^+), Cl^-, SO_4^{2-}, NO_2^-, NO_3^-, total thiosalts, and total and dissolved metals as per the previously noted suite of parameters plus total hardness. In addition, only the cyanide destruction (CND) decant solutions were analyzed for total cyanide, weak acid dissociable cyanide, cyanate, and thiocyanate.

In conclusion

1. Whole rock and elemental analyses determined that the flotation, CND, and combined tailings samples were predominantly comprised of silicate with minor amounts of aluminium, iron, potassium, and calcium, while the neutralization samples were predominantly calcium and iron with very little silica, due a predominant gypsum composition.
2. Analysis of the Duparquet TCLP leachates reported all Directive 019 parameters were at concentrations within the prescribed limits.
3. Modified ABA testing classified the Duparquet flotation and combined tailings samples NPAG due to a readily available carbonate source of alkalinity and insignificant sulphide content, while the CND tailings had sufficient sulphide content to indicate that acid generation would be expected.
4. NAG testing confirmed the NPAG designation provided by the ASA for the flotation and combined tailings samples. The NAG analysis also suggests that the CND samples would not generate PAG conditions.
5. The Duparquet flotation tailings decant solutions were all found to have concentrations of contaminants that were within the regulatory limits.
6. All CND decant solutions, with the exception of the RW-39 CND Decant and Dup-41 CND Decant, had total cyanide concentrations in excess of the regulatory limit (1 ppm).
7. Each of the CND decant solutions were found to have dissolved arsenic concentrations an order of magnitude above the regulatory limit, while elevated iron levels were due to suspended solids.
8. The neutralization decants showed elevated arsenic and iron values but only exceeded the regulatory limits as total values indicating the contaminants were associated with the suspended solids and would be expected to settle out of the water column over time.
9. The combined tailings decant solutions also showed elevated arsenic and iron concentrations in the total solution analyses, with arsenic generally being above the Directive 019 limits.

13.3 Testwork Update and Pilot Plant

13.3.1 SGS Project Reference No. 13054-003 – Final Report – February 5, 2013

SGS Project Reference No. 13054-003 – Final Report summarizes the results of the flotation testwork, conducted to investigate the recovery of gold from six (6) ore samples. This program also included detailed concentrate analysis and cyanidation tests on the flotation tailings samples.

Cleaner flotation tests were conducted to investigate the recovery of the gold in a saleable sulphide concentrate. The rougher concentrate with P_{80} of ~100 µm was cleaned twice and a scavenger stage was applied following the first cleaner tailings. The results of the locked cycle tests conducted on each sample are presented in Table 13.13.

Table 13.13 - Flotation Locked-Cycle test results

Test No. / Sample	Grind P_{80}, µm	Product	Wt%	Assays		% Distribution	
				Au g/t	$S^=$ %	Au g/t	$S^=$ %
LCT1	115	2nd Cleaner Conc.	3.8	83.6	35.1	88.5	88.5
A Zone		1st Cleaner Scav. Tail	20.3	0.99	0.4	5.6	5.4
		Rougher Tail	75.9	0.27	0.12	5.9	6.1
		Head (Calculated)	100	3.55	1.49	100	100
LCT2	113	2nd Cleaner Conc.	1.9	67.5	27.5	84.5	88.6
South		1st Cleaner Scav. Tail	23	0.49	0.12	7.3	4.6
		Rougher Tail	75.1	0.17	0.05	8.3	6.8
		Head (Calculated)	100	1.54	0.6	100	100
LCT3	109	2nd Cleaner Conc.	6.3	50.2	29.7	87.3	92.2
RW Zone		1st Cleaner Scav. Tail	22.7	0.97	0.34	6.1	3.8
		Rougher Tail	71.1	0.33	0.11	6.6	4.0
		Head (Calculated)	100	3.6	2.02	100	100
LCT4	108	2nd Cleaner Conc.	3.2	56.7	35.9	83.1	86.6
Donchester N		1st Cleaner Scav. Tail	16.4	0.91	0.5	6.9	6.2
		Rougher Tail	80.5	0.27	0.12	10	7.1
		Head (Calculated)	100	1.43	0.87	100	100
LCT5		2nd Cleaner Conc.	2.9	39.5	32	82.9	81.2
Donchester S	108	1st Cleaner Scav. Tail	16.5	0.65	0.5	7.8	7.2
		Rougher Tail	80.6	0.16	0.16	9.4	11.5
		Head (Calculated)	100	1.38	1.14	100	100
LCT6	100	2nd Cleaner Conc.	4.8	39	20.9	75.5	83.5
Central		1st Cleaner Scav. Tail	25.1	1.02	0.37	10.2	7.7
Duparquet		Rougher Tail	70.7	0.51	0.15	14.2	8.9
Main		Head (Calculated)	100	2.49	1.21	100.2	100

The gold recovery ranged from 75.5% (Central Duparquet Main sample) to 88.5% (A Zone sample). The cleaner concentrate gold grade ranged from 39.0 to 83.6 g/t Au and the sulphur grade ranged from 20.9% to 35.9% S.

The rougher tailings and cleaner scavenger tailings from each locked cycle test were leached separately to investigate the gold extraction. Table 13.14 indicates the leaching efficiency of cyanidation of the flotation tailings. Depending on the sample, between 3.9% and 11.5% additional gold was leached by cyanidation of flotation tailings.

Table 13.14 - Cyanidation of flotation tailings

Test No. Sample	Product	Wt%	Assays Au, g/t	Distr'n Au,%	Extr'n Au,%	Overall Au Rec. %
LCT1 A Zone	1st Cleaner Scav. Tail	20.3	0.99	5.6	41.5	2.3
	Rougher Tail	75.9	0.27	5.9	27.3	1.6
South	1st Cleaner Scav. Tail	23	0.49	7.3	51.1	3.7
	Rougher Tail	75.1	0.17	8.3	45.5	3.8
RW Zone	1st Cleaner Scav. Tail	22.7	0.97	6.1	39.9	2.4
	Rougher Tail	71.1	0.33	6.6	28.9	1.9
Donchester N	1st Cleaner Scav. Tail	16.4	0.91	6.9	51.2	3.5
	Rougher Tail	80.5	0.27	10	40.3	0.4
Donchester S	1st Cleaner Scav. Tail	16.5	0.65	7.8	35.4	2.8
	Rougher Tail	80.6	0.16	9.4	20.4	1.9
Central Duparquet	1st Cleaner Scav. Tail	25.1	1.02	10.2	51.2	5.2
	Rougher Tail	70.7	0.51	14.2	44.3	6.3

The overall gold recovery ranged from 87.0% (Central Duparquet Main sample) to 92.4% (A Zone sample). The overall reagent consumptions in the cyanidation tests are given in Table 13.15.

Table 13.15 - Overall reagent consumptions for tailing cyanidation

Sample	Rougher Tailing				Cleaner Scavenger Tailing				Combined Tailing			
	Reagent Add'n, kg/t		Reagent Cons, kg/t		Reagent Add'n, kg/t		Reagent Cons, kg/t		Reagent Add'n, kg/t		Reagent Cons, kg/t	
	NaCN	CaO	NaCN	CaO	NaCN	CaO	NaCN	CaO	NaCN	CaO	NaCN	CaO
A Zone	0.58	0.41	0.06	0.41	0.38	0.19	0.1	0.19	0.96	0.60	0.16	0.60
South Zone	0.56	0.34	0.05	0.34	0.52	0.22	0.1	0.22	1.08	0.56	0.14	0.56
RW Zone	0.61	0.53	0.18	0.53	0.62	0.29	0.28	0.29	1.23	0.81	0.47	0.81
Donchester N	0.67	0.43	0.11	0.43	0.34	0.17	0.12	0.17	1.01	0.61	0.23	0.61
Donchester S	0.76	0.56	0.21	0.56	0.35	0.19	0.14	0.19	1.11	0.76	0.35	0.76
Duparquet	0.74	0.34	0.16	0.33	0.6	0.21	0.13	0.21	1.34	0.55	0.29	0.54

The cyanide (NaCN) consumption for leaching the flotation tailings varied between 0.14 kg/t and 0.47 kg/t. For the same tests, the lime (CaO) consumption varied between 0.54 kg/t and 0.81 kg/t.

13.3.2 SGS Project Reference No. 13054-004 – Grindability Report – May 21, 2013

SGS Project Reference No. 13054-004 – Grindability Report includes a series of grindability tests that have been conducted on a pilot plant (PP Feed) sample. The sample was full HQ core, consisting of about 12 tons of material, assumed to be representative of the Duparquet deposit. The results are summarized in Table 13.16.

Table 13.16 - Grindability test of pilot plant feed

Sample Name	Relative Density	JK Parameters			Locked-cycle		HPGR	Batch	CWI	RWI	BWI (kWh/t)		AI
		Axb[1]	Axb[2]	ta	kWh/t	N/mm²	ts/hm³	kWh/t	kWh/t	kWh/t	Feed	HPGR Prod	(g)
PP Feed	2.73	27.0	29.0	0.20	2.82	3.42	233	1.97	7.5	19.1	18.5	16.6	0.547
Bulk PP Feed*	-	-	-	-	-	-	-	-	-	-	17.6	-	-
DON-11-147M	2.74	-	34.4	-	-	-	-	-	-	18.4	18.8	-	0.472
BD-11-333M	2.68	-	25.0	-	-	-	-	-	-	18.2	18.5	-	0.668
BD-11-336M	2.53	-	25.9	-	-	-	-	-	-	18.5	20.3	-	0.618
CD-11-01MM	2.88	-	31.0	-	-	-	-	-	-	17.3	18.4	-	0.836

[1] A x b from DWT
[2] A x b from SMC
* Mill feed sample collected after the bulk sample crushing

The characterization of the composite sample included the following grindability tests:

- Bond low-energy impact test
- JKTech drop-weight test
- SMC test
- Bond rod mill grindability test (RWI)
- Bond ball mill grindability test (BWI)
- Bond abrasion test
- HPGR test

The Pilot Plant Feed sample was characterized as very hard with respect to both resistance to impact (Axb) and abrasion breakage (t_a), as well as in terms of RWI. The sample was characterized as hard with respect to the BWI and HPGR tests. The sample was also found to be very abrasive. All the results were in line with the six (6) previous samples, tested as part of SGS Project 13054-02.

13.3.3 SGS Project Reference No. 13054-004 – Report 1 – June 20, 2013

SGS Project Reference No. 13054-004 – Report 1 summarizes the results of the flotation tests conducted on a pilot plant (PP Feed) sample. The head sample was analyzed at 1.84 g/t Au, 1.16% S, 0.055% As and 7.61% CO_3. The pilot plant was operated to generate bulk sulphide flotation concentrate containing 15-18% S for a subsequent pressure oxidation pilot plant to assess gold recovery and to generate 60-80 kg of a higher grade flotation concentrate assaying over 40 g/t Au for direct sale market evaluation.

In the flotation pilot plant, the ore was ground to a P_{80} of 100 µm and a rougher concentrate was recovered with addition of PAX and R208 collectors. The rougher concentrate was reground in a ball mill, operating in closed circuit with a cyclone. Pilot plant tests PP-01 to PP-06 were conducted with one cleaning stage to generate the pressure oxidation pilot plant feed. Pilot plant tests PP-07 to PP-09 were conducted with two cleaning stages to generate the high grade concentrate. Tests PP-05 and PP-09 were operated over more than 8 hours under relatively stable conditions. The results of these tests are summarized in Table 13.17.

With one cleaning stage (PP-05) the recovery of gold was 91.7% in a concentrate containing 26.8 g/t Au and 16.1% S. The results of PP-09 indicated that a concentrate with 47.8 g/t Au could be produced at 86.5% gold recovery. Flotation tailings were leached with cyanide to recover the gold remaining in this material. The recovery of gold from the PP-05 tailings and the PP-09 tailing were 45.1% and 40.2%, respectively. The overall gold recovery by flotation and cyanidation of the flotation tailings is shown in Table 13.18.

Table 13.17 - Pilot plant flotation test results at steady state

Test	P80 µm	Product	Wt%	Assys, g/t,%		Distribution,%	
				Au	S	Au	S
PP5	99	PP Feed	100	1.81	1.09	100	100
		Rougher Feed	113.9	1.72	1.06	108.4	110.5
		Rougher Conc.	20.1	9.01	5.51	100.1	101.8
		Rougher Tail	93.8	0.16	0.1	8.3	8.6
		1st Cleaner Conc.	6.2	26.8	16.1	91.7	91.4
		1st Cleaner Tail	13.9	1.09	0.82	8.4	10.5
PP9	100	PP Feed	100	1.81	1.12	100	100
		Rougher Feed	108.1	1.91	1.22	113	117.1
		Rougher Conc.	12.4	17.1	11.1	99.5	104.3
		Rougher Tail	95.7	0.25	0.15	13.5	12.8
		1st Cleaner Feed	20.6	16.6	10.9	106.4	112.5
		1st Cleaner Conc.	12.5	39	24.6	93.5	95.4
		1st Cleaner Tail	8.1	3.23	2.64	13	17.1
		2nd Cleaner Conc.	4.3	47.8	29.8	86.5	87.2
		2nd Cleaner Tail	8.2	11.9	8.71	7	8.2

Table 13.18 - Overall gold recovery by flotation and cyanidation of the flotation tails

Sample	Concentrate Grade		Overall Au Recovery,%		
	Au, g/t	S,%	Flot. Conc.	Tail CN	Total
PP-05 (Conc for POX feed)	26.8	16.1	91.7	3.7	95.4
PP-09 (Conc for direct sale)	47.7	29.8	86.5	5.4	91.9

13.3.4 SGS Project Reference No. 13054-004 – DRAFT- Report 2 – August 26, 2013

SGS Project Reference No. 13054-004 – DRAFT – Report 2 summarizes the pressure oxidation pilot test results which consist of feed preparation, pressure oxidation, a hot curing stage and thickening of the final hot cured autoclave residue.

13.3.4.1 Pressure oxidation feed (flotation concentrate)

The feed for the pressure oxidation testwork was produced from a blended mixture of concentrate produced from an earlier flotation pilot plant program (SGS Project 13054-004 Report 1, *A Flotation Pilot Plant Investigation into the recovery of gold from Duparquet Samples*, June 20, 2013). The composition of the concentrate is listed in Table 13.19.

Table 13.19 - Concentrate composition

POX Feed Solids					
Au g/t	25.4	Bi g/t	<20	Pb g/t	234
Ag g/t	32	Ca g/t	22,800	Sb g/t	64
S%	16.9	Cd g/t	<10	Se g/t	<30
S2-%	16.8	Co g/t	113	Sn g/t	<20
Fe %	17.1	Cr g/t	162	Sr g/t	323
Cu %	0.041	Cu	330	Ti g/t	4,890
Hg %	39.1	Fe g/t	179,000	Tl g/t	<30
As %	0.69	K g/t	43,900	U g/t	<40
SiO2 %	35.7	Li g/t	<5	V g/t	177
C (t)	1.11	Mg g/t	5,790	Y g/t	38.9
CO3 2- %	4.35	Mn g/t	745	Zn g/t	214
C (g) %	0.02	Mo g/t	207	Cl g/t	100
Al g/t	55,200	Na g/t	5,080	F %	0.069
Ba g/t	1140	Ni g/t	96	Hg g/t	39.1
Be g/t	2.34	P g/t	844	Te g/t	<50

The flotation concentrate was generated from a composite sample produced from drill core from across the deposit. The drill core selected and the average flotation feed ore grade (Au) were selected to be as representative as possible. The details of sample selection are summarized in the SGS report. The head grade of the blended concentrate is 25.4 g/t gold, 32 g/t silver with sulphide grade at 16.8%. The sulphide content is sufficient to operate the pressure oxidation process under autothermal conditions without the requirement for extra heat. The high carbonate content (4.35%) necessitates an acid pre-treatment process to remove carbonate before pressure oxidation to ensure no build-up of carbon dioxide pressure in the autoclave.

No mineralogy of the concentrate feed was reported but based on the chemical composition the feed was calculated to have the mineralogical composition listed in Table 13.20.

Table 13.20 - Calculated mineral composition of concentrates

Mineral	%	Mineral	%
Quartz	11.4	Calcite	2.4
K-feldspar	25.5	Ankerite	2.4
Albite	5.8	Dolomite	2.4
Pyrite	30.8	Kaolinite	3.7
Arsenopyrite	1.5	Fluoroapatite	1.8
Chalcopyrite	0.1	Biotite	9.0
Haematite	1.5	Gypsum	0.8
Rutile	0.8		

13.3.4.2 Pressure oxidation pilot plant tests

The pressure oxidation tests were carried out using the SGS Lakefield 40L horizontal high pressure titanium vessel. The autoclave was operated and maintained at a temperature and oxygen overpressure of 210°C and 700 kPa respectively. The operating conditions at 60 and 90 minutes residence times were evaluated. The oxygen addition rate equated to 0.45 and 0.48 t of O_2/t_{conc} for 90 and 60 minutes runs respectively (i.e. 2.7 and 2.9 t/t of sulphur in the feed) which is ~40% in excess of the stoichiometric amount.

The feed density was maintained at 18% solids by premixing the concentrate with recycled acid solution. Recycled solution from the hot cure stage at 0.55 L/kg solid, containing 20 g/L Fe^{3+} and 40 g/L free sulphuric acid, was used and found to be sufficient to neutralize the carbonate in the feed concentrate. The carbonate content in the feed concentrates reporting to the autoclave ranged from <0.05 to 0.64%.
A summary of the conditions and results are shown in Table 13.21. The increase in concentration from the final autoclave chamber to the autoclave discharge was attributed to concentration in the evaporation/flashing stage and some basic iron salt redissolution.

Table 13.21 - Summary of pressure oxidation conditions and oxidation

Parameters	Values	
Temperature (°C)	210	
Oxygen overpressure (kPa)	700	
Feed density (%)	18-19	
Retention time (minutes)	60 and 90	
Oxygen addition (O_2 t/t of S)	2.9 and 2.7	
	Sixth Autoclave chamber	Autoclave discharge values
Solids		
Fe (%)	14.6	13.9
S (%)	6.8	6.4
% S oxidation	97.9	99.0
Liquor		
Total Fe (mg/L)	5,363	11,490
Ferrous ion (mg/L)	375	475
Arsenic (mg/L)	245	923
Free acid (g/L)	45	46

The reaction profile through the autoclave showed a similar trend for both 90 and 60 minute retention times but with higher sulphide oxidation taking place earlier in the autoclave for the slower throughput (90 minutes). In both, sulphide oxidation was largely complete after the third chamber of the 6-chambered autoclave. The majority of ferrous iron was converted to ferric after the sixth chamber. Overall, a 60-minute retention time appears to be sufficient to achieve a high level of oxidation.

The autoclave discharge liquor had on average 47 g/L free sulphuric acid and contained ~11-12 g/L iron (>95% as Fe^{3+}). Arsenic in solution was on average < 1 g/L. The Fe/As ratio is sufficiently high in the POX liquor for stabilisation of arsenic as a ferric arsenate product following neutralization. The POX liquor also contained on average 1.7 g/L Al, 1.1 g/L Mg and 0.7 g/L Si. There appeared to be no obvious dissolution of chloride from the concentrate to cause any detrimental effect on both pressure oxidation reactions and gold deportment, with levels reported in the liquor of 23 mg/L largely from the Lakefield river water used to pulp the concentrate. Comparisons of the feed and autoclave residue are provided in Table 13.22 and Table 13.23.

X-ray diffraction studies confirmed the beige, yellowish autoclave discharge product was predominately a jarosite product with little haematite formed. The high free acid accompanied by the dissolution of some phyllosilicate and framework silicate minerals promoted jarosite/alunite formation. The autoclave internals were generally clean indicating scaling was not a major issue.

No work was carried out to optimize the pressure oxidation operating conditions apart from retention time. The merits and amount of acid recycled to neutralize the carbonate in the feed to the autoclave needs to be investigated where lowering the free acidity in the autoclave may promote precipitation of hematite and control the amount of jarosite formed. The trade-off of this approach will be that increased carbonate in the feed to the autoclave will yield more carbon dioxide and reduce the oxygen partial pressure in the autoclave and increase the autoclave venting rate. An increased venting rate will lead to lower oxygen efficiency and higher rates of heat (steam) loss and hence are to be avoided if possible.

Table 13.22 - Autoclave feed and discharge compositions for major components

	Feed concentrate	Solid residue	Filtrate from residue
SG	2.7	2.93	1.086
Free acid (g/L)	-	-	47
	g/t	**g/t**	**mg/L**
Au	25.4	26.3	<0.03
Ag	32	33.7	<0.01
	%	**%**	**mg/L**
S	16.9	6.41	-
S^{2-}	16.8	0.13	-
SO_4	-	18.3	-
Fe	17.1	13.83	11873
Fe^{2+}	-	-	489
Cu	0.041	0.036	91
As	0.69	0.4	937
SiO_2	35.7		
Si	-	8.1	713
$C_{(t)}$	1.11	0.13	-
CO_3	4.35	<0.06	-
C_{org}	0.02	0.13	-

Table 13.23 - Autoclave feed and discharge compositions for other components

	Feed concentrate	Solid residue	Filtrate from residue		Feed concentrate	Solid residue	Filtrate from residue
	g/t	**g/t**	**mg/L**		**g/t**	**g/t**	**mg/L**
Al	55,200	48,750	1,698	Pb	234	238	<2
Ba	1,140	1,014	0.4	Sb	64	70	<1
Be	2.34	1.2	0.35	Se	<30	<30	<3
Bi	<20	<20	<1	Sn	<20	<20	<2
Ca	22,800	18,700	999	Sr	323	300	0.81
Cd	<10	<20	<1	Ti	4,890	5,005	3.72
Co	113	<9	30	Tl	<30	<30	<3
Cr	162	173	32	U	<40	<38	<6
K	43,900	43,600	25.3	V	177	137	5.4
Li	<5	<5	<2	Y	38.9	24.6	1.9
Mg	5,790	1,445	1148	Zn	214	29.8	56.5
Mn	745	20.7	196	Cl	100	37	23
Mo	207	108	28.4	F	0.069	-	-
Na	5,080	4,863	8	Hg	39.1	23.3	4
Ni	96	<20	15.5	Te	<50	<75	-
P	844	540	83.3				

13.3.4.3 Hot cure

The autoclave discharge slurry was pumped directly to the first of four insulated hot cure tanks. The hot cure was initially operated with four tanks providing a total retention time of 4 hours. The latter half of the pilot operated with five tanks having a total retention time of 5 hours.

The operating conditions, iron species content, as well as free acid assays for hot cure feed (AC Flash) and hot cure discharge (HC#4), are summarised in Table 13.24. The target temperature was maintained at 95 °C with steam heated submerged heating coils within tanks.

Iron in the hot cure discharge was ~17 g/L during the 90 minute autoclave retention time period and then increased to ~20 g/L when the retention time was reduced to 60 minutes. Arsenic content remained consistent throughout at an average of ~1.7 g/L. The total sulphur grade of the hot cure solids was consistently about 6% and close to the autoclave discharge slurry average of 6.4%.

For the basic ferric sulphate dissolution, free sulphuric acid is consumed and soluble ferric iron concentration would be expected to increase. However there appeared to have been little evidence of re-dissolution of basic ferric sulphate, with the majority of conversion having taken place within the autoclave flash let down. Changes in concentrations between autoclave discharge and hot cure discharge appear largely related to the flash letdown redissolution of basic iron salts and some loss of solution through evaporation.

Table 13.24 - Hot cure operating conditions, average iron, arsenic and free acid levels

Parameters	Values	
Temperature (°C)	95	
Pulp density (kg/L)	1.20-1.25	
Retention time (hrs)	4-5	
ORP (mV)	700	
	Autoclave discharge	Final hot cure tank discharge
$Fe_{(tot)}$ (mg/L)	11,490	19,050
Fe^{2+} (mg/L)	475	663
Fe^{3+} (mg/L) (difference)	11,015	18,387
As (mg/L)	923	1,818
Free Acid (g/L)	46	44

Optimization of the hot cure conditions of temperature and retention time needs to be carried out in conjunction with further pressure oxidation studies.

13.3.4.4 Hot cure discharge thickening

The hot cure discharge slurry was diluted to ~5-10% solids by recycling thickener overflow and injected in-line with flocculant Magnafloc 455. Delivery into the thickener allowed for a suitable mixing and contact time with the flocculant agent. The hot cure thickener appeared to operate well with the solids settling rate reported to be fast.

Table 13.25 - Thickener parameters

Parameters	Values
Flocculant	Magnafloc 455
Flocculant dosage (g/t of thickener solids)	95
Settling rate	Reasonably fast (not measured)
Average density of thickener underflow (w/w%)	45
Total suspended solids in overflow (g/L)	2.5
Overflow clarity	Poor

The Flocculant Magnafloc 455 dosage averaged ~95 gram per tonne of thickener feed solids basis. The thickener overflow liquor was not clear and contained total suspended solids of ~2.5 g/L.

13.3.4.5 Batch cyanidation test for extracting gold and silver

The recovery of gold and silver from the pilot plant hot cure discharge was determined in bottle roll leach tests. A summary of results, including results from two batch pressure oxidation laboratory tests carried out before the pilot plant, are shown in Table 13.7. Pulp samples from the hot cure discharge were taken from the pilot plant every 6 hours. The collected pulp was filtered, washed and then repulped with fresh water for cyanidation. The cyanide concentration was maintained at 0.5 g/L NaCN and the pH maintained at 10.5-11 with lime addition over the 24 hour leach period.

The recovery of gold from the pilot plant hot cure discharge ranged from 94.7% to 96.5% with gold recovery slightly higher for tests conducted after 90 minute POX time (96.2%, Au in residue 0.96 g/t) than those tests conducted after the 60 minute POX time (95%, 1.21 g/t Au in residue). No testwork was conducted to determine the nature of the gold losses in the pilot plant residue samples.

Lime boil was conducted at 95 °C for 4 hours with the addition of 120 kg/t CaO for two tests (CN15, 24) and at 75 kg/t CaO in a third test (CN25) to break down the jarosite and render silver recoverable by cyanidation. The addition of the lime boil resulted in an increase in the silver recovery from 9.2% to 95.5% as well as an increase in gold recovery from 95.5% to 98.1%. Reducing the lime dosage below 75 kg/t resulted in a reduction in gold and silver extraction. Optimum dosage appeared to be 75 kg/t of lime.

Cyanidation of the products of batch pressure oxidation tests (e.g., CN5-6) yielded better gold recovery of ~98% and sulphur oxidation >99% than the pilot plant operation. Visual observations indicated that the pilot plant product contained more basic iron sulphate and jarosite and less hematite than the lab tests. The reason for the slightly lower gold recovery in the tests conducted on the pilot plant product may be due to differences in the precipitate produced in the pilot plant compared to the batch pressure oxidation tests. The acidity of the solution and the concentration of cations such as Na^+, and K^+ will impact whether hydrolysis reactions favour hematite or jarosite formation. The continuous autoclave operation is also “seeded” with iron precipitates whereas the batch test is unseeded.

There is insufficient data from the pilot plant operation to determine the relationship between sulphide oxidation and gold recovery and whether full sulphide oxidation is required to recover the majority of the gold. Earlier batch testwork showed a direct relationship between sulphide oxidation and gold extraction. Hence, a high overall sulphide oxidation target was maintained.

13.3.4.6 Conclusions

A preliminary pilot test program investigated pressure oxidation and hot curing processing of a Duparquet flotation concentrate to render precious metals extractable by cyanidation.

The pilot plant operated for 48 hours with no significant problems. The 16.8% sulphide sulphur concentrate feed was oxidized efficiently in the autoclave at 210 °C and oxygen overpressure of 700 kPa, with residual sulphide levels averaging 0.13%. A 60 minute retention time appeared sufficient to achieve a high level of oxidation. Recycling a portion of the acidic hot cure thickener overflow allows for neutralization of carbonates present in the feed concentrate. The autoclave discharge was predominately a jarosite product with little hematite formed. The POX liquor contained 47 g/L free acid, 11 g/L Fe (>95% as ferric iron) and ~1 g/L As. No work has been carried out in the pilot plant to optimize the pressure oxidation operating conditions apart from retention time.

The merits and amount of acid recycled to neutralize the carbonate in the feed to the autoclave needs to be investigated. The lowering of the free acidity in the autoclave may promote precipitation of hematite and control the amount of jarosite formed.

The hot cure was operated at 95 °C with autoclave discharge product having a total retention time of either 4 or 5 hours to promote the conversion of any basic iron sulphate formed during pressure oxidation. There appeared to be little evidence of re-dissolution of basic ferric sulphate in the hot cure circuit with the majority of conversion having taken place within the autoclave flash letdown. The re-dissoution that did occur happened relatively quickly. Overall iron in solution increased from ~11 g/L in the autoclave discharge to ~17 g/L in the first hot cure stage and up to ~20 g/L through the hot cure circuit. Arsenic content remained consistent throughout at an average of ~1.7 g/L. Optimization of the size of the hot cure circuit in conjunction with further pressure oxidation studies is required. The cyanidation testwork program has demonstrated that high gold and silver recoveries are obtainable for cyanidation of pressure oxidation residues. The recovery of gold from the pilot plant hot cure discharge from batch testwork ranged from 94.7% to 96.5% with average residue assay from 0.96 g/t to 1.21 g/t Au. Lime (CaO) consumptions ranged from 4 to 8 kg/t with NaCN consumptions between 0.05 and 0.24 kg/t.

Lime boil tests on thickened hot cured discharged material, conducted at 95ºC for 4 hours with the addition of 120 kg/t CaO, were effective in breaking down jarosite and increased silver recovery from 9.2% to 95.5%, and gold recovery from 95.5% to 98.1%. Reducing the lime dosage below 75 kg/t resulted in a reduction in gold and silver extraction, with optimum conditions appear to favour 75 kg/t of lime. Promoting conditions favourable for hematite formation over jarosite in the pressure oxidation would undoubtedly reduce lime consumption in the lime boil step.

Precious metal leaching kinetics, carbon adsorption testwork and process modelling were limited to investigating cyanidation and recovery of gold from hot cured discharge samples. No detailed tests were carried out to determine the leaching and carbon adsorption kinetics for both gold and silver following lime boil processing. The leaching and carbon in pulp circuit was remodelled by Tenova-Bateman using the SIMCIL program (developed in the AMIRA P420 project) to take into account and estimate silver loadings to enable designing the whole precious metal leaching, recovery, elution and electrowinning circuits. The cyanidation circuit was configured with two leach tanks followed by six stages of carbon contacting, similar to the SGS model, with carbon transfer rate increased from 1.5 t/day to 2.5 tpd to take into account silver loading. Further work is required to investigate leaching and adsorption kinetics on lime boiled samples to determine the impact of liberated silver on precious metal leaching and adsorption kinetics to establish the final cyanidation extraction and recovery circuit configuration.

Overall the current design for the process plant is limited to one test program on one concentrate sample. The concentrate was obtained from a blend of samples from across the Duparquet Project. It is recommended that the impact of concentrate variability be investigated with respect to the mine plan and flotation feed variations. Mineralogical characterization and deportment of gold and silver should also be verified. Following further optimization tests on each unit operation, an integrated pilot plant for the main elements of the circuit should be operated for an extended period of time to confirm the results and also test other unit operations within the proposed flowsheet.

13.3.5 SGS Project Reference No. 13054-004 – DRAFT- Report 3 – August 27, 2013

SGS Project Reference No. 13054-004 – DRAFT – Report 3 summarizes the rheometallurgical responses (i.e., solid-liquid separation and rheology) of the flotation tailings (PP-05 Ro Tail), flotation concentrate (PP Cl Conc (POX Feed)), hot cure discharge (HC-4 Discharge), and combined leached tailings (Comb CND + Neut Solids) that were produced as part of the pilot plant test program, conducted at the SGS Canada Lakefield site.

Preliminary static settling test results for flotation tailings, identified as "PP-05 Ro Tail", indicated that the sample settled well in the presence of 20 g/t of BASF Magnafloc 333 flocculant, producing a 66% w/w solids underflow from a 10% w/w solids thickener feed. The resulting supernatant was slightly cloudy after 60 minutes of elapsed settling time. Relevant thickener data included: 0.045 m^2/t/d thickener underflow unit area (TUFUA), 0.016 m^2/t/d thickener hydraulic unit area (THUA), and 883 m^3/m^2/d initial settling rate (ISR). The supernatant total suspended solids (TSS) after 1 hour of elapsed settling was 22 mg/L.

Preliminary static settling test results for the flotation concentrate, identified as "PP Cl Conc. (POX Feed)", indicated that the sample settled well in the presence of 33 g/t of BASF Magnafloc 333 flocculant, producing a 65% w/w solids underflow from a 20% w/w solids thickener feed. The resulting supernatant was slightly cloudy after 60 minutes of elapsed settling time. Relevant thickener data included: 0.059 m^2/t/d thickener underflow unit area (TUFUA), 0.006 m^2/t/d thickener hydraulic unit area (THUA), and 536 m^3/m^2/d initial settling rate (ISR). The supernatant total suspended solids (TSS) after 1 hour of elapsed settling was 16 mg/L.
Preliminary static settling test results for the hot cure discharge, identified as "HC-4 Discharge", indicated that the sample settled in the presence of 92 g/t of BASF Magnafloc 333 flocculant, producing a 30% w/w solids underflow from a 3% w/w solids thickener feed. The resulting supernatant was still cloudy after 60 minutes of elapsed settling time. Relevant thickener data included: 0.374 m^2/t/d thickener underflow unit area (TUFUA), 0.075 m^2/t/d thickener hydraulic unit area (THUA), and

562 $m^3/m^2/d$ initial settling rate (ISR). The supernatant total suspended solids (TSS) after 1 hour of elapsed settling was 625 mg/L.

Preliminary static settling test results for the combined leached tailing, identified as "Comb CND+Neut Solids", indicated that the sample settled well in the presence of 36 g/t of BASF Magnafloc 333 flocculant, producing a 36% w/w solids underflow from a 2.5% w/w solids thickener feed. The overflow was slightly cloudy after 60 minutes of elapsed settling time. Relevant thickener data included: 0.26 $m^2/t/d$ thickener underflow unit area (TUFUA), 0.07 $m^2/t/d$ thickener hydraulic unit area (THUA), and 647 m3/m2/d initial settling rate (ISR). The supernatant total suspended solids (TSS) after 1 hour of elapsed settling was 16 mg/L. The preliminary static settling test results are summarized in Table 13.26

Table 13.26 - Preliminary static settling test results

Sample	Pulp pH	Flocculant BASF	Dosage g/t	Feed[1] % wt,	U/F[2] % wt	CSD U/F[3] % wt	TUFUA[4] $m^2/t/day$	THUA[5] $m^2/t/day$	ISR[6] $m^3/m^2/day$	O/F[7]	TSS mg/L
PP-05 Ro Tail	7.3	Magna-floc 333	20	10	66	70	0. 045	0,016	883	S.C	22
PP Cl Conc (POX Feed)	7.9		33	20	65	65	0.06	0.01	536	S.C	16
HC-4 Discharge	0.6		92	3	30	42	0.37	0.08	562	Cloudy	625
Comb CND+Neut Solids	8.4		36	3	36	44	0.26	0.07	647	S.C	16

All values were calculated without safety factor.

All tests are raked at 1.0 r.pm.

[1] Autodiluted thickener feed

[2] Ultimate Underflow Density

[3] Maximum Underflow Density predicted by Critical Solid Density (CSD)

[4] Thickener Underflow Unit Area corrected with actual CDS

[5] Thickener Hydraulic Unit Area corrected with actual CSD

[6] Initial Settling Rate

[7] Clarity 60 minutes into the test

The results of the preliminary static settling-thickening were used for reagent selection, optimization of thickener feed solid density and flocculant dosage. The flocculant screening and selection tests were conducted at 5% w/w solids. Five (5) different BASF Magnafloc products were tested with the best response from the non-ionic flocculant Magnafloc 333. The settling rate was fast but produced a cloudy supernatant. Using a combination of coagulant BASF Magnafloc 1687 with Magnafloc 333 did not improve liquor clarity.

The optimized dynamic settling conditions for PP-05 Ro Tail predicted 0.100 and 0.011 m^2/tpd for thickener underflow (TUFUA) and hydraulic (THUA) unit areas, respectively. This corresponded to 86.5 m^3/m^2/day net rise rate, 0.415 Um% net solids loading, and 3.60 $m^3/m^2/h$ net hydraulic loading. The overflow TSS was 43 mg/L and the underflow solids density at 65.2% w/w solids under these conditions. A 30 min extended thickening increased the underflow density to 69.3% w/w solids yielding about 23 Pa un-sheared vane yield stress. These results were produced at 10% w/w solid density at a dosage of ~25 g/t BASF Magnafloc 333 flocculant and 1.11 hours residence time.

The optimized dynamic settling conditions for PP Cl Conc. (POX Feed) predicted 0.100 and 0.025 m^2/t/day for thickener underflow (TUFUA) and hydraulic (THUA) unit areas, respectively. This corresponded to 35.9 m^3/m^2/day net rise rate, 0.415 $t/m^2/h$ net solids loading, and 1.496 $m^3/m^2/h$ net hydraulic loading. The overflow TSS was 53 mg/L and the underflow solids density at 61.8% w/w solids under these conditions. A 30 min extended thickening increased the underflow density to 65.5% w/w solids yielding about 36 Pa un-sheared vane yield stress. These results were produced at 20% w/w solid density at a dosage of ~40 g/t BASF Magnafloc 333 flocculant and 1.28 hours residence time.

The overflow stayed cloudy for the entire duration of the test run with TSS consistently exceeding ~1500 mg/L. The thickener underflow unit area (TUFUA) started at 0.401 and decreased to 0.331 m^2/t/day; thickener hydraulic unit area (THUA) started at 0.039 and decreased to 0.037 m^2/t/day. The net rise rate increased from 72.0 to 87.2 m^3/m^2/day. The net solids loading increased from 0.104 to 0.126 t/m^2/h. The net hydraulic loading increased from 3.00 to 3.64 m^3/m^2/h. The underflow solids density reached 41.2% w/w solids at the end of run 1 and 30 min extended thickening increased the underflow density to 42.6% w/w solids yielding about 25 Pa un-sheared vane yield stress. These results were produced at 3% w/w solid density at a dosage of -100 g/t BASF Magnafloc 333 flocculant.

The optimized dynamic settling conditions for Comb CND+Neut predicted 0.22 and 0.011 m^2/t/day for thickener underflow (TUFUA) and hydraulic (THUA) unit areas, respectively. This corresponded to 145.2 m^3/m^2/day net rise rate, 0.190 t/m^2/h net solids loading and 1.496 m^3/m^2/h net hydraulic loading. The overflow TSS was 73 mg/L and the underflow solids density at 42.3% w/w solids under these conditions. 30 min extended thickening increased the Run 1 underflow density to 43.2% w/w solids yielding about 29 Pa un-sheared vane yield stress. These results were produced at 3.0% w/w solid density at a dosage of ~40 g/t BASF Magnafloc 333 flocculant and 2.32 hours residence time.

The dynamic settling test results under optimum conditions are summarized in Table 13.27.

Table 13.27 - Dynamic settling test results

Sample	d_{80}, µm	Flocculant BASF	Dosage, g/t	Feed[1], % wt	U/F[2], % wt	U/F Extended, % wt	TUFUA[3], $m^2/t/day$	TUHUA[4], $m^2/t/day$	Net Rise Rate, $m^3/m^2/day$	Net Solids Loading, $t/m^2/day$	Net Hydraulic Loading, $m^3/m^2/day$	Res.Time, h Solids vs. UF	Overflow Visual	TSS, mg/L
PP-05 Ro Tail	89	Magna-floc 333	25	10	66.2	69.8	0.091	0.01	96.2	0.5	4.01	1.24	Clear	51
PP Cl Conc (POX Feed)	33		40	20	61.8	65.5	0.1	0.025	35.9	0.4	1.496	1.28	S.C	53
HC-4 Discharge	15		100	3										
Comb CND+Neut Solids	12		40	3	42.3	43.2	0.220	0.011	145.2	0.2	1.50	2.32	S.C	73

All values were calculated without a safety factor. S.C. indicates slightly cloudy.

[1] Autodiluted Thickener Feed

[2] Ultimate Underflow(UF) Density

[3] Thickener Underflow Unit Area

[4] Thickener Hydraulic Unit Area

The Critical Solids Density (CSD1) of the PP-05 Ro Tail sample was ~70% w/w solids, which corresponded to a yield stress of 52 Pa under un-sheared flow conditions and 22 Pa under sheared conditions. The CSD of the PP Cl Conc. (POX Feed) thickened underflow sample was -65% w/w, which corresponded to a yield stress of 38 Pa under un-sheared flow conditions and 29 Pa under sheared conditions. The CSD of the HC-4 Discharge thickened underflow sample was ~41.5% w/w, which corresponded to a yield stress of 24 Pa under un-sheared flow conditions, and 8 Pa under sheared conditions. The CSD of the HC-4 Discharge-Washed Solids thickened underflow sample was ~36% w/w., which corresponded to a yield stress of 29 Pa under un-sheared flow conditions and 12 Pa under sheared conditions. The CSO of the Combined CNO+Neut. solids thickened underflow sample was ~43.8% w/w, which corresponded to a yield stress of 30 Pa under un-sheared flow conditions and 17 Pa under sheared conditions.

13.3.5.1 Conclusions

Overall, the aforesaid rheometallurgical test data materially reflect the liquid-solid separation and flow behaviours of the process samples tested, rendering them suitable to be used as design criteria. A possible exception to this assessment involves the hot cure stream which displayed a relatively complex rheometallurgical response. To overcome the high overflow total suspended solids content a further clarification step to remove fine solids from the supernatant may be required.

Further work is still required to carry out specific CCD washing tests and model the process. The wash ratio and washing efficiency through testwork and modelling needs to be evaluated to accurately design the counter current decantation circuit.

[1] The Critical Solids Density (CSD) value is predictive of the maximum underflow solids density achievable in a commercial thickener and of the underflow solids density and pump-ability ranges achievable in practice and with reasonable friction pressure losses for an economically feasible operation.

13.3.6 SGS Project Reference No. 13054-004 – Report 4 – September 12, 2013

SGS Project Reference No. 13054-004 – Report 4 describes the bench-scale testwork program that was conducted to examine various process options relating to the recovery of gold by carbon adsorption from washed hot cure thickener underflow product. The program also investigated neutralization of hot cure thickener overflow solution and cyanide destruction of the cyanide leach product of the pilot plant flotation tailing and hot cure thickener underflow.

The recovery of gold from washed hot cure thickener underflow product by carbon adsorption and neutralization of the hot cure thickener overflow solution were discussed in 13054-002 report 1. The cyanide destruction testwork and its results are described below.

Cyanide destruction using the SO_2/air process was examined briefly for the cyanidation product of PP-05 and PP-09 flotation tailings. All of the samples responded well to the SO_2/air method of cyanide destruction with moderate reagent consumptions.

The results of the tests conducted on the PP-05 rougher tailings showed that the cyanide was effectively destroyed with an SO_2 addition of 5.7 g/g CN_{WAD} (Weak Acid Dissociable Cyanide) with a copper addition as copper sulphate of 0.1 g Cu/g $CN_{WAD.}$ Reducing the copper addition by half resulted in an increase in the CN_T (Total Cyanide) although the CN_{WAD} remained similar. The SO_2 addition in the bulk test was slightly higher than targeted because the feed rate was slightly lower than the target. A polishing stage is required to achieve a CN_T analysis of less than 1 mg/L bringing the total copper addition back to 0.1 g Cu/g $CN_{WAD.}$

Similar conditions were applied to the PP-09 rougher tailing. The testwork indicated that this tailing needs a higher copper addition of 0.2 g Cu/g CN_{WAD} to achieve the target CN_T analysis. The SO_2 requirement to achieve the target was 5.2 g/g $CN_{WAD.}$ The results of the cyanide destruction test are summarized in the Table 13.28.

Table 13.28 - Results of continuous SO2/air cyanide destruction tests on CIP barren pulp

Feed/ Test	Test	Pulp Density	Reactor Vol.	Test Time	Reten. Time	Solution Analysis					Reagent Addition								
		% solids	L	min	min	pH	CN_T mg/L	CN_{WAD} mg/L	Cu mg/L	Fe mg/L	g/g CN_{WAD} SO_2 Equiv.	g/g CN_{WAD} CaO	g/g CN_{WAD} Cu	g/L Feed Pulp SO_2 Equiv.	g/L Feed Pulp CaO	g/L Feed Pulp Cu	kg/t Solids SO_2 Equiv.	kg/t Solids CaO	kg/t Solids Cu
Feed - CN7 Pulp (PP-05 Plot tail)						*10.1*	*266*	*236*	*3.25*	*5.41*									
CND 1-1		50	1	210	64	8.5	0.4	<0.1	<0.05	0.12	5.66	2.12	0.10	0.98	0.37	0.017	1.34	0.50	0.024
CND 1-2	SO_2/air	50	1	180	65	8.5	1.3	<0.1	0.12	0.46	4.91	2.78	0.09	0.85	0.49	0.016	1.16	0.66	0.022
CND 1-3		50	1	180	59	8.5	15.3	0.5	<0.05	3.43	4.42	2.27	0.04	0.76	0.40	0.007	1.04	0.54	0.010
CND 1-4		50	1	180	57	8.5	9.9	0.6	0.10	3.43	3.86	3.22	0.04	0.66	0.57	0.007	0.91	0.76	0.010
CND 1-5	SO_2/air	50	2	390	65	8.5	7.1	<0.1	0.15	1.53	6.86	4.19	0.05	1.18	0.74	0.009	1.62	0.99	0.012
	Polishing	50	20	30	30	8.3	<0.1	<0.1	1.60	0.10			0.04			0.007			0.010
Feed - CN8 Pulp (PP-09 Flot Tail)						*10.0*	*241*	*232*	*4.65*	*6.91*									
CND 2-1		50	1	180	56	8.5	9.1	0.2	0.11	3.14	4.84	3.98	0.09	0.82	0.69	0.015	1.12	0.92	0.020
CND 2-2	SO_2/air	50	1	165	59	8.5	10.0	0.3	<0.05	3.04	3.76	1.74	0.09	0.64	0.30	0.015	0.87	0.40	0.020
CND 2-3		50	1	180	62	8.5	13.4	0.2	0.15	3.25	6.88	3.66	0.10	1.17	0.64	0.017	1.60	0.85	0.023
CND 2-4		50	1	180	59	8.5	4.99	0.3	<0.05	1.65	5.19	3.91	0.14	0.88	0.68	0.023	1.21	0.91	0.032
CND 2-5	SO_2/air	50	1	180	59	8.5	1.65	<0.1	0.08	0.80	5.10	3.50	0.18	0.87	0.61	0.031	1.19	0.81	0.042
	Polishing	50	5	30	30	8.3	0.12	<0.1	0.14	0.06			0.04			0.007			0.010
CND 2-6	SO_2/air	50	1	180	59	8.5	<0.1	<0.1	0.25	0.09	5.18	3.59	0.23	0.88	0.63	0.039	1.20	0.83	0.053
CND 2-7	SO_2/air	50	2	317	61	8.5	0.16	<0.1	<0.05	<0.05	5.24	2.81	0.23	0.89	0.49	0.039	1.22	0.65	0.054
Feed - CN21 Pulp (HC Thk u/f)						*10.1*	*283*	*224*	*2.04*	*0.07*									
CND 3-1		33	1	180	60	8.6	0.37	<0.1	<0.05	0.06	4.48	0.09	0.08	0.85	0.02	0.015	2.04	0.04	0.037
CND 3-2	SO_2/air	33	2	394	62	9.4	0.31	<0.1	<0.05	0.08	4.62	0.00	0.09	0.88	0	0.018	2.10	0	0.043
CND 3-3		33	2	630	61	9.3	0.36	0.1	<0.05	0.06	4.45	0.00	0.09	0.84	0	0.017	2.03	0	0.041

13.3.7 *SGS Project Reference No. 13054-004 – Grindability Report – REV1* – September 20, 2013

SGS Project Reference No. 13054-004 –Grindability Report – REV1 includes the results of the grindability tests that have been conducted on a pilot plant (PP Feed) sample. A single JKSimMet simulation was conducted, using aforementioned grindability test results. The results of the simulations are presented in the Table 13.29.

Table 13.29 - JKSimMet simulation results

SAG Mill Circuit	
Sample Name	PP Feed
Total SAG #	1
Mill Size (Inside Shell Diam. X EGL)	30.0' x 11.0'
Mill Size (Inside Liner Diam.. x EGL)	8.94 m x 3.35 m
Total Feed Rate, t/h	452.9
Total Feed Rate, t/d	10,000
Circuit Availability,%	92
Fresh Feed Size, mm, F_{80}	157
Fresh Feed Size, mm,%-12.7	21.3
Fresh Feed Axb	27.0
Ball Charge,%Vol	12
Mill Speed,% of Cr.	75
SAG Discharge% Solids, w/w	65
Grate Size, mm	64
Class. Slots Size, mm	12.7
SAG Recycle, t/h	99
SAG Recycle,%	22
Pebble Crusher, Y/N	Y
T_{80}, µm	2,125
Total SAG Mill Power Requirements (Motor input), kW	4,551
Total SAG Mill Power Requirements (Motor input), kWh/t	10.0

Ball Mill Circuit	
Sample Name	PP Feed
Total BM #	1
Mill Size (Inside Shell Diam. X EGL)	20.0' x 30.0'
Mill Size (Inside Liner Diam. x EGL)	5.94 m x 8.99 m
Total Feed Rate, t/h	452.9
Total Feed Rate, t/d	10,000
F_{80}, µm	2,125
Make-up Ball Size Diam., mm	57
Ball Charge,%Vol	32
Mill Speed,% of Cr.	75
BM Discharge% Solids, w/w	75
Bench-Scale Work Index (kWh/t), RWI	19.1
Bench-Scale Work Index (kWh/t), BWI	18.5
Recycle, t/h	1,132
Recycle,%	250
P_{80}, µm	100
Total Ball Mill Power Requirements (Motor input), kW	5,674
Total Ball Mill Power Requirements (Motor input), kWh/t	12.5

Overall Circuit	
Sample Name	PP Feed
Total SAG #	1
Total Feed Rate, t/d	10,000
F_{80}, mm	157
P_{80}, µm	100
Total Mill Power Requirements (Motor input), MW	10.2
Total Mill Power Requirements (Motor input), HP	13,706
Total Mill Power Requirements (Motor input), kWh/t	22.6
Operating Work Index (W_{io}), kWh/t	22.0
SAG Motor, MW	5.4
SAG Motor, HP	7,200
SAG Motor,% Utilization	84
BM Motor, MW	6.4
BM Motor, HP	8,600
BM Motor,% Utilization	89
Total Installed Power, MW	12.0
Total Installed Power, H	15,800

The simulations were performed with a SAG mill charge of 25%, 12.7 mm classification screen aperture and a maximum of 25% recycle rate. The SAG mill speed was assumed at 75% of the critical speed.

In the JKSimMet simulation, it was found that a 30.0’ x 11.0’ (nominal diameter x EGL) SAG mill, operated with a 12% ball charge would be required to grind 453 t/h crushed ore with F_{80} of 157 mm to a product with P_{80} of 2 mm. The SAG mill and ball mill specific power requirements were 10.0 and 12.5 kWh/t, respectively for a total power consumption of 22.6 kWh/t. It was also found that 5.7 MW would be required for the ball mill circuit to achieve a final product size of 100 µm. One 20.0' x 30.0' ball mill, with internal dimensions of 5.94 x 8.99 m and assumed 0° cone angle (square mill equivalent), operating with a ball charge of about 32% was selected. The SAG and ball mill installed power should be 4,551 kW and 5,674 kW, respectively. The SAG mill motor was selected to allow a ball charge increase up to 15% as well as an increase to 78% of critical speed.

13.3.8 Outotec Filtration Test Report – August 15, 2013

The *Outotec Filtration Test Report* includes the results of the filtration test on the flotation cleaner concentrate using a Larox Pressure Filter to achieve filter cake with moisture content of less than 8%. The testwork evaluated filter cloth selection, filter cake thickness, filtration rate, moisture content of the cake, and cake handling characteristics. The results are indicated in Table 13.30.

Table 13.30 - Flotation concentrate pressure filtration test results

Sample - Unit	pH	Air Drying Time (min)	Filtration Rate (kg/m2/h)	Filter Cake Moisture (% W/W Water)	Filter Cake Thickness (mm)	Pumping Pressure (Bar)	Pressing Pressure (Bar)	Air Pressure (Bar)
Conc. – Larox 100	8	1-4	699-1097	5-8	46-54	6	12	7-10

14. MINERAL RESOURCE ESTIMATE

An updated Mineral Resource Estimate for the Duparquet Project was prepared in 2013 by InnovExplo and presented in a report titled "Technical Report and Mineral Resource Estimate Update for the Duparquet Project (according to Regulation 43-101 and Form 43-101F1)", dated August 2, 2013 (Williamson et al., 2013b).

The August 2013 Mineral Resource Estimate Update is presented herein. It was performed by Kenneth Williamson, B.Sc., P.Geo., and Karine Brousseau, Eng., under the supervision of Carl Pelletier, B.Sc., P.Geo., using all available results. The main objective was to update the results of InnovExplo's previous Mineral Resource Estimate for the Duparquet Project, dated February 28, 2013 (Williamson et al., 2013a). The updated estimate includes additional new and re-sampled drill holes that were not included in the previous resource estimate. The Dumico area drill holes were also added to this update.

An exception to the above statement is the resource estimate for the tailings pond component, which was not modified during the August 2013 update. The results for this component, also presented herein, are taken directly from InnovExplo's earlier estimate of July 5, 2012 (Brousseau et al., 2012). The reader is referred to Figure 14.9 for the location of the tailings pond with respect to the Duparquet Property boundary.

The mineral resources presented herein are not mineral reserves since they have no demonstrable economic viability. The result of the update was a single Mineral Resource Estimate for sixty (60) mineralized zones, an envelope zone containing the remaining isolated gold intercepts (see below for details), and four (4) tailings zones. The Mineral Resource Estimate includes measured, indicated and inferred resources for both a Whittle-optimized in-pit volume and a complementary underground volume. The effective date of the Mineral Resource Estimate is June 26, 2013.

14.1 In-Pit and Underground Mineral Resource Estimate

14.1.1 Methodology

The Mineral Resource Estimate presented herein was made using 3D block modelling and the inverse distance square interpolation (ID2) method for a corridor of the Duparquet Project with a strike-length of 4.5 km and a width of approximately 1 km, down to a vertical depth of 1,050 m below surface. Sixty (60) mineralized zones have been interpreted in cross-sections spaced 25 m apart (locally 12.5 m where the drill hole density was high enough to allow it).

14.1.1.1 Drill hole and channel sample database

The Geotic / MS Access diamond drill hole database for the Duparquet Project was transferred into GEMS software. It contains 849 surface diamond drill holes and 892 channels with conventional analytical gold assay results, as well as coded lithologies from the drill core logs and channel descriptions. The 849 drill holes cover the 4.5-km strike-length of the Project at a fairly regular drill spacing of 50 m. The 892 channels are centered mostly on two (2) mineralized zones (the South and RWRS zones), with the remaining channels unevenly distributed in the eastern part of the Beattie property and on the Central Duparquet property. The database contains a total of 168,555 sampled intervals taken from 260,948.4 m of drilled core, and a total of 2,371 analyses taken from 1,827 m of channels.

In addition to the basic tables of raw data, the Gemcom database contains several tables with the drill hole and wireframe solid intersection composite calculations required for the statistical evaluation and resource block modelling.

14.1.1.2 Interpretation of mineralized zones

In order to conduct accurate resource modelling of the Duparquet Project, InnovExplo updated the mineralized-zone wireframe model to delineate the geologically defined extent of the mineralized zones within the defined Project area: a 4,500-metre strike-length corridor measuring 1,000 m wide and extending to 1,050 m below surface.

The nine (9) previously defined mineralized zones, once considered as the “main zones”, have been re-interpreted using the new geological and analytical information available.

In particular, the West Zone and the South Zone have been subdivided into several “sub-zones” based on geometrical and geological continuity parameters. Such subdivision was deemed necessary by InnovExplo in order to better reflect the structural style of the mineralization found within the Duparquet deposit and which is now better understood based on the new information available.

The West Zone has been subdivided into three distinct sub-zones: West 1, West 2 and West 3. The West 1 and West 3 sub-zones show a NE orientation with moderate and steep dips, respectively. The West 2 sub-zone is orientated slightly more NNE with a steep dip.

The South Zone has been re-modeled as ten (10) individual mineralized lenses, namely, from west to east, South Zone 1 to South Zone 10. The updated interpretation shows an “en echelon” stacking of ENE oriented mineralized lenses into a broader E-W striking mineralized corridor. This interpretation better reflects the internal geometry of the Donchester Fault Zone (DFZ) where mineralized lenses appear to be associated to second order slightly oblique sigmoidal structures developed within the DFZ itself.

Several other new zones have been interpreted for this mineral resource update. The addition of these new zones was required at Central Duparquet, Dumico and within the unconstrained “inter-zone” mineralized envelope.

In the Central Duparquet area, new drill hole information has led to the interpretation of another mineralized zone (CD S2) to the south of the CD South Zone. The CD S2 Zone is subparallel to the CD and CD South zones and is located about 25 m in the footwall of CD South Zone.

Interpretation of the mineralized zones found at Dumico is based on discussions with Louis Martin from Clifton Star, on grade continuity within the assays point set, and by taking into account the regional geological context of the Dumico property. A total of five (5) zones have thus been modelled at Dumico. Three (3) zones showing a similar geometry as the zones at Central Duparquet have been interpreted as the "natural" extensions of the latter. Two (2) SE-striking zones have also been interpreted based on grade continuity and their regional structural context.

A total of thirty-four (34) secondary mineralized zones have been interpreted within the previously defined "inter-zone" mineralized envelope. The interpretation of these secondary mineralized zones is based on field observations as well as grade continuity throughout the sample point set.

The mineralized zones of the Duparquet Project can be grouped in three (3) groups based on their geometry: 1) a dominant E-W trending group with very steep to vertical dips, 2) a NE to ENE-trending group for which the dip varies from moderate to steep, and 3) an ESE-trending group having moderate to steep dips.

Figure 14.1 shows a 3D isometric view of all the mineralized material zones including Group 2, in yellow and Group 3, in magenta. The geometry defined by these three (3) groups of mineralized zones can be observed on the several outcrops of the Duparquet Project (Figs. 14.2 and 14.3). Crosscutting relationships observed in the field suggest the contemporaneous development of these different groups of structures. Such observations are compatible with the interpreted protracted structural evolution of the DPMFZ as presented in sections 7.1 and 7.3.4.

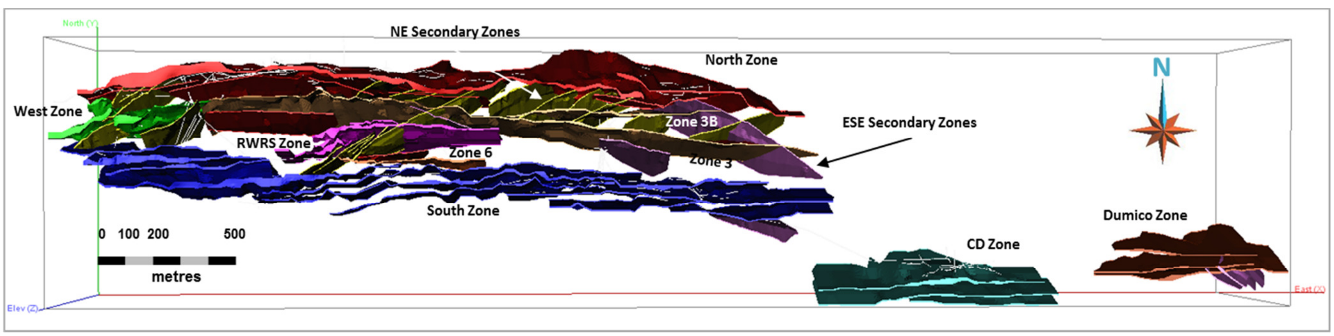

Figure 14.1 – General isometric view showing the main and secondary interpreted mineralized material zones

The following summarizes the changes and additions made to the mineralized material zone wireframe model.

1. The South Zone has been redesigned in such a way that it now presents a series of ten (10) "en-echelon" sub-zones associated with the DFZ;
2. The West Zone has been remodeled into three (3) distinct lenses;
3. A new zone, CD S2, has been created at Central Duparquet;
4. Five (5) mineralized zones have been created at Dumico;
5. Thirty-four (34) new zones have been created within the "inter-zone" mineralized envelope.

The wireframe solids of the mineralized zone model were created and updated by digitizing an interpretation onto sections spaced 25 m apart, or 12.5 m where drill holes density was higher, for a total strike length of 4,500 m, and then using tie-lines between sections to complete the wireframes for each solid.

For the main mineralized zones, the interpretation pushes their extent up to a distance of 100 m away from the last known occurrence of mineralization, unless negative intersects were encountered, in which case the mineralized zones were interpreted up to mid-distance between the last known occurrence of mineralization and the negative hole. For the secondary mineralized zones, a 50 m extension around the zones has been used.

The envelope zone was defined as the parts of the rectangular volume delimiting the block model that are not included in any of the mineralized-zone solids. The envelope zone contains “floating” gold intersects for which continuity has not yet been demonstrated or interpreted.

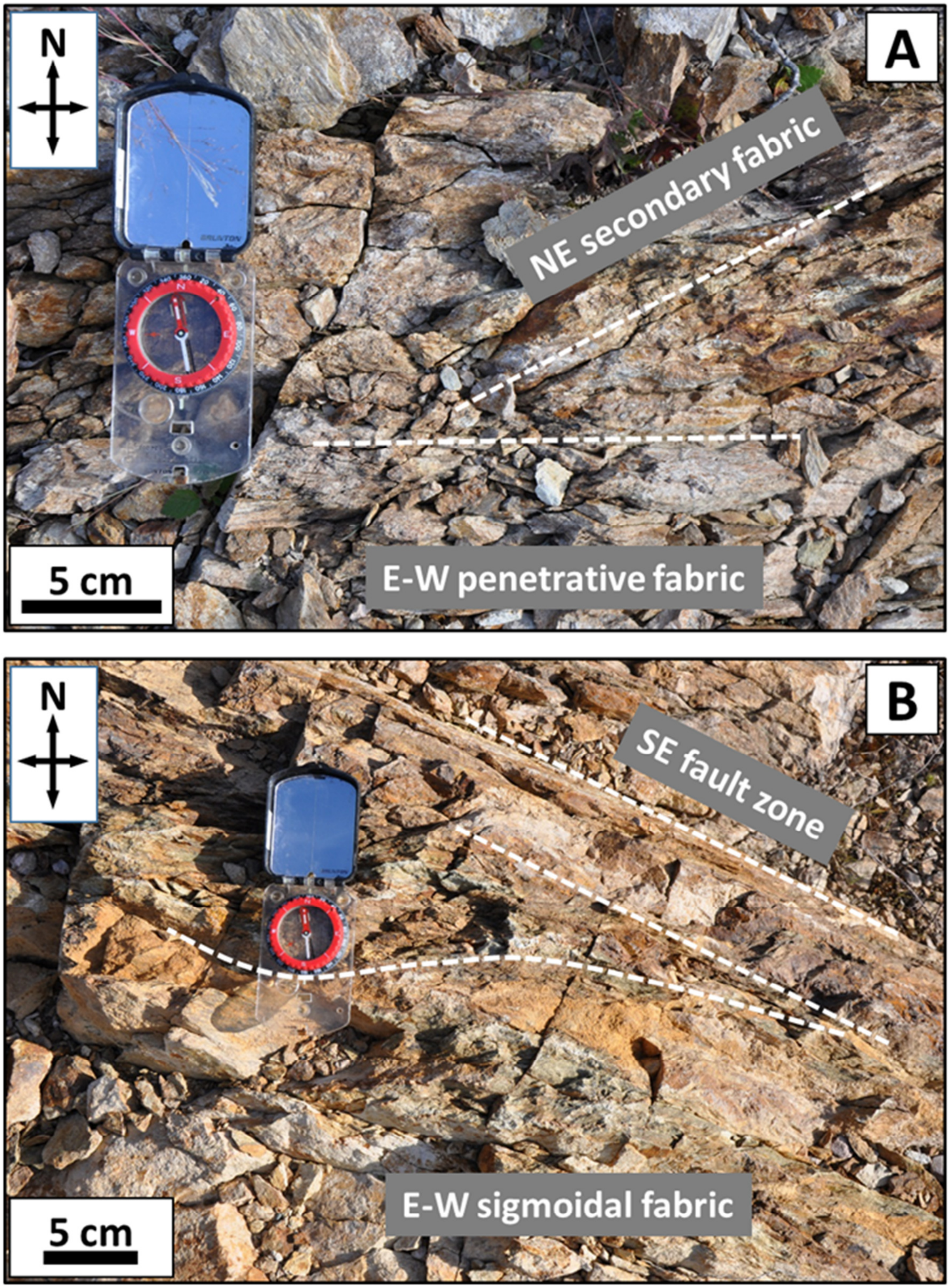

Figure 14.2 – A) Outcrop photograph of the RWRS area showing the relationship between an E-W penetrative fabric and a secondary NE-trending, RWRS sub-parallel, secondary fabric. B) Outcrop photograph showing a dominant SE-trending fault zone superposed onto and deforming the E-W penetrative fabric.

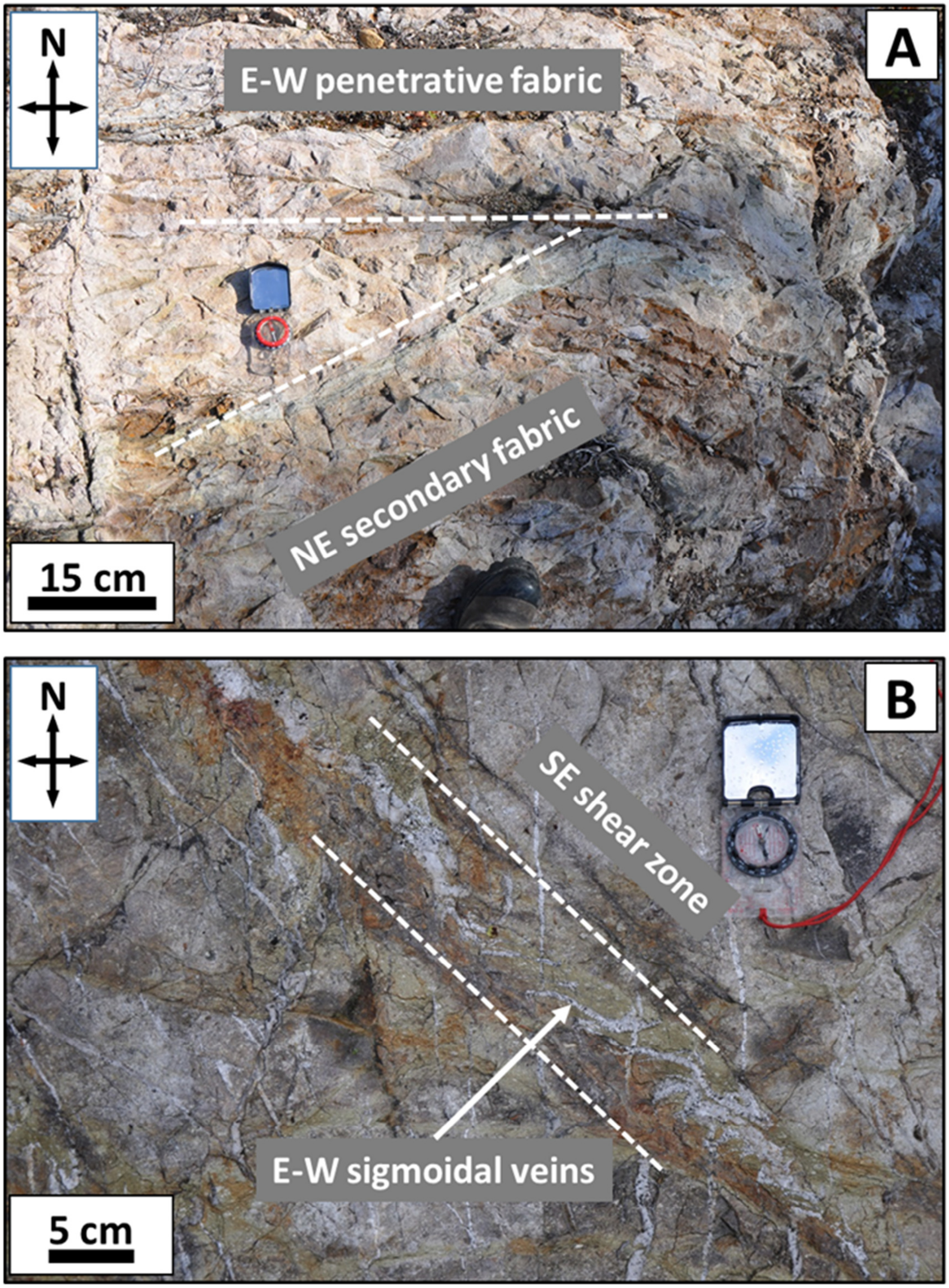

Figure 14.3 – A) Outcrop photograph of the South Zone area showing the relationship between an E-W penetrative fabric and a secondary NE-trending secondary fabric. B) Outcrop photograph of the eastern part of the Dumico area showing a dominant SE-trending shear zone containing E-W trending sigmoidal quartz veins.

14.1.1.3 High grade capping and compositing

Drill hole assays and channel assay intervals that intersect interpreted mineralized zones were automatically coded in the database from 3D solids, and these two (2) databases were used to analyze sample lengths and generate statistics and composites.

<u>High Grade Capping</u>

Basic univariate statistics were performed on the overall assay data and on datasets grouped by zones using point area files containing raw analytical assay data for a total of 168,555 diamond drill hole (DDH) samples and 2,371 channel samples. High grade capping was established at 25 g/t Au for all zones, including the envelope zone (Figs. 14.4 and 14.5).

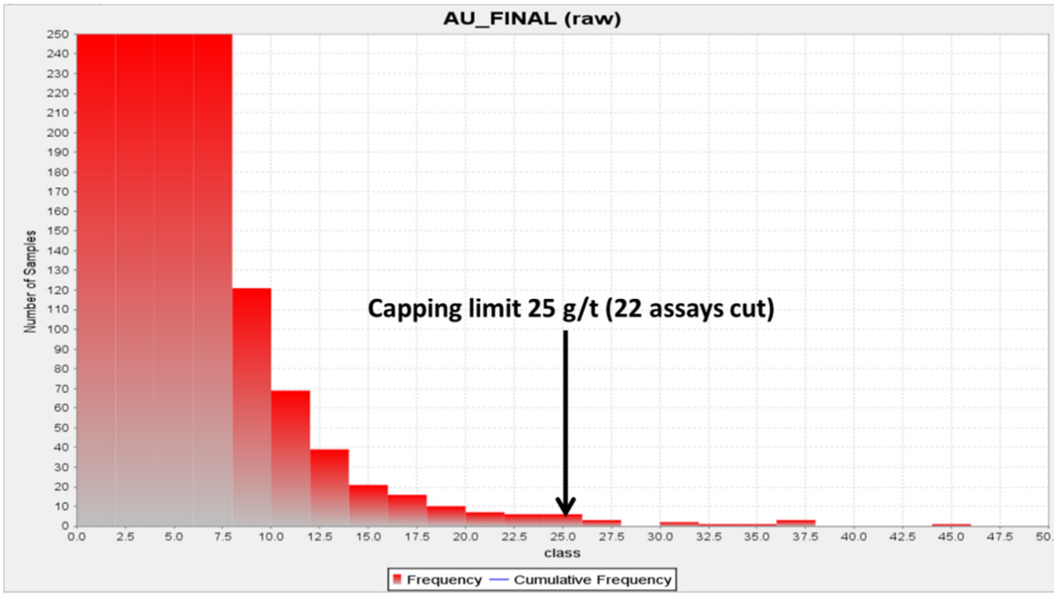

Figure 14.4 – Normal histogram of gold grades for all DDH samples

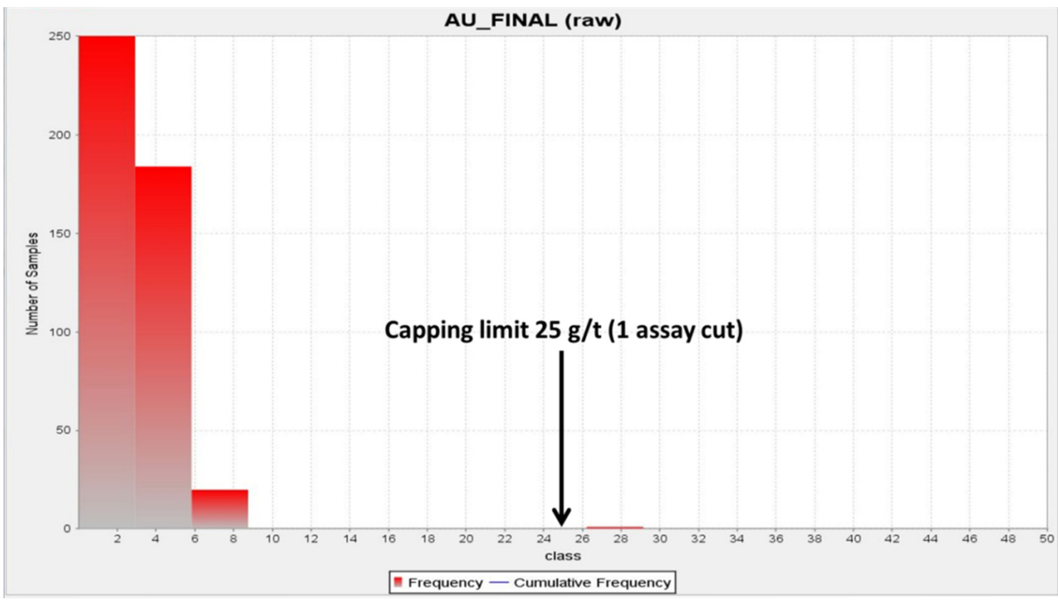

Figure 14.5 – Normal histogram of gold grades for all channel samples

Within the overall database, a total of twenty-two (22) DDH samples and one (1) channel sample were capped at the 25 g/t Au capping limit (Table 14.1). The capping of high assays represented a total loss of 2.77% of the gold metal factor for the DDH population. Table 14.2 presents a summary of the statistical analysis for the zones where high grade samples were capped.

Table 14.1 – List of capped samples in the overall database

DDH ID	FROM	TO	LENGTH	SAMPLE	ZONE	BLOCKCODE	AU_FINAL
BD10-284	158.5	160.0	1.5	D021700	ENVELOPE	4000	341.00
DON10-107	37.9	39.4	1.5	D080981	NORTH	1100	322.00
BD10-117	103.7	105.0	1.3	D000396	WEST_01	1410	119.50
BD10-232	669.8	670.3	0.5	535097	ENVELOPE	4000	98.97
BD10-147	54.0	55.5	1.5	D012305	SOUTH_02	1220	87.80
DON10-107	36.9	37.9	1.0	D080980	NORTH	1100	65.00
D13-03	310.5	311.6	1.1	563602	3B	1310	59.59
DON10-141	27.7	28.9	1.2	D066711	ENVELOPE	4000	54.10
DON10-60	163.0	164.0	1.0	D080762	3	1300	44.90
B88-48	57.62	58.53	0.91	B88-4815	WEST_02	1420	37.67
D08-28	101.9	103.0	1.1	8337	SOUTH_06	1260	37.58
DON10-69	488.0	489.5	1.5	D033676	O_50	2500	37.30
DON10-116	172.5	174.0	1.5	D061971	SOUTH_07	1270	34.90
D12-27	144.0	145.0	1.0	317489	SOUTH_08	1280	33.12
D03-01	202.1	202.7	0.6	D03-64754	NORTH	1100	30.63
B08-41	193.25	194.7	1.45	D909604	WEST_01	1410	30.20
BD10-320	80.0	81.5	1.5	D060141	NORTH	1100	27.90
DUM08-01	60.7	61.7		318124	DUM_03	3130	27.89
B08-45	257.0	258.0	1.0	6961	SOUTH_03	1230	26.90
D09-13	291.3	292.3	1.0	73734	SOUTH_10	1295	25.38
DON10-136	72.5	74.0	1.5	D062158	3B	1310	25.30
B97-01	160.0	160.5	0.5	B97-238667	3	1300	25.03

CHANNEL ID	FROM	TO	LENGTH	SAMPLE	ZONE	BLOCKCODE	AU_FINAL
R1989-116	5.15	5.75	0.6	5585	SOUTH_02	1220	29.14

Table 14.2 – Summary statistics for the raw assays by zone for the DDH population

Zone	Blockcode	Number of samples	Max	Uncut Mean (g/t)	High Grade Capping	Cut Mean	# Samples cut	% Samples capped	% loss Metal factor
NORTH	1100	6209	322.0	1.40	25	1.34	4	0.06%	5.1%
SOUTH_02	1220	2101	87.8	1.10	25	1.07	1	0.05%	3.3%
SOUTH_03	1230	1223	26.9	0.80	25	0.80	1	0.08%	0.1%
SOUTH_06	1260	1788	37.6	0.85	25	0.84	1	0.06%	0.8%
SOUTH_07	1270	397	34.9	1.20	25	1.18	1	0.25%	2.7%
SOUTH_08	1280	351	33.1	0.88	25	0.86	1	0.28%	2.2%
SOUTH_10	1295	128	25.4	0.77	25	0.76	1	0.78%	0.3%
3	1300	1170	44.9	0.89	25	0.87	2	0.17%	1.6%
3B	1310	911	59.6	0.97	25	0.93	2	0.22%	3.8%
WEST_1,2,3	1410-1420-1430	2259	119.5	1.55	25	1.50	3	0.13%	3.1%
O_50	2500	132	37.3	0.71	25	0.62	1	0.76%	14.3%
DUM_01,02,03	3110-3120-3130	246	27.9	1.02	25	1.01	1	0.41%	1.0%
Envelope	4000	139165	341.0	0.08	25	0.07	3	0.00%	4.0%
All Zones		168558	341.0	0.25	25	0.24	22	0.01%	2.8%

Compositing

In order to minimize any bias introduced by the variable sample lengths, the capped gold assays of the DDH data were composited to 1 metre equal lengths ("1m composites") within all intervals that define each of the mineralized zones. Among the composites generated within the assayed interval of the DDH population (257,712 composites), there were 938 (0.36%) tails less than 0.25 m long which were removed from the block model interpolation and variography. The total number of composites used in the DDH dataset is thus 256,774. A grade of 0.00 g/t Au was assigned to missing sample intervals.

No composites were filtered out of the channel sample dataset due to the small amount of data and the fact that such filtering would be statistically meaningless. A total of 2,298 equal-length 1m channel composites were used. There were no missing sample intervals.

14.1.1.4 Search ellipsoids

Each zone and the envelope were estimated separately with their own search ellipsoid. The size of each search ellipsoid was created according to the available geological and geostatistical information.

Panel subdivisions of the geometrically irregular mineralized zones

Eight (8) of the mineralized zones have been subdivided into panels on the basis of variations of their internal geometry. The criteria used for determining if a zone required a subdivision are the intensity of the along strike or down-dip curvature and/or presence of "splays" associated to the given zone. Cutting planes have been created in order to give each of these panels a "panel blockcode" to which specific research ellipsoids could be attributed. Table 14.3 shows a listing of the zone's panels as well as their "panel blockcode" and mean orientation. Figure 14.6 illustrates the panel's subdivision of the North Zone as well as the cutting planes used, and thus represents the methodology used for all of the eight zones that have been sub-divided.

Table 14.3 – Mean orientation of the defined panels for the geometrically irregular mineralized zones

ZONE	PANEL	BLOCKCODE	AZIMUTH	DIP
NORTH	1	1101	80	80
	2	1102	275	80
	3	1103	265	80
	4	1104	90	70
SOUTH_1	1	1211	275	85
	2	1212	95	75
SOUTH_2	1	1221	275	85
	2	1222	85	87.5
SOUTH_3	1	1231	90	70
	2	1232	280	80
SOUTH_4	1	1241	85	87.5
	2	1242	280	70
SOUTH_8	1	1281	265	90
	2	1282	275	85
SOUTH_10	1	1296	275	85
	2	1297	265	80
RWRS	1	1501	90	80
	2	1502	90	75
	3	1503	65	75
	4	1504	85	85

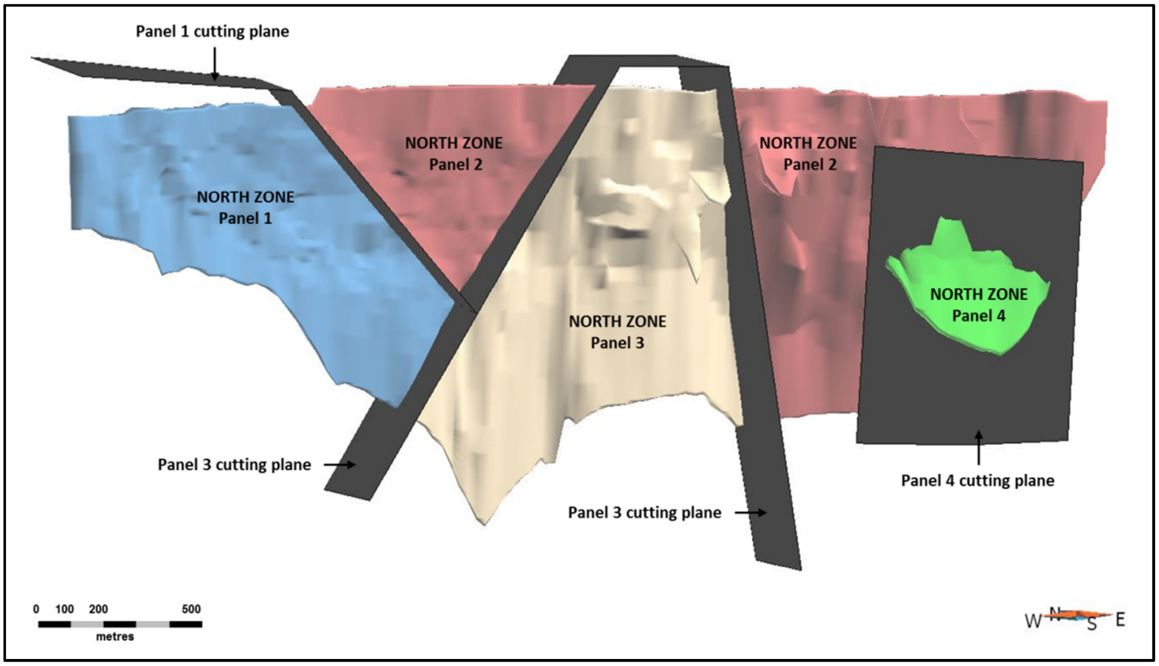

Figure 14.6 – NNE vertical section showing the panels subdivision of the North Zone

Taking into consideration the panels defined above, seventy-two (72) distinct mineralized zones and sub-zones have been considered for the characterization of the search ellipsoids.

Based on the mean azimuth and dip of each zone and sub-zones, a total of twenty-three (23) different search ellipsoids were produced, regrouping zones of similar geometry together. For the envelope zone the best results were obtained using a N090/90 orientation, which corresponds to the mean orientation of the structural grain on the Duparquet Property.

Ellipsoid radiuses were established using a combination of the ranges determined from the geostatistical analysis of the main zones and the interpretation of the true thickness of each zone. The radius of the third axis (Z) varies from 15 to 30 m reflecting the data density available for each zone and the confidence level of the geological interpretation.

Table 14.4 summarizes the parameters of the final ellipsoids used for interpolation.

Table 14.4 – Final search ellipsoid parameters

Ellipsoid	Rotation			Radius			Orientation		ZONE	PANEL
	Z (°)	X (°)	Z (°)	X (m)	Y (m)	Z (m)	Azimuth (°)	Dip (°)	(Blockcode)	
1	50	75	0	80	80	15	40	75	2050	
2	40	70	0	80	80	15	50	70	2280	
3	40	87	0	80	80	15	50	87	2120	
									2080	
									2100	
									2110	
									2070	
									2170	
4	30	75	0	80	80	15	60	75	2420	
									2240	
									2060	
									2310	
									2190	
									2250	
									2300	
									2150	
									1500	1503
5	30	85	0	80	80	15	60	85	2490	
									2500	
6A	15	80	0	80	80	15	65	80	1410	
6B	20	80	0	80	80	15	70	80	2140	
							70	80	2130	
7	15	40	0	80	80	15	75	40	1420	
8	15	85	0	80	80	15	75	85	1400	
9	10	75	0	110	110	15	80	75	2020	
				110	110	25	80	80	1100	1101
10	5	85	0	110	110	15	80	85	2010	
				110	110	25	85	85	1500	1504
				110	110	25	85	85	1240	1241
				110	110	25	85	85	1270	
				110	110	25	85	85	1250	
				110	110	25	85	85	1260	
				110	110	25	85	85	1220	1222
11	0	70	0	110	110	25	90	70	1100	1104
				110	110	25	90	70	1230	1231
				110	110	25	90	75	1500	1502
				110	110	25	95	75	1210	1212
12A1	0	80	0	110	110	25	90	80	1500	1501

Ellipsoid	Rotation			Radius			Orientation		ZONE	PANEL
	Z (°)	X (°)	Z (°)	X (m)	Y (m)	Z (m)	Azimuth (°)	Dip (°)	(Blockcode)	
12A2	-5	80	0	110	110	15	95	80	1300	
				110	110	15	95	80	2330	
				110	110	15	95	80	2270	
12B	0	85	0	110	110	15	90	85	1600	
				110	110	15	95	85	2180	
				110	110	15	95	85	2460	
				110	110	15	90	90	1310	
13	-20	85	0	110	110	15	110	85	2400	
14	-35	65	0	110	110	15	125	65	2380	
15	5	-85	0	110	110	25	265	80	1100	1103
				110	110	25	265	80	1295	1297
				110	110	25	265	90	1280	1281
				110	110	25	265	90	1290	
16	0	-75	0	110	110	30	270	75	1700	
				110	110	30	270	75	1710	
				110	110	30	270	75	1720	
				110	110	30	270	75	3110	
				110	110	30	270	75	3120	
				110	110	30	270	75	3130	
17	-5	-85	0	110	110	25	275	80	1100	1102
				110	110	25	275	85	1280	1282
				110	110	25	275	85	1210	1211
				110	110	25	275	85	1295	1296
				110	110	25	275	85	1220	1221
				110	110	15	275	85	2160	
				110	110	15	275	85	2480	
				110	110	15	275	85	2290	
				110	110	15	275	85	2200	
				110	110	15	275	85	2090	
				110	110	15	275	85	2410	
18	-10	-75	0	110	110	25	280	70	1240	1242
				110	110	25	280	80	1230	1232
19	-45	-70	0	110	110	30	300	65	3510	
				110	110	30	300	65	3520	
20	0	-90	0	50	50	15			4000	

**Block Model system: positive rotation is counter-clockwise.*

***Conventional right-hand rule.*

14.1.1.5 Bulk density

For the current 2013 Mineral Resource Estimate update, no new density data were retrieved from the database. Therefore, based on the analysis of 3,633 bulk density values performed by InnovExplo for a previous study (Brousseau et al., 2012), a density of 2.73 g/cm^3 (mean density of the syenite) is used to characterize all sixty (60) mineralized material zones as well as the envelope zone. A density of 2.00 g/cm^3 has been assigned to the overburden (OB), whereas a density of 1.00 g/cm^3 is used for excavation solids (drifts and stopes) that are considered to be filled with water.

14.1.1.6 Block model geometry

A block model was established for the sixty (60) mineralized zones (and their sub-panels when applicable) and has been extended to cover an area sufficient to host an open-pit. The model has been pushed down to a depth of approximately 1,050 m below surface. The limits of the block model are as follows (UTM NAD83, Zone 17):

Easting:	628500mE	to 634400mE	(1180 columns x 5 m each)
Northing:	5373300mN	to 5375075mN	(355 rows x 5 m each)
Elevation:	370m	to -760m	(226 levels x 5 m each)

The block model has parallel orientation, with the Y-axis oriented along a north grid azimuth. The individual block cells have dimensions of 5 m long (X-axis) by 5 m wide (Y) by 5 m vertical (Z). The block dimensions reflect the dimension of the mineralized zones. Table 14.5 shows the Duparquet Project block model with its interpolated zones and their associated folders. The table also provides details about the corresponding naming convention for the GEMS solids, the rockcodes, blockcodes, and the precedence assigned to each individual solid.

Table 14.5 – Duparquet Project block model and associated interpolated zones

FOLDER	GEMS SOLID NAME			ROCK CODE	BLOCK CODE	PRECEDENCE
	NAME1	NAME2	NAME3			
ENVELOPE	TOPO	OB	MAY2013	OB	6	1
	STOPE	DUPARQUE	FINAL	STOPE	8	2
	DRIFT	DUPARQUE	FINAL	DRIFT	9	3
	DRIFT	CD	FINAL	DRIFT	9	3
	ZONE	ENVELOP	MAY2013_F	4000	4000	400
ZONES	ZONE	NORTH	MAY2013_F	1100	1100	10
	ZONE	SOUTH_01	MAY2013_F	1210	1210	21
	ZONE	SOUTH_02	MAY2013_F	1220	1220	22
	ZONE	SOUTH_03	MAY2013_F	1230	1230	23
	ZONE	SOUTH_04	MAY2013_F	1240	1240	24
	ZONE	SOUTH_05	MAY2013_F	1250	1250	25
	ZONE	SOUTH_06	MAY2013_F	1260	1260	26
	ZONE	SOUTH_07	MAY2013_F	1270	1270	27
	ZONE	SOUTH_08	MAY2013_F	1280	1280	28
	ZONE	SOUTH_09	MAY2013_F	1290	1290	29
	ZONE	SOUTH_10	MAY2013_F	1295	1295	21
	ZONE	3	MAY2013_F	1300	1300	30
	ZONE	3B	MAY2013_F	1310	1310	31
	ZONE	WEST_1	MAY2013_F	1410	1410	41
	ZONE	WEST_2	MAY2013_F	1420	1420	42
	ZONE	WEST_3	MAY2013_F	1430	1430	43
	ZONE	RWRS	MAY2013_F	1500	1500	50
	ZONE	6	MAY2013_F	1600	1600	60
	ZONE	CD	MAY2013_F	1700	1700	70
	ZONE	CD_SOUTH	MAY2013_F	1710	1710	71
	ZONE	CD_S2	MAY2013_F	1720	1720	72
	ZONE	O_01	MAY2013_F	2010	2010	201
	ZONE	O_02	MAY2013_F	2020	2020	202
	ZONE	O_05	MAY2013_F	2050	2050	205
	ZONE	O_06	MAY2013_F	2060	2060	206
	ZONE	O_07	MAY2013_F	2070	2070	207
	ZONE	O_08	MAY2013_F	2080	2080	208
	ZONE	O_09	MAY2013_F	2090	2090	209
	ZONE	O_10	MAY2013_F	2100	2100	210
	ZONE	O_11	MAY2013_F	2110	2110	211
	ZONE	O_12	MAY2013_F	2120	2120	212
	ZONE	O_13	MAY2013_F	2130	2130	213
	ZONE	O_14	MAY2013_F	2140	2140	214
	ZONE	O_15	MAY2013_F	2150	2150	215
	ZONE	O_16	MAY2013_F	2160	2160	116
	ZONE	O_17	MAY2013_F	2170	2170	217
	ZONE	O_18	MAY2013_F	2180	2180	218
	ZONE	O_19	MAY2013_F	2190	2190	219
	ZONE	O_20	MAY2013_F	2200	2200	220
	ZONE	O_24	MAY2013_F	2240	2240	224
	ZONE	O_25	MAY2013_F	2250	2250	225

FOLDER	GEMS SOLID NAME			ROCK CODE	BLOCK CODE	PRECEDENCE
	NAME1	NAME2	NAME3			
	ZONE	O_27	MAY2013_F	2270	2270	227
	ZONE	O_28	MAY2013_F	2280	2280	228
	ZONE	O_29	MAY2013_F	2290	2290	229
	ZONE	O_30	MAY2013_F	2300	2300	230
	ZONE	O_31	MAY2013_F	2310	2310	231
	ZONE	O_33	MAY2013_F	2330	2330	233
	ZONE	O_38	MAY2013_F	2380	2380	238
	ZONE	O_40	MAY2013_F	2400	2400	240
	ZONE	O_41	MAY2013_F	2410	2410	241
	ZONE	O_42	MAY2013_F	2420	2420	242
	ZONE	O_46	MAY2013_F	2460	2460	146
	ZONE	O_48	MAY2013_F	2480	2480	248
	ZONE	O_49	MAY2013_F	2490	2490	249
	ZONE	O_50	MAY2013_F	2500	2500	250
	ZONE	DUM_01	MAY2013_F	3110	3110	311
	ZONE	DUM_02	MAY2013_F	3120	3120	312
	ZONE	DUM_03	MAY2013_F	3130	3130	313
	ZONE	DUM_04	MAY2013_F	3510	3510	314
	ZONE	DUM_05	MAY2013_F	3520	3520	315

14.1.1.7 Mineralized zone block model

A percent block model in the Zones folder "mineralization folder", as well as for the Envelope folder "waste folder" (overburden, envelope, drift and stope), was generated reflecting the proportion of the different respective solids in each block using precedence of solids. In detail, all blocks with at least 0.01% of their volume falling within a selected solid were assigned the corresponding solid blockcode (Table 14.5) and its relative proportion (%) in their respective folder.

As a result, all blocks having at least 0.01% of their volume contained within stope solids were rock-coded stope. InnovExplo deemed such conservative manipulation necessary in order to compensate for the risk that blocks near stopes are either mined out or inaccessible due to safety factors.

The multi-folder percent block model thus generated was used in the mineral resource estimation.

14.1.1.8 Grade block model

A grade model was interpolated using the 1m composite derivates from the conventional capped assay grade data to produce the best possible grade estimate for the defined resources in the Duparquet Project. The interpolation has been done on a point area combining the DDH and channel datasets.

A set of two (2) interpolation profiles was established for grade estimation. The interpolation profiles were customized to estimate grades separately within the Zones and the Envelope folders for the combined DDH and channel composite populations. The method retained for the final resource estimation was inverse distance square (ID2).

The composite points were assigned rock codes and block codes corresponding to the mineralized zone or the mineralized panel in which they occur. The interpolation profiles specify a single target and sample rock code for each mineralized-zone solid, thus establishing hard boundaries between the mineralized zones and preventing block grades from being estimated using sample points with different block codes than the block being estimated. The search/interpolation ellipse orientations and ranges defined in the two interpolation profiles used for grade estimation correspond to those developed in Table 14.4 in Section 14.1.1.4 of this Report (Search ellipsoids).

Other specifications to control grade estimation are as follows:

For the Zones folder corresponding to the sixty (60) mineralized zones:

- inverse distance square interpolation method for data points;
- minimum of two (2) and maximum of twelve (12) sample points in the search ellipse for interpolation;
- maximum of four (4) sample points from any one DDH or channel;
- high-grade values capped.

For the Envelope folder (Envelope zone):

- inverse distance square interpolation method for data points;
- minimum of two (2) and maximum of six (6) sample points in the search ellipse for interpolation;
- no limit on sample points from any one DDH or channel;
- high-grade values capped.

Following the interpolation step, the grade of all the blocks rock-coded stopes was initialized to zero (0) for the Zones and the Envelope folders using “rocktype” selection.

14.1.1.9 Resource Category Block Model

Classification to a <u>Measured category</u> was done for blocks having a volume of at least 25% within an envelope built at a distance of 10 m around existing channels.

Classification to an <u>Indicated category</u> was done for blocks meeting at least one (1) of the conditions below:

- Blocks falling within a 15-m-wide halo surrounding existing stopes
- Blocks for which the average distance to composites is less than 45 m.

A clipping polygon was generated for each of the sixty (60) zones using the criteria described above and the blocks were coded accordingly. All remaining blocks that were interpolated were classified as Inferred category. Figures 14.7 and 14.8 show the resource category for two typical zones out of the sixty (60) mineralized zones, as well as the Whittle-optimized pit shell delimiting the in-pit and underground mineral resources. The reader should refer to sections 14.1.3 and 14.1.4 for further details about the Whittle-optimized pit shell.

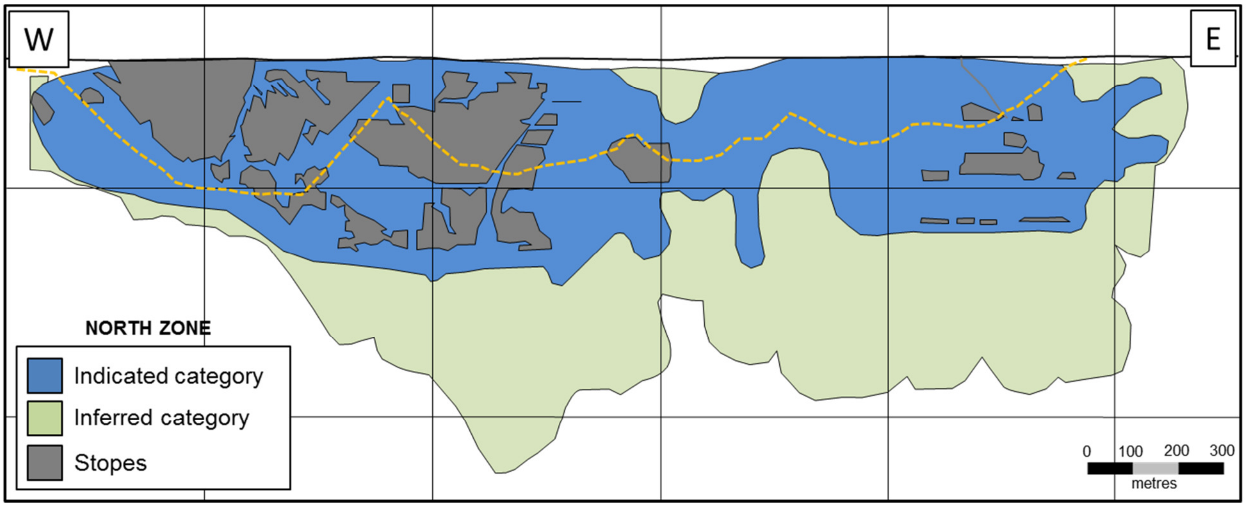

Figure 14.7 – Vertical longitudinal section along the North Zone showing the categorized mineral resources and the Whittle-optimized pit-shell trace.

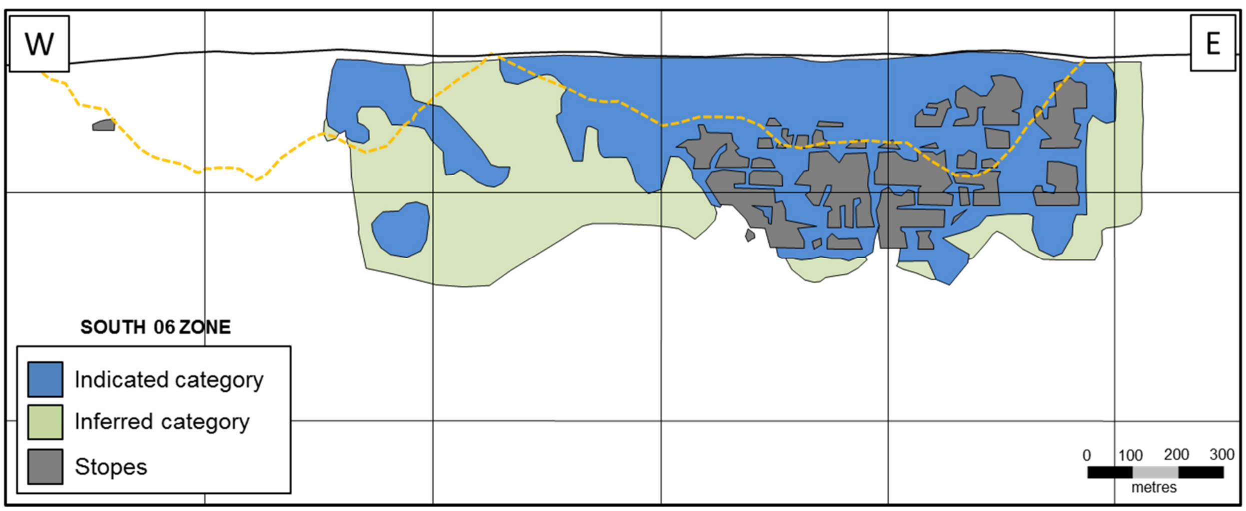

Figure 14.8 – Vertical longitudinal section along the South 06 Zone showing the categorized mineral resources and the Whittle-optimized pit-shell trace.

14.1.2 Mineral Resource Classification, Category and Definition

The resource classification definitions used for this report are those published by the Canadian Institute of Mining, Metallurgy and Petroleum in their document "CIM Definition Standards for Mineral Resources and Reserves".

<u>Measured Mineral Resource</u>

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

Indicated Mineral Resource

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

Inferred Mineral Resource

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

14.1.3 In-Pit Resource Estimation

Given the density of the processed data, the search ellipse criteria, and the specific interpolation parameters, InnovExplo is of the opinion that the current (updated) Mineral Resource Estimate can be classified as Measured, Indicated and Inferred mineral resources. The estimate is compliant with CIM standards and guidelines for reporting mineral resources and reserves. The In-Pit Mineral Resources were estimated using different gold cut-off grades and a minimum width of 3.0 m (true width). The selected cut-off of 0.45 g/t Au allowed the mineral potential of the deposit to be outlined for the in-pit mining option.

The final selected Whittle input parameters and the cut-off grade parameters used for the In-Pit Mineral Resource Estimation are defined in Table 14.6.

Table 14.6 – Input parameters used for the "mill cut-off grade" (MCoG) estimation and Whittle Pit design – Duparquet Project

Input parameter	Value	Provided by
Exchange rate	1.00 USD : 1.01 CAD	InnovExplo
Gold price (CAD/oz)	1,450.00	InnovExplo
Gold selling costs (CAD/oz)	5.00	InnovExplo
Net gold price (CAD/oz)	**1,445.00**	
Processing costs (CAD/t)	13.46	Tenova-Bateman
Transportation costs (CAD/t)	0.25	InnovExplo
Mining costs (CAD/t)	2.40	InnovExplo
Administration costs (CAD/t)	4.18	InnovExplo
Processing recovery (%)	93.9	Tenova-Bateman
Mining dilution (%)	10.0	InnovExplo
Mining Recovery (%)	90.9%	InnovExplo
Overall pit slope	52°	InnovExplo
Overburden slope	30°	InnovExplo

The overall slope angle was set at 52°, which reflects the best approximation since little geotechnical information has been provided. The gold selling, processing and transportation costs, as well as the mining dilution, were based on InnovExplo's recent experiences. The processing costs and recovery were estimated by Tenova-Bateman (see sections 13 and 17 of this Report).

A "mill" or "marginal" cut-off grade was used in Whittle. Using the parameters shown above in Table 14.6, a mill cut-off grade (MCoG) of 0.45 g/t Au was calculated for the Whittle pit shell optimization using the following formula:

$$\text{MCoG} = \frac{(\text{Processing + Transportation + Administration costs}) \times (1 + \text{Mining dilution}) \times 31.1035}{(\text{Processing recovery}) \times (\text{Net gold price})}$$

Volumetrics for the In-Pit Mineral Resource Estimate have been constrained using the topography as the top surface and the Whittle-optimized pit-shell as the bottom surface. The needling has been set to three (3) needles.

Table 14.7 displays the results of the In Situ[2] Mineral Resource Estimate for the In-Pit Whittle-optimized portion of the Duparquet Project (the 60 mineralized material zones and the envelope zone). Figures 14.7 and 14.8 present the Whittle-optimized pit-shell trace along each of the North Zone and the South 06 Zone respectively. The detailed results of the In-pit Mineral Resource Estimate are presented by zone in Appendix II.

[2] The term "in situ" is used to represent all the remaining mineral resources in place at the time of the updated estimate herein.

Table 14.7 – In-Pit Mineral Resource Estimate results (Measured, Indicated and Inferred resources) at different cut-off grades using a gold price of C$1,450/oz

Measured Resources				
Zone	Cut-off g/t Au	Tonnes	Grade g/t Au	Au Oz
All Zones	> 0.90	124,000	1.71	6,802
	> 0.80	134,100	1.64	7,079
	> 0.70	144,700	1.58	7,335
	> 0.65	148,900	1.55	7,424
	> 0.60	153,400	1.52	7,517
	> 0.55	156,700	1.50	7,578
	> 0.50	161,600	1.47	7,658
	> 0.45	**165,100**	**1.45**	**7,711**
	> 0.40	169,500	1.43	7,772
	> 0.35	172,600	1.41	7,811

Indicated Resources				
Zone	Cut-off g/t Au	Tonnes	Grade g/t Au	Au Oz
All Zones	> 0.90	36,141,100	1.98	2,303,349
	> 0.80	39,580,300	1.88	2,397,301
	> 0.70	43,247,100	1.79	2,485,658
	> 0.65	45,156,700	1.74	2,527,119
	> 0.60	47,055,700	1.70	2,565,279
	> 0.55	49,024,800	1.65	2,601,669
	> 0.50	51,027,400	1.61	2,635,488
	> 0.45	**53,070,600**	**1.56**	**2,666,690**
	> 0.40	55,186,300	1.52	2,695,612
	> 0.35	57,306,100	1.48	2,721,190

Inferred Resources				
Zone	Cut-off g/t Au	Tonnes	Grade g/t Au	Au Oz
All Zones and Envelope	> 0.90	10,042,800	1.95	628,621
	> 0.80	11,734,700	1.79	674,741
	> 0.70	14,090,900	1.61	731,414
	> 0.65	15,414,600	1.53	760,124
	> 0.60	17,174,200	1.44	795,413
	> 0.55	19,047,600	1.36	830,036
	> 0.50	21,362,800	1.27	869,085
	> 0.45	**24,092,300**	**1.18**	**910,631**
	> 0.40	27,494,100	1.08	956,942
	> 0.35	31,877,700	0.99	1,009,644

- The Independent and Qualified Persons for the Mineral Resource Estimate, as defined by NI 43-101, are Kenneth Williamson, M.Sc., P.Geo and Karine Brousseau, Eng. under the supervision of Carl Pelletier, B.Sc., P.Geo. (InnovExplo Inc.), and the effective date of the estimate is June 26, 2013.
- Mineral Resources which are not Mineral Reserves, do not have demonstrated economic viability.
- Results are presented undiluted within Whittle-optimized pit shells. The estimate includes 60 gold-bearing zones and the envelope zone containing isolated gold intercepts.
- Resources were compiled at 0.35, 0.40, 0.45, 0.50, 0.55, 0.60, 0.65, 0.70, 0.80 and 0.90 g/t Au cut-off grades.
- Cut-off grades must be re-evaluated in light of prevailing market conditions (gold price, exchange rate and mining cost).
- A fixed density of 2.73 g/cm^3 was used in the mineralized zones and the envelope zone.
- A minimum true thickness of 3.0 m was applied, using the grade of the adjacent material when assayed, or a value of zero when not assayed.
- High grade capping was done on the raw data and established at 25.0 g/t Au for diamond drill hole assays and channel sample assays.
- Compositing was done on drill hole and channel sample sections falling within the mineralized zone solids (composite = 1 m).
- Resources were evaluated from drill hole and surface channel samples using an ID2 interpolation method in a multi-folder percent block model.
- The Measured category is defined by blocks having a volume of at least 25% within an envelope built at a distance of 10 m around existing Channels.
- The Indicated category is defined by the combination of blocks within a maximum distance of 15 m from existing stopes and blocks for which the average distance to drill hole composites is less than 30 m.
- Ounce (troy) = Metric tons x Grade / 31.10348. Calculations used metric units (metres, tonnes and g/t).
- The number of metric tons was rounded to the nearest hundred. Any discrepancies in the totals are due to rounding effects; rounding followed the recommendations in NI 43-101.
- InnovExplo is not aware of any known environmental, permitting, legal, title-related, taxation, socio-political or marketing issues or any other relevant issue that could materially affect the Mineral Resource Estimate.
- Input parameters used for MCoG estimation and Whittle pit design: Gold price: C$ 1,450; Gold selling cost: C$ 5.00; Mining costs: C$ 2.40; Processing cost: C$ 13.46; Transportation cost: C$ 0.25; Administration cost: C$ 4.18; Processing recovery: 93.9%; Mining recovery: 90.9%; Mining dilution: 10.0%; Overall pit slope: 52°.

14.1.4 Underground Resource Estimation

Given the density of the processed data, the search ellipse criteria, and the specific interpolation parameters, InnovExplo is of the opinion that the current Mineral Resource Estimate can be classified as Indicated and Inferred resources. The estimate is compliant with CIM standards and guidelines for reporting mineral resources and reserves. The Underground Mineral Resources were estimated using different gold cut-off grades and a minimum width of 3.0 m (true width). The selected underground cut-off of 2.0 g/t Au allowed the mineral potential of the deposit to be outlined for the underground mining option, outside the Whittle-optimized pit-shell.

The estimation of the underground cut-off grade (UCoG) was based on the parameters presented in Table 14.8.

Table 14.8 – Underground cut-off grade estimation for the Duparquet Project Mineral Resource Estimate

Input parameter	Value	Provided by
Exchange rate	1.00 USD : 1.01 CAD	InnovExplo
Gold price (CAD/oz)	1,450.00	InnovExplo
Selling cost (CAD/oz)	5.00	InnovExplo
Net price (CAD/oz)	**1,445.00**	
Mining costs (CAD/t)	58.00	InnovExplo
Milling costs (CAD/t)	13.46	InnovExplo
Total costs	**71.46**	
Processing recovery (%)	93.9	InnovExplo
Mining dilution (%)	15.0	InnovExplo

Gold Price (US$/oz)	UCoG (g/t)
800	3.42
900	3.04
1000	2.73
1100	2.48
1200	2.28
1300	2.1
1400	1.95
1450	**1.88**
1500	1.82
1600	1.7

The Underground Resource Estimate presented herein uses a rounded value of 2.00 g/t Au for the underground cut-off grade.

A volumetric analysis of the Underground Mineral Resource Estimate was carried out using a constraining surface constructed by merging the Whittle-optimized pit-shell with the bedrock surface in order to calculate the volume of any mineralized or envelope zone material contained within the bedrock but extending beyond the pit boundaries.

Table 14.9 displays the results of the In Situ[3] Mineral Resource Estimate for the underground portion of the Duparquet Project (60 mineralized material zones and the envelope zone). Figures 14.7 and 14.8 present the Whittle-optimized pit-shell trace along the North Zone and South 06 Zone, respectively. The detailed results of the Underground Mineral Resource Estimate are presented by zone in Appendix III.

[3] The term “in situ” is used to represent all the remaining mineral resources in place at the time of the updated estimate herein.

Table 14.9 – Underground Mineral Resource Estimate results (Indicated and Inferred resources) at different cut-off grades using a gold price of C$1,450/oz

Indicated Resources				
Zone	Cut-off g/t Au	Tonnes	Grade g/t Au	Au Oz
All Zones	> 5.0	121,800	6.10	23,882
	> 4.0	251,500	5.22	42,238
	> 3.5	430,700	4.60	63,679
	> 3.0	885,700	3.88	110,402
	> 2.5	1,837,100	3.28	193,536
	> 2.0	**3,520,700**	**2.78**	**314,275**
	> 1.5	6,073,600	2.34	456,603

Inferred Resources				
Zone	Cut-off g/t Au	Tonnes	Grade g/t Au	Au Oz
All Zones and Envelope	> 5.0	286,900	6.51	60,009
	> 4.0	597,100	5.42	104,038
	> 3.5	930,300	4.82	144,071
	> 3.0	1,925,900	3.98	246,315
	> 2.5	3,408,800	3.43	375,934
	> 2.0	**5,592,400**	**2.96**	**532,059**
	> 1.5	10,135,900	2.41	784,026

- The Independent and Qualified Persons for the Mineral Resource Estimate, as defined by NI 43-101, are Kenneth Williamson, M.Sc., P.Geo and Karine Brousseau, Eng. under the supervision of Carl Pelletier, B.Sc., P.Geo. (InnovExplo Inc.), and the effective date of the estimate is June 26, 2013.
- Mineral Resources which are not Mineral Reserves, do not have demonstrated economic viability.
- Results are presented undiluted, outside Whittle-optimized pit shells. The estimate includes 60 gold-bearing zones and the envelope containing isolated gold intercepts.
- Resources were compiled at cut-off grades of 1.5, 2.0, 2.5, 3.0, 3.5, 4.0 and 5.0 g/t Au.
- Cut-off grades must be re-evaluated in light of prevailing market conditions (gold price, exchange rate and mining cost).
- A fixed density of 2.73 g/cm^3 was used in the mineralized zones and in the envelope zone.
- A minimum true thickness of 3.0 m was applied, using the grade of the adjacent material when assayed, or a value of zero when not assayed.
- High grade capping was done on the raw data and established at 25.0 g/t Au for diamond drill hole assays and channel sample assays.
- Compositing was done on drill hole and channel samples sections falling within the mineralized zone solids (composite = 1 m).
- Resources were evaluated from drill hole and surface channel samples using an ID2 interpolation method in a multi-folder percent block model.
- The Indicated category is defined by the combination of blocks within a maximum distance of 15 m from existing stopes and blocks for which the average distance to drill hole composites is less than 30 m.
- Ounce (troy) = Metric tons x Grade / 31.10348. Calculations used metric units (metres, tonnes and g/t).
- The number of metric tons was rounded to the nearest hundred. Any discrepancies in the totals are due to rounding effects; rounding followed the recommendations in NI 43-101.
- InnovExplo is not aware of any known environmental, permitting, legal, title-related, taxation, socio-political or marketing issues or any other relevant issue that could materially affect the Mineral Resource Estimate.
- Parameters used for UCoG estimation: Gold price: C$ 1,450; Gold selling cost: C$ 5.00; Mining cost: C$ 58.00; Milling cost: C$ 13.46; Processing recovery: 93.9%; Mining dilution: 15.0%.

14.1.5 Tailings Mineral Resource Estimate

In August 2009, Genivar was contracted by Clifton Star to produce a NI 43-101 compliant Mineral Resource Estimate for a portion of the tailings area (Fillion, 2009), but the report was never published. On November 7, 2011, Clifton Star assigned InnovExplo the mandate of auditing Genivar's Mineral Resource Estimate in order to incorporate the results into a consolidated Mineral Resource Estimate, which was published as a NI 43-101 compliant Technical Report on July 5, 2012 (Brousseau et al., 2012). Details about the Mineral Resource Estimate for the tailings component of the Duparquet Project can be found in the above report.

The only change with respect to the previous estimate of July 5, 2012 is that the volumetrics have been re-calculated using a cut-off grade similar to the cut-off grade defined for the In-Pit Mineral Resource Estimate presented in Section 14.1.3 of this Report.

Table 14.10 displays the results of the In Situ[4] Resource Estimate for the tailings portion of the Duparquet Project (4 mineralized zones). Detailed results of the Tailings Mineral Resource Estimate are presented by zone in Appendix IV.

A portion of the historical tailings pond is currently located outside the Duparquet Property on government land (MERN property, see Figure 14.9). This portion of tailings was originally produced from the historical Beattie mine (located on the Duparquet Property). ***There is a risk that the MERN will not allow Clifton Star to recover the portion of the Mineral Resources in the tailings that are outside the Property boundary; if this were to happen, the authors wish to point out that this very small portion of the total Mineral Resources will only have a minor impact and will not affect the potential viability of the Project.***

14.2 Duparquet Mineral Resource Estimate Summary

At the request of Clifton Star, InnovExplo produced an updated Mineral Resource Estimate for the Duparquet Project. The Mineral Resource Estimate presented herein includes:

- An in-pit resource estimate, within a Whittle-optimized pit shell (Table 14.7)
- An underground resource estimate, outside the Whittle-optimized pit-shell (Table 14.9)
- A tailings resource estimate, for four (4) of the five (5) Beattie mine tailings ponds (Table 14.8)

Table 14.9 presents the combined resources by category for the Duparquet Project.

[4] The term "in situ" is used to represent all the remaining mineral resources in place at the time of the updated estimate herein.

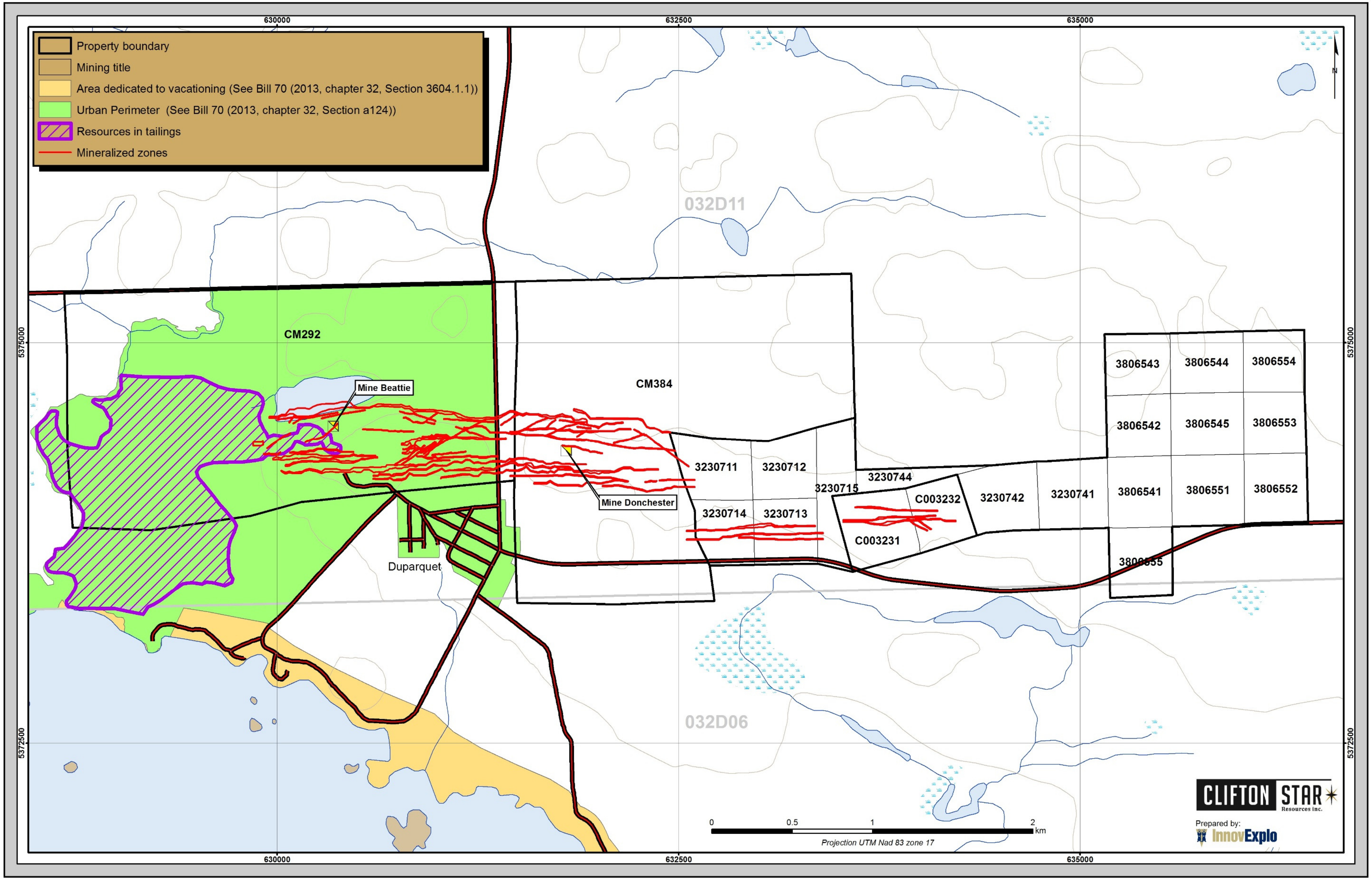

Figure 14.9 – Location map showing resource mining titles claims comprising the Duparquet Project

Table 14.10 – Tailings Mineral Resource Estimate results (Measured and Indicated resources) at different cut-off grades

Measured Resources						Indicated Resources				
Zone	Cut-off (g/t Au)	Tonnes	Grade (g/t Au)	Au (oz)		Zone	Cut-off (g/t Au)	Tonnes	Grade (g/t Au)	Au (oz)
Zones 1 and 2	> 0.90	16,000	2.36	1,215		Zones 3 and 4	> 0.90	2,424,500	1.01	79,036
	> 0.80	16,800	2.28	1,233			> 0.80	3,489,000	0.96	108,111
	> 0.70	18,000	2.18	1,262			> 0.70	4,053,900	0.94	122,102
	> 0.65	18,600	2.13	1,277			> 0.65	4,094,900	0.93	122,994
	> 0.60	19,000	2.10	1,284			> 0.60	4,104,400	0.93	123,189
	> 0.55	19,400	2.07	1,293			> 0.55	4,104,700	0.93	123,195
	> 0.50	19,400	2.07	1,290			> 0.50	4,104,800	0.93	123,196
	> 0.45	**19,600**	**2.06**	**1,295**			**> 0.45**	**4,105,000**	**0.93**	**123,200**
	> 0.40	19,900	2.03	1,297			> 0.40	4,105,200	0.93	123,203
	> 0.35	20,000	2.02	1,299			> 0.35	4,105,400	0.93	123,206

- The Independent and Qualified Persons for the Mineral Resource Estimate, as defined by NI 43-101, are Kenneth Williamson, M.Sc., P.Geo and Karine Brousseau, Eng. under the supervision of Carl Pelletier, B.Sc., P.Geo. (InnovExplo Inc.), and the effective date of the estimate is May 22, 2012.
- Mineral Resources which are not Mineral Reserves, do not have demonstrated economic viability.
- Results are presented undiluted and in situ. The estimate includes four (4) tailings zones.
- Tailings resources were compiled at cut-off grades of 0.35, 0.40, 0.45, 0.50, 0.55, 0.60, 0.65, 0.70, 0.80 and 0.9 g/t Au.
- Cut-off grades must be re-evaluated in light of prevailing market conditions (gold price, exchange rate and mining cost).
- A fixed density of 1.45 g/cm^3 was used in zones and waste.
- High grade capping was done on the raw data and established at 13.0 g/t Au for Zone 1, 3.5 g/t Au for Zone 2, 1.7 g/t Au for Zone 3 and 2.2 g/t Au for Zone 4.
- Compositing was done on drill hole sections falling within the mineralized zone solids (composite = 0.5 m).
- Resources were evaluated from drill hole samples using an ID2 interpolation method in a block model.
- The Measured and Indicated categories were defined based on the drill hole spacing (Measured: zones 1 and 2 = 30m x 30m grid; Indicated: zone 3 = 100m x 100m grid and zone 4 = 200m x 200m grid).
- Ounce (troy) = Metric tons x Grade / 31.10348. Calculations used metric units (metres, tonnes and g/t).
- The number of metric tons was rounded to the nearest hundred. Any discrepancies in the totals are due to rounding effects; rounding followed the recommendations in NI 43-101.
- InnovExplo is not aware of any known environmental, permitting, legal, title-related, taxation, socio-political or marketing issues or any other relevant issue that could materially affect the Mineral Resource Estimate.

Table 14.11 – Global Mineral Resource Estimate results (Measured, Indicated and Inferred Resources) for the Duparquet Project

Resources type	Parameters	Area			TOTAL
		Tailings	In-Pit	Underground	
	Cut-off (g/t)	> 0.45	> 0.45	> 2.00	
Measured	Tonnes (t)	19,600	165,100		184,700
	Grade (g/t)	2.06	1.45		1.52
	Au (Oz)	1,295	7,711		9,006
Indicated	Tonnes (t)	4,105,000	53,070,600	3,520,700	60,696,300
	Grade (g/t)	0.93	1.56	2.78	1.59
	Au (Oz)	123,200	2,666,690	314,275	3,104,165
Measured + Indicated	*Tonnes (t)*	***4,124,600***	***53,235,700***	***3,520,700***	***60,881,000***
	*Grade (g/t)**	***0.94***	***1.56***	***2.78***	***1.59***
	Au (Oz)	***124,495***	***2,674,401***	***314,275***	***3,113,171***
Inferred	Tonnes (t)		24,092,300	5,592,400	29,684,700
	Grade (g/t)		1.18	2.96	1.51
	Au (Oz)		910,631	532,059	1,442,689

*: average weigthed on tonnes

- The Independent and Qualified Persons for the Mineral Resource Estimate, as defined by NI 43-101, are Kenneth Williamson, M.Sc., P.Geo and Karine Brousseau, Eng. under the supervision of Carl Pelletier, B.Sc., P.Geo. (InnovExplo Inc.), and the effective date of the estimate is May 22, 2012 for the tailings and June 26, 2013 for the In-Pit and Underground mineral resources.
- Mineral Resources which are not Mineral Reserves, do not have demonstrated economic viability.
- Tailings results are presented undiluted and in situ. The estimate includes four (4) tailings ponds.
- In-Pit results are presented undiluted and in situ, within Whittle-optimized pit shells. The estimate includes 60 gold-bearing zones and the envelope containing isolated gold intercepts.
- Underground results are presented undiluted and in situ, outside Whittle-optimized pit shells. The estimate includes 60 gold-bearing zones and the envelope containing isolated gold intercepts.
- Tailings resources were compiled at cut-off grades of 0.35, 0.40, 0.45, 0.50, 0.55, 0.60, 0.65, 0.70, 0.80 and 0.9 g/t Au.
- In-Pit resources were compiled at cut-off grades of 0.35, 0.40, 0.45, 0.50, 0.55, 0.60, 0.65, 0.70, 0.80 and 0.9 g/t Au.
- Underground resources were compiled at cut-off grades of 1.5, 2.0, 2.5, 3.0, 3.5, 4.0 and 5.0 g/t Au.
- Cut-off grades must be re-evaluated in light of prevailing market conditions (gold price, exchange rate and mining cost).
- Tailings: A fixed density of 1.45 g/cm3 was used in zones and waste.
- In-Pit and Underground: A fixed density of 2.73 g/cm3 was used in the mineralized zones and in the envelope zone.
- In-Pit and Underground: A minimum true thickness of 3.0 m was applied, using the grade of the adjacent material when assayed, or a value of zero when not assayed.
- Tailings: High grade capping was done on the raw data and established at 13.0 g/t Au for Zone 1, 3.5 g/t Au for Zone 2, 1.7 g/t Au for Zone 3 and 2.2 g/t Au for Zone 4.
- In-Pit and Underground: High grade capping was done on the raw data and established at 25.0 g/t Au for diamond drill hole assays and channel sample assays.
- Tailings: Compositing was done on drill hole sections falling within the mineralized zone solids (composite = 0.5 m).
- In-Pit and Underground: Compositing was done on drill hole and channel sample sections falling within the mineralized zone solids (composite = 1 m).
- Tailings: Resources were evaluated from drill hole and surface channel samples using an ID2 interpolation method in a block model.
- In-Pit and Underground: Resources were evaluated from drill hole and surface channel samples using an ID2 interpolation method in a multi-folder percent block model.
- Tailings: Measured and Indicated categories were defined based on the drill hole spacing (Measured: zones 1 and 2 = 30m x 30m grid; Indicated: zone 3 = 100m x 100m grid and zone 4 = 200m x 200m grid).
- The In-Pit Measured category is defined by blocks having a volume of at least 25% within an envelope built at a distance of 10 m around existing channels.
- The In-Pit and Underground Indicated category is defined by the combination of blocks within a maximum distance of 15m from existing stopes and blocks for which the average distance to drill hole composites is less than 30 m.
- Ounce (troy) = Metric tons x Grade / 31.10348. Calculations used metric units (metres, tonnes and g/t).
- The number of metric tons was rounded to the nearest hundred. Any discrepancies in the totals are due to rounding effects; rounding followed the recommendations in NI 43-101.
- InnovExplo is not aware of any known environmental, permitting, legal, title-related, taxation, socio-political or marketing issues or any other relevant issue that could materially affect the Mineral Resource Estimate.
- Input parameters used for MCoG estimation and Whittle pit design: Gold price: C$ 1,450; Gold selling cost: C$ 5.00; Mining costs: C$ 2.40; Processing cost: C$ 13.46; Transportation cost: C$ 0.25; Administration cost: C$ 4.18; Processing recovery: 93.9%; Mining recovery: 90.9%; Mining dilution: 10.0%; Overall pit slope: 52°.
- Parameters used for UCoG estimation: Gold price: C$ 1,450; Gold selling cost: C$ 5.00; Mining cost: C$ 58.00; Milling cost: C$ 13.46; Processing recovery: 93.9%; Mining dilution: 15.0%.

15. MINERAL RESERVE ESTIMATE

The reserves for the detailed pit design have been calculated in accordance with the definitions and guidelines adopted by the Canadian Institute of Mining, Metallurgy and Petroleum (CIM Standards on Mineral Resources and Reserves) in August 2000.

The effective date of the Mineral Reserve is March 26, 2014, the date on which the latest processing cost estimation was concluded.

The PFS is based on an earlier report titled "Technical Report and Mineral Resource Estimate Update for the Duparquet Project (according to Regulation 43-101 and Form 43-101F1)", dated August 2, 2013.

The mineral resource block model developed by InnovExplo was imported into Whittle™ software from Dassault Systèmes GEOVIA (formerly Gemcom Software International). Design parameters, such as operating costs, mine recovery, dilution and gold price, were used to generate an optimal pit shell.

InnovExplo evaluated dilution and mine recovery by performing an analysis of mineable blocks on typical cross-sections at the mill cut-off grade. After a cross-sectional evaluation, average dilution was estimated at 10% at a grade of 0 g/t, and mine recovery was estimated at 95% assuming good blasting and dilution control practices.

To integrate dilution in Whittle, an initial 5% dilution factor was used when performing reblocking and merging the original small blocks (5x5x5) into grade bands (high and low). This feature defines a maximum number of parcels per reblock blocks. This action regroups the smaller blocks of 5x5x5 into larger parcels, by grade interval. In this evaluation, two parcels were considered. The resulting parcel includes blocks lower than the cut-off grade that are considered internal dilution. If the average grade of the parcel is higher than the cut-off grade, it is considered as ore and sent to the mill. This dilution was estimated at 5%. An additional 5% dilution factor was added to the Whittle parameters for a total dilution factor of 10%.

Iterations were performed to generate a pit design that fits the selected pit shell. To do so, a pit design, including a ramp and catch bench, is made in the selected pitshell. For the Duparquet project, the initial pit wall angle was too steep to accommodate the ramp. The slope angle was smoothed until a ramp could fit in the Whittle pit shell. Many iterations were needed before finding an adequate pit wall angle.

Pit slopes were set at 29° in the overburden, 45° on the north side of the pits, and 48° on the south side of the pit. The ramp was designed on the north side of the pit due to the constraints imposed by the golf course and houses on the south side.

The concurrent prefeasibility work allowed InnovExplo and Roche Ltd Consulting Group ("Roche") to better quantify the operating costs. These costs were used for a final pit optimization in Whittle. These parameters are presented in Table 15.1.

Table 15.1 – Whittle parameters

Input parameters		Value	Provided by
Gold Price		1,417.5 $CAN/oz	InnovExplo
Gold selling cost		5 $/oz	InnovExplo
Dilution		*5 %	InnovExplo
Mining recovery		95 %	InnovExplo
Milling recovery		93.9 %	Roche
Overburden Cost		0.80 $/t	InnovExplo
Mining cost		1.93 $/t	InnovExplo
General & Administration		3.12 $/t	InnovExplo
Processing Cost		16.77 $/t	Roche
Environmental monitoring		0.20 $/t	Roche
Stockpile rehandling cost		0.88 $/t	InnovExplo
Pit slope	North	45 °	InnovExplo
	South	48 °	InnovExplo
	OVB	29 °	InnovExplo

Note: An additional 5% dilution factor is included in the reblocking for a total dilution factor of 10%.

The mining cost of $1.93/t represents the initial cost to mine the rock present at the surface of the open pit. An incremental hauling cost of $0.21 per kilometre of ramp was added, depending on pit depth.

The open pit production is supplemented by 4.1 Mt of available old tailings. The old tailings will be processed during the beginning of the Project at a rate of 750,000 tonnes per year.

The global In-pit and Tailings Proven and Probable Reserves total 39,363,000 tonnes at an average grade of 1.50 g/t (1,895,530 contained ounces of gold). The Whittle pit shell selected for this PFS generates 35,238,429 tonnes of ore, including dilution and losses. Another 4,124,600 tonnes from the old tailings completes the resources for a total of 39,363,029 tonnes of ore from which 0.5% is Proven Reserves and 99.5% is Probable Reserves. A cut-off grade of 0.45 g/t has been considered for the tailings material and 0.51 g/t for the pit. The Mineral Reserve Estimate for the Duparquet Project is presented in Table 15.2.

Table 15.2 – Mineral Reserve Estimate

Reserves type	Parameters	Area		TOTAL
		Tailings	In-Pit 1	
	Cut-off (g/t)	> 0.45	> 0.51	
Proven	Tonnes (t)	19,600	175,100	194,700
	Grade (g/t)	2.06	1.31	1.38
	Au (Oz)	1,295	7,372	8,667
Probable	Tonnes (t)	4,105,000	35,063,400	39,168,400
	Grade (g/t)	0.93	1.56	1.50
	Au (Oz)	123,200	1,763,664	1,886,864
Proven + Probable	***Tonnes (t)***	***4,124,600***	***35,238,400***	39,363,000
	Grade (g/t)	***0.94***	***1.56***	**1.50**
	Au (Oz)	***124,495***	***1,771,035***	**1,895,530**

16. MINING METHODS

16.1 Introduction

This section describes the results of the technical work undertaken by InnovExplo to produce a mine plan for the current Prefeasibility Study (PFS) for the Duparquet open pit project. The PFS is based on a Mineral Resource Estimate produced by InnovExplo in an earlier report prepared for the issuer titled "Technical Report and Mineral Resource Estimate Update for the Duparquet Project (according to Regulation 43-101 and Form 43-101F1)", dated August 2, 2013. The PFS is compliant with NI 43-101.

Mining of the Duparquet deposit has been designed as an open pit with a planned production of 3,650,000 tonnes per year (3.65M tpy) or 10,000 tonnes per day (tpd), 365 days per year of mill operation and 360 working days operation for the pit. The open pit production is supplemented by 4.1 Mt of available old tailings. The old tailings will be processed during the beginning of the project at a rate of 750,000 tonnes per year.

16.2 Geotechnical Study

A preliminary geotechnical review was prepared by Jane Alcott, M.Sc., Eng. of InnovExplo under the supervision of Sylvie Poirier, Eng., for the Preliminary Economic Assessment ("PEA") study published on February 28, 2013. No additional geotechnical data was evaluated for the current study. The preliminary geotechnical assessment and design recommendations are shown below.

16.2.1 Structural Data

The Duparquet geotechnical review was limited to a review of existing data. Following discussions with Louis Martin (Vice-President Exploration, Clifton Star), structural data was taken from the report titled *Analyse Structurale de Syenite de Beattie à Duparquet, Abitibi, Canada* (Bigot, 2011). This study was conducted by researchers at the Université de Montréal for Osisko Mining Corporation. The study intended to identify and map the faults, joints and tension structures associated with the Duparquet mineralization. The data set consisted of 79 measurements representing faults (48), joints (16), and tension veins (15). All data measurements were taken from surface outcrops. The majority of the measurements correspond to N-S and E-W orientations with dips typically in excess of 60° (Bigot, 2011). Figure 16.1 summarizes the pole plots, orientation rosettes, pole contours and major planes for the entire data set. The major planes are summarized in Table 16.1.

Table 16.1 – Summary of structural data analysis

Structural set	Major planes (dip / dip direction)
1	65 / 169
2	76 / 005
3	74 / 273

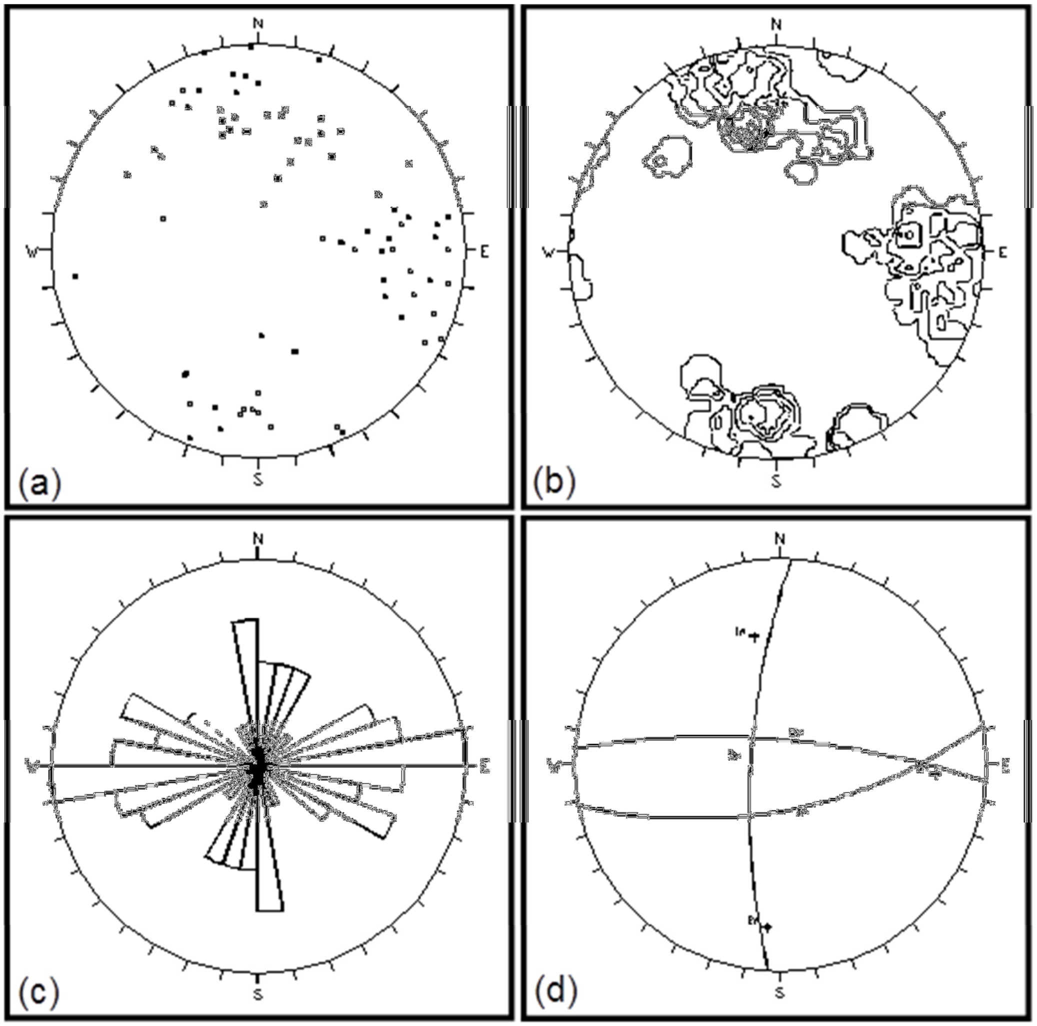

Figure 16.1 – Structural data analysis: (a) poles, (b) contours, (c) rosette and (d) major planes

16.2.2 Duparquet Project Pit Slope Recommendations

Based on the available structural data, the fault structures will control the proposed pit slope configurations. At this time there is not enough geotechnical data to permit zoning and zone specific pit slope configurations. Therefore, two (2) pit-wide slope configurations are proposed. These are summarized in Table 16.2 and Figure 16.2.

Table 16.2 – Summary of pit slope recommendations for the Duparquet Project

	Bench Face Angle (BFA)	Bench Height (Hv)	Berm Width (W)	Overall Angle
3 x 5 m benches	75°	15 m	7.5 m	52.4°
2 x 10 m benches	75°	20 m	8.5 m	55°

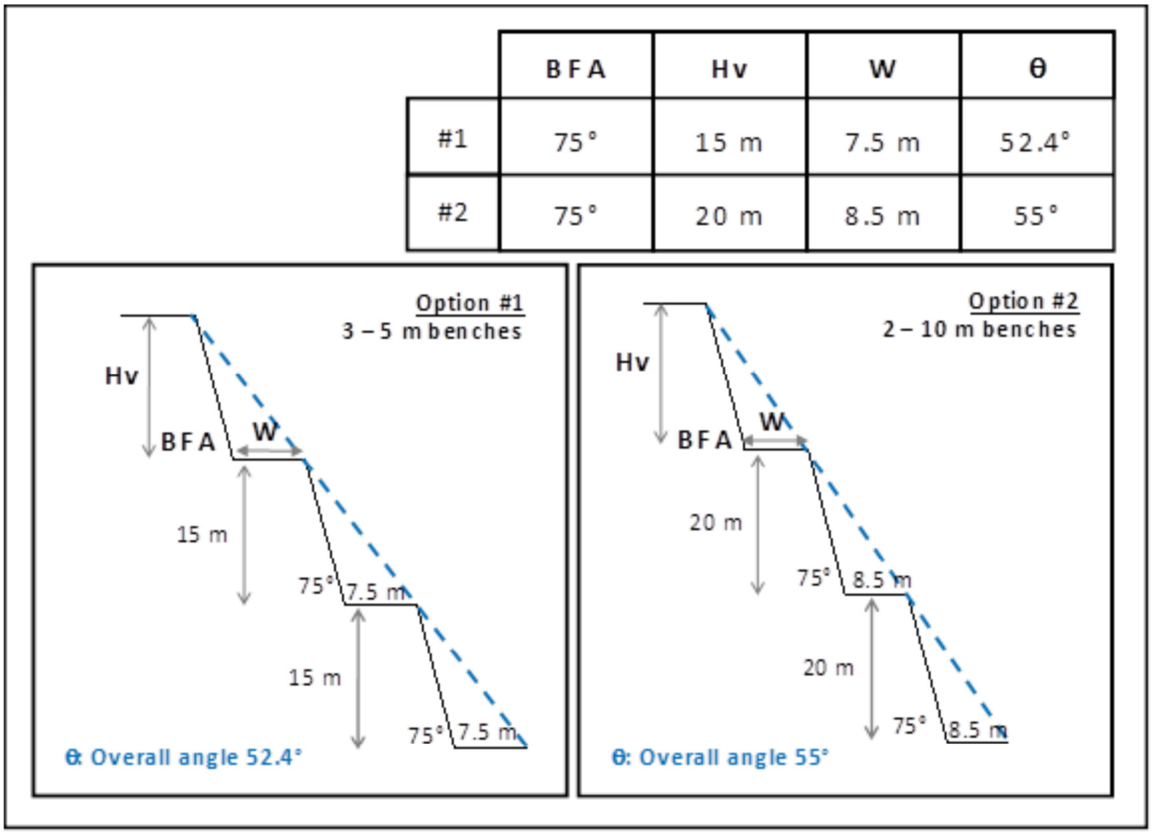

Figure 16.2 – Recommended slope configurations for the Duparquet Project

In order to advance to a feasibility study, it is recommended that the proposed slope configurations be verified. This involves confirming the structural data over the proposed open pit footprint. Ideally, this would involve a geotechnical drilling program with a minimum of one hole oriented perpendicular to each of the four pit walls (North, South, East and West).

16.3 Open Pit Optimization Methodology

The objective of pit optimization and mine design is to evaluate the mineable resource and use it as a basis for mine scheduling and economic analysis.

The mineral resource block model developed by InnovExplo has been imported into Whittle™ software from Dassault Systèmes GEOVIA (formerly Gemcom Software

International). Design parameters such as operating costs, mine recovery, dilution and gold price were used to generate an optimal pit shell.
The types of material included in the block model used in the optimization are:

- Overburden;
- Waste host rock;
- Water filled-in mined area, including:
    - Glory Hole Lake (old mine breaking through at surface);
    - Old stopes and drifts;
- Indicated and measured resources;
- Blocks coded according to their location east or west of Route 393

In the block model, all blocks having at least 0.01% of their volume contained within stope solids were rock-coded as stope. Following the interpolation step, the grades of all blocks rock-coded as stopes were set to zero (0). InnovExplo decided that such conservative manipulation was necessary in order to compensate for the risk that blocks near stopes are either mined out or inaccessible due to safety factors. The optimization in Whittle does not take into consideration changes in productivity around the old underground workings.

16.3.1 Whittle Parameters and Constraints

In Whittle, reblocking was used at various levels of integration. The original resource block model has a block size (in metres) of 5x5x5. A first reblocking was done at 20x20x10. The first reblocking was used to perform a dilution study by preserving the original small blocks (5x5x5) for mining selectivity and merging them in grade bands (high and low grade). The resulting dilution is estimated to be equivalent to about 5% of the original block model. An additional 5% dilution factor was added to the Whittle parameters for a total dilution factor of 10%.

Another reblocking of 60x60x20 was performed. This larger reblocking is done to smooth pit walls and floor bottoms, to make room for operating mining equipment, to make room for ramps, and for blast design optimization. It facilitates the pit design when integrating the ramps and berms using the GEOVIA pit design module.

In order to generate multiple pit shells, the slope angle was set at 55°, as provided in the geotechnical section. Preliminary up-to-date parameters were used including economical, production and variable mining costs according to pit depth. Also, a number of constraints were integrated:

- A limit was added to constrain the pit size within the claim limit;
- A constraint of 30 m was added on each side of Route 393 in order to preserve the provincial highway.
- A constraint was put on the golf property perimeter;
- A constraint was added to preserve the access road leading to the golf course and to preserve the houses on the south side of this road;

Although pit shells produced by the Lerchs-Grossman algorithm are optimal from a financial point of view, in many cases the resulting “onion skin” pits it produces cannot be practically mined. More realistic pits can be achieved using mining direction constraints in Whittle by specifying or forcing the mining to start from a particular direction in the pit shell.

In order to develop a pit design for the Duparquet Project that is more practical to mine, an eastward mining direction was used in Whittle. By applying a mining direction it is possible to control pit development more efficiently to produce a more realistic mining scenario. This allowed the pit to start on the west side and to gradually expand towards the road. It allows a better use of equipment by reducing movements as a result of more centralized mining activities. The mining direction also reduced the size of the waste stockpiles for the Duparquet Project, since it makes it possible to dump waste from the east pit into the west pit once it is completed.

Considering all parameters and constraints listed above, an optimal preliminary pit shell was selected and used for further optimization to serve as a basis of a mine plan for the current Prefeasibility Study. At this point in the process, advanced optimization features were used to optimize the mining phases, the mine schedule (Milawa NPV), and the stockpiles and cut-off grades (SPCO). Simultaneous Optimization (SIMO) was also used but this analysis was not successfully completed.

Following the pit shell optimization, mine design was initiated. The mine design process converts the LG shells into operational open pit mine designs. This is an iterative process and once completed, the total contents of the designed pit should not differ considerably from the contents of the shell on which it is based.

The first step in the iteration process is to make a pit design that fits the selected pit shell. To do so, a pit design, including a ramp and catch bench, is made in the selected pitshell. For the Duparquet Project, the initial pit wall angle of 55º was too steep to accommodate the ramp. The slope angle was smoothed until a ramp could fit in the Whittle pit shell. Many iterations were needed before finding an adequate pit wall angle.

Pit slopes were set at 29° in the overburden, 45° on the north side of the pits, and 48° on the south side of the pit. The ramp was designed on the north side of the pit due to the constraints imposed by the golf course and houses on the south side and due to the location of the mill.

The mining sequence obtained from the Whittle pit shell that suited the pit designwas used to initiate cost estimation for the current PFS.

16.3.1.1 Final Whittle Parameters

The concurrent prefeasibility work allowed InnovExplo and Roche Ltd Consulting Group (“Roche”) to better quantify the operating costs. These costs were used for a final pit optimization. These parameters are presented in Table 16.3.

Table 16.3 – Whittle parameters

Input parameters		Value	Provided by
Gold Price		1,417.5 $CAN/oz	InnovExplo
Gold selling cost		5 $/oz	InnovExplo
Dilution		*5 %	InnovExplo
Mining recovery		95 %	InnovExplo
Milling recovery		93.9 %	Roche
Overburden Cost		0.80 $/t	InnovExplo
Mining cost		1.93 $/t	InnovExplo
General & Administration		3.12 $/t	InnovExplo
Processing Cost		16.77 $/t	Roche
Environmental monitoring		0.20 $/t	Roche
Stockpile rehandling cost		0.88 $/t	InnovExplo
Pit slope	North	45 °	InnovExplo
	South	48 °	InnovExplo
	OVB	29 °	InnovExplo

Note: An additional 5% dilution factor is included in the reblocking for a total dilution factor of 10%.

The mining cost of $1.93/t represents the initial cost to mine the rock present at the surface of the open pit. An incremental hauling cost of $0.21 per kilometre of ramp was added, depending on pit depth.

16.4 Final Optimization Results

A series of potential pit shells were generated by Whittle for the Duparquet Project based on varying revenue factors. The pit shells were generated from revenue factors ranging from 0.25 to 1.0. The revenue factors were applied to the base case gold price of C$1,417.50/oz. The results of the potential pit shells are summarized in Figure 16.3.

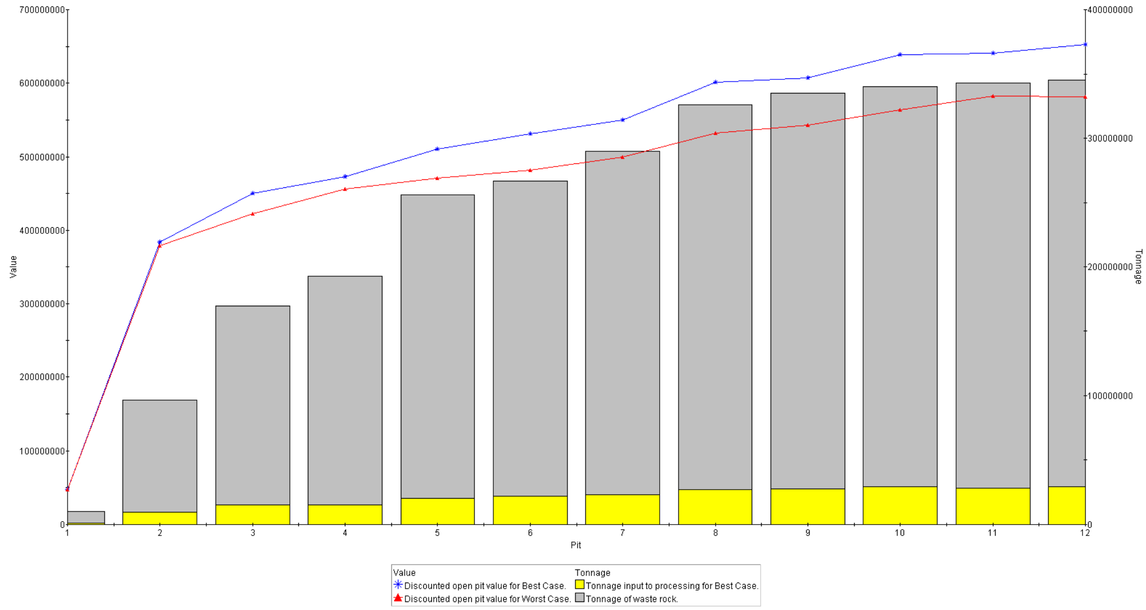

Figure 16.3 – Whittle pit shell optimization results

For each pit shell, Whittle calculated a base case scenario assuming that every internal pit shell is mined out sequentially, and a worst case scenario assuming that mining takes place bench by bench. The best case scenario provides the upper NPV boundary limit while the worst case scenario provides the lower NPV boundary limit. The optimal pit selected usually ranges between the best and worst case boundary limits. Based on the analysis of the Whittle pit shells, pit shell 11 with a revenue factor of 0.8 was chosen as the base case pit shell for further phasing and scheduling for the Duparquet Project. This pit shell has been chosen because it was the best case in the worst case scenario; this ensures that optimization remains on the conservative side and that a robust mine plan is proposed.

Phasing of the pit was investigated by using smaller pit shells to determine project pushbacks. Three (3) pushbacks were necessary on the Beattie pit and two (2) pushbacks were necessary on the Donchester and Central Duparquet pit.

The Whittle pit shell selected for this PFS generates 35,238,429 tonnes of ore. Another 4,124,600 tonnes from the old tailings completes the resources for a total of 39,363,029 tonnes of ore from which 0.5% is Measured Resources and 99.5% is Indicated Resources. The pit also generates 291,213,881 tonnes of waste and 23,398,085 tonnes of overburden, resulting in a LOM strip ratio of 8.26 to 1. Taking into account the overburden, the LOM strip ratio is 8.92 to 1. The estimated life-of-mine (LOM) average grade is 1.50 g/t, including the ore from the pit and from the old tailings. A total of 1,894,365 ounces of gold would be produced over the LOM. Table 16.4 presents the distribution of mineralized material.

Table 16.4 - Mineralized material distribution

		Proven reseves			Probable reserves		
Area	Cut-off	Tonnes (t)	Grade (g/t)	Au (Oz)	Tonnes (t)	Grade (g/t)	Au (Oz)
In Pit	≥ 0.51	175,100	1.31	7,373	35,063,400	1.56	1,763,666
Old tailings	≥ 0.45	19,600	0.93	586	4,105,000	0.93	122,740
Total		**194,700**	**1.27**	**7,959**	**39,168,400**	**1.50**	**1,886,406**

16.5 Final Open Pit Mine Design

From the selected Whittle optimized pit shell, a ramp was designed into the pit and the design was smoothed to remove irregularities that are inconsistent with operating practices. The detailed pit design work was carried out using the GEOVIA™ package software. The pit design includes a haulage ramp access to all benches, except the last two (2) benches that will only have temporary access ramp, as well as the pit slope design and the safety berm as presented in the previously described geotechnical study.

The designed pits are approximately 3,508 m long by 1,120 m wide and 360 m deep.

16.5.1 Haul Road

The ramp was designed for double lane traffic in general, except for the Central Duparquet pit in the east. The northeast side of the Beattie pit is also designed for single-lane traffic (Fig. 16.3). The detailed open pit design is shown in Figure 16.4.

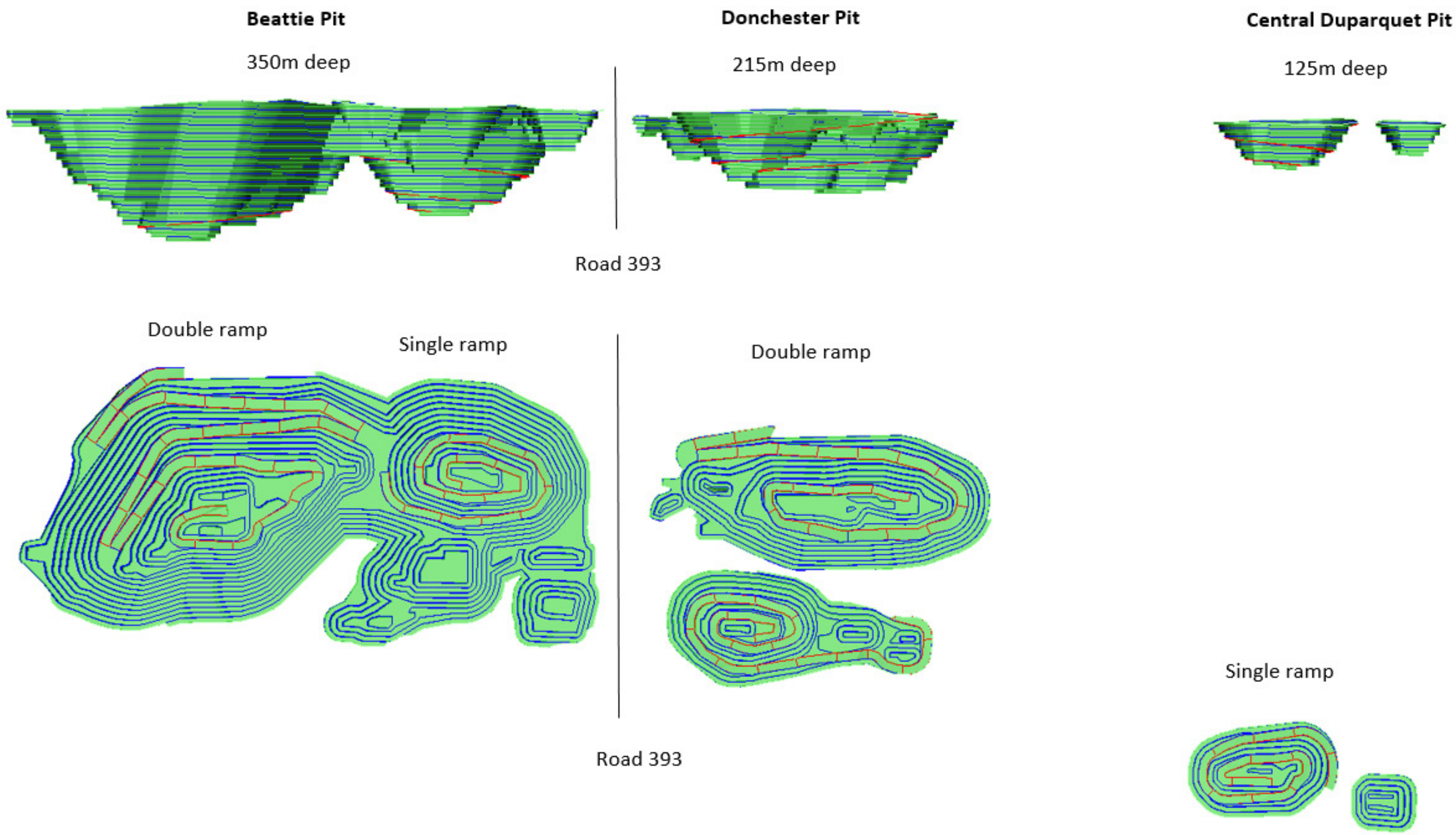

Figure 16.4 – Longitudinal and plan view of the Duparquet Pit

For the single lane traffic, the travel width should be not less than 2 times the width of the biggest equipment, with allowance for a bumper when a fall of more than 3 m is possible. For the double lane traffic, the travel width should be not less than 3 times the width of the biggest equipment, with allowance for a bumper when a fall of more than 3 m is possible. The muck bumper needs to be at least half the height of the largest tire on any equipment hauling on the road. The berm slope was estimated at 35°, which is lightly less than the angle of repose. It must be placed on the edge of the haulage road whenever a drop is greater than 3 m. Ditches are included in the travel width.

The biggest equipment in this project is the 136-tonne capacity off-road haul truck, and road widths are based on its specifications. The 136-tonne rear-dump truck operating width is 7.06 m and half the tire height is 1.5 m. Based on this truck, the haul road design is as follows:

- Single lane traffic with one bumper: 18.5 m
- Single lane traffic with two bumpers: 23 m
- Double lane traffic with one bumper: 25.5 m
- Double lane traffic with two bumpers: 30 m

Based on the productivity rate and sensitivity to the stripping ratio, it was decided that for the last five (5) benches, three would be served by a one-way ramp. In order to reduce the volume of waste to be stripped, the last two (2) benches will not have permanent ramp access.

The ramp gradient is 10%, and the ramp exit is on the north side of the pit to minimize the haulage distance to the waste dump, stockpile or mill, the latter of which presently located on the north side of the Beattie and Donchester pits.

16.5.2 Bench Height

The Duparquet pit design is based on double benching with 10-m single benches. This option was chosen based on the preliminary geotechnical assessment presented in Section 16.2 of the Report.

16.5.3 Berm Width

A minimum catch bench of 8.5 m has been considered in the preliminary geotechnical study when using double benches.

16.6 Stockpile Design

A stockpile is used in a way that optimizes the cut-off grade throughout the mining period. A maximum of 5.7 Mt of ore will be directed to the stockpile. The minimum cut-off of that stockpile is estimated at 0.51 g/t.

16.7 Waste Management

Three waste piles are planned, two on the west side Route 393 and one on the east side. This is meant to limit equipment traveling between each side of the road. Most of the waste generated from the west pit will remain on this side of the road and the same for the waste generated on the east side. According to the mine plan sequence, it will be possible to send approximately 25 Mt of waste from the east pit to the west pit once mining of the west pit is complete. This will allow the size of the waste pile to be reduced.

16.8 Mine Dilution and Mine Recovery Estimation

InnovExplo evaluated dilution and mine recovery by performing an analysis of mineable blocks on typical cross-sections at the mill cut-off grade. After a cross-sectional evaluation, average dilution was estimated at 10% at a grade of 0 g/t, and mine recovery was estimated at 95% assuming good blasting and dilution control practices.

To integrate dilution in Whittle, an initial 5% dilution factor was used when performing reblocking and merging the original small blocks (5x5x5) into grade bands (high and low). This feature defines a maximum number of parcels per reblock blocks. This action regroups the smaller blocks of 5x5x5 into larger parcels, by grade interval. In this evaluation, two parcels were considered. The resulting parcel includes blocks lower than the cut-off grade that are considered internal dilution. If the average grade of the parcel is higher than the cut-off grade, it is considered as ore and sent to the mill. This dilution was estimated at 5%. An additional 5% dilution factor was added to the Whittle parameters for a total dilution factor of 10%.

16.9 Mine Production Schedule

The life-of-mine (LOM) for the Duparquet Pit was based on supplying the mill with 3,650,000 tonnes of ore per year. Initially, 4.1 Mt of tailings would be reserved to supplement the mill when necessary, but it was later decided to process the tailings at a rate of 750,000 tonnes per year right from the start in order to clean the tailings area to make room for the waste stockpile #2 for the west pit.

The mining schedule will require the extraction of 39,363,029 tonnes of ore and 291,213,881 tonnes of waste rock, resulting in a LOM strip ratio of 8.26 to 1. The overburden consists of 23,398,919 tonnes. Taking into account the overburden, the LOM strip ratio is 8.93 to 1. The stripping ratio (waste to ore) will not necessarily be constant. Stockpiles will be used to vary the cut-off grade in order to optimize the economics of the Project. The ore, waste and tailings production distribution plan is presented in Table 16.5 on a yearly basis and in Table 16.7 on a daily basis.

The LOM for the Duparquet Pit will be spread over 11 years, preceded by a 4-year pre-production period. This schedule will yield a yearly production of 3,650,000 tonnes (Table 16.6).

In order to provide material for construction of road, foundations, dyke and tailings, waste pre-stripping work will be initiated in PP2. Also, the existing pit contains approximately 5.7 Mm3 of water. At the start of the projectProject, it is estimated that 2 Mm3 need to be dewatered, and that it will take approximately six (6) months.

The pre-production period is estimated at 4 years and identified as year PP1 to year PP4. During Year PP4, 894,717 tonnes of ore will be mined from which 400,000 tonnes at an average grade of 1.14 g/t will be used for the mill start up and the rest will be send to the stockpile to help support the first year of production. Also 135,450 tonnes at an average grade of 0.93 g/t from the old tailings will be processed during the start-up period. A total of 16,608 ounces of gold will be recovered from which

3,398 ounces will come from processing the old tailings with a recovery factor of 83.95%, and 13,210 ounces will come from processing the mineralized pit material with a recovery of 90.1%.

From Year 1 to Year 5, 2,900,000 tonnes of mineralized material processed will come from the pit or the stockpile containing mineralized material from the pit, and 750,000 tonnes will be from the old tailings. During these years, full production of 3,650,000 tonnes is reached, producing an average of 173,000 ounces per year.

During Year 6, the rest of the old tailings will be processed, representing 239,150 tonnes at a grade of 0.93 g/t. In addition, 3,275,400 tonnes of mineralized material from the pit will be processed at an average grade of 1.32 g/t. A total of 131,004 ounces of gold will be recovered during that year.

From Year 7 to Year 10, all the ore will be provided from the pit or the stockpile. The yearly process rate will be constant and will be 3,650,000 tonnes. A total of 636,405 ounces of gold will be recovered. Year 10 corresponds to the final year of pit operation, and manpower requirements will gradually decrease over the course of the year.

During Year 11, only the stockpile will feed the mill, for a total of 2,463,029 tonnes at a grade of 0.70 g/t, producing 50,214 ounces of gold.

Table 16.5 – Production by year and stripping ratio

Year	Stripping		Mineralized material		Total production		Stripping ratio
	Overburden (tpy)	Waste (tpy)	From pit (tpy)	From old tailings (tpy)	Without overburden (tpy)	With overburden (tpy)	
PP1	-	-	-	-	-	-	
PP2	513,834	800,000	-	-	800,000	1,313,834	
PP3	3,467,354	5,398,400	-	-	5,398,400	8,865,754	
PP4	402,075	626,000	894,717	135,450	1,520,717	1,922,792	1.15
1	6,272,956	27,162,421	3,288,695	750,000	30,451,116	36,724,072	10.17
2	-	28,864,822	5,727,596	750,000	34,592,418	34,592,418	5.04
3	1,859,667	29,360,581	3,646,700	750,000	33,007,281	34,866,948	8.56
4	1,260,908	31,612,674	2,236,841	750,000	33,849,515	35,110,423	14.70
5	1,291,105	29,831,261	3,562,544	750,000	33,393,805	34,684,910	8.74
6	4,446,345	28,281,662	2,884,832	239,150	31,166,494	35,612,839	11.34
7	30,288	33,081,261	2,831,059	-	35,912,320	35,942,608	11.70
8	2,954,360	29,211,166	3,649,988	-	32,861,154	35,815,514	8.81
9	743,041	31,605,924	3,650,216	-	35,256,140	35,999,181	8.86
10	156,985	15,377,709	2,865,243	-	18,242,952	18,399,937	5.42
Total	**23,398,919**	**291,213,881**	**35,238,429**	**4,124,600**	**326,452,310**	**349,851,229**	**8.93**

Table 16.6 – Mineralized material processing

Year	To the mill from								
	ROM		Stockpile		Old tailings		Total		Total ounces
	Tonnage (t)	Grade (g/t)	Tonnage (t)	Grade (g/t)	Tonnage (t)	Grade (g/t)	Tonnage (t)	Grade (g/t)	Au recovered
PP4	400,000	1.14			135,450	0.93	535,450	1.09	16,608
1	2,900,000	1.90			750,000	0.95	3,650,000	1.70	178,823
2	2,900,000	2.01			750,000	0.93	3,650,000	1.79	187,928
3	2,899,853	1.60	147	2.32	750,000	0.93	3,650,000	1.47	153,598
4	1,913,865	2.22	986,135	1.51	750,000	0.93	3,650,000	1.76	185,127
5	2,897,632	1.68	2,368	1.42	750,000	0.93	3,650,000	1.53	159,869
6	2,543,520	1.44	731,880	0.89	239,150	0.93	3,514,550	1.29	131,004
7	2,827,860	1.44	822,140	0.97			3,650,000	1.33	140,702
8	3,649,988	1.59	12	0.71			3,650,000	1.59	168,114
9	3,650,000	1.42					3,650,000	1.42	149,781
10	2,865,243	1.95	784,757	0.71			3,650,000	1.68	177,808
11			2,463,029	0.70			2,463,029	0.70	50,214
Total	**29,447,961**	**1.69**	**5,790,468**	**0.90**	**4,124,600**	**0.93**	**39,363,029**	**1.50**	**1,682,968**

Table 16.7 – Daily production rates

Year	Stripping		Mineralized material		Total production	
	Overburden (tpd)	Waste (tpd)	From Pit (tpy)	From old tailings (tpy)	Without overburden (tpd)	With overburden (tpd)
PP1	-	-	-	-	-	-
PP2	1,408	2,192	-	-	2,192	3,600
PP3	9,500	14,790	-	-	14,790	24,290
PP4	1,102	1,715	2,451	371	4,537	5,639
1	17,186	74,418	9,010	2,055	85,483	102,669
2	-	79,082	15,692	2,055	96,829	96,829
3	5,095	80,440	9,991	2,055	92,486	97,581
4	3,455	86,610	6,128	2,055	94,793	98,248
5	3,537	81,729	9,760	2,055	93,545	97,082
6	12,182	77,484	7,904	655	86,043	98,225
7	83	90,634	7,756	-	98,390	98,473
8	8,094	80,031	10,000	-	90,031	98,125
9	2,036	86,592	10,001	-	96,592	98,628
10	430	42,131	7,850	-	49,981	50,411
Average	**4,579**	**56,989**	**6,896**	**807**	**64,692**	**69,271**

16.10 Mine operations

16.10.1 Drilling

The drilling pattern has been selected in order to control blast induced vibrations and airblast overpressures in the Town of Duparquet. The spacing and burden for the Project were estimated at 6.5 m and 6 m respectively. Percussion drills with 215 mm bit size will be used in ore and waste. A penetration rate of 22 m/hr was used when estimating equipment requirements. A total of four (4) drills will be required to achieve the production target.

Pre-split blasting will be done in order to maximize stable bench face and inter-ramp angles along the final walls. The pre-split consists of a row of closely-spaced holes along the final excavation limit. One percussion drill with 140 mm bit size will be used for pre-shear and holes will be drilled at an interval of 1.5 m.

16.10.2 Blasting

An explosives supplier will provide the mine with explosives. In this study, it was assumed that the explosives will be truck to Duparquet from the supplier plant located in Malartic.

The proximity of both the town and much of the underground and surface water requires that emulsion be used as an explosive. At this stage of the Project, only electronic detonators have been considered in the cost estimation. Electronic detonators allow for more precise blasts and consequently provide better control on rock projection and vibrations. In addition, blasting mats will be used as a supplementary precaution to avoid any potential risk of rock projectiles.

The loading of explosives into the drill holes will be done by truck and the service will be provided by the supplier. The holes will be stemmed to avoid fly-rock and excessive air blasts. A small loader will be used to dump the crushed rock into the drill holes.

A powder factor of 0.31 kg/t was estimated considering the parameters presented in Table 16.8. These parameters will need to be re-evaluated in a future detailed blasting study.

Table 16.8 – Blasting parameter assumptions

Elements	Parameters
Spacing	6.5 m
Burden	6 m
Hole size	8 ½ "
Explosive in hole density	1.2 g/cc
Bench height	10 m
Subdrilling	1 m
Collar	2.5 m
Loaded column	8.5 m
Charge per hole	330.78 kg/hole
Rock density	2.72 t/m^3
Yield per hole	1060.8 t/hole
Powder factor	0.31 kg/t

16.10.3 Loading

The fleet recommendation is based on a Fleet Production and Cost (FPC) analysis performed by Hewitt Equipment Ltd on the basis of the mine plan data supplied by InnovExplo. The FPC analysis compared different hauler and loader combinations in order to minimize the cost per tonne while enabling sufficient flexibility and backup capacity for efficient mining operations. The outcome of the analysis showed that the 6030FS Electric Hydraulic Shovels combined with the 785D Mining Truck was the best combination to meet that objective. Based on the data available on the site physical characteristics, Hewitt believe that the 785D Mining Trucks are suitable for all material to be excavated from the mine site including the overburden.

Electric shovels were not retained; instead, diesel shovels were chosen for the Project. Although electric shovels are more economical, they present less manoeuvrability than diesel shovels, which is a key consideration for the Duparquet Project is as the pits are not uniform. A 6030 Backhoe Diesel shovel was added to the fleet for overburden removal and to serve as a backup loader for waste and ore production.

Only one model of shovel is considered in this study. This will simplify the spare equipment requirements, thereby reducing the equipment capital. Three (3) 6030 FS shovels will be needed. Two (2) of them will be used for the production and one (1) 6030 Backhoe Diesel shovel will be added to the fleet for overburden removal and to serve as a backup loader for waste and ore production. One (1) CAT 994H wheel loader will also be necessary in the loading fleet to load ore from the ore stockpile to the mill and as a backup. Table 16.9 presents parameters used in the fleet analysis.

Table 16.9 – Parameters considered in the fleet analysis

Loading unit	**Units**	**CAT 6030 FS Diesel**	**CAT 994H**
Loading unit weight	t	293	195
Haulage unit		**CAT 785D**	**CAT 785D**
Truck payload	t	132	132
Production parameters			
Bucket capacity	m^3	16.5	15
Worked Hours/Day	Hours	20	20
Worked Days/Year	Days	360	360
Annual Worked Hours	Hours	7200	7200
Truck Mechanical Availability	%	90	90
Loaders Basic Availability	%	90	90
Utilisation	%	95	95
Operator Efficiency	%	94	94
Overall Deration	%	72	72
Passes (whole)	#	5.0	4.0
Cycle time			
First bucket dump	min	0.1	0.1
Spot time	min	0.75	0.75
Load time	min	2	1.5
Clean/Bench/Travel	min	0.1	0.3
Total cycle time	min	2.95	2.65

16.10.4 Hauling

The truck fleet will be composed of a maximum of fourteen (14) 136-tonne trucks. The ore and waste will be hauled in those trucks. The fleet study performed by Hewitt Equipment determined the number of trucks required each year. InnovExplo provided information on rock movement quantities and travel distances on a yearly based, calculated from the centroid of excavated tonnage. This information was used to estimate the truck fleet and loading equipment requirements.

For the current study, it is assumed that trucks will be crossing Route 393 to access the different parts of the Property. However, a detailed analysis of other options will need to be investigated in a future study.

16.10.5 Working Around Existing Underground Openings

In order to mitigate the risk of working around underground workings, it will be necessary to develop strict safety procedures prior to commencing operations. In the current study, it has been assumed that openings will be backfilled and that a crane will be used as an anchor for equipment and personnel when working around openings.

16.10.6 Pit Maintenance

Pit maintenance services include haul road maintenance, waste pile construction, mobilizing operating supplies, and relocating equipment. Haul road maintenance is a major aspect of pit maintenance since it enhances the life of the production equipment (particularly the tires) which represents a significant portion of the operating cost. Keeping haul roads in good condition also improves truck speed and thereby helps maintain production levels.

16.11 Mining Equipment

Hewitt Equipment recommended the principal preliminary mining equipment for the PFS. Three (3) 6030FS shovels have been chosen for the Project as well as one (1) 994H front-end loader. It has been determined that 785D trucks will be used for the Project. A total of 14 trucks will be necessary during the production peak, which covers Years 2 to 7. Sanvik recommended two types of drill for the Project, one DR540 for the pre-shear drilling, which will drill 140 mm holes, and four (4) D55SP drills for the production drilling, which will drill 215 mm holes. Crane operation will be contracted when required for safety purposes while working around old openings. InnovExplo created a list of the utility equipment required for the Project; these are presented in Table 16.10.

Table 16.10 – Mining equipment

Mining Equipment	Pre-production			Production										
	-2	-1	0	1	2	3	4	5	6	7	8	9	10	11
Production equipment														
Truck 785D (150t)	1	2	4	12	14	14	14	14	14	14	9	9	10	1
Hydraulic Shovel 6030FS			1	3	3	3	3	3	3	3	3	3	2	0
Loader 994H	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Drill DR540 (6 ½")	1	1	1	1	1	1	1	1	1	1	1	1	1	
Drill D55SP (8 ½")		1	1	4	4	4	4	4	4	3	3	3	3	
Support equipment														
Grader 16M		1	1	2	2	2	2	2	2	2	2	2	2	1
Dozer D9T			1	3	3	3	3	3	3	3	3	3	3	1
Wheel Dozer 844H			1	1	1	1	1	1	1	1	1	1	1	
Water Truck 76,000L (777G)			1	1	1	1	1	1	1	1	1	1	1	1
Excavator (rockbreaker) CAT 349				1	1	1	1	1	1	1	1	1	1	1
Excavator CAT 349		1	1	2	2	2	2	2	2	2	2	2	2	
Wheel loader CAT 980K		1	1	1	1	1	1	1	1	1	1	1	1	1
Small loader (hole stemming)			1	1	1	1	1	1	1	1	1	1	1	
Fuel truck CT660		1	1	1	1	1	1	1	1	1	1	1	1	1
Tow lowboy			1	1	1	1	1	1	1	1	1	1	1	
Tow truck 777G			1	1	1	1	1	1	1	1	1	1	1	
Pickup truck		10	10	19	19	19	19	19	19	19	19	19	19	10
Pit busses			1	2	2	2	2	2	2	2	2	2	2	
Maintenance equipment														
Service truck CT660		1	1	2	2	2	2	2	2	2	2	2	2	1
Boom truck			1	1	1	1	1	1	1	1	1	1	1	1
Tool carrier			1	1	1	1	1	1	1	1	1	1	1	1

16.12 Manpower

A total of 339 employees will be needed for the Duparquet Project. This assumes an operation that runs 24 hours a day, 7 days a week, 52 weeks per year.

The working schedule for most yearly compensated employees will be a standard 40-hour week at 8 hours per day, 5 days per week, Monday to Friday. Some yearly compensated employees will be working 12-hour shifts, equivalent to 84 hours per week, as part of a two-week repeating schedule: the first week working 4 days followed by 3 days off, the second week working 3 days followed by 4 days off. The hourly workers will be working 12-hour shifts as part of the same two-week repeating schedule. Most activities require 24-hour per day operation, which is split into 4 shifts.

The Duparquet Project is close to a number of the Abitibi region's main cities and towns. From Year 10, manpower needs will gradually decrease until the end of the mine during Year 11. Once in full production, the workforce will consist of 339 employees.

Table 16.11 – Manpower requirements per year and division

Manpower	Pre-production				Production										
	-3	-2	-1	0	1	2	3	4	5	6	7	8	9	10	11
Administration															
General Manager	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Administrative secretary	0	0	0	1	1	1	1	1	1	1	1	1	1	1	0
Administrative superintendant	0	0	1	1	1	1	1	1	1	1	1	1	1	1	1
Accountant	0	0	0	1	2	2	2	2	2	2	2	2	2	2	1
Purchasing Officer & warehouse supervisor	0	0	0	1	2	2	2	2	2	2	2	2	2	2	2
Warehouse deputy	0	0	0	1	4	4	4	4	4	4	4	4	4	4	2
IT technologist	0	0	0	1	2	2	2	2	2	2	2	2	2	2	1
HR & safety superintendant	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Human resources coordinator	0	0	0	1	2	2	2	2	2	2	2	2	2	2	0
Health & safety coordinator	0	0	0	1	1	1	1	1	1	1	1	1	1	1	1
Public relation coordinator	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Nurse	0	0	0	1	1	1	1	1	1	1	1	1	1	1	1
Training coordinator	0	0	0	1	1	1	1	1	1	1	1	1	1	1	1
Dryman	0	0	0	2	2	2	2	2	2	2	2	2	2	2	2
Surface crew operator & logistic	0	0	1	1	2	2	2	2	2	2	2	2	2	2	2
Sub-Total	**2**	**3**	**5**	**16**	**24**	**24**	**24**	**24**	**24**	**24**	**24**	**24**	**24**	**24**	**17**
Technical Services															
Geology															
Chief Geologist	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0
Senior Geologist	0	0	0	1	1	1	1	1	1	1	1	1	1	1	1
Production geologist	0	0	0	0	2	2	2	2	2	2	2	2	2	2	0
Exploration geologist	0	0	2	2	2	2	2	2	2	2	2	2	2	2	0
Sample man	0	0	2	6	6	6	6	6	6	6	6	6	6	4	0
Grade control & data processing tech.	0	0	0	0	2	2	2	2	2	2	2	2	2	2	1
Sub-Total	**1**	**1**	**5**	**10**	**14**	**14**	**14**	**14**	**14**	**14**	**14**	**14**	**14**	**12**	**2**
Engineering															
Chief Engineer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0
Senior Engineer	0	0	0	1	1	1	1	1	1	1	1	1	1	1	1
Drill & blast engineer	0	0	0	1	1	1	1	1	1	1	1	1	1	1	0
Geotechnical Engineer	0	0	0	0	1	1	1	1	1	1	1	1	1	1	0
Geotechnical technician	0	0	0	0	1	1	1	1	1	1	1	1	1	1	1
Chief surveyor	0	0	1	1	1	1	1	1	1	1	1	1	1	1	0
Surveyor - Production	0	0	1	1	4	4	4	4	4	4	4	4	4	3	0
Drill and Blast Technician	0	0	1	1	2	2	2	2	2	2	2	2	2	2	0
Mine planner technician	0	0	1	1	1	1	1	1	1	1	1	1	1	1	0
Sub-Total	**1**	**1**	**5**	**7**	**13**	**13**	**13**	**13**	**13**	**13**	**13**	**13**	**13**	**12**	**2**
Environment															
Senior environmental engineer	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Environmental monitor	0	0	1	1	1	1	1	1	1	1	1	1	1	1	1
Sub-Total	**0**	**1**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**
Maintenance															
Maintenance Superintendant	0	0	1	1	1	1	1	1	1	1	1	1	1	1	0
Maintenance foreman	0	1	1	1	4	4	4	4	4	4	4	4	4	3	0
Maintenance Planner	0	0	0	1	1	1	1	1	1	1	1	1	1	1	0
Maintenance shift planner	0	0	0	1	2	2	2	2	2	2	2	2	2	2	0
Mechanics	0	2	2	6	24	24	24	24	24	24	24	24	24	15	3
Mechanics helper	0	2	8	8	8	8	8	8	8	8	8	8	8	5	0
Welders	0	0	0	2	6	6	6	6	6	6	6	6	6	4	0
Machnist	0	0	0	1	1	1	1	1	1	1	1	1	1	1	0
Sub-Total	**0**	**5**	**12**	**21**	**47**	**47**	**47**	**47**	**47**	**47**	**47**	**47**	**47**	**32**	**3**
Operation Supervision															
Mine Superintendent	0	0	1	1	1	1	1	1	1	1	1	1	1	1	0
Senior general foreman	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine supervisor - Production	0	0	1	2	4	4	4	4	4	4	4	4	4	3	0
Mine supervisor - Drill & Blast	0	0	0	2	4	4	4	4	4	4	4	4	4	3	0
Mine supervisor - Project	0	0	0	1	2	2	2	2	2	2	2	2	2	2	0
Clerk	0	0	0	1	1	1	1	1	1	1	1	1	1	1	0
Sub-Total	**0**	**1**	**3**	**8**	**13**	**13**	**13**	**13**	**13**	**13**	**13**	**13**	**13**	**11**	**1**
Operation															
Production loader operator	0	1	2	4	16	16	16	16	16	16	16	16	16	11	3
Mining truck driver	0	1	4	12	48	56	56	56	56	56	56	36	36	22	3
Mine helper - General	0	0	0	2	8	8	8	8	8	8	8	8	8	5	0
Production drill operator	0	1	3	3	20	20	20	20	20	20	20	20	20	11	0
Drill helper	0	0	0	0	8	8	8	8	8	8	8	8	8	5	0
Trainer	0	0	0	1	2	2	2	2	2	2	2	2	2	2	0
Sharpener	0	0	0	1	2	2	2	2	2	2	2	2	2	2	0
Grader operator	0	0	0	2	8	8	8	8	8	8	8	8	8	5	0
Dozer Operators	0	0	0	3	12	12	12	12	12	12	12	12	12	7	0
wheel dozer operator	0	0	0	1	4	4	4	4	4	4	4	4	4	3	0
Wheel loader operator	0	0	0	1	4	4	4	4	4	4	4	4	4	3	0
Fuel truck operator	0	0	0	1	4	4	4	4	4	4	4	4	4	3	0
Utility equipment operator	0	1	2	2	8	8	8	8	8	8	8	8	8	5	3
Sub-Total	**0**	**4**	**11**	**33**	**144**	**152**	**152**	**152**	**152**	**152**	**152**	**132**	**132**	**84**	**9**
Total Mine manpower	**4**	**16**	**43**	**97**	**257**	**265**	**265**	**265**	**265**	**265**	**265**	**245**	**245**	**177**	**36**

Mill manpower															
Staff employee															
Process plant superintendant	0	0	1	1	1	1	1	1	1	1	1	1	1	1	1
Senior metallurgist	0	0	0	1	1	1	1	1	1	1	1	1	1	1	1
Metallurgist	0	0	0	1	1	1	1	1	1	1	1	1	1	1	1
Metallurgical technician	0	0	0	1	2	2	2	2	2	2	2	2	2	2	2
Chief chemist	0	0	0	1	1	1	1	1	1	1	1	1	1	1	1
Lab technician (sample preparation)	0	0	0	1	4	4	4	4	4	4	4	4	4	4	3
Lab technician (assay lab)	0	0	0	2	6	6	6	6	6	6	6	6	6	6	5
Clerk	0	0	0	1	1	1	1	1	1	1	1	1	1	1	1
Sub-Total	**0**	**0**	**1**	**9**	**17**	**17**	**17**	**17**	**17**	**17**	**17**	**17**	**17**	**17**	**15**
Hourly															
Shift supervisor/control room	0	0	0	2	4	4	4	4	4	4	4	4	4	4	3
Crushing operator	0	0	0	2	4	4	4	4	4	4	4	4	4	4	3
Grinding operator	0	0	0	2	4	4	4	4	4	4	4	4	4	4	3
Flotation and CIL operator	0	0	0	2	4	4	4	4	4	4	4	4	4	4	3
Pre leach, autoclave operator	0	0	0	2	4	4	4	4	4	4	4	4	4	4	3
Lime boil, cyanide leach operator	0	0	0	2	4	4	4	4	4	4	4	4	4	4	3
Elution circuit, gold room operator	0	0	0	2	4	4	4	4	4	4	4	4	4	4	3
Tailings & reagents operator	0	0	0	2	4	4	4	4	4	4	4	4	4	4	3
Labour	0	0	0	1	2	2	2	2	2	2	2	2	2	2	2
Sub-Total	**0**	**0**	**0**	**17**	**34**	**34**	**34**	**34**	**34**	**34**	**34**	**34**	**34**	**34**	**26**
Mill maintenance manpower															
Mill maintenance foreman	0	0	0	1	1	1	1	1	1	1	1	1	1	1	1
Mill maintenance planner	0	0	0	1	1	1	1	1	1	1	1	1	1	1	1
Mechanical leader	0	0	0	1	2	2	2	2	2	2	2	2	2	2	2
Electrical leader	0	0	0	1	2	2	2	2	2	2	2	2	2	2	2
Machanical/Boilermaker/pipefitter	0	0	0	4	7	7	7	7	7	7	7	7	7	7	5
Helper	0	0	0	2	4	4	4	4	4	4	4	4	4	4	3
Electrician	0	0	0	2	4	4	4	4	4	4	4	4	4	4	3
Instrument technician	0	0	0	1	2	2	2	2	2	2	2	2	2	2	2
Sub-Total	**0**	**0**	**0**	**13**	**23**	**23**	**23**	**23**	**23**	**23**	**23**	**23**	**23**	**23**	**19**
Total Mill manpower	**0**	**0**	**1**	**39**	**74**	**74**	**74**	**74**	**74**	**74**	**74**	**74**	**74**	**74**	**60**
Grand-Total Mine and Mill	**4**	**16**	**44**	**136**	**331**	**339**	**339**	**339**	**339**	**339**	**339**	**319**	**319**	**251**	**96**

17. RECOVERY METHODS

Based on the testwork described in Section 13, two processing options were selected for the recovery of the gold from the Duparquet Project ores: the Concentrate option and the pressure oxidation option ("POX option"). While gold doré would be produced on site with the POX option, the Concentrate option would produce a sulphide concentrate product for sale.

17.1 Process Description – Concentrate Option

The Concentrate option would produce a sulphide rich concentrate having the highest gold content possible. Maximizing the gold content in the concentrate has an impact on the flotation gold recovery, and justifies a CIL circuit to recover the remaining gold from the flotation tailings.

A conventional open circuit primary crushing stage was chosen followed by two grinding stages, coupled with a flotation circuit including rougher, 1st cleaner, 2nd cleaner and regrinding of the rougher concentrate, as well as cyanidation of the flotation tails with CIL recovery of the gold. The gold will be recovered from the carbon by elution, followed by electrowinning and doré smelting. Carbon regeneration has also been included. A simplified flow diagram is shown in Figure 17.1.

17.1.1 Process Flowsheets – Concentrate Option

Thirteen (13) flowsheets have been developed for the Concentrate option. They are described below. A simplified flowsheet of the overall process is shown in Figure 17.1 at the end of this section.

104812-N-P-FS-0002, Primary Crushing, Stockpile and Ore Handling Circuits, Area 100 & 120

The Run-of-Mine (ROM) ore will be crushed to P_{80} 157mm, stockpiled, and fed to the plant at a constant rate of 453 t/h.

104812-N-P-FS-0003, Grinding Circuit, Area 200

The grinding circuit will further reduce the crushed ore from F_{80} 157mm to P_{80} 100µm and deliver the ground ore to the flotation circuit

104812-N-P-FS-0004, Flotation and Regrind Circuit, Area 300

The flotation circuit will produce a sulphide gold concentrate with 50 g/t Au grade, using rougher flotation, regrind of the rougher concentrate, and two stages of cleaner flotation. The rougher tails will be sent to the CIL circuit for the recovery of the remaining gold.

104812-N-P-FS-0005, Concentrate Thickener and Filtration Circuit, Area 320

The concentrate thickener and filtration circuit will thicken and filter the flotation concentrate to 6.5% moisture, which is suitable for transport by truck or train.

104812-N-P-FS-0006, Flotation Tails Pre-Leach Thickener, Area 500

The flotation tails pre-leach thickener will thicken the flotation tails prior to their transfer to the leach and CIL circuit. Water recovered will be recirculated to the process.

104812-N-P-FS-0007, Leach and CIL Circuit, Area 520

The leach and CIL circuit will dissolve the gold using a sodium cyanide/caustic solution and adsorb it on activated carbon, where the gold will be stripped from carbon in the elution circuit.

104812-N-P-FS-0008, Final Tailings Thickener, Cyanide Destruction and Process Water Circuit, Area 600 and 800

The CIL tailings will be thickened in the final tailings thickener to 50-55% solid density, and then be sent to the cyanide destruction tank to destroy the remaining cyanide in the slurry. The final tailings are then transferred to the tailings pond.

104812-N-P-FS-0009, Carbon Stripping Circuit, Area 540

The carbon stripping circuit is designed to strip the gold from the loaded carbon and send the gold-pregnant solution to the refinery circuit.

104812-N-P-FS-0010, Carbon Reactivation Circuit, Area 540

The carbon reactivation circuit uses an indirectly fired propane kiln to reactivate the stripped carbon for reuse in the process.

104812-N-P-FS-0011, Refinery Circuit, Area 540

In the refinery, gold is electrolytically plated from the pregnant carbon strip solution onto steel wool cathodes in the electrowinning cells. The gold laden steel wool will be smelted in a tilting furnace with the addition of fluxes and will be poured into the doré ingot moulds.

104812-N-P-FS-0012-13, Reagents Circuit, Area 700

The reagents area will be used to prepare and distribute the required reagents for the flotation circuit: lime (pH Control), MIBC (frother), PAX (collector), R208 (collector); for the leach and CIL circuit: sodium cyanide (lixiviant), lime (pH control); and for the cyanide destruction circuit: sulphur dioxide (oxidant), lime (pH control), copper sulphate (catalyst), and flocculant for the thickening circuits.

104812-N-P-FS-0014, Air Services, Area 820

The air services will provide the compressed air for the plant services as well as the leach, CIL, and cyanide destruction circuits. Moreover, low pressure air will be produced by air blowers and will be delivered to the flotation circuit.

104812-N-P-FS-0015, Water Services, Area 800

The water services will transfer water from the old Central Duparquet shaft to the fresh/fire water tank and distribute it for feed and reagent preparation and fire water.

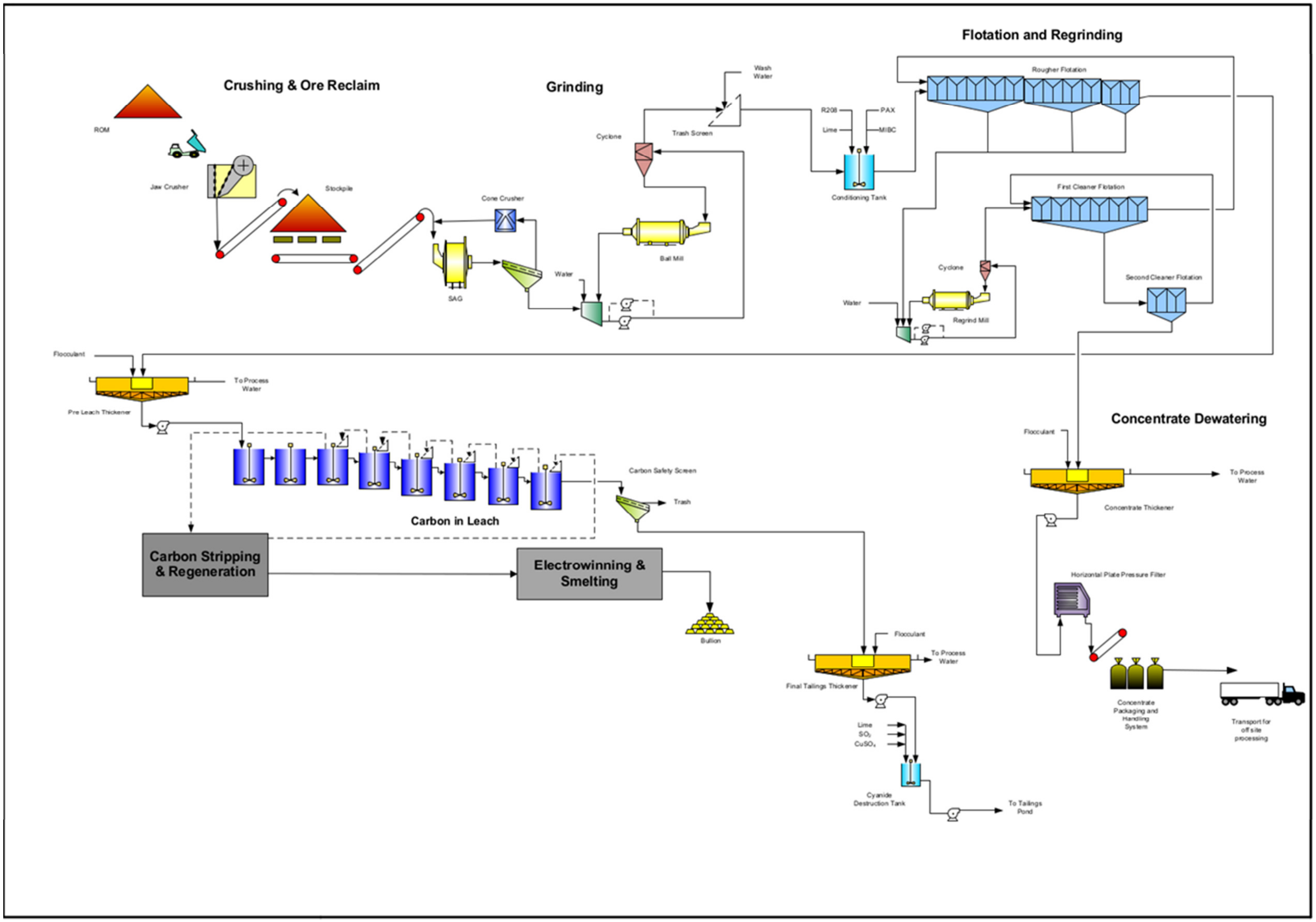

Figure 17.1 – Concentrate option simplified flow diagram

17.1.2 Process Design Criteria – Concentrate Option

The process design criteria have been established based on the results of the pilot plant testwork results conducted by SGS on 12 tonnes of full HQ core samples, the Outotec filtration test results, information from Roche and Tenova in-house databases (similar projects), and standard industry practices. The key items have been listed in Table 17.1.

Table 17.1 – Concentrate option process design criteria

Criteria	Value
Throughput	
Mill Availability	92%
Operating Days	365
Annual Throughput	3,650,000 t/a
Daily Throughput	10,000 t/d
Hourly Throughput	453 t/h
Feed Gold Grade	1.43 g/t
Ore Specific Gravity	2.73
Grinding	
SAG Mill Specific Power Consumption	10 kWh/t
Bond Abrasion Index	0.55
SAG Mill Recirculating Load	30%
SAG Mill Feed Size, F_{80}	157 mm
SAG Mill Transfer Size, T_{80}	2.1 mm
Ball Mill Work Index (BWI)	18.5 kW/h
Ball Mill Product Size, P_{80}	100 µm
Ball Mill Recirculating Load	250%
Flotation and Regrind	
Conditioning Time	12 min
Rougher Flotation Retention Time	70 min
First Cleaner Flotation Retention Time	30 min
Second Cleaner Flotation Retention Time	10 min
Regrind Mill Product Size, P_{80}	35 µm
Concentrate Weight Recovery	2.47%
Concentrate Production	247 t/d
Gold Recovery	86.4%
Concentrate Gold Grade	50 g/t
Flotation Tails Cyanidation	
Leach Time	6 h
Leach Pulp Density	45%
Cyanide Concentration	1 g/L
Lime Addition	0.58 kg/t ore
Pulp PH	10.5
Gold Recovery	41.5%
Overall Gold Recovery	92.8%
CIL	
Adsorption Time	18 h

Criteria	Value
Carbon Loading	2000 g/t
Cyanide Destruction	
Retention Time	2 h
SO_2 Consumption Rate	1.2 kg/t
$CuSO_4$ Consumption Rate	34 g/t
Thickening and Filtration	
Concentrate Thickener U/F Density	65%
Concentrate Thickener Sizing Criteria	0.1 m²/t/d
Flotation Tails Pre-Leach Thickener U/F Density	53%
Flotation Tails Pre-Leach Thickener Sizing Criteria	0.1 m²/t/d
Final Tailings Thickener U/F Density	50%
Final Tailings Thickener Sizing Criteria	0.22 m²/t/d

17.1.3 Concentrator Plant Design – Concentrate Option

The plant is designed to operate 24 h/d, 365 d/yr, and process 3.65 Mt of mineralized material (dry) annually, at a plant availability of 92%. The nominal daily throughput will be 10,000 tonnes of dry material.

The ROM ore will be trucked by mine haulage trucks to the crushing plant, dumped into the crusher feed hopper, and conveyed by apron feeder into the primary jaw crusher. The crusher product will then be conveyed by two conveyors to the crushed ore stockpile. The live capacity of the stockpile will be 10,000 tonnes with a total capacity around 36,000 tonnes. Apron feeders will then draw off the material at 452.9 t/h from below the stockpile and conveyors will deliver the material to the SAG mill feed chute.

The SAG mill is driven by a 5,670 kW (7,200 hp) motor. The SAG mill product will be discharged to the SAG mill discharge screen; the coarse material (screen oversize) will be conveyed to the pebble crusher driven by a 315 kW motor and crushed pebbles will be recirculated to the SAG mill. The screen undersize will gravity-feed the pump box and mix with the ball mill discharge. The secondary grinding circuit consists of one 6,410 kW (8,600 hp) ball mill which will operate in closed circuit with one hydrocyclone cluster. The hydrocyclone underflow will return into the ball mill while the overflow with a P_{80} of 100 µm will gravity-feed to the trash screen prior to transfer to the conditioner tank of the flotation circuit.

The flotation circuit consists of one (1) conditioning tank, eleven (11) rougher tank cells, six (6) 1st cleaner cells, two (2) 2nd cleaner cells, and a rougher concentrate regrind mill. Reagents will be added to the conditioning tank and to different points of the rougher and cleaner flotation stages. The rougher flotation tails will be pumped to the flotation tailings pre-leach thickener feed tank ahead of the cyanidation and CIL circuits. The rougher flotation concentrate will be pumped to the regrind cyclone feed pump box. The regrind ball mill will operate in closed circuit with one cyclone cluster. The cyclone underflow will be returned to the regrind mill while the cyclone overflow with P_{80} of 35 µm will be pumped to the 1st cleaner cells. The 1st cleaner tails will return to the rougher feed while the concentrate will be pumped to the 2nd cleaner cells. The second cleaner tails will be pumped back to the 1st cleaner feed, while the

2nd cleaner concentrate will be the final product of the flotation circuit with 2.47% weight recovery (247 tpd concentrate), 50 g/t Au content, and 86.4% gold recovery.

The flotation tails will be delivered to the pre-leach thickener. The thickener will raise the solid density of the slurry to approximately 60-65%. The thickener underflow will then be mixed with final tailings thickener overflow to achieve a slurry with 45-50% solid density that will feed the cyanidation tanks. Lime will be added to the thickener to raise the pH level to 10.5, which will ensure that no hydrogen cyanide evolves from the first leach tank following the addition of sodium cyanide. The thickener overflow will be returned to the process and used as process make-up water.

Lime and sodium cyanide will be added to the first leach tank and the third and fourth CIL tanks to maintain a constant extraction rate. The pulp will decant from first leach tank to the second tank and from the first CIL tank to the sixth tank over the course of 24 hours. The gold in solution in the CIL tanks will be adsorbed onto the activated carbon.

The carbon will be contained by in-tank screens which will reverse the carbon flow counter-current to the incoming slurry. Fresh/reactivated carbon will be fed to the last tank in the CIL train, and a safety screen will extract any broken carbon that escapes through the in tank screens. The undersize (slurry) from the screen will be pumped to the final tailings thickener.

The gold-loaded carbon will be pumped from the first CIL tank to the loaded carbon screen. The oversize (carbon fraction) from the screen will be discharged into the loaded carbon tank and will then be pumped to the elution circuit, while the undersize will be recycled back into the cyanidation tank.

In the elution circuit, the loaded carbon will be washed with recirculating hydrochloric acid in the acid wash vessel, neutralized with caustic soda (sodium hydroxide), and then washed with process water prior to being pumped to the carbon stripping vessel. In the carbon stripping vessel the gold will be stripped from the carbon using a hot caustic cyanide solution. The solution will be maintained at a temperature of 135 °C and an internal pressure of 350 kPa. The stripping solution will be heated with solution heaters and heat exchangers, and then pumped into the stripping vessel. The gold laden solution will exit the stripping vessel via the heat exchanger and flowing into the pregnant solution storage tank.

The pregnant solution will be pumped into the electrolytic (electrowinning) cells where the gold will be plated out of solution onto the steel wool cathodes. The stripped solution will exit the electrowinning cells and will gravity-feed the barren solution tank for reuse during the next elution cycle once the cyanide concentration is restored.

The gold laden steel wool will be smelted in a tilting furnace with the addition of fluxes, typically sodium borate, sodium carbonate, and silica. The impurities are slagged off and the gold will be poured into the doré ingot moulds. The slag will be recycled back to the jaw crusher feed to re-enter the process to minimize the gold losses. The gold recovery from the flotation tails CIL circuit will be 47.8% of the CIL feed or 6.5% of the plant feed.

The stripped carbon will be regenerated at 650 °C in an indirectly-fired kiln for 30 minutes in the absence of oxygen. It will then be cooled in a quench tank. The reactivated carbon will be pumped into the carbon attrition tank, where the fresh carbon will be added as make-up to replace attrited carbon. Prior to return into the CIL circuit, the reactivated carbon will be screened to remove any fine carbon which may adsorb gold. If the fines are allowed back into the CIL circuit, they would cause gold losses for the process.

The final tailings will be thickened in the final tailings thickener and detoxed in the cyanide destruction circuit (CND circuit) prior to transfer to the tailings pond. The CND will be performed at 50-55% solid density using the SO_2-air method.

It is expected that approximately 417 m^3/d of water enters in the plant as moisture in the ore. The outgoing water will be the water content in the products of the plant: 17 m^3/d with the concentrate and 9,311 m^3/d with the final tailings. The total process water requirement will be 27,806 m^3/d which will be recirculated from the flotation concentrate and the tailings thickener overflows, as well as tailings pond reclaim. The total fresh water requirement for the reagents and flocculant preparation is estimated at 252 m^3/d.

17.1.4 Long Lead Items – Concentrate Option

Based on quotes submitted for the major equipment, long lead items are listed in Table 17.2.

Table 17.2 – Long lead equipment, Concentrate option

Equipment	Lead Time (weeks)
SAG Mill and Drive	52-54
Ball Mill and Drive	57-59
Gold Elution Package	35-40
Thickeners	35-39
Concentrate Filter	35
Flotation Tank Cells	34
Pebble Crusher	34
Jaw Crusher	30-32
Lime Package	30

17.1.5 Plant Layout and General Arrangements – Concentrate Option

The process plant facilities from crusher to final tailings thickener are shown in Figure 17.2.

The crusher building covers an area of approximately 400 m2 and has a maximum height of 35.5 m. The main building covers an area of approximately 4,100 m2 and has a maximum height of 22 m, while the CIL circuit covers 2000 m2.

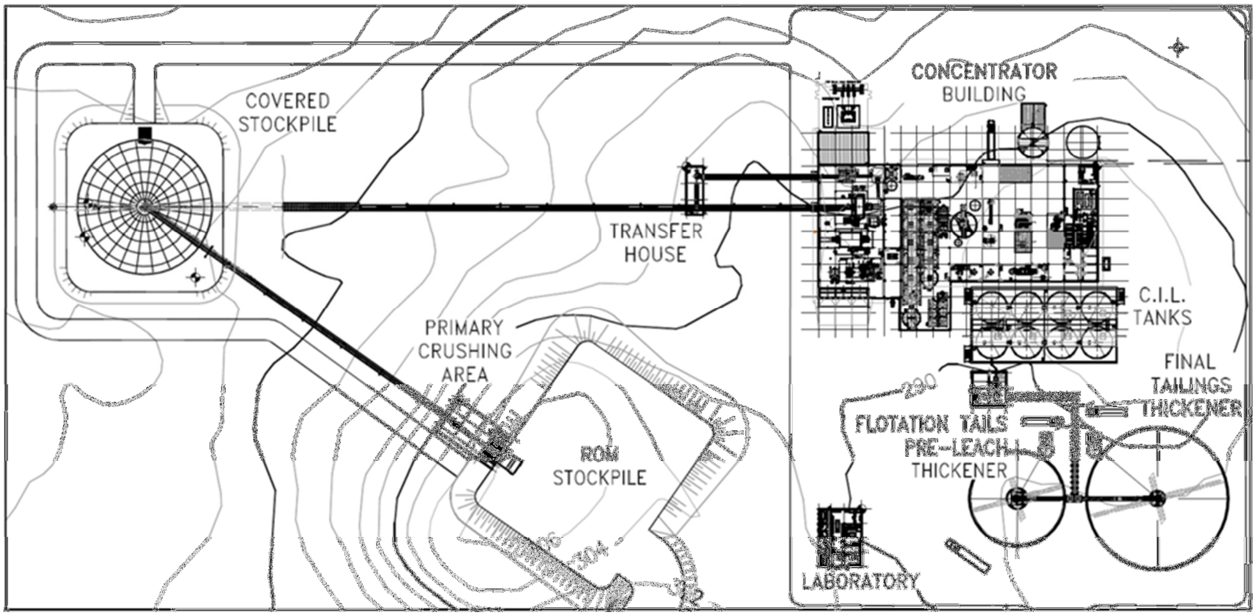

Figure 17.2 - Plant layout – Concentrate option

17.1.6 Major Equipment List – Concentrate Option

The major pieces of equipment that have been used to determine the capital and operating costs of the Project are provided in Table 17.3.

Table 17.3 - Major equipment list, Concentrate option

Equipment	Quantity	Size	Power (kW/unit)
Primary Jaw Crusher	1	63" x 47"	220
Stockpile Feed Conveyor	1	42" width,	93
Stockpile Discharge Apron Feeders	3	1320 mm x 6100 mm	20
SAG Mill Feed Conveyor	1	42" width,	93
SAG Mill and Drive	1	28' Ø X 14' length	5369
Pebble Crusher	1	93"	315
Mills Liner Handler	1		22
Grinding Area O-H Crane	1	40 tonne, 40 m span	97
Ball Mill and Drive	1	20' Ø X 31' length	6413
Rougher Flotation Tank Cells	11	Tankcell 200 m^3	186
First Cleaner Flotation Tank Cells	6	Tankcell 30 m^3	45
Second Cleaner Flotation Tank Cells	2	Tankcell 20 m^3	37
Regrind Ball Mill	1	3.96 m Ø x 5.49 m L ball mill	1119
Flotation Area O-H Crane	1	15 tonne	45
Concentrate Thickener	1	10.0 m Ø	22
Concentrate Filter	1	Pressure Filter 55m^2	30
Pre-Leach Thickener	1	40.0 m Ø	22
Final Tailings Thickener	1	60.0 m Ø	22

Equipment	**Quantity**	**Size**	**Power (kW/unit)**
Leach Tanks	2	13.3 m Ø x 13. 6m H	75
CIL Tanks	6	13.3 m Ø x 13.6 m H	75
Cyanide Destruction Tank	1	13.0 m Ø x 13.6 m H	250
Gold Elution Package	1	1 tonne package	
Carbon Regeneration Kiln	1	Package	
Refinery and smelting	1	Package	

17.2 Process Description – POX Option

The POX processing option also uses crushing, grinding, and flotation, but followed by pressure oxidation and cyanidation of the oxidized flotation concentrate. For the POX option, the flotation circuit is operated to maximize gold recovery to flotation concentrate instead of maximizing the gold concentrate grade as in the Concentrate option. With much less gold being rejected in the flotation tailings, cyanidation of the tailings is currently not economically feasible in light of the CAPEX and OPEX associated with a tailings CIL circuit, as well as the price of gold used ($1,350/oz).

The POX option uses the same crushing and grinding circuits as the Concentrate option. The flotation circuit will include the rougher, regrind, and 1st cleaner circuits (2nd cleaner will not be needed). After thickening, the flotation concentrate will pass through the pressure oxidation circuit to be oxidized followed by a carbon-in-pulp (CIP) circuit for cyanidation and adsorption for gold recovery. The flotation tailings will be considered as final tailings. In the CIP circuit, gold will be cyanide leached and adsorbed on the carbon and then recovered from the carbon by an elution circuit, followed by electrowinning and doré smelting as previously described for the Concentrate option tailings CIL circuit. Carbon regeneration has also been included. A simplified flow diagram is shown in the Figure 17.3.

17.2.1 Process Flowsheets – POX Option

Twenty-four (24) flowsheets have been developed for the POX option. They are described below. A simplified flowsheet of the overall process is shown in Figure 17.3 at the end of this section.

<u>104812-N-P-FS-1002, Primary Crushing, Stockpile and Ore Handling Circuits, Area 100 & 120</u>

ROM ore will be crushed to P_{80} 157mm, stockpiled, and fed to the plant at a constant rate of 453 t/h.

<u>104812-N-P-FS-1003, Grinding Circuit, Area 200</u>

In the grinding circuit crushed ore will be ground from F_{80} 157mm to P_{80} 100µm and delivered to the flotation circuit

<u>104812-N-P-FS-1004, Flotation and Regrind Circuit, Area 300</u>

The flotation circuit will produce a sulphide gold concentrate at 21 g/t Au, using rougher flotation, regrind of the rougher concentrate and one stage of cleaner flotation.

104812-N-P-FS-1005, Concentrate Thickener, Area 320

In the concentrate thickener circuit the flotation concentrate will be thickened to 50%w/w and delivered to the POX circuit. Water recovered as thickener overflow will be recirculated to the process.

104812-N-P-FS-1006, Flotation Tails Thickener and Process Water Circuit, Area 500 and 800

In the flotation tailings thickener the flotation tails will be thickened to 50% solids density. The flotation tailings will then be sent to the tailings pond. Water recovered as thickener overflow will be recirculated to the process.

104812-N-P-FS-1008, Cyanide Destruction and POX Tailing, Area 600

The tailings from the carbon in pulp circuit will be sent to the cyanide destruction tank to destroy the remaining cyanide in the slurry. The final tailings are then sent to the tailing pond.

104812-N-P-FS-1009, Concentrate Pre-Leach, Area 400

The concentrate feed is treated with acid produced by the pressure oxidation circuit downstream to neutralize and eliminate carbonates in the concentrate preleach circuit.

104812-N-P-FS-1010-11, Pressure Oxidation Autoclave Slurry Discharge, Hot Cure and Gas Scrubbing, Area 410

The pressure oxidation of acidified preleached concentrate is done in an autoclave to oxidise sulphide minerals. Autoclave discharge is flashed before flowing to a hot cure stage to eliminate basic ferric sulphate from the pressure oxidized solid product. Gases from the autoclave and vapour from the hot cure are treated through a scrubber.

104812-N-P-FS-1012, CCD and Neutralization, Area 420

The pressure oxidized residue is washed in a counter current decantation (CCD) circuit. Excess acid from CCD thickener overflow liquor is then neutralized with lime before being sent to the cyanide neutralization circuit.

104812-N-P-FS-1013, Lime Boil, Area 420

CCD underflow discharge is sent to the lime boil circuit to liberate silver locked in jarosite formed during pressure oxidation.

104812-N-P-FS-1014, Cyanidation, Area 440

In the cyanidation circuit, gold and silver is dissolved from the pressure oxidized residue, using a sodium cyanide/caustic solution and adsorbed onto activated carbon.

<u>104812-N-P-FS-1015-16-17, Gold Recovery, Area 540</u>

The Zadra elution circuit is used for stripping the gold and silver from carbon. Carbon from the cyanidation circuit for treating pressure oxidized concentrate is processed through this elution plant. The carbon regeneration circuit is used for regenerating carbon following elution. The electrowinning and smelter circuit is used to produce doré bars.

<u>104812-N-P-FS-1018-19, Reagents Circuit, Area 700</u>

Reagents will be prepared and distributed for the flotation circuit: MIBC (frother), PAX (collector), R208 (collector); for the cyanide destruction circuit: sulphur dioxide, lime (pH Control), copper sulphate, and flocculant for the thickening circuits.

<u>104812-N-P-FS-1020, Air Services, Area 820</u>

The air services will provide the compressed air for the plant services as well as the leach, CIP, and cyanide destruction circuits. Moreover, low pressure air will be produced by air blowers and will be delivered to the flotation circuit.

<u>104812-N-P-FS-1021, Water Services, Area 800</u>

The water services will transfer the water from the old Central Duparquet shaft to the fresh/fire water tank and distribute it for reagent preparation and fire water.

<u>104812-N-P-FS-1022-23-24, Reagents Circuit, Area 710</u>

Reagents will be prepared and distributed for cyanidation and gold recovery circuit: caustic, cyanide, lime, limestone, and concentrate pre-leach circuit: sulphuric acid reagents.

<u>104812-N-P-FS-1025-26, Utilities, Area 900</u>

Utilities area for oxygen plant and boiler plant for autoclave startup and operation.

<u>104812-N-P-FS-1027, Water Services, Area 800</u>

Water services area for seal water for autoclave operation.

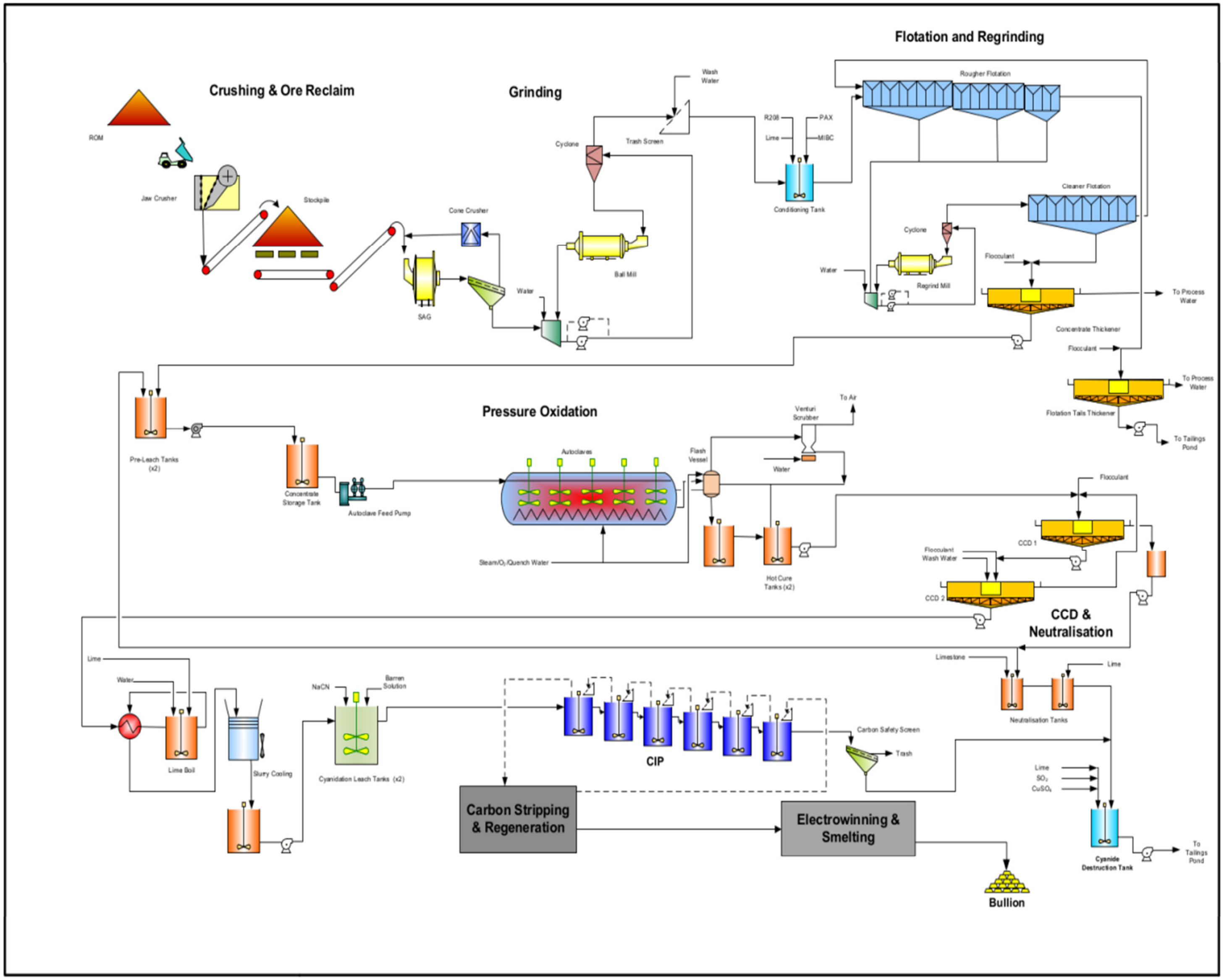

Figure 17.3 - Pressure oxidation (POX) option simplified flow diagram

17.2.2 Process Design Criteria – POX Option

The process design criteria have been established based on results of the pilot plant testwork conducted by SGS on 12 tonnes of full HQ core samples, the Outotec filtration test results, information from Roche and Tenova-Bateman in-house databases (similar projects), and standard industry practices. The key items have been listed in Table 17.4.

Table 17.4 – POX option process design criteria

Criteria	Value
Throughput	
Mill Availability	92%
Operating Days	365
Annual Throughput	3,650,000 t/a
Daily Throughput	10,000 t/d
Hourly Throughput	453 t/h
Feed Gold Grade	1.43 g/t
Ore Specific Gravity	2.73
Grinding	
SAG Mill Specific Power Consumption	10 kWh/t
Bond Abrasion Index	0.55
SAG Mill Recirculating Load	30%
SAG Mill Feed Size, F_{80}	157 mm
SAG Mill Transfer Size, T_{80}	2.1 mm
Ball Mill Work Index (BWI)	18.5 kW/h
Ball Mill Product Size, P_{80}	100 µm
Ball Mill Recirculating Load	250%
Flotation and Regrind	
Conditioning Time	12 min
Rougher Flotation Retention Time	70 min
First Cleaner Flotation Retention Time	30 min
Regrind Mill Product Size, P_{80}	35 µm
Concentrate Weight Recovery	6.2%
Concentrate Production	620 t/d
Gold Recovery	91.66%
Concentrate Gold Grade	21.1 g/t
Concentrate Pre-Leach	
Total Retention Time	4 h
Slurry Density	30%
Pressure Oxidation Circuit	
Retention Time	60 min
Autoclave Operating Temperature	201 °C
Maximum Pressure	3,600 kPa
Hot Cure	
Retention Time	4 h

Criteria	Value
Counter Current Decantation	
Number of Stages	2
Target Underflow Density	45%
Targeted free acid concentration in overflow	47 g/l
Neutralization	
Number of Tanks	2
Total Retention Time	4 h
Neutralization Agent (1^{st} stage)	Limestone
Neutralization Agent (2^{nd} stage)	Lime
Lime Boil	
Number of Tanks	4
Total Retention Time	4 h
Temperature	95 °C
Cyanidation Circuit	
Leach Time	24 h
Leach Pulp Density	45%
Cyanide Addition	1 $kg/t_{leach\ feed}$
Lime Addition	3 .6 $kg/t_{leach\ feed}$
Design Gold Dissolution	98.3%
Design Silver Dissolution	89.9%
Overall gold recovery (ore)	90.1%
Overall gold recovery (tailings)	83.9%
Adsorption Time (CIP)	18 h
Carbon Loading	4,630 ppm Au 7,273 ppm Ag
Cyanide Destruction	
Retention Time	2 h
SO_2 Consumption Rate	1.2 kg/t
$CuSO_4$ Consumption Rate	34 g/t
Thickening and Filtration	
Concentrate Thickener U/F Density	50%
Concentrate Thickener Sizing Criteria	0.1 $m^2/t/d$
Flotation Tails Thickener U/F Density	50%
Flotation Tails Pre-Leach Thickener Sizing Criteria	0.1 $m^2/t/d$

The overall gold recovery of 90.1% on the ore is calculated based on the 91.66% gold recovery in floatation concentrate and the 98.3% gold recovery in the POX circuit including the lime boil and cyanidation circuit. For the existing tailings processed the overall gold recovery is 83.9%.

17.2.3 Concentrator Plant Design – POX Option

The front end of the process is identical to the Concentrate option: crushing, stockpiling, and grinding (as described in Section 17.1).

The flotation circuit consists of one (1) conditioning tank, eleven (11) rougher cells, six (6) cleaner cells, and a rougher concentrate regrind circuit. Reagents will be added to the conditioning tank and to several points of the rougher and cleaner flotation stages. The rougher flotation tails will be pumped to the flotation tailings thickener feed tank. The rougher flotation concentrate will be pumped to the regrind cyclone feed pump box. Regrind ball mill will operate in closed circuit with one cyclone cluster. The cyclone underflow will be return to the regrind mill while the cyclone overflow with P_{80} of 35 µm will be pumped to the 1st cleaner cells. The cleaner tails will return to the rougher feed, while the cleaner concentrate will be the final product of the flotation circuit with 6.2% weight recovery (620 tpd concentrate), 21.1 g/t Au content, and 91.7% gold recovery.

The flotation tailings will be thickened by the flotation tailings thickener prior to being sent to the tailings pond.

The flotation sulphide concentrate contains ~7.2% carbonate minerals which consume acid and release carbon dioxide. The build-up of carbon dioxide gas within the pressure oxidation stage has a detrimental impact on controlling the partial pressure of oxygen required for the oxidation of sulphide minerals. Therefore a pre-leach acidification step is used to limit the amount of carbonate material reporting to the autoclave. The acid required for the pre-leach is recovered from the solid-liquid separation stage following pressure oxidation in the CCD and neutralization circuit. This minimizes the amount of acid that has to be imported to the site.
The concentrate pre-leach circuit consists of two agitated tanks with a total residence time of 4 hours. Thickened concentrate slurry from flotation circuit is pumped at a rate of 28.1 t/h of solids to the first of the two pre-leach tanks where acidic liquor from the CCD overflow is added at a rate of 51 m^3/h (acid at 50 g/L).

The reactions associated with the decomposition of carbonates during the pre-leach are as follows:

$$CaCO_3 + H_2SO_4 \rightarrow CaSO_4 + H_2O + CO_2$$

$$CaMg(CO_3)_2 + 2\ H_2SO_4 \rightarrow MgSO_4 + CaSO_4 + 2\ H_2O + 2\ CO_2$$

$$CaSO_4 + 2H_2O \rightarrow CaSO_4 \cdot 2H_2O$$

$$CaFe(CO_3) + 2\ H_2SO_4 \rightarrow 2\ CO_2 + 2\ H_2O + FeSO_4 + CaSO_4$$

The acidified slurry is then pumped by one of two pumps to the autoclave feed tank at a rate of 92.3 t/h (27% solids).

POX is used to oxidize sulphides in the concentrate in an autoclave to enable treatment in a conventional oxide gold recovery treatment plant.

Acidified concentrate slurry from the pre-leach passes through a fixed screen to remove any oversize material before reporting to the autoclave feed tank. The agitated slurry feed tank has an 8 hour buffer of sulphide concentrate before the autoclave. The autoclave processes concentrate at a feed rate of 28 t/h with a solids content of 30%.

The POX autoclave design incorporates a large autoclave (4.1 m x 18.81 m tan-to-tan) consisting of five agitated compartments, one flash tank vessel and a gas handling system. The autoclave will operate at 210 ºC and 3,100 kPa with a retention time of approximately 60 minutes.

A cryogenic oxygen plant supplies high pressure oxygen to the autoclave chambers at 98% purity. Process water is used as quench water to control the temperature within the autoclave.

The reactions that occur in the autoclave are as follows:

Sulphide oxidation and iron and arsenic precipitation reactions

$4\ FeS_2 + 2\ H_2O + 15\ O_2 \rightarrow 2\ H_2SO_4 + 2\ Fe_2(SO_4)_3$

$CuFeS_2 + 4\ O_2 \rightarrow CuSO_4 + FeSO_4$

$2\ FeAsS + 7\ O_2 + 2\ H_2O + H_2SO_4 \rightarrow 2\ H_3AsO_4 + Fe_2(SO_4)_3$

$2\ H_3AsO_4 + 4\ H_2O + Fe_2(SO_4)_3 \rightarrow 2\ FeAsO_4H_4O + 3\ H_2SO_4$

$Fe_2(SO_4)_3 + 2\ H_2O \rightarrow 2\ Fe(OH)SO_4 + H_2SO_4$

$3\ Fe_2(SO_4)_3 + 14\ H_2O \rightarrow 2\ H_3O\text{-}Fe_3(SO_4)_2(OH)_6 + 5\ H_2SO_4$

$Fe_2O_3 + 3\ H_2SO_4 \rightarrow Fe_2(SO_4)_3 + 3\ H_2O$

Phyllosilicate and framework silicate decomposition reactions

$2\ KAlSi_3O_8 + H_2SO_4 \rightarrow H_2O + K_2SO_4 + 6\ SiO_2 + Al_2O_3$

$2\ NaAlSi_3O_8 + 4\ H_2SO_4 \rightarrow Al_2(SO_4)_3 + Na_2SO_4 + 4\ H_2O + 6\ SiO_2$

$2\ KMgAl_3Si_3 + H_2SO_4 + O_2 \rightarrow K_2SO_4 + 6\ SiO_2 + 3\ H_2O + 3\ Al_2O_3 + 2\ MgO$

$4\ KFeAl_3Si_3 + 2\ H_2SO_4 + 3\ O_2 \rightarrow 6\ H_2O + 12\ SiO_2 + 2\ K_2SO_4 + 6\ Al_2O_3 + 2\ Fe_2O_3$

$Al_2Si_2O_9H + 3\ H_2SO_4 \rightarrow 2\ SiO_2 + 5\ H_2O + Al_2(SO_4)_3$

Jarosite and alunite formation

$K_2SO_4 + 3\ Fe_2(SO_4)_3 + 12\ H_2O \rightarrow 2\ KFe_3(SO_4)_2(OH)_6 + 6\ H_2SO_4$

$4\ Ag + 6\ Fe_2(SO_4)_3 + 22\ H_2O + O_2 \rightarrow 4\ AgFe_3(SO_4)_2(OH)_6 + 10\ H_2SO_4$

$3\ Al_2(SO_4)_3 + K_2SO_4 + 12\ aH2O \rightarrow 6\ H_2SO_4 + 2\ KAl_3(SO_4)_2(OH)_6$

$Na_2SO_4 + 12\ H_2O + 3\ Fe_2(SO_4)_3 \rightarrow 2\ NaFe_3(SO_4)_2(OH)_6 + 6\ H_2SO_4$

Sulphide minerals are fully oxidized with jarosite minerals being the predominate product. The arsenic is precipitated as ferric arsenate. The overall POX process is exothermic and keeps the autoclave at operating temperature.

Leached slurry residue from the autoclave is let down to just above atmospheric pressure in a single stage flash vessel. This results in the removal of excess heat by the release of steam. The excess pressure in the flash vessel provides the driving force for the vent gas through the splash heater and venturi scrubber. The autoclave residue from the flash vessels is collected in the autoclave discharge holding tank. The autoclave discharge holding tank has a slurry residence time of 1 hour.
The splash heater gas discharge is vented to a venturi scrubber, where the gas is scrubbed using raw water sprays to remove entrained liquor and solids. The autoclave gas vent line and autoclave emergency pressure relief line also feed to the scrubber. The scrubber also services the hot curing circuit. Scrubbed residual gas is vented to the atmosphere.

Autoclave residue from the autoclave discharge holding tank is treated through a hot cure stage consisting of two agitated tanks in series. The hot curing process is required to convert all basic ferric sulphate formed during the pressure oxidation process to ferric sulphate so as to reduce lime consumption during neutralization. The reaction associated with hot curing is as follows:

$$2\ Fe(OH)SO_4 + H_2SO_4 \rightarrow Fe_2(SO_4)_3 + 2\ H_2O$$

The hot curing of oxidized slurry is maintained at a temperature of 95 °C for 4 hours. The retention time can be shortened by reducing the number of tanks used in series through launders to bypass tanks. The slurry from the hot cure circuit is pumped via one of two pumps to the counter current decantation and neutralization circuit.
The hot cured autoclave residue is washed to remove acidic liquor and soluble mineral components from the residue by Counter Current Decantation (CCD). Some of the acidic liquor is used in the concentrate pre-leach stage to remove carbonate minerals. The rest of the acidic liquor is neutralized and pumped to the tails treatment facilities.
The CCD circuit consists of two 18-m diameter thickeners in series. The flow of solids from the hot cure circuit proceeds in a direction counter current to the flow of the wash solution, with each stage composed of a mixing step followed by settling of the solids from the suspension. The wash ratio (wash water flow/slurry liquor flow) is 1.2.
The hot cured autoclave residue from the hot cure circuit is pumped to the CCD#1 thickener. The slurry is mixed with overflow solution from the CCD#2 thickener in the thickener#1 feed tank. The thickener#1 feed tank is a small vessel designed to accept the autoclave residue stream and also thickener area sump pump material. The slurry gravitates to the thickener feed well, during which a dilute flocculating agent is contacted with the stream. The solid settles to the underflow and is pumped to CCD#2 thickener at a pulp density of 45% w/w solids. The solution overflow from CCD#1 thickener reports to a thickener overflow holding tank where ~40% of the solution is recycled to the pre-leach circuit (Area 400) and the rest pumped to the neutralization circuit.
The thickened slurry from CCD#1 thickener underflow is pumped to the CCD#2 thickener, where it is mixed with process water in the thickener #2 feed tank. The slurry gravitates to the thickener feed well where dilute flocculating agent is contacted with the stream. The solid settles to the underflow and is pumped to the lime boil circuit at a pulp density of 45% w/w solids. The solution overflow from CCD#2 thickener gravitates to the thickener 1 feed tank above CCD#1 thickener.

The extra solution not used for pre leach acidification reports to the first of two neutralization agitated tanks in series with a total retention time of 4 hours. The neutralization circuit is used to neutralize the acid and promote the precipitation of residual arsenic and other cations from solution. Limestone is added to raise the pH to 5 followed by lime to further raise the pH to 8. The reactions taking place are:

In the presence of limestone

$$FeSO_4 + CaCO_3 + H_2O \rightarrow Fe(OH)_3 + CaSO_4 + CO_2$$

$$CuSO_4 + CaCO_3 + 1\ H_2O \rightarrow Cu(OH)_2 + CaSO_4 + CO_2$$

$$Fe_2(SO_4)_3 + 3\ CaCO_3 + 3\ H_2O \rightarrow 2\ Fe(OH)_3 + 3\ CaSO_4 + 3\ CO_2$$

$$Al_2(SO_4)_3 + 3\ CaCO_3 + 3\ H_2O \rightarrow 2\ Al(OH)_3 + 3\ CaSO_4 + 3\ CO_2$$

$$H_2SO_4 + CaCO_3 \rightarrow CaSO_4 + H_2O + CO_2$$

$$2\ H_3AsO_4 + Fe_2(SO_4)_3 + 4\ H_2O \rightarrow 2\ FeAsO_4.2H_2O + 3\ H_2SO_4$$

$$H_2O + CaCO_3 \rightarrow Ca(OH)_2 + CO_2$$

In the presence of lime

$$H_2SO_4 + Ca(OH)_2 \rightarrow 2\ H_2O + CaSO_4$$

$$CuSO_4 + Ca(OH)_2 \rightarrow Cu(OH)_2 + CaSO_4$$

$$Fe_2(SO_4)_3 + 3\ Ca(OH)_2 \rightarrow 2\ Fe(OH)_3 + 3\ CaSO_4$$

$$Al_2(SO_4)_3 + 3\ Ca(OH)_2 \rightarrow 2\ Al(OH)_3 + 3\ CaSO_4$$

Limestone is added to the first tank and lime added to the second tank. Provisions are also made to allow addition of limestone or lime into either of the two tanks. The resulting neutralized slurry with a pulp density of 18% w/w solids is then pumped via one of two pumps to the tails treatment facilities.

The argentojarosite formed during pressure oxidation must be decomposed to render the silver recoverable by cyanidation. The breakdown and liberation of silver from jarosite is carried out by the process known as lime boil at 95ºC for 4 hours with the addition of lime.

The washed autoclave residue from the solid liquid separation circuit reports to the lime boil feed tank. The autoclave residue is preheated by heat exchange with the lime boil discharge stream. The autoclave residue then reports to the first of four lime boil agitated tanks and contacted with lime slurry to raise the pH to 10.5. The temperature is maintained in the lime boil tanks via hot gases fed directly from the autoclave discharge flash vessel. A scrubbing circuit is provided to remove vapours from the lime boil circuit.

The reactions that take place in the lime boil circuit are as follows:

$$4\ AgFe_3(SO_4)_2(OH)_6 + 8\ Ca(OH)_2 = 4\ Ag + 6\ Fe_2O_3 + 20\ H_2O + 8\ CaSO_4 + 1\ O_2$$

$$H_2SO_4 + Ca(OH)_2 = 2\ H_2O + CaSO_4$$

$$CuSO_4 + Ca(OH)_2 \rightarrow Cu(OH)_2 + CaSO_4$$

$$Fe_2(SO_4)_3 + 3\ Ca(OH)_2 \rightarrow 2\ Fe(OH)_3 + 3\ CaSO_4$$

$$FeSO_4 + Ca(OH)_2 = Fe(OH)_2 + CaSO_4$$

$$Al_2(SO_4)_3 + 3\ Ca(OH)_2 \rightarrow 2\ Al(OH)_3 + 3\ CaSO_4$$

$$MgSO_4 + Ca(OH)_2 \rightarrow Mg(OH)_2 + CaSO_4$$

$$Fe(OH)SO_4 + Ca(OH)_2 \rightarrow 1\ Fe(OH)_3 + 1\ CaSO_4$$

$$2\ KFe_3(SO_4)_2(OH)_6 + 3\ Ca(OH)_2 \rightarrow 6\ Fe(OH)_3 + 3\ CaSO_4 + K_2SO_4$$

$$H_3O\text{-}\ Fe_3(SO_4)_2(OH)_6 + 2\ Ca(OH)_2 \rightarrow 2\ CaSO_4 + 3\ Fe(OH)_3 + 2\ H_2O$$

$$2\ NaFe_3(SO_4)_2(OH)_6 + 3\ Ca(OH)_2 \rightarrow 3\ CaSO_4 + 6\ Fe(OH)_3 + Na_2SO_4$$

$$2\ KAl_3(SO_4)_2(OH)_6 + 3\ Ca(OH)_2 \rightarrow 6\ Al(OH)_3 + 3\ CaSO_4 + K_2SO_4$$

Provision is made in the design to bypass any one of the lime boil tanks for maintenance or descaling purposes. The slurry in the last lime boil tank is pumped to one of two cooling towers to cool the slurry. The cooled slurry reports to the cooling tower discharge tank to be pumped and treated in the cyanidation circuit. The cooling tower is operated to reduce the slurry temperature from 95 °C to 40 °C by contacting the slurry with counter current flow fan forced air. The cooling tower discharge tank has a slurry retention time of 2 hours.

The cyanide leach circuit consists of two agitated leach tanks and six agitated carbon in pulp (CIP) tanks in series. The two leach tanks each have 12 hours residence time, whereas the six CIP tanks each have 3 hours residence time. Each tank is baffled with a downcomer to minimize short circuiting. The slurry flows by gravity from tank to tank with launders designed to allow the ability to bypass any of the tanks.

Lime boiled autoclave residue in the cooling tower discharge tank is pumped directly to the first leach tank. Provisions are also made to pump to the second leach tank.

Oxygen used as the oxidant in the cyanide leaching process is introduced into the two leach tanks separately via a spear mounted in the side of each tank below the agitators. Sodium cyanide solution as lixiviant for leaching precious metals in the autoclave residue and lime to maintain pH is also added to the first two leach tanks. The pH is monitored and maintained at a value of 10.5. The density of the feed to tanks is controlled to 48 wt% solids. Cyanide addition points (Tanks 1-2) are controlled by an online cyanide controller that measures free cyanide levels in the first, second and final CIP tanks.

The simplified reactions occurring in the cyanide leach are:

$$4\ Au + 2\ H_2O + 8\ CN^- + O_2 \rightarrow 4\ Au(CN)_2^- + 4\ OH^-$$

$$4\ Ag + 2\ H_2O + 8\ CN^- + O_2 \rightarrow 4\ Ag(CN)_2^- + 4\ OH^-$$

Regenerated carbon is added to the last CIP tank 6 and pumped counter currently from CIP tank 6 to CIP tank 1 using vertical spindle pumps with recessed impellors. Fresh carbon is added to the circuit to replenish the carbon lost to fine size fractions. The carbon concentration in each tank is maintained at 25 g/L.

Loaded carbon is extracted from CIP tank 1 and pumped to the loaded carbon recovery screen for cleaning, before acid wash and elution.

The CIP circuit carbon is retained in the adsorption tanks by interstage pumping screens with apertures of 800 µm. This interstage pumping screening method enables the height differential required between tanks to be minimized. An overhead gantry crane is available for cleaning and maintenance of all screens.

Cyanide leached residue is passed through one of two carbon safety screens to a hopper and then pumped to the cyanide detoxification circuit. Carbon fines from the linear screen overflow is collected for treatment later.

Cyanide gas monitors are installed at location points over the tanks to monitor hydrogen cyanide gas concentrations.

For recovering the gold and silver from loaded carbon the Zadra elution process is utilised and involves stripping of gold and silver from the carbon with a hot caustic/cyanide solution (eluant) and electrowinning the gold/silver on cathodes in a closed loop

Loaded carbon is transferred from the loaded carbon recovery screen to the loaded carbon hopper via a sieve bend. Excess water used to transfer the carbon is drained off. Water is introduced at the bottom of the loaded carbon hopper during the carbon transfer. Material entrained in the carbon bed floats to the top of the tank and overflows to the spillage pump. The carbon in the loaded carbon hopper is then transferred to the elution column. The elution circuit is designed for 3 tonnes of loaded carbon per day.

The eluant used to strip the gold and silver off the loaded carbon is made up of 3% NaOH and 0.1% NaCN and stored in the eluant tank. The eluant is pumped from the eluant tank, passing through two heat exchangers where the temperature is elevated to 140 °C, before flowing upstream through the carbon bed in the elution column. The solution exits the strainer located at the top of the column into the primary recovery heat exchanger where waste heat is recovered by transferring the heat into incoming solution from the eluant tank. The solution is cooled to below 85 °C in the flash tank located in the gold room and flows through an electro-winning cell. In the electro-winning circuit gold and silver is deposited onto cathodes and the eluant overflows from the cells and gravitates back to the eluant tank.

The eluant heating system consists of a liquid petroleum gas (LPG) fuelled heater and two heat exchangers, a primary and secondary heat exchanger. For the primary heat exchanger, a plate heat exchanger recovers waste heat from existing eluant from the elution column and transfers the heat to the eluant entering the column. For the secondary heat exchanger, hot oil at a set temperature circulates through a shell and tube heat exchanger, elevating the eluant temperature to 140 °C. The eluant temperature is controlled by regulating the oil flow through the heat exchanger. The eluant temperature is reduced to below boiling point before flowing to the electro-winning cells.

The eluted carbon is hydraulically transferred to the kiln feed hopper via a dewatering sieve bend. The carbon regeneration kiln has a capacity for 3 t of carbon.

Thermal regeneration of the carbon is carried out at 700 °C in a LPG fired rotary horizontal kiln. Under these conditions, the organics which may have absorbed on the carbon are removed. The presence of organics deactivates carbon and therefore must to be removed prior to returning to the cyanidation circuit. Carbon exiting the kiln is quenched with water in the quench pan. The object of quenching is to open up the micro pores in the carbon. Water and carbon overflows onto the carbon vibrating screen to remove water and fine carbon generated by thermal shock to the carbon.

Carbon discharges from the screen into the acid wash tank. One bed volume of 3% dilute HCl is pumped from the acid make up tank into the acid wash tank to remove entrained calcium carbonate and acid soluble impurities from the carbon. The carbon is soaked in dilute acid for 1 hour, followed by a caustic neutralizing and a water rinse cycle. The neutralized spent acid is pumped back as the process water.

The carbon inventory is checked before the carbon is pumped back to the cyanidation circuit. When required fresh carbon is loaded onto the carbon vibrating screen and sprayed with water to wash out the fines from the fresh carbon. The regenerated carbon and/or fresh carbon in the acid wash tank are pumped to the carbon return screen in the cyanidation circuit area (Area 440) for use in the carbon in pulp circuit.

The electro-winning and smelting is carried out in the high security area. Electro-winning is part of the Zadra elution cycle where eluate flows from the elution column via the recovery heat exchanger into the electro-winning cell. The solution flows through cathodes prior to exiting the cell via the overflow weir. The solution is recirculated back to the eluant tank. Barren levels are targeted at 5 mg/L Au.

During electro-winning, a voltage is applied by the rectifier across the anode and cathode resulting in reduction and subsequent deposition of gold and silver on the cathodes. The electro-winning current is monitored and the voltage is adjusted automatically to supply sufficient current for gold and silver deposition. The cathodes are constructed from stainless steel wool and the anodes from stainless steel plates.

On completion of the electro-winning cycle the rectifier is switched off and the cathodes are manually removed from the cell. The cathodes are washed with high pressure water to recover the metal sludge that has deposited. The cathode washings gravitate to the cathode wash bay/tank. Once the cathodes have been washed they are returned to the cell and the electro-winning process can recommence.

The settled gold/silver sludge in the cathode wash-bay/tank gravitates into the cylindrical filter press located below the tank. The press is sealed and compressed air is introduced on top of the filter press.

The filter cake is collected and dried in the drying oven. Fluxes are added to the dried filter cake and then smelted in the LPG fired furnace and gold/silver bars produced.

The circuit discharge will feed the cyanide destruction circuit (CND circuit) prior to being sent to the tailings pond. The CND circuit will be performed at the 50-55% solid density using Air/SO_2 method.

Regarding water circulation in the process plant, it is expected that approximately 417 m^3/d of water enters in the plant as moisture in the ore. The outgoing water will be the water content in the products of the plant: 620 m^3/d with concentrate and 13,491 m^3/d with final tailings. The total process water requirement will be 30,957 m^3/d, which will be recirculated from the flotation concentrate, the tailings thickeners overflow, as well as tailings pond reclaim. The total fresh water requirement for the reagents and flocculant preparation is estimated at 144 m^3/d.

17.2.4 Long Lead Items – POX Option

Based on quotes submitted for the major equipment, long lead items are listed in Table 17.5.

Table 17.5 – Long lead equipment, POX option

Equipment	Lead Time (weeks)
Oxygen Plant	117
Autoclave Shell and Lining	65 - 88
Autoclave Agitators	42
SAG Mill and Drive	52-54
Ball Mill and Drive	57-59
Lime Package & Slaking System	50
Gold Elution Package	22-26
Thickeners	35-39
Autoclave Feed Pumps	35
Flotation Tank Cells	34
Pebble Crusher	34
Jaw Crusher	30-32

17.2.5 Plant Layout and General Arrangements – POX Option

The process plant facilities from crusher to final tailings thickener are illustrated in Figure 17.4.

The crusher building covers an area of approximately 400 m^2 with a maximum height of 35.5 m. The main building covers an area of approximately 3,000 m^2 with a maximum height of 22 m, with secondary building and the 2,000 m^2 for the CIL circuit.

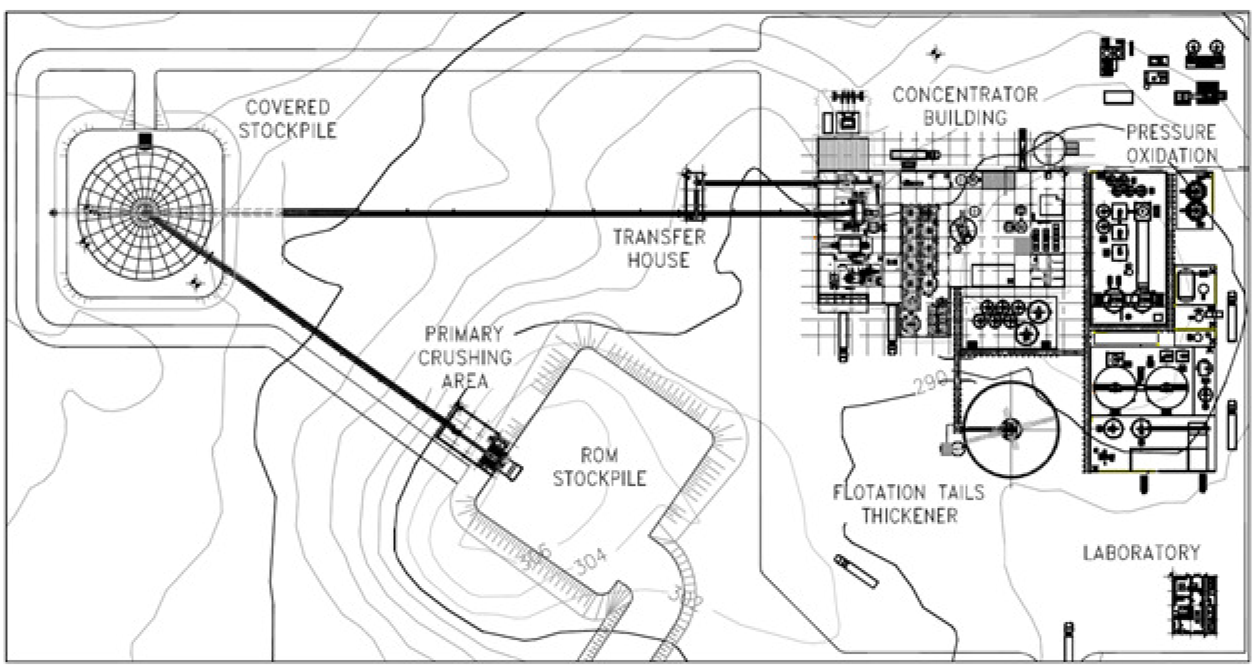

Figure 17.4 - Plant layout, POX option

17.2.6 Major Equipment List – POX Option

The major pieces of equipment for the Project are provided in Table 17.6.

Table 17.6 - Major equipment list, POX option

Equipment	Quantity	Size	Power (kW/unit)
Primary Jaw Crusher	1	63” x 47”	220
Stockpile Feed Conveyor	1	42” width,	93
Stockpile Discharge Apron Feeders	3	1320 mm x 6100 mm	20
SAG Mill Feed Conveyor	1	42” width,	93
SAG Mill and Drive	1	28’ Ø X 14’ length	5369
Pebble Crusher	1	93”	315
Mills Liner Handler	1		22
Grinding Area O-H Crane	1	40 tonne, 40 m span	97
Ball Mill and Drive	1	20’ Ø X 31’ length	6413
Rougher Flotation Tank Cells	11	Tankcell 200 m^3	186
First Cleaner Flotation Tank Cells	6	Tankcell 30 m^3	45
Regrind Ball Mill	1	3.96 m Ø x 5.49 m L ball mill	1119
Flotation Area O-H Crane	1	15 tonne	45
Concentrate Thickener	1	10.0 m Ø	22
Flotation tails Thickener	1	40.0 m Ø	22
Concentrate Pre-leach Tank	2	4.8 m Ø x 5.3 m H	11
Autoclave	1	4.1 m Ø x 21.0 m l, 223m^3	465
Flash Vessel	1	4.4 m Ø x 6.6 m H	0
Hot cure tank	2	7.3 m Ø x 7.8 m H	18.5
Neutralization Tank	2	5.2 m Ø x 5.7 m H	15
CCD Thickener	2	18.3 m Ø	18.5
Lime Boil Tank	4	4.2 m Ø x 5.2 m H	5.5
CIP Tank	6	5.4 m Ø x 8.5 m H	15
Gold Elution Package	1	1 tonne package	
Carbon Regeneration Kiln	1	Package	
Refinery and Smelting	1	Package	
Limestone VertiMill	1		225
Cyanide Destruction Tank	1	5.0 m Ø x 5.3 m H	50

18. PROJECT INFRASTRUCTURE

This section summarizes Project infrastructure, such as power lines, access roads, on-site buildings and tailings storage facilities (TSF), as well as site services that are required to complement the processing of ore from the Duparquet deposit.

All topographic information for infrastructure locations was derived from stereo satellite images.

No geotechnical investigations for the surface infrastructure have been performed for this PFS. The process plant buildings and site buildings are located in an area where outcrops are found. Detailed geotechnical investigations, such as boreholes and test pits, are recommended at the feasibility level in order to confirm assumptions about bedrock depth and capacity, and to optimize civil design criteria related to the foundations of the mill and process plant. It is expected that field investigations will begin as the Project progresses to a feasibility phase.

The site layout and general site arrangement are presented in Figure 18.1.

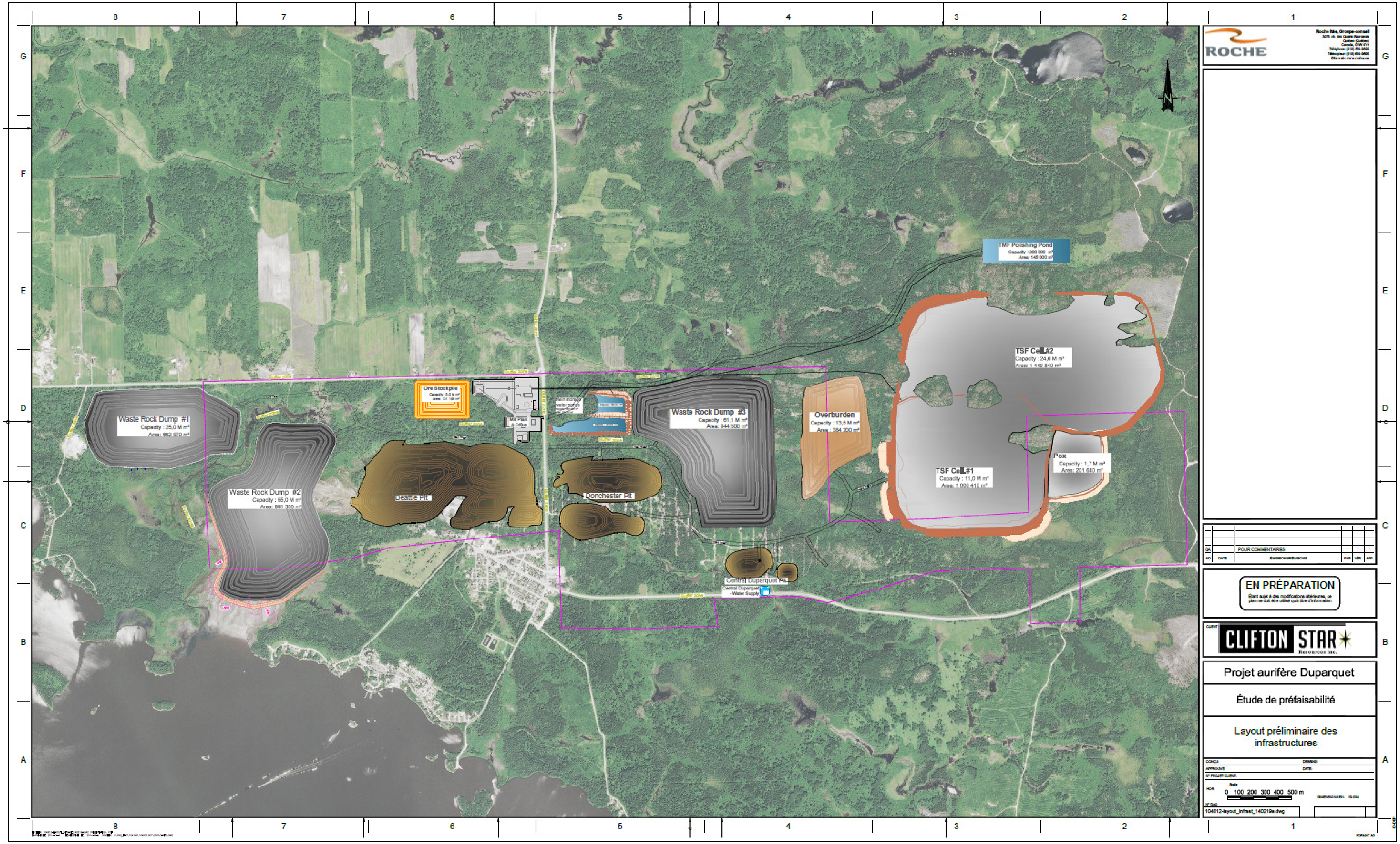

Figure 18.1 – Duparquet Project infrastructure

18.1 Off-Site Infrastructure

18.1.1 Power Line

Electric power will be provided by Hydro-Québec through a new 15 km long 120 kV power line that could be available in 36 months depending on permiting and construction. At the present time, Hydro-Québec's Abitibi network does not have the capacity to supply the required energy for the project, except by importing or the temporary rerouting of its power network. Hydro-Québec is currently planning on upgrading their network for 2019. A 120 kV main substation will be located near the process plant. The main substation will include a concrete base, a 30 MVA 120 / 13.8 kV transformer and accessories for the Concentrate option (50 MVA for the POX option), and a capacitor bank for power factor correction. Power will be delivered to the site's remote buildings at 4.16 kV via a 5 km power line.

The total power capacity will be 50 MW and the estimated power requirement will amount to approximately 35 MW. Provisions to the transformer capacity are required for starting large motors; an additional capacity of 5 to 10 MVA is therefore included. Final capacity will be reviewed in the detailed engineering phase.

Metso's proposal for the SAG Mill and Ball Mill includes wound rotor induction motors with liquid resistance. Wound rotor induction motors require less starting current than synchronous motors, but their utilisation requires power factor correction (capacitor bank). On another hand, synchronous motors need no power factor correction and can be used to control the power factor of a plant. Synchronous motors, however, require specific devices for starting, such as an air clutch.

18.1.2 Access Road

The Duparquet Project is located 45 km northwest of Rouyn-Noranda in the Abitibi-Témiscamingue region of Québec. The Property partly overlaps the municipal boundary of the Town of Duparquet. The future mine site is easily accessible using the already existing provincial highway, Route 393, and no access road other than a connection between the site facilities and Route 393 will be required for project development.

18.2 On-Site Infrastructure

18.2.1 Tailings Storage Facility and Water Management

18.2.1.1 Site layout and staged construction, and conceptual dam sections

The proposed beneficiation plant will produce approximately 46.5 Mt of tailings over the life of mine. Two options are considered for the processing as described in Section 17 of the Report: 1) a Concentrate option where all tailings will result from a CIL circuit; and 2) a POX option where approximately 6% of the tailings will have passed through the CIP process. Since the CIL circuit involves cyanidation, the tailings and process water resulting from this circuit will require being stored in a lined or impermeable tailings cell to prevent seepage into groundwater aquifers.

<u>Concentrate option</u>

The Concentrate option involves a volume of approximately 34.5 Mm^3 of tailings requiring containment in an impermeable TSF. Due to the cyanidation process involved in this option, this TSF will be fully lined to prevent seepage into groundwater aquifers. This volume is based on a bulk density of 1.35 t/m^3, which is a conservative estimate based limited information. Consolidated tailings are likely to achieve a higher bulk density resulting in lower storage volume requirements and costs for dam construction.

A preliminary assessment of disposal facility requirements to store and manage the tailings was prepared for the Duparquet Project LOM. The TSF for the Concentrate option is conceived as a staged construction with two separate cells (see Figures 18.2, 18.3, and 18.4).

- Cell 1 in the southwest – for cyanidation process tailings (fully lined);
- Cell 2 in the north – for cyanidation process tailings (fully lined);

The tailings dams will be constructed in stages to minimize the initial capital construction costs. The TSF is conceived to be constructed in three phases:

- **Phase 1:** Cell 1 for approximately 10.7 Mm^3 (elev. 331.1 m);
- **Phase 2:** Cell 2 construction (elev. 325.1 m);
- **Phase 3:** Raise Cell 1 and Cell 2 to final crest height (elev. 334.5 m). May be carried out as single lift or as multiple lifts to make use of cycloned underflow tailings as construction material.

The concept is presented in the Figure 18.2, 18.3 and, Figure 18.4, which illustrate the construction stages and the use of natural ridges to help form parts of the containment structure and reduce construction costs.

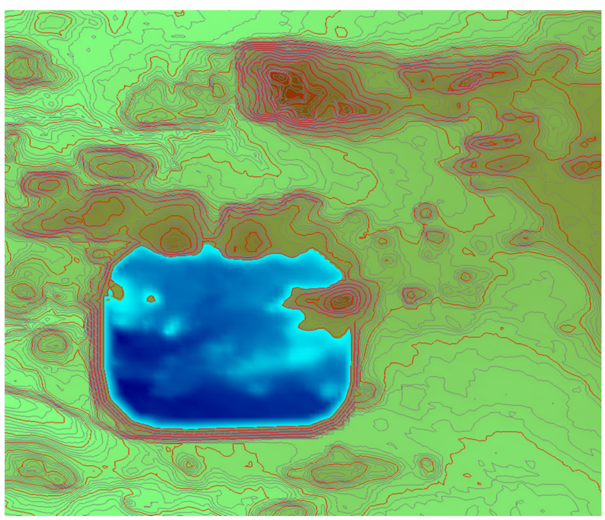

Figure 18.2 - Phase 1 configuration for Concentrate option

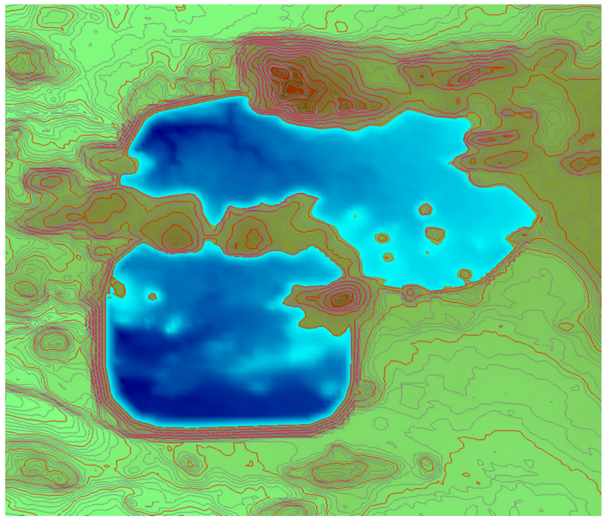

Figure 18.3 - Phase 2 configuration for Concentrate option

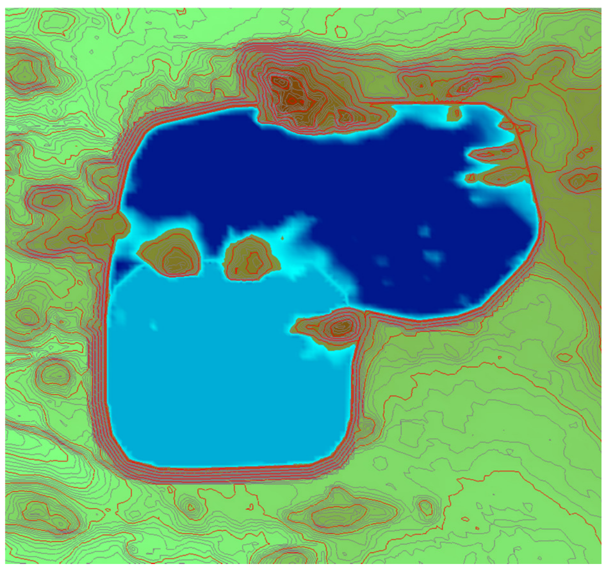

Figure 18.4 - Phase 3 configuration for Concentrate option

POX option

The POX option involves treating only about 6% of the tailings with a CIP (cyanidation) circuit. Since only the tailings of the CIP circuit need to be stored in a lined cell, less membrane is required in construction than for the Concentrate option, and water management and treatment is simplified.

The TSF layout is similar to that of the Concentrate option described above, except a third smaller **lined** cell is added to handle the reduced tailings quantity from the CIP circuit. The large northern and southwest cells are **unlined** in this scenario since early geochemical results indicate that the tailings are not acid generating, nor will they be treated by cyanide as was the case for the Concentrate option.

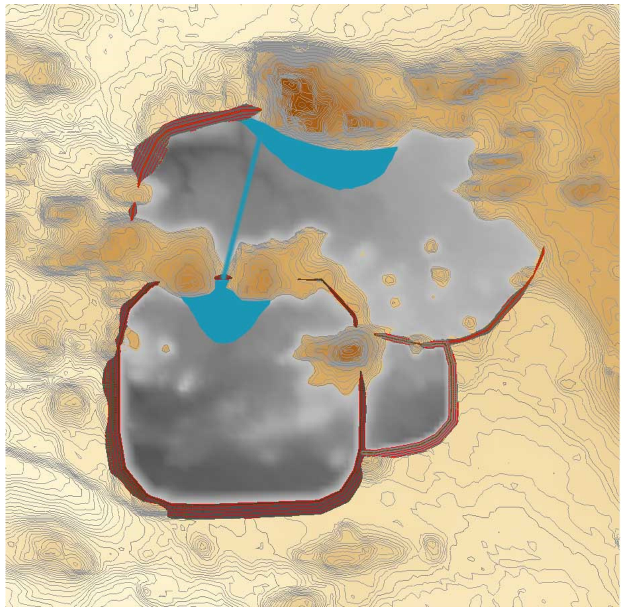

Figure 18.5 - Phase 3 configuration for POX option. POX cell constructed during Phase 1.

As described previously, the proposed beneficiation plant will produce approximately 46.5 Mt (34.7 Mm^3) of tailings. Of this, 44.2 Mt (32.7 Mm^3) will come directly from the flotation process while 2.3 Mt (1.9 Mm^3) will be produced using the POX process, which includes a CIP component. The volumes in the POX option are slightly higher due to the assumed lower bulk density of the tailings resulting from the POX process (1.2 t/m^3). The remaining 94% of the tailings from flotation are still estimated at 1.35 t/m^3, as was the case for the Concentrate option.

The two types of tailings will require separate handling and disposal at the tailings storage facility utilising the tailings delivery pipelines. Separate pipelines will be required for the POX/CIP and flotation tailings.

For the POX option, the TSF is conceived as a staged construction with three separate cells (see Figure 18.5 above):

- Cell 1 in the southwest – for low-risk tailings (unlined);
- POX cell in southeast – for cyanidation process tailings (fully lined);
- Cell 2 in north – for low risk tailings (unlined).

As was the case for the Concentrate option, the tailings dams will be constructed in stages to minimize the initial capital construction costs. The tailings storage facility is conceived to be constructed in 3 phases:

- **Phase 1:** Cell 1 and POX cell for approximately 10.7 Mm^3 and 1.9 Mm^3 of tailings, respectively (elev. 331.1 m);
- **Phase 2:** Cell 2 construction (elev. 325.1 m);
- **Phase 3:** Raise Cell 1 and Cell 2 to final crest height (elev. 334.5m). Can be carried out as a single lift or as multiple lifts to make use of cycloned underflow tailings as construction material.

18.2.1.2 Conceptual dam sections

The current conceptual plan is a centreline dam configuration, predominantly constructed using waste rock from the open pit mining operation with upstream portions of the dam being founded on compacted coarse cycloned underflow tailings or low plasticity tailings (see Figure 18.6 to Figure 18.13 below). Compaction of the tailings foundation material will be required to reduce the risk of liquefaction. Where tailings foundation material is fine, dewatering of the foundation tailings must be considered. In addition, an upstream filter and drain below the dam is recommended to increase stability of the embankment and facilitate the tailings dewatering, as well as prevent piping of the fine tailings deposited next to the embankment. This will be handled with a filter design based on the embankment and tailings grain size distributions on either side of the filter (to be assessed during the feasibility study and design phase). For the purposes of the PFS, a 1 m thick filter on the upstream side was considered, both for proper drainage and ease of construction.

Drainage ditches on the downstream toe are to manage runoff and seepage through the dam. Water collected from the seepage of the dam can be pumped back to the tailings decant pond or polishing ponds for recirculation via the reclaim water pipeline.

Tailings cells 1 and 2 will be managed as a permeable facility which necessitates that the tailings are classified as low risk as per Directive 019 (MDDELCC, March 2012). Preliminary tailings geochemistry support that tailings resulting from the flotation circuit for the POX option (i.e., not requiring cyanidation) are considered low risk and non-acid generating. As per the guidelines on the characterization of mine tailings (MDDELCC, 2003), the testing of the tailings material and crushed core material should be carried out as outlined in Table 18.1 during production.

Table 18.1 - Recommended minimum geochemical analysis for tailings classification

Geological Mass (in Tonnes) that will be either extracted or minerally treated	Minimum number of samples required for analysis purpose
≤ 10,000	3
> 10,000 and ≤ 100,000	8
> 100,000 and ≤ 1,000,000	26
> 1,000,000 and ≤ 10,000,000	80
> 10,000,000	144

All the tailings resulting from the Concentrate option and 6% of the tailings mass in the POX process tailings are classified as “cyanide-containing” since the gold recovery involves a carbon-in-leach (CIL) circuit. Level-A leak-proofing measures will be required for the tailings resulting from the CIL circuit.

For the purposes of this report, only the maximum final dam heights (Phase 3) are presented. The configuration of the conceptual dams for areas with clay foundation (purple) and granular foundation (yellow) materials are shown in Figure 18.6 to Figure 18.13 below.

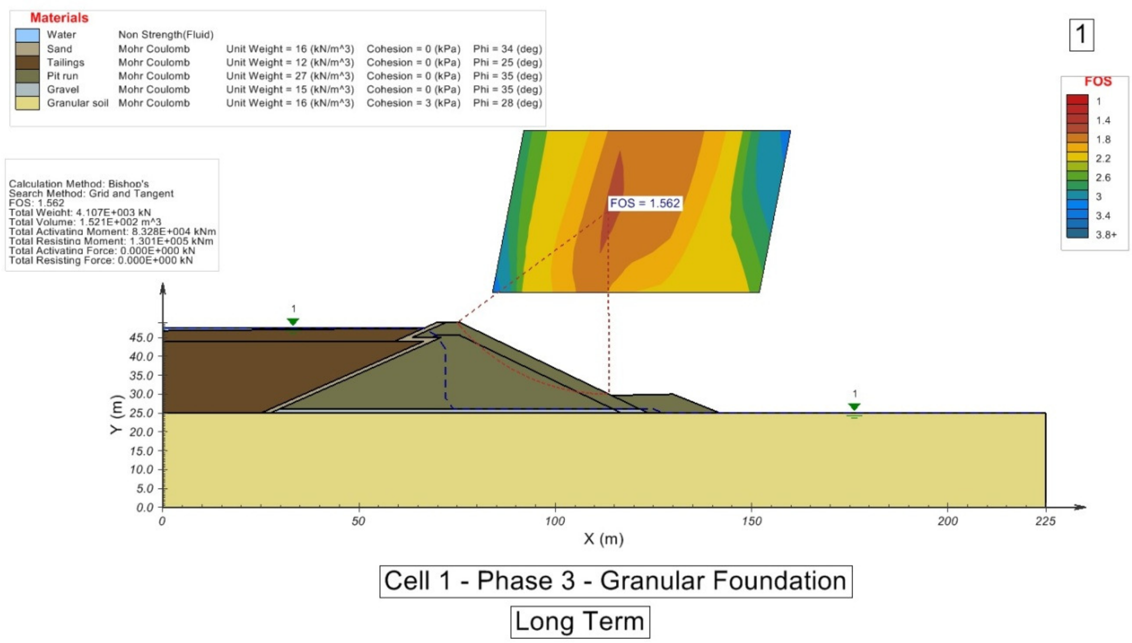

Figure 18.6 - Maximum tailings dam cross section and stability factor of safety.

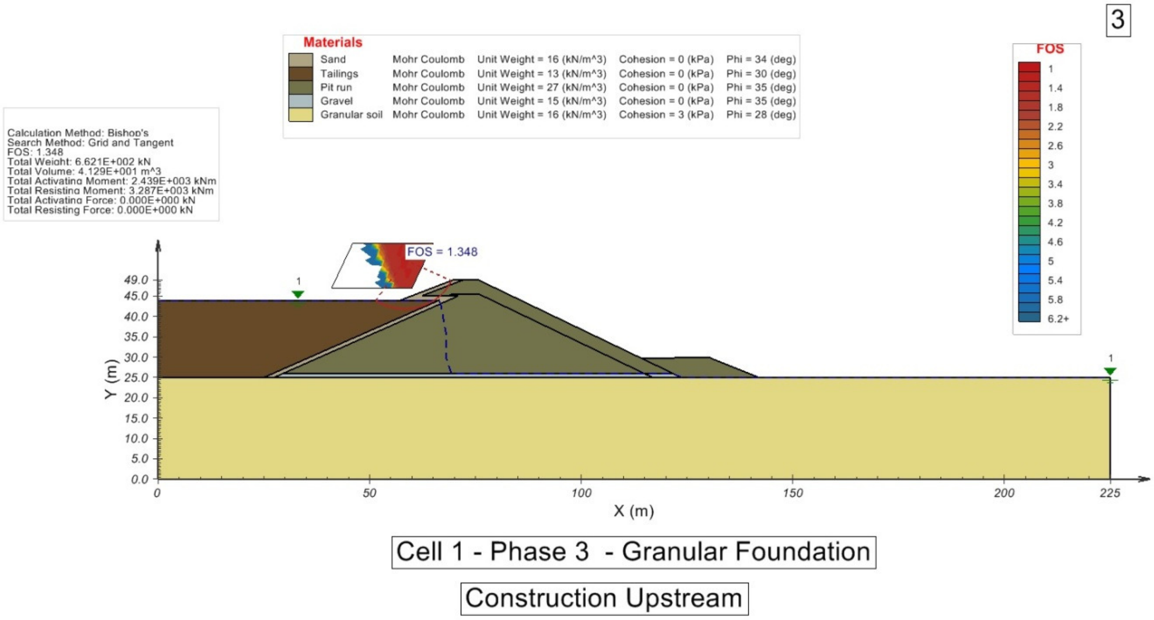

Figure 18.7 - Maximum tailings dam cross section and stability factor of safety.

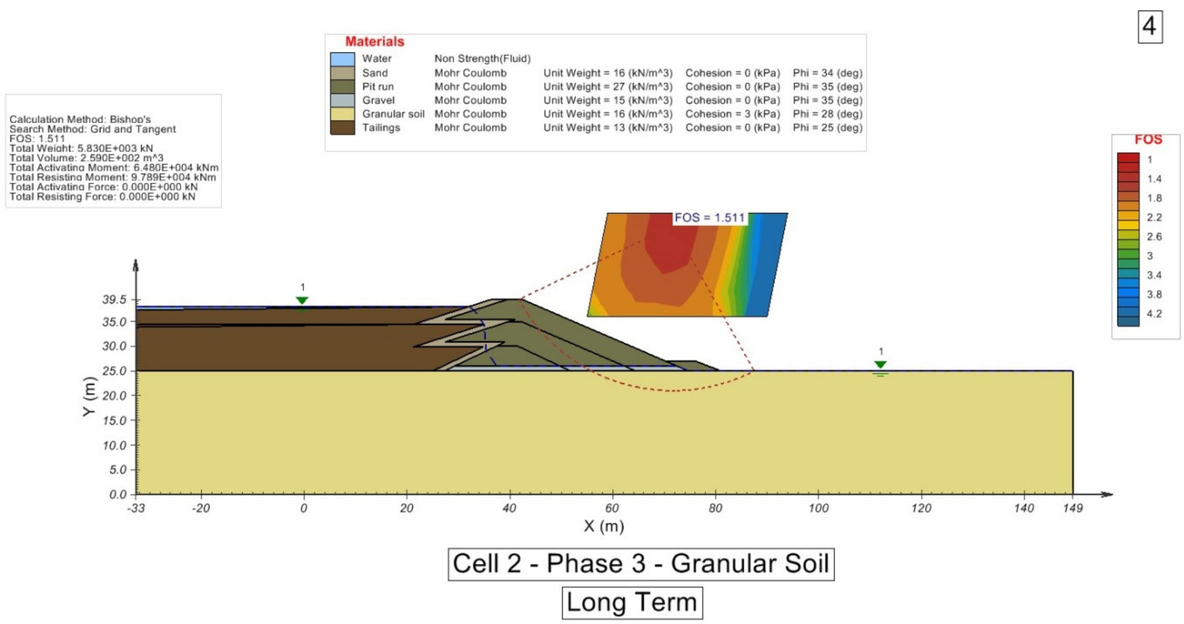

Figure 18.8 - Maximum tailings dam cross section and stability factor of safety.

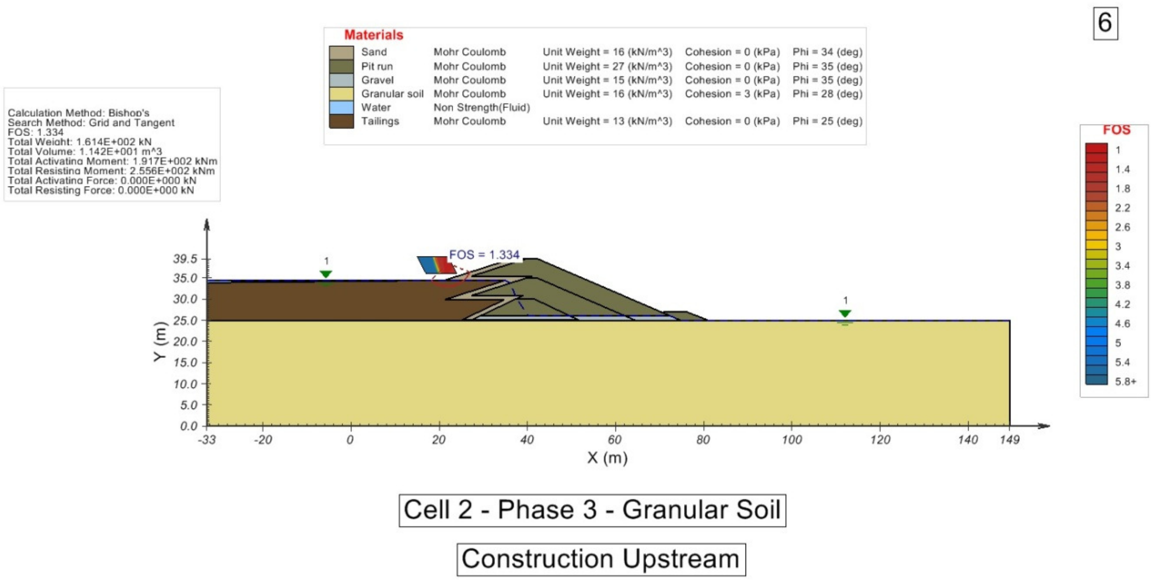

Figure 18.9 - Maximum tailings dam cross section and stability factor of safety.

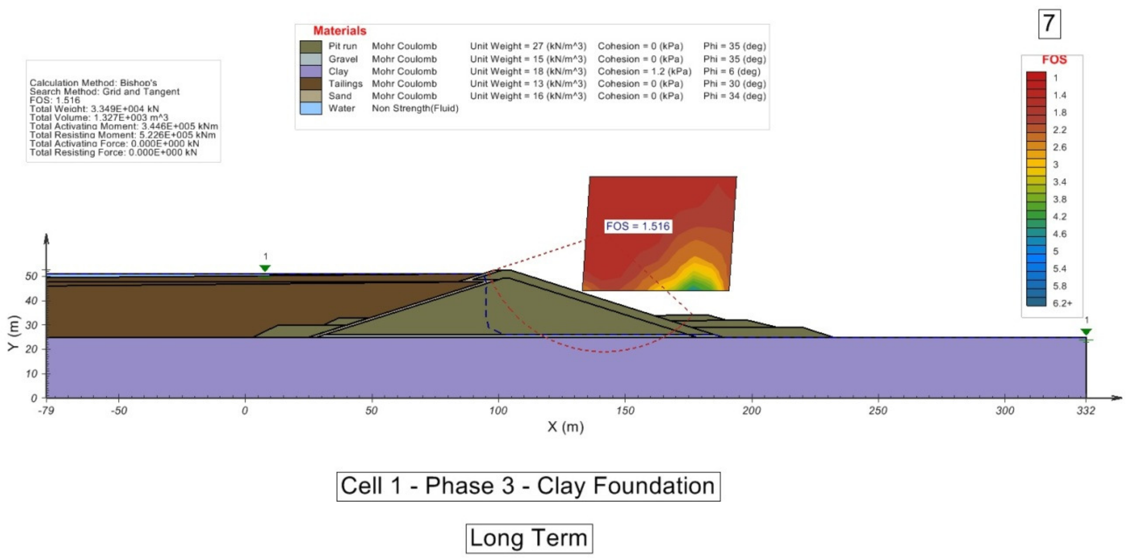

Figure 18.10 - Maximum tailings dam cross section and stability factor of safety.

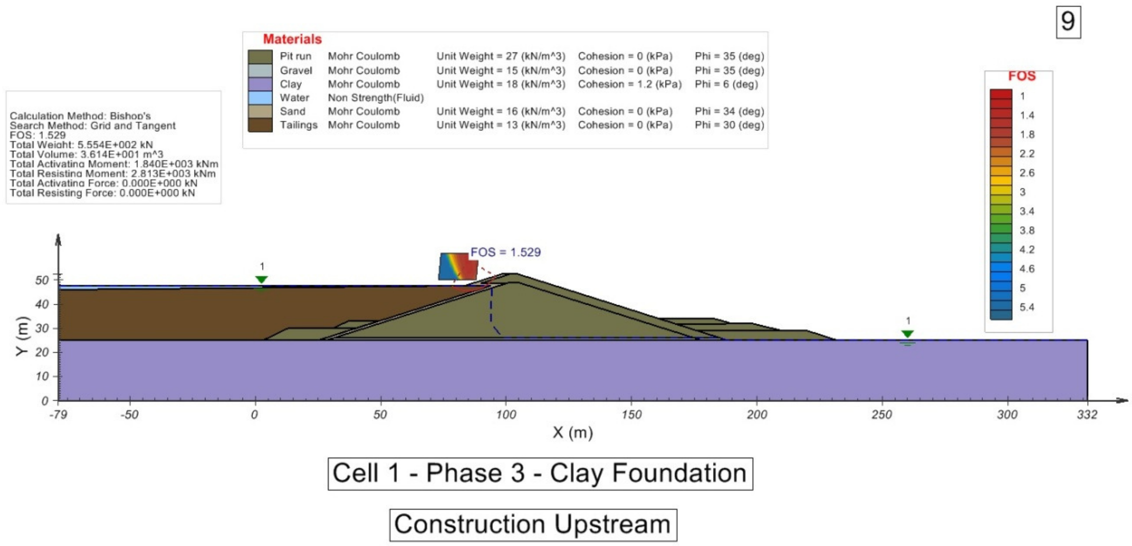

Figure 18.11 - Maximum tailings dam cross section and stability factor of safety.

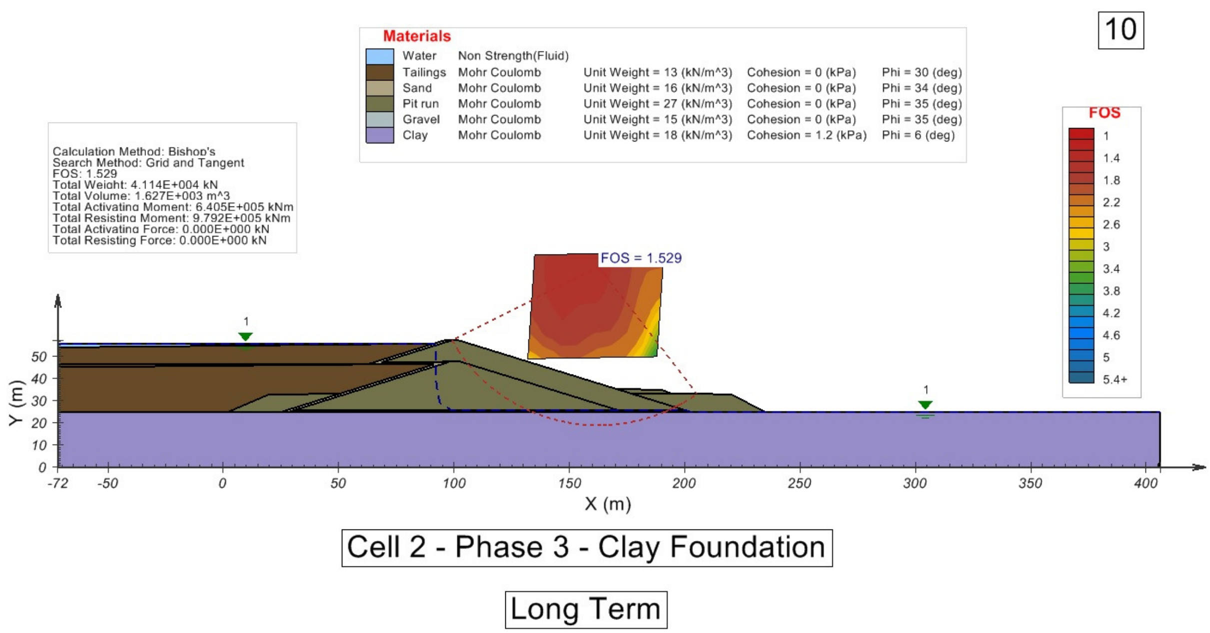

Figure 18.12 - Maximum tailings dam cross section and stability factor of safety.

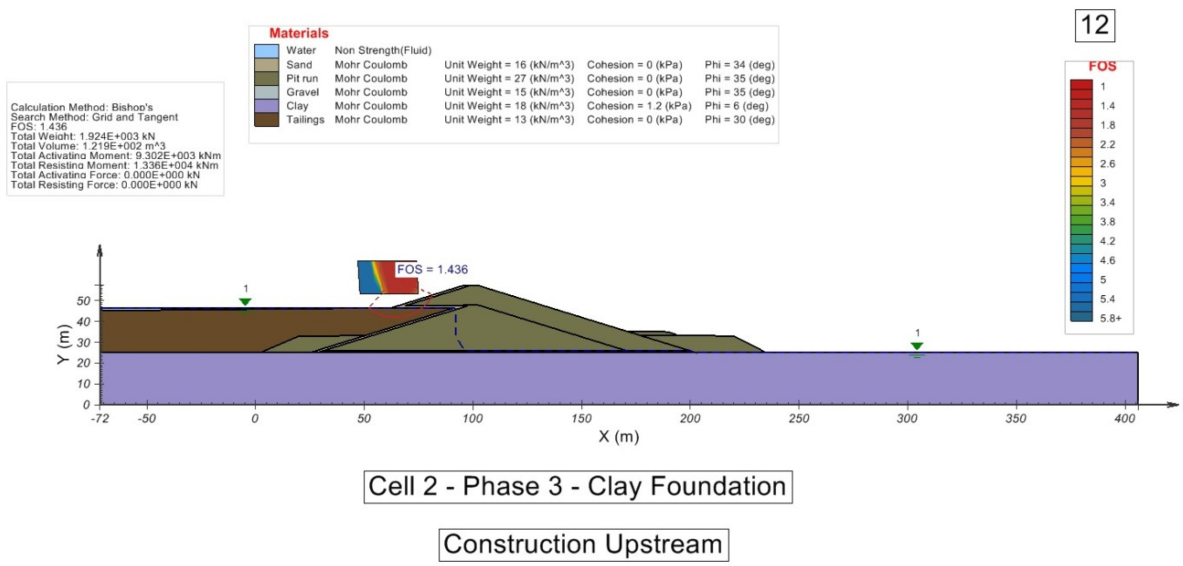

Figure 18.13 – Maximum tailings dam cross section and stability factor of safety.

It should be noted that the clay observed in the field was saturated beyond its liquid limit resulting in highly conservative designs for the dams founded on clay soil. In some cases, upstream berms were required to maintain stability during construction. Further geotechnical studies may result in reduction or removal of these berms. Trenching in the clay to firmer foundation materials may also be considered during the feasibility study.

18.2.1.3 Embankment slope angles and volume estimates

A summary of the preliminary stability analysis is presented below. The parameters used in the analysis were chosen from results of the fall 2013 field program and from values commonly found at other sites and in the literature. This assessment was based on preliminary geotechnical data in the form of test pits in and around the TSF. Due to access restrictions, the geotechnical testing was not necessarily performed below the dam foundation, but in regions where the foundation conditions are expected to be similar to those in the region of the TSF. The results of the testing provide preliminary foundation characteristics and help guide the future geotechnical investigation during a bridging program and/or during the feasibility study. At the end of this section is a summary of recommendations for the future geotechnical investigation required to better define the tailings storage facility foundation.

The values chosen tended towards conservative, which ultimately leads to greater material volume requirements and costs. Improved definition of the geotechnical properties of the foundation during the feasibility stage may allow for a further reduction in dam construction costs. For the current estimates, the slope ratios and safety factors are shown in Table 18.2.

Table 18.2 - Summary of preliminary embankment slope ratios and safety factor estimates.

Summary Table of the Different Factors of Safety for each Dam section and each Phase									
Cell	Soil foundation type	Phase	Crest elevation (m)	Slope (uptream)	Slope (downstream)	Factor of Safety			
						Construction phase		Seismic	Long term
						Downstream	Upstream		
1	Granular	1	331.1	2:1	2:1	1.43	1.38	1.36	1.55
	Granular	3	334.5	2:1	2:1	1.56	1.35	1.38	1.56
	Clay	1	331.1	3:1	3:1	1.55	1.50	1.23	1.55
	Clay	3	334.5	3:1	3:1	1.52	1.53	1.14	1.52
2	Granular	2	325.1	2:1	2:1	1.55	1.39	1.40	1.59
	Granular	3	334.5	3:1	2:1	1.51	1.33	1.33	1.51
	Clay	2	325.1	3:1	3:1	1.70	1.37	1.22	1.59
	Clay	3	334.5	3:1	3:1	1.53	1.44	1.25	1.53

Note: POX tailings dams will have the same configuration and crest elevation as cell-1, phase-1 granular foundation.

The material volume estimates, based on the above slopes, are presented in Table 18.3.

Table 18.3 - Volume and membrane estimates – Concentrate and POX options

Phase of TMF dam construction	Material	Units	Concentrate Option	POX Option
Phase-1	Drain (Gravel)	m^3	211,460	288,394
	Filter (Clean sand)	m^3	95,244	130,152
	Waste-rock	m^3	1,843,991	2,262,359
	Geomembrane	m^2	**1,006,410**	**190,000**
Phase-2	Drain (Gravel)	m^3	60,169	60,169
	Filter (Clean sand)	m^3	27,703	27,703
	Waste-rock	m^3	344,902	344,902
	Geomembrane	m^2	**1,449,840**	**0**
Phase 3	Drain (Gravel)	m^3	115,978	115,978
	Filter (Clean sand)	m^3	97,258	97,258
	Waste-rock	m^3	1,204,771	1,204,771

During the current PFS, berms for dam stabilization are considered in the volume and cost estimates since they will potentially be an important volume of material due to the relatively soft clay observed during the test pit program. Alternatively, following further characterization of the clay, some may be excavated from the dam foundation and placed as a liner below the membrane in the POX tailings cell. This requires further geotechnical characterization of the dam foundations during the FS to assess the deeper clay properties or presence of more stable dam foundation materials at depth.

Based on observations of the bedrock outcrops at the site, the bedrock is generally massive with little open jointing or weathering at the surface. However, a proper mapping, drilling, and hydrogeological assessment should be carried out during the feasibility study. At the current time, little grouting or excavation of bedrock is expected along the abutments or foundation.

The slope ratios described in the table above depend partly on the characteristics of the waste rock material used for the embankment. While the material from the pit is expected to provide suitable construction material, the characteristics of the material should be defined at the feasibility and detailed design levels of the Project.

18.2.1.4 POX tailings cell

The conceptual POX tailings storage cell is founded largely on granular soil to reduce the strain on the liner resulting from potential settlement and consolidation of a soft clay foundation soil. In addition, its placement makes use of the dams already forming cells 1 and 2.

Due to the lack of geotechnical information at this phase regarding the suitability or availability of clay as a construction or foundation material, the POX storage cell is expected to be a fully lined storage structure. Clay was found on the site in the fall 2013 test pit program, but proved to be highly saturated with low shear strength.

However, further field studies should be carried out during the feasibility study to assess alternative POX tailings storage locations, including valleys with firm clay or locations closer to the plant in order to reduce dam construction costs and piping/pumping requirements. In addition, it may be possible to stage the POX tailings dam construction to reduce the initial CAPEX. Water management of the POX tailings cell is to be assessed at the FS phase.

Construction of the POX impoundment area involves grubbing and removal of the topsoil, which is stockpiled for future restoration, followed by compaction of the exposed foundation soils. A geosynthetic clay liner (GCL) may be required below the membrane depending on the foundation soil characteristics (i.e., permeability, grain sizes, etc.). Alternatively, if suitable clay is located on the site, it may be placed below the membrane where the foundation soils are inadequate. A protective layer should be considered over the membrane to prevent damage from sun, temperature, and ice before the membrane is adequately covered by tailings. Finally, the drainage below the liner on the sides of the POX cell adjacent to cells 1 and 2 should be evaluated in detail to prevent seepage water from these cells from building up pressure below the POX cell liner. Further assessment of these needs should be considered during the feasibility study and detailed engineering.

18.2.1.5 Reclaim water system and effluent quality control

Runoff water and supernatant water from the tailings will accumulate in the northern sections of Cell 1 and Cell 2 next to the bedrock where dam construction is not required. The water accumulated will either spill over into polishing ponds or be pumped directly back to the mill for re-use. It is proposed that the northwest dam of Cell 2 is started during Phase 1 to form the initial polishing pond. Allowance should be made for regular hydraulic dredging of this pond to maintain residence time and avoid accumulation of slimes that would negatively impact the stability of future dam lifts in Cell 2. The need for a polishing pond in Phase 2 can be assessed based on the performance of the decant pond in Cell 1 during Phase 1 operations.

18.2.1.6 Conceptual design assumptions

The following assumptions were used at the current PFS level study, but these items will need to be defined during the feasibility study, associated geotechnical studies, and laboratory testing:

- Condemnation drilling for the proposed tailings site(s) is assumed to be complete or will be completed;
- Centreline dam construction (requires laboratory testing on tailings at feasibility stage);
- Contractor-placed fill from borrow source/open pit development;
- A 4-year permeable starter dam with decant pond at the north end (Cell-1);
- Staged tailings dam lifts in Year 1 and Year 7 to maintain a total tailings storage facility operation for 15 years, followed by closure;
- Assumed waste rock production schedule will be consistent with tailings dam construction schedule;
- Permeable dam construction for Cell 1 and Cell 2; lined basin for the POX tailings cell;
- Null tailings slope;
- Contingency of 10% for the total tailings tonnage;
- Surficial geology from MERN Canada, 2010, Map 6061 (including some verification with test pits in fall 2013);
- Spigotting / cycloning of tailings to occur around west, south, and east perimeters to maximize storage space and direct decant pond towards more stable bedrock outcrops;
- Deposited bulk density of 1.35 t/m^3 for desulphurized tailings, and 1.2 t/m^3 for POX tailings;
- Total required tailings storage volume of 34.7 Mm^3;
- Dam raises will be completed partially on compacted coarse or low plasticity tailings (to be verified at feasibility stage by lab testing);
- Generally dam lifts not to exceed 5 m per year for centreline construction. 10 m and 5 m raises evaluated for stability. 10 m raises can be revised to 5 m raises if a 10 m lift is not supported by geotechnical and laboratory results;
- Starter dams constructed with waste-rock (permeable). Feasibility study seepage analysis and water balance to verify sufficient recirculation water and the need for a low permeability core in the starter dams;
- Seepage analysis not assessed at PFS. Phreatic surface estimated assuming full drainage via upstream filter and drain in dam;
- Construction material quantities do not consider consolidation and settlement of dams or foundation soils. Lab testing at feasibility stage to characterize settlement.
- Cost calculations for liner assume single liner system. Clay soil liner and under drainage requirements to be assessed at feasibility stage;
- Trenching, cut-off slurry walls, and impermeable starter dams to be assessed following feasibility level geotechnical study and borrow pit investigation;
- Water management structure conceptual designs, including spillways, decant towers, and/or decantation barges to be assessed during the feasibility study;
- Freeboard assumed to be 1.5 m;
- Dam crest width of 6 m;

- Seismic coefficient (peak ground acceleration) for Rouyn-Noranda obtained from Natural Resources Canada (seismic hazard design values): vertical peak ground acceleration set at 0.11, and horizontal acceleration estimated at half the vertical (0.06).

18.2.1.7 Minimum recommended future work for the feasibility study

For the purposes of the geotechnical investigation and laboratory testing leading into the feasibility stage, the following is the minimum recommended for the bridging or feasibility studies:

- Early test pit program to delineate surficial geology and identify borrow pits for construction materials, and identify potential alternative POX cell locations if the POX option is selected; followed by:
- Geotechnical drilling program below the dam foundations and related infrastructure; program to be defined by results of test pit program;
- Bulk density of placed tailings (coarse & fine portions, drained & undrained);
- Permeability for tailings and clay material at various compaction levels;
- Consolidation testing of clay foundation material and tailings material;
- Undrained shear strength for foundation materials, including tailings;
- Atterberg limits for clay and tailings foundation materials;
- Proctor testing for dam construction materials;
- Dam filter and drain permeability under maximum loads expected;
- Interface shear testing of the liner system, including clay/membrane, foundation/membrane, and under drainage/membrane materials;
- Packer and permeability testing of foundation and rock abutments;
- Nilcon vane testing of clay foundation materials;
- Grain size distribution and borehole profiles;
- Packer and pumping tests around bedrock ridges that form natural containment structures for tailings.
- The following design and study considerations should be assessed at the FS:
- Under drainage design and material requirements for the lined POX tailings cell;
- Spillway designs, including decant towers, decant barges, or other methods for managing supernatant water in the tailings cells;
- Seepage analysis and water balance for process water and runoff;
- Stability analysis using updated geotechnical results;
- Storage capacity of tailings cells with updated tailings bulk densities and probable tailings deposition slopes;
- Scheduling of waste rock production to allow increased frequency of lifts and reduce capital construction costs;
- A hydrogeology study to assess potential impacts to groundwater and classification of the aquifers.

18.2.1.8 Waste rock, ore, overburden and tailings management

The criteria used for waste rock, ore, overburden and tailings management are depicted in Table 18.4. The values used to estimate bulk density and angle of repose were established following a review of other projects similar in terms of geology and ore processing. No geotechnical study was performed to assess soil bearing capacity under the proposed infrastructures except for the tailings storage facilities where preliminary geotechnical investigations were performed. Water management and associated leakproofing measures, following applicable guidelines (i.e., Québec's Directive 019), are presented in sections 20.1.2.4 and 20.3.

Table 18.4 – Criteria for waste rock, ore, overburden and tailings management

Criteria	Value
Flotation Tailings Storage Facility (3 phases)	
Tonnage (Mt)	47.5
Bulk Density (t/m³)	1.35
Volume (Mm³)	35.2
Phase 1 - Volume (Mm³)	11.4
Phase 1 crest elevation (masl)	331.1
Phase 2 - Volume (Mm³)	8.1
Phase 2 crest elevation (masl)	325.1
Phase 3 - Volume (Mm³)	15.7
Phase 3 crest elevation (masl)	334.5
Volume of waste rock used for dike construction (Mm³)	3.4
POX Tailings Storage Facility	
Tonnage (Mt)	2.0
Bulk Density (t/m³)	1.2
Volume (Mm³)	1.7
Max crest elevation (masl)	334.5
Volume of waste rock used for dike construction (Mm³)	0.4
Waste Rock Stockpiles (3 units)	
Tonnage (Mt)	282.2
Bulk Density (t/m³)	2.00
Volume (Mm³)	141.1
Height (m)	72/102/118
Bench height (m)	15.0
Angle of repose (°)	24.8
Ore Stockpile	
Tonnage (Mt)	10
Bulk Density (t/m³)	2.00
Volume (Mm³)	5.0
Height (m)	65.0
Angle of repose (°)	23.0
Overburden Stockpile	
Tonnage (Mt)	-
Bulk Density (t/m³)	
Volume (Mm³)	13.5
Height (m)	54.0
Angle of repose (°)	25.0

Note: All tonnages and volumes include a +10% contingency

18.2.2 Site Roads and Surface Pads

Local roads will be used to access the various industrial sites and services. Mining roads will be used for activities relating to mining production. The on-site roads will provide access to the following areas:

- Process plant facility and surrounding buildings;
- Open pits;
- Ore stockpile;
- Waste dumps and overburden dump;
- Explosives storage;
- Garage;
- Pumping stations;
- Tailings disposal area.

The selected location of haul roads will minimize the distances between long-term infrastructures. With the information available, only a small brook on the road to waste dump #1 will need to be crossed so neither major bridges nor river dykes will be required for the haul roads construction.

To estimate the haul road dimensions, the following assumptions have been taken in consideration:

- The haul trucks used are 150-ton payload trucks with a width of 6.64 m and a tire height of 3 m;
- The standard haul road construction design consists of four main layers:
    - the sub-grade (various materials),
    - the sub-base (sand),
    - the base course (coarse gravel),
    - the surface course (crushed gravel).

Most of the haul roads will be double lane. According to article 45.1 of the *Regulation respecting Occupational Health and Safety in Mines*, such roads need to be at least three times the width of a truck. Adding a 3 m berm on each side, the width of the double-lane roads are planned to be 26 m.

A few haul roads, such as the tailings pond access road, will only be single-lane due to the low traffic. According to the same regulation, these roads need to be twice the width of a truck. Including the same berm specifications, the width of the single-lane roads will be 20 m.

The haul roads require a maximum grade of 10%. According to the available topography, a major steep gradient to overcome is on the haul road between the central pit and waste rock dump #3. To smooth the road angle from 18% to 10%, about 40,000 m^3 of additional material will be required.

A total of about 13 km of mine roads will be required. All roads will be equipped with the appropriate information, regulatory and warning signs.

The work listed below will be performed at the industrial and service sites:

- Land clearing and grubbing;
- Stripping of topsoil;

Excavation and backfilling with adequate material up to infrastructure elevation, for construction of work platforms.

18.2.3 House Relocation

In the prefeasibility scenario, it is estimated that six (6) houses will need to be relocated or bought by Clifton Star. One of those houses is curently the property of the current mine owner and is used at times to house Clifton Star employees. The five remaining houses are located on Rue Principale, two are on the north side of the road, and three are on the south side of the road and the end of Rue Principale. No discussions have yet been held between Clifton Star and the owners of these houses.

18.2.4 Site Buildings

In addition to the crusher, ore storage and concentrator buildings which will house common circuits for both options (crushing, grinding, flotation, concentrate dewatering and gold room) and a specific circuit for the POX option (CIP circuit, POX circuit, neutralization), the site will include the following:

- The administrative building and mine dry (change house);
- The mine equipment maintenance facility and warehouse;
- The assay laboratory;
- The fuel farm;
- And the explosive plant and storage.

Each building will have its own electric room for power distribution equipment. These electrical rooms will be fed by individual 4160 / 600 V pad-mounted transformers connected to the overhead 4.16 kV power line.

18.2.4.1 Administrative building

The administration building (42.0 m x 24.0 m) will include offices for administration, mine supervision and technical support, and the infirmary. It will also contain the gatehouse, conference room, change house, as well as space for employee and mine rescue training.

Medical facilities will be installed in the administration building and will provide space and equipment needed for the medical staff and a nurse. An ambulance vehicle will also be parked in the administration building.

Fire protection will be available via water sprinklers. There will be fire protection cabinets with water hoses and fire extinguishers.

The building will be heated with electric unit heaters in the dry and ambulance sections, and electric baseboard heaters will heat offices and the bathroom.

18.2.4.2 Mine equipment maintenance facility

The garage building (80.0 m x 28.0 m) will have 6 bays with 9.14-m-wide doors. There will have washing, lubrication, welding, and repair bays for the mine vehicles. There will be also one repair bay for small vehicles and another one for miscellaneous jobs. A storage area will be available for parts and for oil and greases. On the first floor, there will be office space to be used by the maintenance staff, as well as a lunch room and a conference room.

The building will be heated with electric unit heaters in the maintenance and warehouse sections, and electric baseboard heaters will heat offices.

Ventilation of workspaces will be done through air exchangers and each of them will provide variable air flow in case of deleterious gas detection. The building will have water sprinklers for fire protection. There will be fire protection cabinets with water hoses and fire extinguishers.

18.2.4.3 Assay laboratory

The assay laboratory building will measure 24.4 m x 18.3 m for a total area of about 440 m^2. Double doors will allow reception of materials inside the building. A dust collector will be installed for the fusion, comminution, and preparation areas. Laboratory hoods will be installed in the wet lab section.

Water sprinklers will be installed for fire protection. There will be fire protection cabinets with water hoses and fire extinguishers.

Most of the building will be heated with electric baseboard heaters except for the storage area where a unit heater will be used.

18.2.5 Explosives Storage and Handling

The explosive storage will include the installation of garage/storage on site. The storage capacity will be 68,000 kg bulk emulsion. In addition, there will be a heated garage for the storage of the MMU (Mobile Manufacturing Unit), which is the emulsion pumping unit.

Only authorized and trained personnel will have access to the explosives storage area. The site will be enclosed by a fence and a lockable gate. Cameras and electrical lighting will be installed near the magazines site.

18.2.6 Telecommunications

Radio communication service will be available for site traffic and emergency response.

Site telecommunication will be provided by an optical fiber network with an IT network subsystem. VoIP type telephones will be installed.

The optical fiber backbone could also be used for process control and surveillance camera.

18.2.7 Security and Access Control System

The security officer will control the site access and will be located in a gate at the entrance of the mine site. This gatehouse will be part of the administration building. There will be an alarm system for fire protection and a surveillance system for the site via cameras. The cameras will be Ethernet type (Power Over Ethernet if possible) and all linked to the fibre-optic network.

Surveillance cameras will be installed to monitor the following areas: the gatehouse, overall plant site, the open pit, the mine vehicle maintenance shop, the sub-station, the pumping stations, and the explosive storage; all cameras will have pan, tilt and zoom functions.

Also cameras for process monitoring will be installed in the process plant (with adequate dedicated quartz or LED lighting):

- in the crusher area, including the ore bin chute;
- in the tunnel and process plant feed conveyor;
- in the concentrator sector;
- in the POX sector – for POX option only.

A DVR server with recording capability will be at the gatehouse. This server allows the operator to organize views on the screen, operate the cameras, and watch previously recorded images. The security officers will have access to the cameras via a TV screen in the gatehouse. The process plant and crusher operators will also have access to the process cameras via another TV screen.

18.2.8 Green Wall

A buffer zone 50 m wide will be developed along the southern limit of the Beattie pit to mitigate the impacts of the mining activities on the citizens of the Town of Duparquet. Inside this buffer zone, a green wall will be built mainly made of rock and topsoil from the prestripping mining work. This wall will be 10 m high and 33 m wide. Soil and organic matter will cover the surface and then different types of vegetation will be planted, such as shrubs trees and grasses. Figure 18.14 illustrates the green wall.

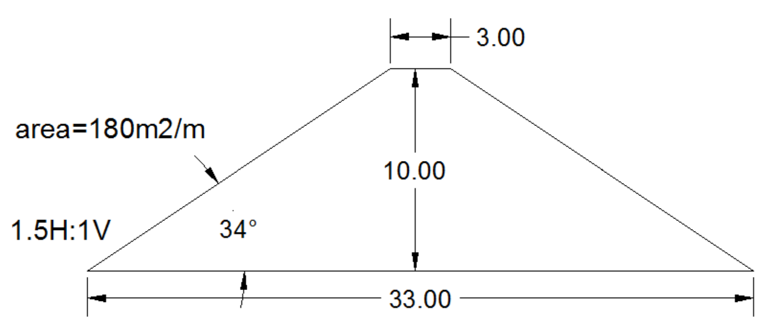

Figure 18.14 – Green wall illustration

18.2.9 Fence

Because of the operation's proximity to the Town of Duparquet and provincial roads, and to ensure security by preventing access to the property by intruders, fences will be installed around the mine site and facilities. A simple fence will be installed along the green wall, on east and west of Route 393, and around the site pad for process facilities and the administrative building with access control. Pedestrian turnstiles and motorized gates will be installed to control the flow to and from this area.

18.2.10 Fuel Storage and Distribution System

The mine site fuel storage facilities will have a concrete slab with containment berms for the vehicle filling area, bollards to protect the installation, and a membrane to prevent soil contamination in the event of fuel leakage or spill.

The fuel storage will include three diesel tanks of 50,000 litres for the mine fleet. This fuel will be used for filling the fuel tanker, mine trucks, and emergency power generation. A high flow dispenser with a hose reel is selected to fill the mine trucks, along with a low flow dispenser for other vehicles. It will also include a gasoline reservoir of 5,000 litres for small vehicles and other equipment.

An additional fuel tank will be installed next to the 5 MW emergency generating set (genset). This tank will be exclusively used for emergency power generation purpose.

18.2.11 Solid Waste, Water Treatment, and Management

At the mine site, domestic waste water will be treated through a standard system located in a container which will allow for water disinfection and phosphate removal before being returned to the environment.

Solid waste will be removed from the site by a contractor on a regular basis.

18.2.12 Process and Fresh Water

18.2.12.1 Site fresh water

The fresh water pumping station will be located at the old Central Duparquet mine shaft. This water source will be used for the first 6-7 years of operation. After this period, the Central Duparquet pit will be developed and another water source will be needed.

The fresh water pump will pump the water to the plant through a 2.8-km insulated and heat traced steel pipeline from the old Central Duparquet shaft. The nominal capacity of this pumping station will be 2400 m^3/d.

For that purpose, a building with an electrical room will be erected at that location. The main purpose of the building and its equipment will be to allow the installation and maintenance of the two pumps, one in operation and one on stand-by. The plant pumping station will have a fan for air evacuation and a power-driven louver. Heating will be provided by electric unit heaters and a heat trace will be installed to prevent the piping from freezing.

The fresh water pumping station will have a 4160 / 600 V transformer and will include an electrical room with distribution and control equipment. Remote signals will be available through the optical fiber link. Electric power will be provided by the 4.16 kV overhead power line installed along the pipeline.

18.2.12.2 Reclaim water

Tailings water reclaims will be pumped from the settling pond at the discharge of the tailings pond through a 4.8 km HDPE pipeline to be reused as process water in the process plant. The total pumping capacity will be 9,000 m^3/d.

For that purpose, a building with an electrical room will be erected at that location. The main purpose of that building and its equipment is to allow the installation and maintenance of the two pumps. The plant pumping station will have fan for air evacuation and a power-driven louver. Heating will be provided by electric unit heaters.

The tailings water reclaims pumping station will have a 4160 / 600V transformer and will include an electrical room with distribution and control equipment. Remote signals will be available through the fiber-optic link. Electric power will be provided by the 4.16 kV overhead power line installed along the pipeline. The water reclaim pumps will be on emergency power to allow flushing and avoiding sedimentation of the tailings pipeline in case of emergency shutdown. The water reclaim pipeline will not be insulated or heat traced because the system will recirculate if necessary to avoid freezing.

18.2.12.3 Fire protection

In the plant fresh water tank, there will be water capacity allocated for fire protection. The lower section of the tank will be kept for that purpose, representing around 175 m^3. There will be three dedicated pumps for fire protection at the process plant pump house. There will be two high pressure pumps, one electrical and one diesel with a small capacity (Jockey pump) to maintain pressure in the fire protection system and all required monitoring control panels.

18.2.12.4 Drinking water supply

Drinking water will be supplied through the local Duparquet city water distribution system for the administrative building, assay laboratory and process plants.

The explosives storage area will be serviced by an artesian well and will include a minimum operating reserve. No water treatment facilities are planned for this zone; bottled-water dispensers will be added if the water drawn from the well does not meet quality standards and requirements for human consumption.

19. MARKET STUDIES AND CONTRACTS

19.1 Market Studies

Markets for doré are readily available and the doré bars produced from Duparquet could be sold on the spot market. Gold markets are considered mature, despite a current gold price that is lower than the 3-year trailing average; in recent years, the demand has been high and the gold price has risen significantly. To better reflect current market expectation, this study uses a price of gold of US$1300/oz at a CAD/USD exchange rate of 1.1 : 1.

19.2 Contracts

No contracts have been assigned for the Duparquet Project considering the early stage of the Project.

20. ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Baseline Studies

20.1.1 Duparquet Project Area

In November 2013, an environmental baseline study (EBS) report was issued by Roche Ltd Consulting Group ("Roche") for the Duparquet Project area in order to refine a study previously completed by STAVIBEL/SNC-Lavalin in February 2013. The latter covered past environmental work performed on the area from 1994 to 2012. Components of the human environment characterizing the study area were also described by STAVIBEL and Roche in 2013. The following sections summarize the methodology used and the main results for each component.

20.1.1.1 Physical Environment

The physical environment of the Duparquet Project area was described based on information collected from various sources:

1. Field surveys;
2. Aerial photographs and/or satellite images, maps, and geomatics tools;
3. Information gathered from various governmental agencies as well as by other project proponents active in the territory (other mine projects);
4. Studies from the scientific and technical literature.

An evaluation of the site's existing environmental conditions was performed based on two environmental assessment studies (SNC-Lavalin, 1995 and 2012). Soil, groundwater and surface water as well as lake and river sediment analyses were performed in order to assess the physical environment before the start of mining operations. These preliminary characterizations mainly established the characteristics and natural variability of the environment in anticipation of site restoration objectives. Sample analyses were conducted by independent laboratories in Québec, following standard quality control programs (QA/QC).

Findings showed that arsenic contamination, generated by the initial milling processes which began in 1933 and the later addition of ore roasting activities that took place between 1937 and 1956 is present on the industrial site and the neighbouring residential and institutional sites. In addition, in certain areas, molybdenum and occasional mineral oil and grease contamination were also found.

<u>Hydrology</u>

No hydrogeological assessment has been performed to date. Soils on the site consist of thin layers of sand or silty sand and clay. In general, groundwater flows to the south, towards Lake Duparquet.

The main water course whose watershed overlaps the Property is Creek #1. The source of this creek is located northeast of the Property. It starts flowing from the east to west and then turns south and flows into Lake Duparquet, just west of the existing tailings beach. The Creek #1 watershed drainage area is approximately 1,600 ha. Close to one third of the Property is located within this watershed. The remainder of the Property is located on the upstream area of several watersheds flowing to the south, i.e. also towards Lake Duparquet. The old mine pit, also called Glory Hole, is located in the Creek #1 watershed.

Soil quality

During the 1995 study, several test pits were carried out in areas of concern. The waste deposit, the sawdust storage area as well as another storage area and some soil samples were all analyzed for arsenic, molybdenum and mineral oils and grease. Some soil and waste samples were submitted to leaching tests for arsenic and molybdenum.

In the 2010-2011 study, 104 boreholes were drilled to the south, northwest and west of the mine tailings area in order to determine the quality of soils and tailings. The Beattie mine tailings contained in the area where samples were collected are not acid generating. The tailings samples yielded results above applicable criteria for certain metals (arsenic, manganese, molybdenum, and in some cases copper, nickel, lead and selenium). Soil samples collected from the native soil layer below the tailings also exceeded applicable criteria for some metals.

Site rehabilitation will take place as part of the Duparquet Project (see Section 20.2.6 of this Report).

Groundwater quality

The monitoring wells that were installed to assess groundwater vulnerability during the 1995 study have been sampled regularly since 1999. This monitoring consisted of 21 sampling programs conducted between June 1999 and May 2011.

In the 1995 study, groundwater analysis indicated very high arsenic and sulphur concentrations, as well as a high molybdenum concentration (above the MDDELCC's Groundwater Criteria for Drinking Water) in the farthest and uppermost well (F-1A). In the same location, analytical results indicated high molybdenum and sulphur concentrations (above the Drinking Water criteria) in the farthest and lower-most well (F-1). The other wells indicated low arsenic and molybdenum concentrations, but high sulphur contents (above the Drinking Water criteria). Figure 20.1 locates the wells.

Between 1999 and 2011, the groundwater analysis indicated very high arsenic concentrations in well F-1A, at levels above the criteria for Drinking Water and for Seepage into Surface Water or Infiltration into Sewers. Numerous concentrations above the criteria for Drinking Water were also measured in well F-1. Some arsenic concentrations above the criteria for Drinking Water have also been measured in wells F-2, F-3 and F-3A since 2008. As well, high molybdenum concentrations (above Drinking Water criteria) were also detected in wells F-1, F-1A and F-2.
These results indicate that the local sandy deposits, where wells F-1 and F-1A are located, are characterized by groundwater with high concentrations of arsenic, molybdenum and sulphur. On the other hand, the results indicate that the clay deposit where F-3 and F-3A are located contains groundwater with low arsenic and molybdenum concentrations, but a high sulphur content.

These results may indicate that the clay unit is less susceptible to contamination transport.

2010-2011 Study (mine tailings area)

Twenty-one (21) monitoring wells were installed in the tailings and in the underlying natural material in order to determine groundwater quality. Groundwater analytical results collected in tailings showed arsenic and chromium concentrations above the criteria for Seepage into Surface Water or Infiltration into Sewers, for almost the entire surface of the tailings area. Numerous exceedances of the arsenic and chromium criteria were also observed in wells installed in the natural soils underlying the aquifer.

Reservoir No. 1 - Complementary sampling of the observation wells

A groundwater sampling program was carried out near Reservoir No. 1 in October 2012. In well PZ-3, water analysis results indicated high concentrations (above the MDDELCC's criteria for Seepage into Surface Water or Infiltration into Sewers) of aluminum, arsenic, copper, mercury and total phosphorus. In PZ-2, the water analysis results indicated high concentrations (above the criteria for Seepage into Surface Water or Infiltration into Sewers) of aluminum, arsenic, silver, cadmium, copper, mercury, molybdenum, lead, selenium and total phosphorus. It has, however, recently been determined that several of these wells were placed within the limits of previously existing reservoirs that were used to store arsenic trioxide material, thus explaining the high values.

Surface water quality

In the 2010-2011 study, a total of 24 surface water samples were collected along Creek #1, Lake Duparquet and their tributaries. Samples were collected over time and at different periods to evaluate the hydrochemistry of the various watercourses in the vicinity of the study area. During the summer of 2013, ten (10) samples were collected (Figure 20.1). Of these, eight (8) were taken in rivers and two in lakes.

Samples were all collected in accordance with applicable provincial and federal guidelines. Analysis parameters meet the recommendations of Directive 019.

In general, surface waters of the study area are characteristic of shallow aquatic environments rich in organic matter. The waters are relatively turbid, mildly acidic to alkaline and they have moderate to high buffering capacities. The conductivity, total dissolved solids and hardness results from the water samples vary considerably from one station to the next. Observed variability is likely due to the hydrological differences between streams and lakes and, particularly at station ST-3 (Figure 20.1), to sediment transport in the runoffs coming in from the village of Duparquet. No petroleum hydrocarbon contamination was detected in any of the water samples.

Surface water mineral content is highly variable. Nitrogen concentrations are relatively low. Total phosphorus concentrations are characteristic of mesotrophic (0.01-0.03 mg/l) to hyper-eutrophic (>0.10 mg/l; MDDELCC, 2013) water environments. The proximity of the village of Duparquet and of its sewage treatment plant as well as the agricultural areas found close to stations ST-6, ST-7 and ST-8 likely explain the high mineral and phosphorus concentrations that were observed (Figure 20.1).

Arsenic concentrations vary between 0.0027 and 0.190 mg/l. It is not unusual to observe high arsenic concentrations in samples collected close to or on the old Beattie gold mine site and its tailings accumulation areas. Observed results and relevant literature suggest that high arsenic results obtained from the samples collected outside of the old mine site are, to a large extent, of natural origin. In fact, the Archean sedimentary rock that composes the geology of the area is known to be a potential source of arsenic. Other metals found in concentrations higher than the detection limit are aluminum, barium, copper, iron, manganese, mercury, lead and zinc.

The concentrations of dissolved oxygen, total phosphorus, aluminum, arsenic, copper, iron, manganese, mercury and lead all exceeded the criteria for the protection of aquatic life in at least one water sample. No real trend in the spatial distribution of these exceedances was observed, with some being located in past mine areas while others were located in reference areas out of the Property limits and in a separate sub-watershed. It is important to note that, in the surface waters of mineralized areas with an economical potential, measuring high metal concentrations exceeding aquatic life protection criteria or recommendations is not unusual.

In order to complete the surface water characterization program, it would be important to collect samples at different hydrological periods such as during spring freshet since this important hydrological period is usually different from the others. In addition, all waterbodies likely to be affected by the projected infrastructure should be comprehensively portrayed.

Figure 20.1 – Location of sampling stations for sediment, surface and ground water quality assessment

Sediment Quality

In the summer of 2013, sediment samples were collected at the same sites used for water quality assessment (Figure 20.1).

Sediments found in water bodies of the study area are mostly composed of sand or of clay and silt. Total organic carbon content of the collected samples varies between 0.6 and 5.0% and it is higher in the sediments composed of clay and silt. Sulphur content is higher in samples collected on the old Beattie gold mine site. Measured levels of arsenic, chromium, mercury, copper, cadmium, lead and zinc all exceeded the criteria for the protection of aquatic life in at least one sediment sample. As for surface water, no real trend in the spatial distribution of these exceedances was observed. Sample concentrations for iron, manganese, arsenic, copper, molybdenum and mercury were highly correlated to sulphur concentrations whereas aluminum, barium, chromium, lithium, nickel, vanadium and zinc results were correlated to organic carbon concentrations of the sediment samples. Additional sediment quality programs should be done at different periods of the year in order to monitor changes in their quality.

20.1.1.2 Biological Environment

Vegetation and Wetlands

No exhaustive floral inventory has been made on the study zone sector to date, although the flora of the Kekeko hills, located some 50 km south of the Project, have been studied. STAVIBEL reported that this data constitutes the only publicly available inventory for this area. As part of the future development of the Duparquet Project, an inventory of terrestrial habitats and wetlands will need to be conducted in order to list plant species present on the Property as well as those that may potentially be present.

According to the Department of Energy and Natural Resourcesof Québec (MNR), there are designated exceptional forest ecosystems (EFE) along Lake Duparquet. Those have been classified and protected since 2002 under section 14 of the *Sustainable Forest Development Act* (R.S.Q., c A-18.1, Québec Government). The *Forêt d’enseignement et de recherche du Lac Duparquet* (“Lake Duparquet Research and Teaching Forest”; FERLD) has three EFEs within its boundaries: the ancient Akotekamik forest, the ancient Lac-Bayard forest and the rare Lake Duparquet forest. Four other EFEs have been proposed by the MNR and the FERLD. None of those are located in the immediate vicinity of the Duparquet Project.

The regional sector under study for the Duparquet Project contains a number of naturally occurring lakes, none of which lie directly within the current limits of the mining project.

According to the regional pedological map, the Duparquet Project area is generally surrounded by heavy clay deposits, but a peatland is observed about 400 to 500 m west of it. According to the MNR, there are also two peatlands located about 400 to 500 m south and east of the study area. Moreover, a number of beaver dams contribute to altering the hydric nature of the habitat.

Terrestrial Wildlife

No exhaustive terrestrial wildlife inventory has been completed to date. Based on the literature, 198 bird species have been identified on the land under the responsibility of the FERLD and 54 species of mammals and micromammals are likely to be found within that same area.

Information obtained from the MNR indicates that there are two white-tailed deer (*Odocoileus virginianus*) yards in the Duparquet region.

Finally, the MNR also provided information on wildlife species likely to be found in the Duparquet Project area. The presence of an aquatic bird concentration area, heron colonies and bald eagle nests (special status species) were confirmed in the area. In addition, four large mammal species, two small mammal species (including one special status species) and eight micromammal species (including one special status species) are known to be present within that area according to the MNR's data. Moreover, during the course of the August 2013 surveys, one species of snake was recorded.

No specific bird or wildlife surveys were done as part of the Project. Thus, bird and terrestrial wildlife surveys should be done for the subsequent realisation of the project impact study.

Fisheries and Fish Habitats

According to STAVIBEL, three spawning sights are known to be present in the Duparquet region: two for walleye (*Stizostedion vitreum*) and one for northern pike (*Esox lucius*). Four potential spawning areas have also been identified, two of them for freshwater drum (*Aplodinotus grunniens*). According to the MNR, fish were also identified in the Glory Hole water body.

As part of the fish habitat surveys, nine water courses that may be impacted by future mining operations were characterized in August 2013 using standard methods recommended by provincial and federal authorities (Figure 20.1). The water courses were characterized over a distance of 100 m. Each 100-m segment was first divided into homogeneous sub-sections following Boudreault (1984). Homogeneity was defined using criteria such as type of watercourse, particle size distribution of the riverbed, width and depth. Other biophysical characteristics – such as width at the natural high water mark, water flow, presence and type of aquatic vegetation, etc. – were also noted. In addition, the potential ecological functions of each sub-section were also noted.

Three principal types of flow facies were identified: ponds, channels and zones alternating between rapids and weirs. Almost all streams found on the Duparquet Property presented a channel type flow facies with a clay substrate. Most streams could be considered to be a poor to good fish habitat. A potential impassable fish passage was located on watercourse #1, which is the only stream or channel to cross the project site.

No experimental fisheries were achieved during the August 2013 field surveys. Only a specific fish field survey could confirm fish presence/absence in streams as well as which ones can be considered as fish habitats. This survey would also allow characterizing streams that were not surveyed in August 2013. The MNR's answers to information requests indicate that 34 species of fish have been found in the area during the course of their experimental fisheries. According to Bernatchez and Giroux (2012), 44 species of fish are likely to be found in the area.

20.1.2 Waste Rock, Ore and Tailings Environmental Characterization

20.1.2.1 Methods

In order to determine the geochemical characteristics of mine materials to be produced as part of the Duparquet Project and to define the associated management requirements, Maxxam Analytics, an MDDELCC-accredited laboratory based in Québec City, Québec, and SGS Canada, based in Lakefield, Ontario, performed several tests and/or measurements on various samples of waste rocks (CS-1 to CS-12) and ore (CS-13) selected by Clifton Star's geologists. Flotation (CS-17) and POX tailings (NT7 + CND3-2) samples were produced by SGS Minerals Services (Table 20.1).

Table 20.1 List of samples used for geochemical characterization

ID	Material State	Sample Description	Sampling
CS-1-3	Rock / core sample	Waste Rock - Barren Sediment	Mean of 3 samples
CS-4-6	Rock / core sample	Waste Rock - Barren Volcanic	Mean of 3 samples
CS-7-9	Rock / core sample	Waste Rock - Barren Syenite	Mean of 3 samples
CS-10-12	Rock / core sample	Waste Rock - Barren Composite	Mean of 3 samples
CS-13	Dry Solids	Ore - Pilot Plant (PP) Composite	Mean of 3 samples
CS-17	Liquid + Solids	PP-09 Flotation Tailings - Pulp	Mean of 3 samples
NT7 + CN3-2	Liquid + Solids	PP-POX Tailings after neutralization, cyanidation and cyanide destruction	1 sample

<u>Waste Rock, Ore and Tailings (solid portion)</u>

- Metal content for all samples except POX tailings by partial digestion using aqua regia followed by measurements of Al, Ag, As, B, Ba, Be, Bi, Ca, Cd, Co, Cr, Cu, Fe, Hg, K, Mg, Mn, Mo, Na, Ni, Pb, Sb, Se, Sn, Ti, U, V and Zn, according to Method MA. 200 – Mét. 1.2 (Centre d'expertise en analyse environnementale du Québec (CEAEQ), March 2011) and Method MA. 200 Hg 1.1 for mercury or equivalent (CVAAS);
- Metal content for POX tailings samples by total digestion using a mixture of HNO_3, HF, $HClO_4$ and HCl followed by measurements of Al, Ag, As, B, Ba, Be, Bi, Ca, Cd, Co, Cr, Cu, Fe, Hg, K, Mg, Mn, Mo, Na, Ni, Pb, Sb, Se, Sn, Ti, U, V and Zn by ICP-OES/MS;
- US EPA Toxicity Characteristic Leaching Procedure (TCLP 1311) leaching test according to Method MA.100-Lix.com.1.0 (CEAEQ, September 2010) and measurement of Al, Ag, As, B, Ba, Be, Bi, Ca, Cd, Co, Cr, Cu, Fe, Hg, K, Mg, Mn, Mo, Ni, Pb, Sb, Se, Sn, Ti, U, V and Zn in leachates according to CEAEQ Method MA. 200 – Mét. 1.2 and Method MA. 200 Hg 1.1 for mercury or equivalent;

- US EPA Synthetic Precipitation Leaching Procedure (SPLP 1312) leaching test according to Method MA.100-Lix.com.1.0 and measurement of Al, Ag, As, B, Ba, Be, Bi, Ca, Cd, Co, Cr, Cu, Fe, Hg, K, Mg, Mn, Mo, Na, Ni, Pb, Sb, Se, Sn, Ti, U, V and Zn in leachates according to CEAEQ Method MA. 200 – Mét. 1.2 and Method MA. 200 Hg 1.1 for mercury or equivalent (not performed on POX tailings samples);
- Environment Canada's Wastewater Technology Centre Laboratory 9 (CTEU-9) leaching test according to Method MA.100-Lix.com.1.0 and measurement of Al, Ag, As, B, Ba, Be, Bi, Ca, Cd, Co, Cr, Cu, Fe, Hg, K, Mg, Mn, Mo, Ni, Pb, Sb, Se, Sn, Ti, U, V and Zn in leachates according to CEAEQ Method MA. 200 – Mét. 1.2 and Method MA. 200 Hg 1.1 for mercury or equivalent (not performed on POX tailings samples);
- Shake Flask Extraction (SFE) testing on POX tailings samples only and measurement of Al, Ag, As, B, Ba, Be, Bi, Ca, Cd, Co, Cr, Cu, Fe, Hg, K, Mg, Mn, Mo, Ni, Pb, Sb, Se, Sn, Ti, U, V and Zn in leachates according to CEAEQ Method MA. 200 – Mét. 1.2 and Method MA. 200 Hg 1.1 for mercury or equivalent
- Neutralization Potential (NP) according to Modified Sobek ABA Method and Acidification Potential (AP) according to CEAEQ Method MA.110 – ACISOL 1.0 or equivalent.

Tailings (decant solution)

- Al, Ag, As, B, Ba, Be, Bi, Ca, Cd, Co, Cr, Cu, Fe, Hg, K, Mg, Mn, Mo, Na, Ni, Pb, Sb, Se, Sn, Ti, U, V and Zn according to CEAEQ Method MA. 200 – Mét. 1.2 and Method MA. 200 Hg 1.1 for mercury or equivalent.
- pH, hardness and conductivity;
- total acidity and total alkalinity;
- ammonia (CEAEQ Method MA.300 – N 2.0 or equivalent);
- dissolved organic carbon and dissolved inorganic carbon (CEAEQ Method MA.300 – C 1.0 or equivalent);
- fluorides (CEAEQ Method MA.300 – F 1.2 or equivalent);
- phosphorus (CEAEQ Method MA.315 – P 2.0 or equivalent);
- chlorides (CEAEQ Method MA.300 – Ions 1.3 or equivalent);
- nitrates and nitrites (CEAEQ Method MA.300 – NO3 2.0 or equivalent);
- sulphates (CEAEQ Method MA.300 – Ions 1.3 or equivalent);
- sulphides (CEAEQ Method MA.300 – S 1.2 or equivalent);
- total cyanide and free cyanide (CEAEQ Method MA.300 – CN 1.2 or equivalent);
- cyanates (CEAEQ Method MA.315 – CNO 1.1 or equivalent);
- thiocyanates and thiosulphates (CEAEQ Method MA.304 – Ions 1.1 or equivalent).

20.1.2.2 Applicable Management Requirements

In Québec, Directive 019 for the mining industry is the document used by the Department of Sustainable Development, Environment, Wildlife and Parks (MDDELCC) to authorize and monitor mining projects. It should be noted that in that document, waste rock, tailings, treatment sludge and dusts are all defined as “mining residues” (or mining waste).

Directive 019 outlines requirements with regards to the characteristics of the residues (low-risk, leachable, acid-generating, cyanide-containing or high-risk) and defines leakproofing measures to be applied to mining waste management facilities, thereby ensuring groundwater protection. Figure 20.2 specifies criteria to be considered in determining the leakproofing measures as depicted in Directive 019.

Defined as "low risk" is any mining residue with metal concentrations that comply with criterion A of the Soil Protection and Rehabilitation of Contaminated Sites Policy (SPRCSP) for a specific geological province (in this case, the Superior Province) or the natural local background levels of the area. However, residues that produce a leachate with concentrations lower than the groundwater protection criteria specific to "leachable" mining residues are also considered "low risk", even if total metal content is greater than criterion A of the SPRCSP.

A sample is considered "leachable" for a given element if its metal concentration is higher than Québec's SPRCSP criterion A and its leachate concentration obtained with TCLP 1311 test is higher than the SPRCSP groundwater protection criteria for surface water seepage. Leachable residues require groundwater protection measures. It is important to note that there are no criterion A values for some metals.

Directive 019 specifies that "acid-generating" mining wastes are characterized by a sulphur content of more than 0.3% and for which an acid generating potential has been confirmed by one of the two following conditions: 1) the net acid neutralizing potential (NP-AP) is lower than 20 kg $CaCO_3$/t of residues; or 2) the acid neutralizing potential ratio (NP/AP) is lower than 3. "Acid-generating" residues require groundwater protection measures.

It should also be noted that any mining residues produced as part of a process involving cyanidation are considered as "cyanide-containing". Such residues require groundwater protection measures.

Finally, defined as "high risk" is any material that produces leachate with levels higher than those defined in Appendix II of Directive 019 or material that is considered radioactive according to the definition set out in Directive 019. "High-risk" residues require the strictest groundwater protection measures.

20.1.2.3 Results

The following tables present all results obtained as part of the above-mentioned tests:

- Table 20.2: Metal content in solids for waste rock, ore and tailings;
- Table 20.3: Characteristics of the decant solution of tailings samples prepared by SGS Minerals Services;
- Table 20.4: Metal content in leachates produced following TCLP 1311 leaching tests on solids for waste rock, ore and tailings;
- Table 20.5: Metal content in leachates produced following SPLP 1312 leaching tests on solids for waste rock, ore and tailings;
- Table 20.6: Metal content in leachates produced following CTEU-9 leaching tests or Shake Flask Extraction tests on solids for waste rock, ore and tailings;

- Table 20.7: Acid-generating potential for waste rock, ore and tailings.
- According to those results and following Directive 019, Duparquet's mining residues are classified as:

    - Not "high risk";
    - Not potentially "acid-generating", except for POX tailings which are potentially acid-generating;
    - Potentially "leachable" according to TCLP 1311 test results for Al, Ba, Be, Cu, Fe, Pb, Ni, Ag and Zn. However, it should be noted that results obtained from SPLP 1312 and CTEU-9 (or SFE) leaching tests for waste rock, ore and both types of tailings tend to show that such materials would most likely not be leachable in "real-life" field conditions. SPLP 1312, CTEU-9 and SFE tests are more representative of such conditions than TCLP 1311, but are still over-conservative (SPLP grain size is <9.5 mm, CTEU-9 is <150 µm and SFE is 1.0 mm to 2.5 mm; inadequate solid/liquid ratio, etc.).
    - Therefore, based on the information gathered so far, waste rock, ore and both types of tailings are considered as potentially non-leachable. Kinetic tests will be performed to confirm that those mining residues are not leachable.

POX tailings are also classified as "cyanide-containing" since their production involves a carbon-in-leach (CIL) circuit during which carbon adsorbs the gold from the solution as cyanidation of the ore proceeds. Flotation tailings do not go through cyanidation. See Section 17 of this Report for more information on ore processing.

Table 20.2 - Metal content in solids for waste rock, ore, concentrate and tailings

Parameter	Unit	Reported detection limit(s)	Soil Protection and Rehabilitation of Contaminated Sites Policy (SPRCSP)			Metal Content (Solids)						
						Waste Rock				Ore	Tailings	
						Barren Sediment	Barren Volcanic	Barren Syenite	Composite		Float	POX
			Criterion A[1]	Criterion B	Criterion C	CS-1-3	CS-4-6	CS-7-9	CS-1-12	CS-13	CS-17	NT7 + CND3-2
Metals and Metalloids												
Aluminum (Al)	mg/kg	20	-	-	-	29,333	22,000	5,400	18,233	5,100	3,000	29,000
Antimony (Sb)	mg/kg	0.1	-	-	-	5	0	1	2	3.8	2	44
Arsenic (As)	mg/kg	2	5	30	50	**170**	**19**	**17**	**60**	**567**	**160**	**2900**
Barium (Ba)	mg/kg	4	200	500	2000	115	115	**663**	**363**	**253**	197	**590**
Beryllium (Be)	mg/kg	0.1	-	-	-	0.4	0.7	1.3	0.8	0.7	0	1.2
Bismuth (Bi)	mg/kg	2	-	-	-	<2	<2	<2	<2	<2	<2	5.8
Boron (B)	mg/kg	2	-	-	-	6	8	3	6	7	3	23
Cadmium (Cd)	mg/kg	0.1	0.9	5	20	0.1	<0.1	<0.1	<0.1	0.1	0.1	0.8
Calcium (Ca)	mg/kg	20	-	-	-	21,967	20,333	15,633	20,000	34,333	37,333	90,000
Chromium (Cr)	mg/kg	1	85	250	800	**697**	54	5	**234**	7	**99**	**150**
Cobalt (Co)	mg/kg	1	20	50	300	**45**	19	11	**22**	17	3	**57**
Copper (Cu)	mg/kg	1	50	100	500	48	**77**	20	42	26	**58**	**190**
Iron (Fe)	mg/kg	10	-	-	-	48,333	52,000	23,000	41,667	26,333	18,333	91,000
Lead (Pb)	mg/kg	1	40	500	1,000	14	2	14	21	23	13	**120**
Magnesium (Mg)	mg/kg	5	-	-	-	38,000	11,433	4,633	16,867	4,867	5,033	3,100
Manganese (Mn)	mg/kg	2	1,000	1,000	2,200	793	883	610	730	960	997	350
Mercury (Hg)	mg/kg	0.01	0.3	2	10	0.08	0.03	0.08	0.09	**2.8**	**1.1**	**10**
Molybdenum (Mo)	mg/kg	0.5	6	10	40	1.9	1.4	2.8	3.7	**19.7**	**13**	**98**
Nickel (Ni)	mg/kg	0.5	50	100	500	**393**	**56**	4	**130**	5	4.0	32
Potassium (K)	mg/kg	20	-	-	-	2,274	4,667	2,900	3,833	3,800	1,400	25,000
Selenium (Se)	mg/kg	0.5	3	3	10	0.9	1.2	2.2	1.8	2.6	1.7	**7.1**
Silver (Ag)	mg/kg	0.5	0.5	20	40	<0.5	<0.5	<0.5	**0.7**	**2.2**	<0.5	**15**
Sodium (Na)	mg/kg	10	-	-	-	210	343	1,200	830	357	217	2800
Tin (Sn)	mg/kg	1	5	50	300	1	1	1	2	1	1	1.7
Titanium (Ti)	mg/kg	2	-	-	-	38	45	624	46	21	15	920
Uranium (U)	mg/kg	2	-	-	-	<2	<2	4	3	4	3	5.3
Vanadium (V)	mg/kg	2	-	-	-	71	42	47	58	20	14	74
Zinc (Zn)	mg/kg	5	120	500	1,500	71	88	53	67	46	40	110

Bold (grey)	Value is higher than criterion A
Bold (yellow)	Value is higher than criterion B
Bold (orange)	Value is higher than criterion C

[1] *Backround levels or criterion A of the Soil Protection and Rehabilitation of Contaminated Sites Policy for the Superior geological province*

[2] *Measured by Strong Acid Total Digestion, and consequently results cannot be compared with criteria from SPRCSP (based on partial digestion).*

Table 20.3 - Characteristics of the decant solution of tailings samples prepared by SGS Minerals Service

Parameter	Unit	Reported detection limit(s)	Groundwater Protection (Seepage in surface water)[1]	Tailings (Liquid Fraction)	
				CS-17	NT7 + CND3-2
Conventional Parameters					
pH	pH	-	[a]	8.12	7.45
Total alcalinity (as $CaCO_3$) (pH 4.5)	mg/l	1	-	180	48
Acidity (as $CaCO_3$)	mg/l	10 / 2	-	10	< 2
Total hardness (as $CaCO_3$)	mg/l	10	-	653	-
Conductivity	mS/cm	0.001	-	3.9	3.3
Carbon Compounds					
Dissolved Inorganic Carbon	mg/l	0.9	-	44	-
Dissolved Organic Carbon	mg/l	0.2	-	9.8	-
Major Ions and Nutrients					
Ammoniacal nitrogen (N-NH_3)	mg/l	0.05	19[b]	**30**	1.9
Nitrites (N-NO^{2-})	mg/l	0.01; 0.3	0.06[c]	0.03	< 0.3
Nitrates (N-NO^{3-})	mg/l	0.01; 0.6	-	0.05	< 0.6
Nitrates (N) and Nitrites (N)	mg/l	0.2	-	0.08	-
Total Phosphorus	mg/l	0.01	-	0.02	-
Chlorides (Cl)	mg/l	0.05	860[d]	30.7	28.0
Fluorides (F)	mg/l	0.1	4[e]	0.6	2.05
Sulphates (SO_4)	mg/l	0.5	500[f]	**1,800**	**1,800**
Sulfides (as S^{2-})	mg/l	0.02	-	2.5	-
Thiosulphates	mg/l	0.1; 2	-	29	< 2
Cyanates (CNO^-)	mg/l	0.5	-	22	170
Free cyanides (CN^-)	mg/l	0.01	0.022	**0.03**	< 0.01
Total cyanides	mg/l	0.003; 0.01	-	0.106	< 0.01
Thiocyanates	mg/l	2	2.1[g]	**35**	< 2
Metals and Metalloids					
Aluminum (Al)	mg/l	0.01	0.75[h]	0.02	0.04
Antimony (Sb)	mg/l	0.001	1.1	0.113	0.0053
Arsenic (As)	mg/l	0.001	0.34[j]	0.088	0.112
Barium (Ba)	mg/l	0.002	0.47[i]	0.06	0.00127
Beryllium (Be)	mg/l	0.0032; 0.00002	0.00212[i]	<0.0032	< 0.00002
Bismuth (Bi)	mg/l	0.0016; 0.00001	-	<0.0016	< 0.00001
Boron (B)	mg/l	0.05	28	0.085	0.163
Cadmium (Cd)	mg/l	0.00032	0.0008[i]	<0.00032	0.000412
Calcium (Ca)	mg/l	5.0	-	210	574
Chromium (Cr)	mg/l	0.008	-	<0.008	0.0032
Cobalt (Co)	mg/l	0.001	0.37	0.083	0.0493
Copper (Cu)	mg/l	0.001	0.0059[i]	**0.0075**	**0.0070**
Iron (Fe)	mg/l	0.096	30	<0.096	0.083
Lead (Pb)	mg/l	0.0008; 0.0005	0.05[i]	<0.0008	0.00051
Magnesium (Mg)	mg/l	0.1	-	32	62.5
Manganese (Mn)	mg/l	0.001	1.9[i]	0.29	0.018
Mercury (Hg)	mg/l	0.00001	0.0016[l]	0.00004	**0.02**
Molybdenum (Mo)	mg/l	0.001	29	0.72	7.52
Nickel (Ni)	mg/l	0.0032	0.22[i]	<0.0032	0.0097
Potassium (K)	mg/l	0.5	-	153	4.1
Selenium (Se)	mg/l	0.0048	0.062[m]	<0.0048	**0.109**
Silver (Ag)	mg/l	0.0016	0.00042[i]	<0.0016	0.00003
Sodium (Na)	mg/l	0.5	-	573	268
Tin (Sn)	mg/l	0.0032	-	<0.0032	0.0135
Titanium (Ti)	mg/l	0.016	-	<0.016	0.0008
Uranium (U)	mg/l	0.001	0.32[i]	0.018	0.000451
Vanadium (V)	mg/l	0.0032	0.11	<0.0032	0.00008
Zinc (Zn)	mg/l	0.007	0.055[i]	0.017	0.005

Bold Value is higher than the groundwater protection (seepage in surface waters) criteria

[1] According to the Soil Protection and Rehabilitation of Contaminated Sites Policy. These criteria are applicable to the contamination of underground waters. They are based on the MDDEFP criteria for water quality (2009, updated in 2012 and 2013). Notes associated to these criteria are presented in Appendix 1.

Table 20.4 – Metal content in leachates produced following TCLP 1311 leaching tests on solids for waste rock, ore, concentrate and tailings

Parameter	Unit	Reported Detection Limit(s)	High Risk Residues (Directive 019)	Groundwater Protection (Seepage in surface water)[1]	Leachate						
					Waste Rock				Ore	Tailings	
					Barren Sediment	Barren Volcanic	Barren Syenite	Composite		Float	POX
					CS-1-3	CS-4-6	CS-7-9	CS-1-12	CS-13	CS-17	NT7 + CND3-2
Conventional Parameters of the Leachate											
Weight of the sample	g	-	-	-	20	20	20	20	25	20	100
pH of the pre-test	-	-	-	-	4.80	5.98	5.80	6.29	-	6.43	4.9
Final pH of the leachate	-	-	-	-	5.47	4.97	4.97	4.82	-	4.83	5.14
Extraction fluid volume 1	ml	-	-	-	400	-	400	-	-	-	2,000
Extraction fluid volume 2	ml	-	-	-	400	400	400	400	-	400	-
Extraction fluid volume	ml	-	-	-	-	-	-	-	-	-	-
Addition of extraction fluid	-	-	-	-	-	-	-	-	2,000	-	-
Leaching stopped	-	-	-	-	-	-	-	-	2,000	-	-
pH of the extraction fluid	-	-	-	-	-	-	-	-	-	-	-
pH after 7 days of mixture	-	-	-	-	-	-	-	-	8.4	-	-
Metals and Metalloids											
Aluminum (Al)	mg/l	0.03; 0.3	-	0.75[h]	**1.23**	**12.21**	**7.30**	**11.37**	**2.87**	**2.13**	0.17
Antimony (Sb)	mg/l	0.006; 0.06	-	1.1	0.014	<0.006	<0.006	<0.006	0.005	<0.006	0.0031
Arsenic (As)	mg/l	0.002; 0.02	5.0	0.34[j]	0.174	0.004	0.008	0.046	0.029	0.008	0.0795
Barium (Ba)	mg/l	0.005; 0.05	100	0.47[i]	0.33	**0.92**	**5.20**	**4.03**	**3.10**	**1.23**	0.134
Beryllium (Be)	mg/l	0.002; 0.02	-	0.00212[i]	<0.002	**0.005**	**0.008**	**0.006**	**0.004**	**0.002**	0.0004
Bismuth (Bi)	mg/l	0.05; 0.00001	-	-	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	< 0.00001
Boron (B)	mg/l	0.05; 0.5	500	28	0.055	0.075	0.037	0.055	0.057	<0.05	0.184
Cadmium (Cd)	mg/l	0.001; 0.01	0.5	0.0008[i]	<0.001	0.0007	**0.0020**	**0.0013**	**0.0010**	<0.001	**0.0026**
Calcium (Ca)	mg/l	0.5; 5; 50	-	-	473	1,013	670	803	1300	1167	1,020
Chromium (Cr)	mg/l	0.007; 0.07	5.0	-	0.024	0.010	0.023	0.134	<0.007	0.010	0.0029
Cobalt (Co)	mg/l	0.01; 0.1	-	0.37	0.10	0.16	0.29	0.17	0.34	0.01	0.28
Copper (Cu)	mg/l	0.003; 0.03	-	0.0059[i]	**0.007**	**0.144**	**0.098**	**0.035**	<0.003	**0.297**	**0.0285**
Iron (Fe)	mg/l	0.1; 1	-	30	12.4	**125.5**	**86.3**	**120.7**	**74.0**	**60.7**	0.2
Lead (Pb)	mg/l	0.001; 0.01	5.0	0.05[i]	0.017	0.001	0.006	0.030	0.039	0.008	0.0006
Magnesium (Mg)	mg/l	0.2; 2	-	-	218	57	82	165	68	51	80.8
Manganese (Mn)	mg/l	0.003; 0.03	-	1.9[i]	**11.1**	**35.7**	**22.2**	**24.0**	**24.0**	**20.3**	**2.91**
Mercury (Hg)	mg/l	0.0005; 0.00001	0.1	0.0016[l]	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	< 0.00001
Molybdenum (Mo)	mg/l	0.01; 0.1	-	29	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	0.00686
Nickel (Ni)	mg/l	0.006; 0.06	-	0.22[i]	**1.48**	0.04	0.03	**0.32**	0.02	0.01	**0.256**
Potassium (K)	mg/l	0.2; 2	-	-	37.4	61.7	45.3	50.3	53.7	8.3	2.45
Selenium (Se)	mg/l	0.001; 0.01	1.0	0.062[m]	0.001	0.0015	0.008	0.006	0.003	0.003	0.008
Silver (Ag)	mg/l	0.0003; 0.003	-	0.00042[i]	**0.0006**	<0.0003	0.0002	**0.0005**	<0.0003	<0.0003	0.00002
Tin (Sn)	mg/l	0.05; 0.5	-	-	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	0.00008
Titanium (Ti)	mg/l	0.05; 0.5	-	-	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	0.0023
Uranium (U)	mg/l	0.0006; 0.006	2.0	0.32[i]	0.0004	0.0004	0.0167	0.0075	0.0058	0.0033	0.0053
Vanadium (V)	mg/l	0.01; 0.1	-	0.11	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	0.00011
Zinc (Zn)	mg/l	0.05; 0.005	-	0.055[i]	0.023	**0.204**	**0.430**	**0.323**	**0.220**	**0.157**	**0.213**

Bold (green)	Value is higher than the groundwater protection criteria for seepage in surface water but metal content for this parameter is lower than criterion A (see table 20.2).
Bold (yellow)	Value is higher than the groundwater protection criteria for seepage in surface water and no criterion A exists for this parameter (see table 20.2).
Bold (orange)	Value is higher than the groundwater protection criteria for seepage in surface water and metal content for this parameter is higher than criterion A (see table 20.2).

[1] *According to the Soil Protection and Rehabilitation of Contaminated Sites Policy. These criteria are applicable to the contamination of underground waters. They are based on the MDDEFP acute toxicity criteria for the protection of aquatic life (2009, updated in 2012 and 2013. Notes associated to these criteria are presented in Appendix 1).*

Table 20.5 – Metal content in leachates produced following SPLP 1312 leaching tests on solids for waste rock, ore and tailings

Parameter	Unit	Reported Detection Limit(s)	High Risk Residues (Directive 019)	Groundwater Protection (Seepage in surface water)[1]	Leachate						
					Waste Rock				Ore	Tailings	
					Barren Sediment	Barren Volcanic	Barren Syenite	Composite		Float	POX
					CS-1-3	CS-4-6	CS-7-9	CS-1-12	CS-13	CS-17	NT7 + CND3-2
Conventional Parameters of the Leachate											
Weight of the sample	g	-	-	-	20	20	20	20	20	20	-
pH of the pre-test	-	-	-	-	-	-	-	-	6.63	-	-
Final pH of the leachate	-	-	-	-	-	-	-	-	5.05	-	-
Extraction fluid volume 1	ml	-	-	-	-	-	-	-	-	-	-
Extraction fluid volume 2	ml	-	-	-	-	-	-	-	400	-	-
Extraction fluid volume	ml	-	-	-	100	100	100	100	-	100	-
Addition of extraction fluid	-	-	-	-	-	-	-	-	-	-	-
Leaching stopped	-	-	-	-	-	-	-	-	-	-	-
pH of the extraction fluid	-	-	-	-	8.8	8.8	9.3	9.1	-	8.6	-
pH after 7 days of mixture	-	-	-	-	-	-	-	-	-	-	-
Metals and Metalloids											
Aluminum (Al)	mg/l	0.03; 0.3	-	0.75[h]	**0.98**	**1.11**	**1.75**	**0.94**	0.69	0.38	-
Antimony (Sb)	mg/l	0.006; 0.06	-	1.1	0.114	<0.006	<0.006	0.028	0.033	<0.006	-
Arsenic (As)	mg/l	0.002; 0.02	5.0	0.34[j]	0.227	0.005	0.044	0.061	0.029	0.091	-
Barium (Ba)	mg/l	0.005; 0.05	100	0.47[i]	0.004	0.006	0.197	0.149	0.140	0.081	-
Beryllium (Be)	mg/l	0.002; 0.02	-	0.00212[i]	<0.002	<0.002	<0.002	<0.002	<0.002	<0.002	-
Bismuth (Bi)	mg/l	0.05; 0.00001	-	-	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	-
Boron (B)	mg/l	0.05; 0.5	500	28	0.03	0.03	<0.05	<0.05	<0.05	<0.05	-
Cadmium (Cd)	mg/l	0.001; 0.01	0.5	0.0008[i]	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	-
Calcium (Ca)	mg/l	0.5; 5; 50	-	-	8	11	6	9	12	13	-
Chromium (Cr)	mg/l	0.007; 0.07	5.0	-	0.006	<0.007	<0.007	<0.007	<0.007	<0.007	-
Cobalt (Co)	mg/l	0.01; 0.1	-	0.37	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	-
Copper (Cu)	mg/l	0.003; 0.03	-	0.0059[i]	<0.003	<0.003	0.002	<0.003	<0.003	<0.003	-
Iron (Fe)	mg/l	0.1; 1	-	30	0.1	0.2	0.5	0.1	<0.1	0.4	-
Lead (Pb)	mg/l	0.001; 0.01	5.0	0.05[i]	<0.001	<0.001	0.001	<0.001	<0.001	<0.001	-
Magnesium (Mg)	mg/l	0.2; 2	-	-	9.4	2.4	2.2	4.3	5.2	1.7	-
Manganese (Mn)	mg/l	0.003; 0.03	-	1.9[i]	<0.003	0.010	0.015	0.003	0.011	0.011	-
Mercury (Hg)	mg/l	0.0005; 0.00001	0.1	0.0016[l]	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	-
Molybdenum (Mo)	mg/l	0.01; 0.1	-	29	<0.01	<0.01	0.03	0.03	0.02	<0.01	-
Nickel (Ni)	mg/l	0.006; 0.06	-	0.22[i]	<0.006	<0.006	<0.006	<0.006	<0.006	<0.006	-
Potassium (K)	mg/l	0.2; 2	-	-	17.9	30.0	18.3	23.7	30.3	3.8	-
Selenium (Se)	mg/l	0.001; 0.01	1.0	0.062[m]	0.001	0.001	0.001	0.001	0.003	0.001	-
Silver (Ag)	mg/l	0.0003; 0.003	-	0.00042[i]	<0.0003	0.0004	<0.0003	<0.0003	<0.0003	<0.0003	-
Sodium (Na)	mg/l	0.2; 2	-	-	3.1	6.2	20.0	12.3	4.0	4.7	-
Tin (Sn)	mg/l	0.05; 0.5	-	-	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	-
Titanium (Ti)	mg/l	0.05; 0.5	-	-	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	-
Uranium (U)	mg/l	0.0006; 0.006	2.0	0.32[i]	<0.006	<0.0006	0.004	0.0007	0.0016	<0.0006	-
Vanadium (V)	mg/l	0.01; 0.1	-	0.11	<0.01	<0.01	0.02	<0.01	<0.01	<0.01	-
Zinc (Zn)	mg/l	0.05; 0.005	-	0.055[i]	<0.005	<0.005	0.004	<0.005	<0.005	<0.005	-

Bold (green) Value is higher than the groundwater protection criteria for seepage in surface water but metal content for this parameter is lower than criterion A (see table 20.2).

Bold (yellow) Value is higher than the groundwater protection criteria for seepage in surface water and no criterion A exists for this parameter (see table 20.2).

Bold (orange) Value is higher than the groundwater protection criteria for seepage in surface water and metal content for this parameter is higher than criterion A (see table 20.2).

[1] According to the Soil Protection and Rehabilitation of Contaminated Sites Policy. These criteria are applicable to the contamination of underground waters. They are based on the MDDEFP acute toxicity criteria for the protection of aquatic life (2009, updated in 2012 and 2013. Notes associated to these criteria are presented in Appendix 1).

Table 20.6 – Metal content in leachates produced following CTEU-9 leaching tests or Shake Flask Extraction tests on solids for waste rock, ore and tailings

Parameter	Unit	Reported Detection Limit(s)	High Risk Residues (Directive 019)	Groundwater Protection (Seepage in surface water)[1]	Leachate						
					Waste Rock				Ore	Tailings	
					Barren Sediment	Barren Volcanic	Barren Syenite	Composite		Float	POX
					CS-1-3	CS-4-6	CS-7-9	CS-1-12	CS-13	CS-17	NT7 + CND3-2
Conventional Parameters of the Leachate											
Weight of the sample	g	-	-	-	25	25	25	25	20	25	250
pH of the pre-test	-	-	-	-	-	-	-	-	-	-	8.9
Final pH of the leachate	-	-	-	-	-	-	-	-	-	-	8.92
Extraction fluid volume 1	ml	-	-	-	-	-	-	-	-	-	-
Extraction fluid volume 2	ml	-	-	-	-	-	-	-	-	-	-
Extraction fluid volume	ml	-	-	-	100	100	100	100	-	100	-
Addition of extraction fluid	-	-	-	-	2,000	2,000	2,000	2,000	-	2,000	-
Leaching stopped	-	-	-	-	2,000	2,000	2,000	2,000	-	2,000	-
pH of the extraction fluid	-	-	-	-	-	-	-	-	8.9	-	-
pH after 7 days of mixture	-	-	-	-	7.7	8.4	8.9	8.6	-	8.0	-
Metals and Metalloids											
Aluminum (Al)	mg/l	0.03; 0.3	-	0.75^{h}	0.27	0.26	**2.52**	0.35	0.08	0.04	0.0025
Antimony (Sb)	mg/l	0.006; 0.06	-	1.1	0.381	0.011	0.025	0.084	0.110	0.034	0.0041
Arsenic (As)	mg/l	0.002; 0.02	5.0	0.34^{j}	**0.472**	0.007	0.160	0.070	0.013	0.036	0.144
Barium (Ba)	mg/l	0.005; 0.05	100	0.47^{i}	0.0115	0.020	0.183	0.158	0.183	0.066	0.0859
Beryllium (Be)	mg/l	0.002; 0.02	-	0.00212^{i}	<0.002	<0.002	<0.002	<0.002	<0.002	<0.002	< 0.00002
Bismuth (Bi)	mg/l	0.05; 0.00001	-	-	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	< 0.00001
Boron (B)	mg/l	0.05; 0.5	500	28	0.09	0.13	0.11	0.12	0.16	<0.05	0.216
Cadmium (Cd)	mg/l	0.001; 0.01	0.5	0.0008^{i}	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	**0.00226**
Calcium (Ca)	mg/l	0.5; 5; 50	-	-	10	15	5	10	40	74	564
Chromium (Cr)	mg/l	0.007; 0.07	5.0	-	<0.007	<0.007	<0.007	<0.007	<0.007	<0.007	0.0019
Cobalt (Co)	mg/l	0.01; 0.1	-	0.37	<0.01	<0.01	<0.01	<0.01	<0.01	0.01	0.0135
Copper (Cu)	mg/l	0.003; 0.03	-	0.0059^{i}	<0.003	0.0023	0.0055	<0.003	<0.003	0.0043	0.003
Iron (Fe)	mg/l	0.1; 1	-	30	<0.1	<0.1	0.8	<0.1	<0.1	0.3	0.004
Lead (Pb)	mg/l	0.001; 0.01	5.0	0.05^{i}	<0.001	<0.001	0.001	<0.001	0.001	<0.001	< 0.00002
Magnesium (Mg)	mg/l	0.2; 2	-	-	35.3	8.8	3.0	11.8	33.7	19.0	53.4
Manganese (Mn)	mg/l	0.003; 0.03	-	1.9^{i}	0.003	0.019	0.015	0.009	0.079	0.19	0.00319
Mercury (Hg)	mg/l	0.0005; 0.00001	0.1	0.0016^{l}	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	< 0.00001
Molybdenum (Mo)	mg/l	0.01; 0.1	-	29	0.03	0.02	0.18	0.19	0.17	0.15	4.22
Nickel (Ni)	mg/l	0.006; 0.06	-	0.22^{i}	0.004	<0.006	<0.006	<0.006	<0.006	<0.006	0.0097
Potassium (K)	mg/l	0.2; 2	-	-	66	107	54	84	140	32	1.61
Selenium (Se)	mg/l	0.001; 0.01	1.0	0.062^{m}	0.001	<0.001	0.001	0.002	0.014	0.003	0.057
Silver (Ag)	mg/l	0.0003; 0.003	-	0.00042^{i}	<0.0003	<0.0003	<0.0003	<0.0003	0.0003	<0.0003	< 0.00001
Tin (Sn)	mg/l	0.05; 0.5	-	-	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	0.00002
Titanium (Ti)	mg/l	0.05; 0.5	-	-	<0.05	<0.05	0.04	<0.05	<0.05	<0.05	0.0007
Uranium (U)	mg/l	0.0006; 0.006	2.0	0.32^{i}	<0.0006	0.0004	0.028	0.004	0.006	0.0060	0.000173
Vanadium (V)	mg/l	0.01; 0.1	-	0.11	<0.01	<0.01	0.08	<0.01	<0.01	<0.01	0.00029
Zinc (Zn)	mg/l	0.05; 0.005	-	0.055^{i}	<0.005	<0.005	0.005	<0.005	<0.005	<0.005	0.002

Bold (green) Value is higher than the groundwater protection criteria for seepage in surface water but metal content for this parameter is lower than criterion A (see table 20.2).

Bold (yellow) Value is higher than the groundwater protection criteria for seepage in surface water and no criterion A exists for this parameter (see table 20.2).

Bold (orange) Value is higher than the groundwater protection criteria for seepage in surface water and metal content for this parameter is higher than criterion A (see table 20.2).

[1] *According to the Soil Protection and Rehabilitation of Contaminated Sites Policy. These criteria are applicable to the contamination of underground waters. They are based on the MDDEFP acute toxicity criteria for the protection of aquatic life (2009, updated in 2012 and 2013. Notes associated to these criteria are presented in Appendix 1).*

[2] *Measured using the Shake Flask Extraction Method.*

Table 20.7 – Acid-generating potential for waste rock, ore, ore concentrate and combined tailings

Maxxam Sample No	Sample ID	Quebec's Directive 019 Criteria for not potentially acid-generating mine residues			Paste pH	Total S	HCl Extractable Sulphur	HNO_3 Extractable Sulphur	Non Extractable Sulphur (by diff.)	Acid Generation Potential (AP)	Mod. ABA Neutralization Potential (NP)	Fizz Rating	Net Neutralization Potential (NNP)	Neutralization Potential Ratio
	Units	NP/AP ratio	NNP	Total S	pH Units	wt%	wt%	wt%	wt%	Kg $CaCO_3$/t	Kg $CaCO_3$/t	-	Kg $CaCO_3$/t	NP/AP
HG3257-59	CS-1-3 (barren sediment)				8.91	0.14	0.03	0.09	0.03	2.8	116.3	Moderate	113.4	337.3
HG3260-62	CS-4-6- (barren volcanic)				8.88	0.05	0.02	0.02	<0.02	0.5	61.9	Moderate	61.5	72.7
HG3263-65	CS-7-9 (barren syenite)				9.92	0.02	0.01	<0.01	0.02	<0.3	54.0	Moderate	54.0	-
HG3266-68	CS-10-12 (composite)	> 3	> 20 Kg $CaCO_3$/t	< 0.3%	9.37	0.05	0.01	0.01	0.03	0.4	67.5	Moderate	67.2	186.7
HG3269-71	CS-13 (ore, pilot plant composite)				9.16	1.14	0.01	0.98	0.15	30.7	108.3	Moderate	77.6	3.5
HG3281-83	CS-17 (flotation tailings				8.61	0.17	0.02	0.09	0.06	2.9	115.3	Moderate	112.4	39.8
-	NT7 + CND 3-2 (POX tailings)				8.39	**9.74**	7.54	2.20	-	68.8	19.0	-	**-49.6**	**0.28**
Detection Limits		-			*N/A*	*0.02*	*0.01*	*0.01*	*0.02*	*0.3*	*0.1*	-	*0.1*	*N/A*
Maxxam SOP #					*BBY0-00003*	*Acme*	*BBY0-00010*	*BBY0-00010*	*Calculation*	*Calculation*	*BBY0SOP-00020*	*BBY0SOP-00020*	*Calculation*	*Calculation*

Modified ABA according to: Lawrence, R.W. 1991. Acid Rock Drainage Prediction Manual

References:

Acid Generation Potential = HNO_3 Extractable Sulphide Sulphur*31.25

Fizz Rating - Reference method used is based on NP method.

Non Extractable Sulphur = (Total Sulphur)-(HCl Extractable Sulphate Sulphur)-(HNO_3 Extractable Sulphide Sulphur)

Net Neutralization Potential = (Modified ABA Neutralization Potential)-(Acid Generation Potential (HNO3 Extr))

Mod. ABA Neutralization Potential - MEND Acid Rock Drainage Prediction Manual, MEND Project 1.16.1b (pages 6.2-11 to 17), March 1991.

Neutralization Potential Ratio = (Neutralization Potential)/(Acid Generation Potential)

Paste pH - Field and Laboratory Methods Applicable to Overburdens and Minesoils, (EPA 600 / 2-78-054, March 1978).

HCl Extractable Sulphur and HNO_3 Extractable Sulphur is based on a modified version of ASTM Method D 2492-02

Total sulphur, total carbon & carbonate carbon (CO_2; HCl direct method) by Leco done at Acme Labs.

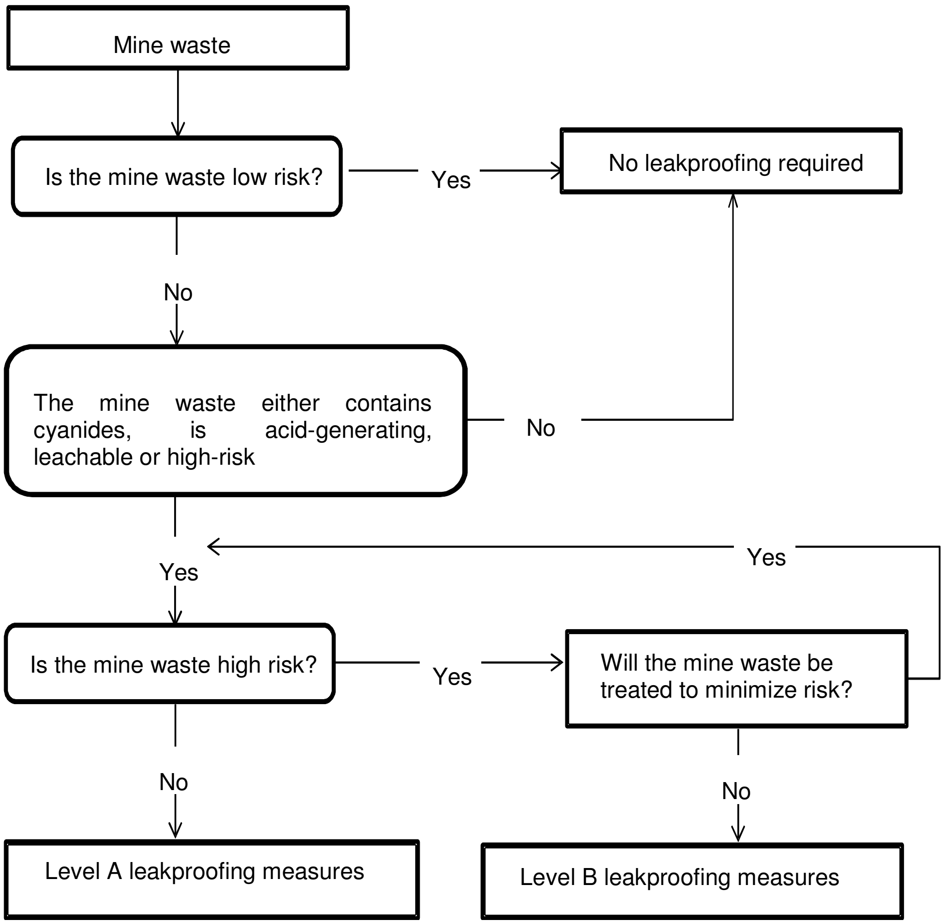

Figure 20.2 – Criteria for determining leakproofing measures to be applied to tailings accumulation areas

Source: *Directive 019 for the mining industry - March 2012*

20.1.2.4 Waste Rock, Ore and Tailings Management

Based on the above results and considering not only the TCLP leaching test results but also the SPLP, CTEU-9 and SFE results, the waste rock, ore and flotation tailings would be classified as “low-risk” at the provincial level, meaning that no specific leakproofing measures have to be implemented for management purposes. Again, kinetic tests will have to be performed to confirm such classification.

On the other hand, POX tailings, as stated above, are classified as potentially acid-generating and cyanide-containing since their production involves cyanidation. Consequently, level-A leakproofing measures would be required for POX tailings following Directive 019:

- Maximal daily infiltration (percolation) rate of 3.3 L/m²;
- Hydrogeological study demonstrating full compliance with the groundwater protection objectives listed in Appendix III of Directive 019;
- Monitoring of groundwater quality and elaboration of a water balance for each stockpile.

Considering the type of soils present in the area targeted for the proposed tailings storage facility (TSF), a geomembrane will have to be used to cover all of the area where POX tailings will be disposed.

Tailings management will consequently be done in such a way that flotation (desulphurized) tailings will be managed separately from POX tailings, therefore using two distinct tailings storage facilities. Such separate management enables cost reduction not only at construction phase – since no geomembrane is required under the Flotation TSF – but also at closure since legal requirements applicable to potentially acid-generating POX tailings are also stricter.

20.1.2.5 Alternate Scenario for Tailings Storage

As explained in Section 17 of this Report, a different production scenario was also considered as part of the present study, i.e. the production of an ore concentrate that would be shipped directly out of the Property without any additional on-site POX processing. In that scenario, the production of such a concentrate would require the flotation tailings to be classified as “cyanide-containing” since their production would involve a carbon-in-leach (CIL) circuit during which carbon adsorbs the gold from solution as cyanidation of the ore proceeds. Such a classification would mean that flotation tailings would require Level-A leakproofing measures in compliance with Directive 019 and, considering the type of soils in the area targeted for the Flotation TSF, the proposed infrastructure would have to be entirely lined with a geomembrane during construction phase. Closure requirements would however not be impacted by such classification since flotation tailings would still not be considered as potentially acid-generating or leachable.

20.2 Jurisdictions and Applicable Laws and Regulations

The legal framework for the construction and operation of the projected facilities is a combination of provincial, national, and municipal policies, regulations and guidelines. The design and the environmental management of the Project facilities and activities must be done in accordance with this legal framework.

20.2.1 Recent Modifications to the Mining Act

The Québec *Mining Act* was substantially amended and modernized by Bill 70, which the Québec National Assembly adopted on December 9, 2013. This fourth attempt to update Québec’s mining legislation follows on the heels of the defeat of Bill 43 and that of Bill 79 and Bill 14 in previous legislative sessions. Within the specific context of the Duparquet Project, the following amendments are relevant:

- Provisions specific to aboriginal communities and referring to an aboriginal community consultation policy specific to the mining sector (obligation to consult aboriginal communities and requirement that the Minister consult aboriginal communities separately if the circumstances so warrant).

- On each anniversary date of a mining lease or mining concession, the lessee or grantee will have to send the Minister a report showing the quantity of ore extracted during the previous year, its value, the duties paid under the *Mining Tax Act* during that period and the overall contributions paid.
- Mining leases to be issued will require the prior approval of a rehabilitation and restoration plan and the issue of a CA under the *Environment Quality Act* (EQA), unless the time needed to obtain a certificate is unreasonable.

On July 23, 2013, the Government of Québec passed amendments to the *Regulation respecting Mineral Substances other than Petroleum, Natural Gas and Brine* in order to set new rules concerning the financial guarantees required for the restoration of mining sites. Among other things, those result in an increase of the financial guarantee from 70% to 100% of the projected costs for the work required under the rehabilitation and restoration plan. The guarantee must cover not only restoration costs associated with the accumulation areas, but all costs for the entire mine site and associated infrastructures. It must be paid in three annual instalments. The first instalment corresponds to 50% of the total amount of the guarantee and must be paid within 90 days following the receipt of the approval of the plan. The second and third instalments each represent 25% of the guarantee and must be paid in full by the first and second anniversary date.
Lastly, Bill 70 amends the *Regulation respecting Environmental Impact Assessment and Review* in order to require an environmental impact assessment for all metal ore processing plant construction projects, and all metal mine openings and operation projects where the processing or production capacity of the plant or the mine is 2,000 metric tons or more per day.

20.2.2 Québec Procedure Relating to the Environmental Assessment of the Project

20.2.2.1 Overview

“Section 31.1 of the Environment Quality Act (EQA) states that “No person may undertake any construction, work, activity or operation, or carry out work according to a plan or program, in the cases provided for by regulation of the Government without following the environmental impact assessment and review procedure and obtaining an authorization certificate from the Government.”

Moreover, section 2 of the *Regulation respecting Environmental Impact Assessment and Review* provides the list of projects subject to the environmental impact assessment and review procedure, namely:

(n.8) the construction of an ore processing plant for metalliferous ore or asbestos ore, where the processing capacity of the plant is 2,000 metric tons or more per day, except in the case of rare earth deposits;

(p) the opening and operation of a metals mine or an asbestos mine that has a production capacity of 2,000 metric tons or more per day, except in the case of rare earth deposits”

Thus, the Duparquet Project is subject to the provincial environmental impact assessment and review procedure.

Section 31.2 of the EQA states that: “Every person wishing to undertake the realization of any of the projects contemplated in section 31.1 must file a written notice with the Minister describing the general nature of his project; the Minister, in turn, shall indicate to the proponent of the project the nature, the scope and the extent of the environmental impact assessment statement that he must prepare.”

Clifton Star will table a Project Notice as soon as possible with the MDDELCC. A month later, following the review of the project notice, a Directive (“guidelines”) defining the required scope and content of the environmental impact assessment of the Project will be sent by the MDDELCC to Clifton Star.

According to the *Regulation respecting Environmental Impact Assessment and Review*, and following the filing of the Project Notice, the Minister must submit the application record to the Government for approval within a maximum of 15 months. However, one should note that the 15-month period does not include the time taken by the ‘’proponent’’ to draft the EIA report or to answer any request for additional information, nor the time taken by the Government (Cabinet) to make its decision. Figure 20.3 shows the steps involved in this procedure.

20.2.2.2 General Contents of an Environmental Impact Assessment Statement

Section 3 of the *Regulation respecting Environmental Impact Assessment and Review* defines the contents of an environmental impact assessment statement:

a) A description of the project mentioning, in particular, the desired objectives, the site [...], the project timetable, any subsequent operation and maintenance activities, the amounts and characteristics of types of borrowed materials required, power sources, methods of management of waste or residue other than road construction residue, transportation activities inherent in the construction and subsequent operation of the project, any connection with land use planning and development plans, urban zoning plans or agricultural zoning and reserved areas within the meaning of the act to preserve agricultural land [...];

b) A qualitative and quantitative inventory of the aspects of the environment which could be affected by the project, such as fauna, flora, human communities, the cultural, archaeological and historical heritage of the area, agricultural resources and the use made of resources of the area;

c) A list and evaluation of positive, negative and residual impacts of the project on the environment, including indirect, cumulative, latent and irreversible effects [...];

d) A description of the different options to the project, in particular regarding its location, the means and methods of carrying out and developing the project, and all other variables in the project as well as reasons justifying the option chosen;

e) A list and description of measures to be taken to prevent, reduce or attenuate the deterioration of the environment, including [...] In particular, any equipment used or installed to reduce the emission, deposit, issuance or discharge of contaminants into the environment, any control of operations and monitoring, emergency measures in case of accident, and reclamation of the area affected.”

20.2.2.3 Summary of the Environmental Impact Assessment Statement

Section 4 of the *Regulation respecting Environmental Impact Assessment and Review* indicates that an environmental impact assessment statement prepared pursuant to section 31.1 of the EQA must be accompanied by a non-technical summary of the main elements and conclusions of the studies, documents or research. The summary is published separately.

20.2.2.4 Evaluation of the Environmental Impact Assessment (EIA)

A well-defined project is essential to producing an EIA report that not only accounts for the project to be carried out, but that would also be considered acceptable by the authorities early on in the procedure. After reviewing the EIA report, the MDDELCC will determine its admissibility. This assessment involves consultations with several government departments and agencies. The proponent should generally expect to receive questions and comments to be addressed before the EIA report can be determined to be admissible. Following the filling of the response to this first set of questions, a second set of questions may be issued. To avoid delays associated with this procedure, it is essential to produce an environmental impact assessment that covers as specifically as possible every aspect raised in the Directive issued by the MDDELCC.

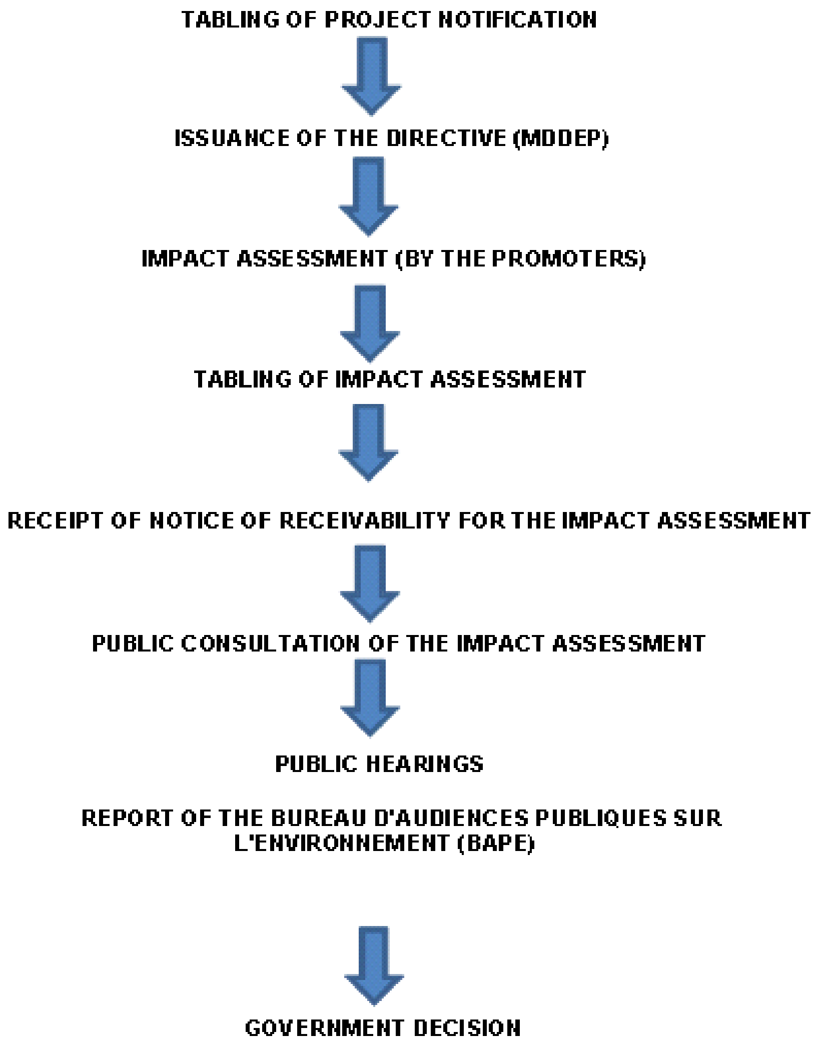

Figure 20.3 – Steps of the environmental impact assessment procedure (source: MDDELCC website)

20.2.2.5 Public Consultations

Section 31.3 of the EQA states that: “After receiving the environmental impact assessment statement, the Minister shall make it public and indicate to the proponent of the project to initiate the stage of public information and consultation provided for by regulation of the Government.”

Once the impact study is found to be admissible, the Minister will then direct the *Bureau d'audiences publiques sur l'environnement* (BAPE) to prepare the project for public consultation. This consultation process lasts 45 days (section 11 of the *Regulation respecting Environmental Impact Assessment and Review*).

20.2.2.6 Public Hearings

Section 31.3 of the EQA also states that: “Any person, group or a municipality may, within the time prescribed by regulation of the Government, apply to the Minister for the holding of a public hearing in connection with such a project. Unless he considers such application to be frivolous, the Minister shall direct the [BAPE] to hold a public hearing and report its findings and its analysis thereof to him.”

Public hearings are governed by the Rules of Procedure relating to the Conduct of Public Hearings (Q 2, r. 45).

Following the public hearings, the BAPE commission files its report to the MDDELCC. The commission is required to complete its mandate and file its report within 4 months. The Minister then has 60 days to publicly release the BAPE report.

20.2.2.7 Government Decision

On the basis of the BAPE report and of the MDDELCC’s evaluation of the EIA (the Environmental Analysis Report; in French: “*Rapport d’analyse environnementale*”), the Minister analyses the project and makes a recommendation to the Government. As specified in section 31.5 of the EQA, the Government renders its decision by Decree: it authorizes the project, with or without changes and conditions, or rejects it. The maximum period between the publication of the BAPE report and the Government's decision is not specified in the EQA nor its regulations.

20.2.3 Federal Procedure

The Canadian Environmental Assessment Act (CEAA 2012) was introduced on July 6, 2012. Consequently, projects are now examined according to the requirements of this new law. Thus, under the CEAA 2012, an environmental assessment focuses on potential adverse environmental effects that are within federal jurisdiction, including:

- Fish and fish habitat;
- Other aquatic species;
- Migratory birds;
- Federal lands;
- Impacts that will or could potentially cross provincial or international boundaries;
- Impacts on Aboriginal peoples, such as land use and traditional resources;
- Impacts that are directly linked or necessarily incidental to any federal decisions about a project.

An environmental assessment will consider a comprehensive set of factors that include any cumulative effect, mitigation measure and comments received from the public.

Regulations Designating Physical Activities determines the specific activities which constitutes designated projects that may require an environmental assessment by the Canadian Environmental Assessment Agency (“the CEA Agency”, or “the Agency”) or by the Canadian Nuclear Safety Commission or the National Energy Board. It should be noted that this Regulation was officially amended on October 24, 2013 (*Regulation amending the Regulations Designating Physical Activities*). The schedule specifies the designated projects that may require an environmental assessment under the responsibility of the CEA Agency (sections 1 to 30), the Canadian Nuclear Safety Commission (sections 31 to 38) or the National Energy Board (sections 39 to 45). With regards to the Duparquet Project, it has been determined that the following designated activities are to be considered:

"16. The construction, operation, decommissioning and abandonment of a new […] (c) rare earth element mine or gold mine, other than a placer mine, with an ore production capacity of 600 t/day or more."

According to CEAA (2012), proponents of designated projects are required to submit a description of the designated project to the Agency to inform on whether or not an environmental assessment of the designated project is required. The project description must include the prescribed information as set out in the Prescribed Information for the Description of a Designated Project Regulations, including information about the possible adverse environmental effects which will be generated by the project. Clifton Star will table a Project Description as soon as possible with the CEA Agency.

After approving the Project Description, the Agency posts a notice on the Canadian Environmental Assessment Registry Internet site ("the registry") to inform whether or not an environmental assessment must be conducted. A summary of the Project Description is also displayed, along with a notice that the public has 20 days to submit comments on the project.

Within 45 days after the posting of the Project Description in the registry, the Agency must decide whether a federal environmental assessment is required or not. The Agency must consider the following factors while making a decision:

- The description of the designated project provided by the proponent;
- The possibility that the designated project may cause adverse environmental effects;
- Any comments received from the public during the 20 days after posting the project description summary on the registry;
- The results of any relevant regional studies.

Considering the above information and recent experience with the CEAA 2012, the Duparquet Project is a designated project and it will consequently have to go through the Canadian environmental assessment process.

After having determined that an environmental assessment is required, the Agency shall post a Notice of Commencement of the environmental assessment on the registry. The Agency shall then prepare a preliminary version of the guidelines relative to the environmental impact assessment. These guidelines are posted on the registry allowing the public to comment on the proposed studies and methods as well as on the information that will be required for the environmental impact assessment.

The Agency takes into account the general public's comments, including the observations made by Aboriginal groups and federal ministries before providing the final version of the environmental impact assessment guidelines to the proponent.

The proponent then has to submit to the Agency an environmental impact assessment identifying the environmental effects of the project and propose measures to mitigate these effects, while accounting for the Agency's guidelines

Following the submission of the environmental impact assessment to the Agency, the latter will ensure of its relevancy and accuracy. The Agency may require that the proponent provides further clarifications or additional information to better understand the potential environmental effects and the proposed mitigation and preventive measures.

The Agency will then prepare a preliminary version of the environmental assessment report, which will include the Agency's conclusions on the potential environmental effects of the project, the proposed mitigation measures, the significance of the residual adverse environmental effects of the project and the requirements of the monitoring program. The Agency will then invite the public to comment on this preliminary report before finalizing it and submitting it to the Minister of Environment.

If the Minister decides that the project is not likely to cause any significant adverse environmental effects or that the latter are considered justifiable by the Governor in Council, then the conditions relative to the mitigation measures and the monitoring program to be respected by the proponent as part of its project, are set out in the assessment decision statement issued by the Minister

An environmental assessment to be conducted by the Agency must be completed within 365 days. This timeframe begins when the environmental assessment's Notice of Commencement is released on the website of the registry and ends when the Minister of Environment issues its decision as to whether or not the designated project is likely to cause significant adverse environmental effects.

The Minister may extend the deadline for a maximum of 3 months in order to allow for a partnership with another authority or because of particular circumstances to the project. The Federal Cabinet has the authority to extend this deadline for more than three months

20.2.4 Canada-Québec Agreement on Environmental Assessment Cooperation (2010)

The federal and provincial governments reached an agreement in 2004 ("the Agreement") aiming for a simultaneous analysis of impact assessments submitted to both governmental levels in order to minimize the time required for obtaining the environmental authorizations. The Agreement which expired in 2009 was renewed in 2010.

This Agreement applies to projects located in Québec and conducted on provincial and federal lands that are subject to an assessment under the CEAA and the Québec EQA. However, it does not apply to projects governed by the James Bay and Northern Québec Agreement.

The Agreement has as objective to ensure that a single environmental assessment is conducted for projects that must comply with the federal and provincial requirements for environmental assessments. The Parties agreed to respect the environmental assessment timetables stipulated in the provincial and federal legislations. Each jurisdiction has a single point of contact and they are responsible for ensuring that the requirements of each party participating in the cooperative environmental assessment are respected. The Agreement does not delegate any federal powers to the province, or vice versa. Each government maintains its authority in the areas under its jurisdiction and remains responsible for the environmental assessment decisions required by its legislation.

As specified on the CEA Agency's website, the Agreement:

- constitutes an administrative framework within which the parties will collaborate to exercise their respective powers and duties with respect to environmental assessment as set out in the CEAA and in Division IV.1 of the Québec EQA;
- must be interpreted in accordance with the CEAA and the Québec EQA, as well as any other legal requirements, including, but not limited to, the legislative requirements;
- does not establish new powers or duties nor does it alter the powers and duties established by the CEAA and the Québec EQA, and is not legally binding on the parties;
- does not affect in any way the independence and autonomy of any commission of the BAPE or joint review panel which may participate in the process of a cooperative environmental assessment.

20.2.5 Environmental Permitting

Even if the project undergoes an environmental impact assessment and is authorized by the Government pursuant to section 31.5 of the EQA, it would still be subject to section 22 of the EQA and must therefore obtain a CA as stated in section 6 of the *Regulation Respecting the Administration of the Environment Quality* Act (c. Q-2, r. 3).

"6. Notwithstanding sections 1 to 3 of this Regulation, any project arising from a project authorized by the Government pursuant to section 31.5 of the Act is subject to the application of section 22 of the Act."

In addition to the mitigation measures set out as part of the environmental impact assessment, the final project design must comply with all applicable standards relating to the proposed infrastructure and equipment.

The issuance of the CA, however, should only be a formality as the certificate issued pursuant to section 31.5 of the EQA binds the Minister as to where he exercises the powers provided in section 22 and as specified in section 31.7 of the Act.

"31.7. Every decision rendered under section 31.5 or 31.6 is binding on the Minister, where he subsequently exercises the powers provided in section 22, 32, 55, 70.11 or in Division IV.2."

In addition to the CA required under section 22 of the EQA, the proponent must obtain the permits, authorizations, approvals, certificates and leases required from the appropriate authorities. Those are described in the upcoming sections.

The authorization application and permitting process is expected to take one full year. Applications may be filed concurrently with the construction work and should therefore not impact on the project schedule.

20.2.5.1 Certificates of Authorization

In order to carry out the Duparquet Project, one or more certificates of authorization (CAs) will be required from the MDDELCC under section 22 of the EQA. 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 Environment Quality Act* is included with a CA application. For mining activities, CA applications must also comply with the Directive 019 requirements.

Moreover, because the Duparquet Project will involve discharges into the aquatic environment, it will be necessary to complete the effluent discharge objectives application form (*"Demande d'objectifs environnementaux de rejet (OER) pour les industries"*) and attach it to the CA application. The CA application forms and all required documents must be sent to the MDDELCC's Abitibi-Témiscamingue and Northern Québec regional branch. The time required to analyze an application for a CA directly depends on the complexity of the project. Under the Declaration of Services to the Public, the Department is committed to providing an official response within 75 days following the receipt of the application for a CA or approval.

The number of CA applications to prepare will depend on the timeline of the project activities and its associated work items. By dividing the project into predefined items, it will enable a step-by-step implementation process.

Under the Ministerial Order concerning the fees payable under the EQA, fees are payable by the company seeking an authorization under the Act. The fees which are indicated in the Ministerial Order are approximately $1,000 to $5,000 for each request for an authorization under sections 22, 31, 32, 48 and 70.8 of the EQA.

- Authorization for Water Supply Intakes and Devices for the Treatment of Drinking Water and Disposal of Wastewater

An authorization under section 32 of the EQA is needed to connect the Duparquet Project facilities to the municipal water system. Two forms must be completed and signed by the project engineer, and the required documents must be attached to them. The required documents are administrative documents and a technical document to be signed by the project engineer. The application for an authorization must be submitted to the MDDELCC regional branch.

- Authorization to Install an Apparatus or Equipment to Prevent, Reduce or Cause the Cessation of the Contaminants Release into the Atmosphere

Under section 48 of the EQA, an application for an authorization must be submitted for the installation of an apparatus or equipment which will prevent, reduce or cause the cessation of the release of contaminants into the atmosphere. The application for the authorization of an industrial project ("*Demande d'autorisation pour un projet industriel*") must be completed and submitted to the MDDELCC regional branch. The documents to be attached to this application are listed in the form.

20.2.5.2 Approval

Approval for the Location of the Process Concentration Plant and Mine Tailings Storage Facility

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 its 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 Environment Quality Act, 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 MERN before activities begin at the Duparquet gold mine. This request must include the information and documents as set out in sections 124 and 125 of the *Regulation respecting Mineral Substances other than Petroleum, Natural Gas and Brine*.

20.2.5.3 Attestation

Depollution Attestation

In accordance with the Order in Council 515-2002 issued on May 1, 2002, the Duparquet Project requires a depollution attestation from the MDDELCC. This certificate, which is renewable every 5 years, identifies the environmental conditions that must be met by the industrial facilities when carrying out its activities. The certificate compiles all of the environmental requirements relating to the operation of an industrial facility. The depollution attestation differs from the CA issued under section 22 of the EQA. The latter is a statutory document issued prior to the implementation of a project or activity, whereas the former applies strictly to the operation of an industrial facility. The steps included in the depollution attestation process are described below.

Order in Council

The process for the issuance of a depollution attestation was implemented through the adoption by the Québec Government of an order in council that subjects certain categories of industrial facilities to Subdivision 1 of Division IV.2 of Chapter 1 of the EQA. This Subdivision of the Act establishes the legal framework for the depollution attestation.

Application for a Depollution Attestation

The operator of an industrial facility which is subject to an order in council must apply to the Ministry for a depollution attestation within 30 days following the issuance of the CA issued under section 22 of the EQA for the operation of its mine project. This application must be made using the form provided by the Ministry that identifies all of the required information.

First Draft of Depollution Attestation

The Ministry will prepare and submit a first draft of the depollution attestation to the industrial facility. The facility management has then 30 days to provide comments, as stipulated by the regulation.

Public Consultations

As stipulated by the regulation, the Ministry must publish a notice for public consultation in a daily or weekly newspaper within 90 days of the mailing date of the first draft of the depollution attestation. The Ministry must also make the request and project attestation accessible to public consultation. These consultations must take place over a period of at least 45 days. The facility management is also informed of the project attestation being submitted to public consultation.

Second Draft of Depollution Attestation

Following the public consultations, the Ministry will review the comments that were received and will prepare a second draft of the depollution attestation. The second draft is submitted to the industrial facility management, which has 30 days to provide comments.

Issuing of Depollution Attestation

The Ministry will review the final comments provided by the industrial facility management and will prepare the final version of the depollution attestation, which will be issued to the industrial facility management for a period of five years.

The facility management for its part will be responsible for requesting a renewal of its depollution attestation at least six months before it expires. The original certificate will remain in effect until a new certificate is issued.

20.2.5.4 Permits

Forest Management Permit for Mining Activities

Under section 73 of the *Sustainable Forest Development Act* holders of mining rights can obtain forest management permits relating to mining activities in order to exercise their rights under the *Mining Act*. This permit holder is allowed to cut timber on the land covered by its mining rights for the construction of buildings or any other operations necessary for its mining activities, in compliance with the *Sustainable Forest Development Act* and its regulations. The applicant must have already obtained the right to operate the site for mining purposes, a right which is granted by the Mines Division of the MERN. Prior to proceeding with its timber cutting operations, the holders of mining rights must submit a written request to the MERN forest management unit in order to obtain a permit for its mining operations. The request can be for the clearing of a site for mining activities, the exploratory boring of a gravel bed or the clearing of a gravel or sand pit.

It is important to note that the holder of a forest management permit for mining activities must scale all timber harvested in public land according to the standards prescribed by the Government regulation and as specified in section 75 of the Sustainable Forest Development Act. The holder is responsible for paying the prescribed duty as stipulated in section 6 of the Regulation respecting Forest Royalties.

High-Risk Petroleum Equipment Operating Permit

Under section 120 of the *Safety Code*, "The owner of a petroleum equipment installation that includes at least one component that is high-risk petroleum equipment must obtain a permit for the use of all the high-risk petroleum equipment situated at the same address, until the equipment is removed from its respective place of use".

A "high-risk" petroleum equipment as defined in section 8.01 of the *Construction Code* is having one of the following characteristics:

- For underground storage systems:
  - Capacity of 500 or more litres, used to store gasoline or diesel;
  - Capacity of 4,000 or more litres, used to store heating oil and heavy fuel oil except for equipment used for heating a residential single-family dwelling;

- For aboveground storage systems:
  - Capacity of 2,500 or more litres, used to store gasoline;
  - Capacity of 10,000 or more litres, used to store diesel;
  - Capacity of 10,000 or more litres, used to store heating oil and heavy fuel oil except for equipment used for heating a residential single-family dwelling;

- Storage tanks used to store gasoline, diesel, heating oil and heavy fuel oil for profit, regardless of their capacity.

The form "*Application for a Permit for the Use of a High-Risk Petroleum Equipment*" must be completed and submitted to the *Régie du bâtiment*. This application must include all of the information and documents identified in section 121 of the *Safety Code*. A permit is valid for 24 months. The issuing and renewal of a high-risk petroleum equipment permit are subject to compliance and performance monitoring under the provisions of the *Construction Code* and the *Safety Code*.

Explosives Permit

Under the *Act respecting Explosives*, no person shall possess, store, sell or transport any explosives unless he is holding a permit for such purpose. Depending on the intended usage, several permits are required for the possession of explosives for industrial or commercial purposes. Division II of the Regulation under the *Act respecting Explosives* describes the different types of permits that are required. A general explosives permit entitles the holder to have explosives in his possession. Solely the holder of a general permit can obtain a magazine, sale or transport permit. A magazine permit entitles the holder of a general permit to purchase and store explosives in a container or a building that complies with the regulations. A transport permit entitles the holder of a general permit to transport explosives.

In order to obtain these permits, the forms "*Application for a General Explosives Permit*" and the "*Application for a Sales, Magazine or Transport Permit*", which are available from the Sûreté du Québec (SQ), must be completed. The required documents and fees must be submitted to the SQ. Permits are valid for a period of 5 years.

20.2.5.5 Leases

Mining Lease

Under section 100 of the *Mining Act*, "no person may mine mineral substances, except surface mineral substances, petroleum, natural gas and brine, unless he has previously obtained a mining lease from the Minister or a mining concession under any former Act relating to mines". In order to obtain a mining lease, a claim holder must establish the existence of the presence of an economic deposit. Applications must be submitted to the Registrar's Office or to the regional office. The initial term of a mining lease is 20 years. The lease can then be renewed every 10 years for the duration of the mining operation. The procedure for obtaining a mining lease is described in the MERN's online publication "Mining Leases and Concessions".

Non-Exclusive Lease for the Mining of Surface Mineral Substances

According to section 109 of the *Mining Act*, "a lessee or a grantee may use, for their mining activities, sand and gravel that is part of the domain of the State except where the land that is subject to the lease is already subject to an exclusive lease to mine surface mineral substances in favour of a third person". The mining of sand and gravel located outside of mining leases requires a non-exclusive lease for the mining of surface mineral substances, under section 140 of the *Mining Act*. The applicant must make a request for a non-exclusive lease by completing the form "*Application for Non-Exclusive Lease (BNE) for Mining Surface Mineral Substances*" and providing the documents identified in section 3 of the form.

Lease for the Occupation of the Domain of the State

Under section 239 of the Mining Act, "the holder of mining rights or the owner of mineral substances may, in accordance with the Act respecting the lands in the domain of the State (chapter T-8.1), obtain that public lands be transferred or leased to him to establish a storage site for tailings, or a site for mills, shops or facilities necessary for mining activities". Several components of the Duparquet Project might be located outside of the lands covered by the mining lease. Since a portion of the Duparquet Project is located on public lands, the land in question will need to be leased under section 47 of the *Act respecting the Lands in the Domain of the State*.

20.2.5.6 Rehabilitation of the historical Beattie gold mine site

As depicted in Section 6 of this Report, infrastructure associated with the historic Beattie Gold Mine site will have to be dismantled. The former operator of this mine, Beattie Gold Mines Limited, was set up in 1932. Production started in 1933 and operations ended in 1956, after 23 years of almost continuous production. Except for a small surface exploration program in 1966, the property remained dormant from 1956 to 1987. From 1987 until 2007, several mining and exploration companies, such as SOQUEM, Cambior, Forbex Mining Resources, Beattie Gold Mine, and

Fieldex Inc, conducted drilling on the Beattie, Donchester, Central Duparquet and Dumico properties. From 2007 until today, the exploration of the Beattie and Donchester properties was accelerated in order to gather enough information to conduct the present NI 43-101.

The environmental characterization work completed in 2012 by Stavibel for the demolition of the on-site infrastructure indicates and confirms the presence of hazardous materials and materials contaminated by hazardous materials. These materials will have to be managed during demolition to limit their impact on human health, flora and fauna, and the environment. Demolition work will have to be carried out methodically, in stages, so that proper material segregation can be maintained. A demolition specification (technical specifications) with a demolition material management plan (dismantling plan) will have to be prepared for this purpose to guide the contractor during the demolition work. It is recommended that the main Roaster and Cottrell buildings and the ore conveyor building be demolished mechanically. The preliminary risk assessment associated with demolishing the stack recommended demolition by blasting. The waste and debris from the stack demolition will subsequently be sorted on the ground and then managed, characterized, and disposed of in an authorized facility. It is recommended that Clifton Star perform an exhaustive stack demolition risk and feasibility study before proceeding with any demolition work. Different options will have to be analyzed, taking into account the risks and costs associated with the demolition of this infrastructure.

Regulation Respecting Hazardous Materials

Activities such as dismantling, temporary storage, treatment and/or cleaning, reuse, recycling and revalorization as well as disposal and transport will all need specific permits, mostly to be issued under the Regulation respecting hazardous materials, but also in compliance with the EQA and the Regulation regarding the landfilling and incineration of residual materials.

20.2.5.7 Federal Permitting

Authorization to Alter Fish Habitat

Section 35 of the *Fisheries Act* specifies that:

(1) No person shall carry on any work, undertaking or activity that results in the harmful alteration or disruption, or the destruction, of fish habitat.
(2) A person may carry on a work, undertaking or activity without contravening subsection (1) if:
(a) the work, undertaking or activity is a prescribed work, undertaking or activity, or is carried on in or around prescribed Canadian fisheries waters, and the work, undertaking or activity is carried on in accordance with the prescribed conditions;
(b) the carrying on of the work, undertaking or activity is authorized by the Minister and the work, undertaking or activity is carried on in accordance with the conditions established by the Minister;
(c) the carrying on of the work, undertaking or activity is authorized by a prescribed person or entity and the work, undertaking or activity is carried on in accordance with the prescribed conditions;

(d) the harmful alteration or disruption, or the destruction, of fish habitat is produced as a result of doing anything that is authorized, otherwise permitted or required under this Act; or
(e) the work, undertaking or activity is carried on in accordance with the regulations.

When a project includes a known risk of affecting fish and fish habitat, such a project must be submitted to Fisheries and Oceans Canada (DFO) for its 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 have previously completed 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 the 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 to be acceptable – an authorization for a harmful alteration, disruption or destruction of fish habitat (HADD) and a compensation for fish habitat loss are required;
- The project will have unacceptable impacts on fish and fish habitat – the project cannot proceed as designed.

The Ministry will issue in most cases an authorization if the compensation plan results in no net loss of fish habitat. The Duparquet Project is expected to require an authorization for HADD and a compensation for the loss of habitat.

Clifton Star will have to establish an Environmental Effects Monitoring Program (EEMP). This is a requirement for regulated mines in accordance with the Metal Mining Effluent Regulations (MMER) under the authority of the *Fisheries Act*. The objective of the EEMP is to evaluate the effects of mine effluents on fish, fish habitat and the use of fisheries resources by humans. Directive 019 sets at the provincial level the criteria that mine effluents must comply with at the end-of-pipe. The EEMP examines the effectiveness of the environmental protection measures directly in the aquatic ecosystems, i.e. downstream of the final discharge point. The EEMP consists of biological monitoring and effluent and water quality monitoring as follows:

- Effects on fish are assessed through comparison of adult fish exposed to effluent with unexposed fish;
- Effects on fish habitat are assessed through a comparison on benthic invertebrate communities from areas exposed and unexposed to effluent;
- Effects on the use of fisheries resources are assessed by comparing the designated contaminants (i.e., mercury for metal mines) in fish tissue against health guidelines for fish consumption.

Moreover, the effluent quality is monitored through sub-lethal toxicity testing. For metal mines, an effluent characterization and water quality monitoring studies are also required.

The requirement of an EEMP is to be reviewed as more information is collected and when a better assessment of the impact of effluents on the aquatic environment is available.

Licence for Explosives Factories and Magazines

Under section 7(1) a) of the federal *Explosives Act*, a licence issued by the Minister of Natural Resources Canada is required for the operation of explosives plants and magazines in Canada.

It is reported that there will be no explosives plant on site. Also, 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”.

In Québec, the *Act respecting Explosives* provides for the issuing of permits, the inspection and the control of activities associated with explosives (see section 1.4.4 of the Act: E*xplosives permit*).

Thus, no licence for explosives should be required from Natural Resources Canada. If modifications were to be implemented to the Project so that it would require the operation of an explosives plant or magazine, such license would be necessary. However, obtaining it would not have any impact on the permitting schedule after consideration of the recent modifications applied to the CEAA 2012.

Appendix 2 of the Metal Mining Effluent Regulations

Project design was done in a way that, most likely, no fish habitats will be directly impacted by the implementation of any accumulation areas. Consequently, those infrastructures will not have to be listed in Appendix 2 of the MMER.

The use of a natural waterbody considered as a fish habitat to store mine residues indeed requires a modification to the MMER which was adopted in compliance with sections 34(2), 36(5) and 38(9) of the *Fisheries Act* in order to control mine effluent discharge and the implementation of tailings and waste rock management facilities in fish-bearing waterbodies.

20.2.6 Rehabilitation and Mine Closure Plan

Section 232.1 of the *Mining Act* states that:
The following must submit a rehabilitation and restoration plan to the Minister for approval and must carry out the work provided for in the plan:

(1) every holder of mining rights who engages in exploration work determined by regulation or agrees that such work be carried out on the land subject to his mining rights;
(2) every operator who engages in mining operations determined by regulation in respect of mineral substances listed in the regulations;
(3) every person who operates a concentration plant in respect of such substances;
(4) every person who engages in mining operations determined by regulation in respect of tailings.

The obligation shall subsist until the work is completed or until a certificate is issued by the Minister under section 232.10.

As stated in section 101, "the [mining] lease cannot be granted before the rehabilitation and restoration plan is approved in accordance with this Act, and the CA mentioned in section 22, 31.5, 164 or 201 of the EQA (chapter Q-2) has been issued."

Hence, a rehabilitation plan will have to be prepared as part of the project and approved by the MERN. The rehabilitation and restoration plan should be elaborated in accordance with the provincial Guidelines for Preparing a Mining Site Rehabilitation Plan and General Mining Site Rehabilitation Requirements (1997) which provides to the proponents the rehabilitation requirements. This study accounts for costs of all works needed for the rehabilitation of a mining site following the *Regulation respecting Mineral Substances other than Petroleum, Natural Gas and Brine*.

20.2.6.1 General Principles

The main objective of mine site rehabilitation is to restore the site to a satisfactory condition by:

- Eliminating unacceptable health hazards and ensuring the public safety;
- Limiting the production and circulation of substances that could damage the receiving environment and trying to eliminate long-term maintenance and monitoring;
- Restoring the site to a condition which is visually acceptable to the community;
- Reclaiming the areas where the infrastructures are located (excluding the accumulation areas) for future use.

Specific objectives are to:

- Restore degraded environmental resources and land uses;
- Protect important ecosystems and habitats of rare and endangered flora and fauna, which favors the re-establishment of biodiversity;
- Prevent or minimize future environmental damage;
- Enhance the quality of specific environmental resources;
- Improve the capacity of eligible organizations to protect, restore and enhance the environment; and
- Undertake resource recovery and waste avoidance projects and prevent and/or reduce pollution.

The general guidelines of a rehabilitation plan include:

- Favouring a progressive restoration to allow for a rapid re-establishment of biodiversity;
- Implementing a monitoring and surveillance program;
- Maximizing recovery of previous land uses;
- Establishing new land uses;
- Promoting habitat rehabilitation using operational environmental criteria;
- Ensuring sustainability of restoration efforts.

The mine site rehabilitation plan focuses on land reclamation, reclamation of tailings area and water basins, and of surface drainage to prevent erosion. The successful completion of a rehabilitation plan will ensure that the project will result in a minimum of disturbance. Site inspections will be carried out before the property is returned to the Government.

The rehabilitation concept for the current project is described below and complies with the requirements described in the Guidelines for Preparing a Mining Site Rehabilitation Plan and General Mining Site Rehabilitation Requirements and the current legislation.

20.2.6.2 Mining Site Rehabilitation Plan Concept

The rehabilitation and restoration plan concept is summarized as follows:

- Tailings Accumulation Cells and Waste Rock Piles
  - Exposed surfaces of the accumulation areas (tailings accumulation cells, waste rock and overburden piles) will be covered with a layer of top soil/overburden and revegetated when feasible.

- Haul Roads
  - Surface will be scarified and revegetated.

- Industrial Complex and Buildings
  - No building will be left in place. Whenever possible, buildings will be sold with the equipment they contain, completely or partially. During dismantling works, beneficiation/recycling of construction material will be maximized. Remaining waste will be disposed of in an appropriate site.
  - All equipment and machinery will be disposed of or recycled off-site.
  - The explosives magazine, if any, and related facilities will be dismantled.
  - The drinking water supply and domestic wastewater treatment facilities will be dismantled.
  - Infrastructure relating to electricity supply and distribution will be dismantled with the exception of Hydro-Québec requirements.

All underground services (power lines, pipelines, water and sewer pipes, etc.) shall remain in place since they are unlikely to cause any environmental damage. Openings and access to such pipelines, however, shall be sealed.

Open Pit

The surface exploitation of a mineral substance is common in Québec. Many open pits that are created to extract a mineral substance or ore are therefore found throughout the province. Unlike quarries that are essentially developed on rock outcrops, ore deposits can be located below the surface, which means pits could be filled with groundwater. In many open pit mines, water could rise to the overburden contact without the dewatering wells.

Once mining activities cease, the pit will gradually fill up to its equilibrium level with rainfall and groundwater. The overburden slope around the pit will have already been established for a safe operation of the mine. No special work in this regard will be required upon the cessation of mining activities.

To permanently close pit access roads, an embankment 2 m high will be built using waste rock, along with an equivalent crest line. A ditch 2 m wide and 1 m deep will be excavated in front of the embankment.

Environmental Aspects

- Drainage
  - Whenever possible, the surface water drainage pattern will be re-established to a condition similar to the original hydrological system.

- Topsoil Management
  - During the site construction period and overburden stripping over the orebody, overburden and topsoil will be stored separately and used for revegetation purposes. Slopes of the overburden storage area and flat surfaces will ultimately be seeded and revegetated.

- Waste Management
  - Waste material from demolition activities will be:
    - Decontaminated when required;
    - Recycled when cost-effective;
    - Buried in an appropriate site.
  - All non-contaminated waste will be sent to an appropriate site.

- Hazardous Materials
  - Facilities containing petroleum products, chemicals, solid waste, hazardous waste, and/or contaminated soil or materials will be dismantled and managed according to regulatory requirements.
  - All hazardous waste will be managed according to existing laws and regulations and will be transported off site.

- Characterization Study

The *Land Protection and Rehabilitation Regulation*, which came into force on March 27, 2003, contains several provisions concerning land protection in the new section IV.2.1 of the EQA. The term “land” also includes groundwater and surface waters. The Regulation sets limit values for a range of contaminants, and specifies the categories of targeted commercial or industrial activities. The mining industry is one of the categories subject to the Regulation.

For the mining industry, this generally entails an undertaking of a site characterization study within 6 months following the termination of the mine operations. In cases where the contamination were to exceed the criteria set for in the Regulation, a rehabilitation plan which would specify the environmental protection measures to be undertaken must be submitted to the MDDELCC for its approval.

Waste rocks and mine tailings are not soils and are not covered by this Regulation. The characterization study will address the areas that are likely to have been contaminated by human activities, specifically the handling of petroleum products.

20.2.6.3 Monitoring Program and Post-Closure Monitoring

According to Directive 019 for the mining industry, a Monitoring Program will have to be implemented during the mine operation to account for all the requirements specified in that Directive, especially with regards to noise levels, vibrations, surface and ground waters.

Physical Stability

The physical stability of the tailings dams and of the waste rock piles will need to be assessed, and signs of erosion will be noted. This monitoring will be conducted on an annual basis for a minimum of five years following mine closure.

Environmental Monitoring

Monitoring of the water quality (surface and groundwater) will continue for a minimum of five years after the completion of the restoration work.

Agronomic Monitoring

The agronomic monitoring program is designed to assess the effectiveness of the revegetation which will be done as part of mining rehabilitation efforts.
To document the success of the revegetation efforts over the waste dumps areas, an agronomic monitoring will be undertaken following the establishment of a vegetative cover on the areas subject to the progressive restoration program. This monitoring will be conducted annually for three years following the revegetation efforts. Reseeding will be carried out, as required, in areas where revegetation is found unsatisfactory.

20.3 Water Management

20.3.1 Water Balance

All surface infrastructure components are located on the general site layout plan (see Figure 18.1). Table 20.8 depicts data on which the following preliminary water management plan is based, and Figure 20.4 illustrates it.

Table 20.8 – Preliminary water management plan design criteria for the Duparquet Project

Infrastructures	Area (m^2)	Annual Rainfall[1] (m^3/d)	Evapotranspiration[2] (m^3/d)	Net Runoff (m^3/d)	Groundwater[3] (m^3/d)	Effluent[4] (m^3/d)	FINAL EFFLUENT[5]
Considered in the final effluent							
Floatation TMF - Cell 1	1,449,840	3,507	877	2,631			
Floatation TMF - Cell 2	1,006,410	2,435	609	1,826		4,822	
POX TMF	201,640	488	122	366	n/a		
Floatation TMF Polishing Pond	145,000	351	140	210		230	
POX TMF Polishing Pond	13,658	33	13	20			
Beattie Pit	884,130	2,139	428	1,711	1,400		
Donchester Pit	434,340	1,051	210	841	890	5,493	m^3/d 8,079
Central Duparquet Pit	207,140	501	100	401	250		m^3/h 336.6
Mill and Office	162,359	393	79	314			m^3/s **0.09**
Ore Stockpile	131,190	317	95	222	n/a	690	
Main Storage Water Ponds (280 000 m^3)	105,760	256	102	154			
Other discharges							
Overburden	384,200	929	232	697		697	
Waste Rock #1	662,970	1,604	481	1,123	n/a	1,123	n/a
Waste Rock #2	991,300	2,398	719	1,679		1,679	
Waste Rock #3	944,500	2,285	685	1,599		1,599	

[1] *Annual mean total precipitation at the Kinojevis River Weather Station is 883 mm.*

[2] *Based on average runoff for different land types on a mine site in Northern Quebec (annual mean total precipitation = 798 mm). See below.*

[3] *Hydrogeological analysis not performed yet.*

[4] *It is considered that for every dry m^3 of tailings sent to the TMF, 0.5 m^3 of interstitial water will stay in the TMF. Flotation tailings are 49.8% solid while POX tailings are 13% solid. Average milling rate is 10,000 t/d.*

[5] *Make-up water (recirculation from TMF Polishing Pond to the mill) is estimated at 13,288 m^3/d. Freshwater supply to the plant is 159 m^3/d.*

	Tailings Area, Overburden	Waste Rock, Ore Stockpile	Open Pits, Built Areas	Natural Areas (lands, lakes)
Annual Average Runoff	75%	70%	80%	60%
Annual Average Evapotranspiration	25%	30%	20%	40%

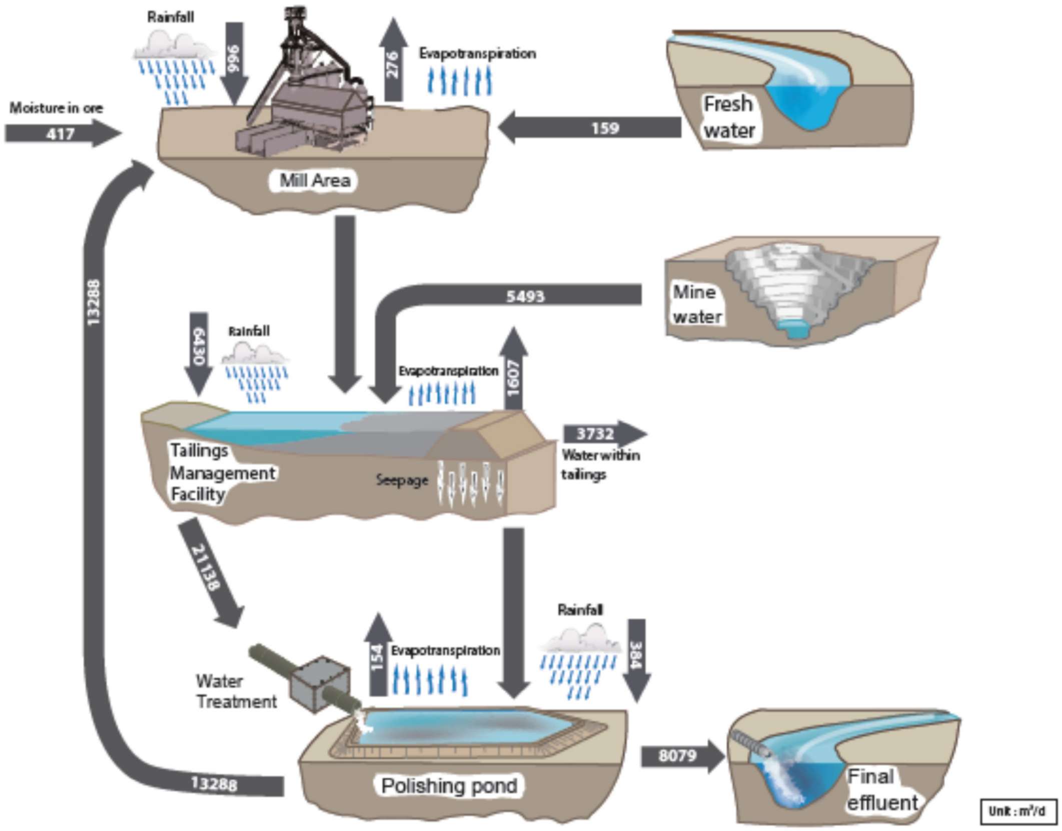

Figure 20.4 – Water balance for the Duparquet Project

20.3.2 Assumptions

Rainfall and Evapotranspiration

Weather data from the Kinojevis River Station, near Rouyn-Noranda, was used to estimate rainfall and snowfall in the area. The annual average rainfall is estimated at 648 mm while snowfall is 235 cm annually. Total precipitation is therefore annually estimated at 883 mm on average. Evapotranspiration rates were used considering land types (tailings, waste rock, open pit, etc.; Table 20.8). Those rates were determined following numerical simulations performed for a different mine project located in a region with similar weather conditions.

Pit Dewatering

A preliminary estimate of the water inflow to mine pits was made despite the lack of available information on the hydrogeological properties of the sector. The water inflow during dewatering was calculated using the simple analytical equations by Marinelli and Niccoli (2000). The Marinelli and Niccoli (2000) method is divided into two parts where Q1 represents the inflow from the pit walls and Q2 is the flow through the pit bottom and is calculated by the following analytical equations:

eq 1: $Q1 = W\pi\,(ro^2 - rp^2)$

eq 2: $Q2 = 4rp\,(Kh/m^2)(ho\text{-}d)$

Where:

W = recharge flux

rp = effective pit radius

ro = radius of influence (maximum extent of the cone of depression)

Kh = horizontal hydraulic conductivity

Kv = vertical hydraulic conductivity

m = anisotropy parameter

d = depth of pit lake

ho = initial saturated thickness above the base of the pit

In this method, the pit is treated as a well, and the long-term pumping is considered as steady state discharge. The estimation was made for the last year of operation when each pit will be at their maximum depth.

- Assumptions associated with this method include:
- Pit walls are considered as if they were forming a right circular cylinder;
- Water table is approximately horizontal;
- Recharge is uniformly distributed across the influenced area;
- All recharge within the influenced area of the pit is assumed to be captured by the pit; and
- Ground water flow towards the pit is axially symmetric.

In 2013, Stavibel carried out four (4) pumping tests in monitoring wells located in the vicinity of the project and installed in the top part of the bedrock. The measured hydraulic conductivity values range from 1.1x10-8 to 8.1x10-7 m/s. The geometric mean value of 8.3x10-8 m/s was selected in order to calculate the inflow in the pits. This value is consistent with the theoretical values for fractured rock between 4.3x10-5 m/s and 5.0x10-9 m/s as identified in the Abitibi-Témiscamingue Groundwater Knowledge Acquisition Program (*Projet d'acquisition de connaissances sur les eaux souterraines* [PACES]; Cloutier et al., 2013). For this preliminary estimation, the inflow of water has been calculated for the three main pits of the Project, i.e. Beattie, Donchester and Central Duparquet. The initial hydraulic head was estimated from data collected by Stavibel (2013) in monitoring wells installed in the rock aquifer and the recharge rate (W) was estimated from regional meteorological information. According to Marinelli and Niccoli (2000), the inflow to the pits are maximized when the seepage face depth at the pit walls is set to 0. This will be the case where mine dewatering is active since mining will be done in dry conditions. This will also act as a safety for determining the maximum possible inflow to the mine. Table 20.9 shows data that were used for the preliminary estimation of water inflow for each pit.

Table 20.9 – Estimated groundwater inflow in the proposed pits

Projected Pits	Initial saturated thickness	Pit area at the end of mining	Equivalent pit radius	Recharge within the system	Horizontal hydraulic conductivity	Vertical hydraulic conductivity	Calculated radius of influence	Total groundwater inflow
	m	m^2	m	m/s	m/s	m/s	m	m^3/d
	ho	A	rp	W	Kh	Kz	ro	Q
Beattie	300	1 024 400	52	$6.7x10^{-9}$	$8.3x10^{-8}$	$8.3x10^{-8}$	725	1400
Donchester	175	585 000	79	$6.7x10^{-9}$	$8.3x10^{-8}$	$8.3x10^{-8}$	525	890
Central Duparquet	100	139 000	37	$6.7x10^{-9}$	$8.3x10^{-8}$	$8.3x10^{-8}$	285	250

Based on these data, the total inflow rate of groundwater is estimated to be 1,400 m^3/d for the Beattie pit, 890 m^3/d for Donchester and 250 m^3/d for Central Duparquet (Table 20.8).

Note that the current estimation is based on geometric mean value of hydraulic conductivity measured in the surface of the bedrock. The Marinelli and Niccoli (2000) method is sensitive to the hydraulic conductivity value; therefore, hydraulic conductivity values in the entire thickness of the aquifer should be investigated in a future phase of the Project. In addition, *in situ* hydrogeological values should be used in the future in order to confirm water inflow by numerical modelling. This estimate provides (within an order of magnitude) the potential daily flow that will need to be pumped in order to dewater the pits. Since these values are estimated from theoretical data and a simplified model, it must under no circumstances be used for design purposes.

As part of the future steps of the Duparquet Project, a comprehensive hydrogeology assessment will have to be completed in order to adequately estimate the groundwater input in the water balance. Such assessment will include, among other things, mathematical modelling of the successive mine operation phases to estimate the daily flow rate required to keep the pit dry.

Tailings Interstitial Water Content

According to the mining plan for the Duparquet Project, a maximum of 3.65 Mt of tailings will be generated annually and sent to the TSF for disposal. Considering that, based on data depicted in Section 17 of this Report, solid content of flotation tailings is 49.8% (w/w) and of POX tailings is 13% (w/w), it is considered that 13,576 m^3/d of water is associated with those tailings. However, based on data collected as part of another gold project located in the same region and using similar ore processing methods, it was calculated that tailings water content (interstitial water), once disposed in the TSF, typically is about 50%; that is, for every dry m^3 of tailings there is 0.5 m^3 of water that stays in the TSF ("within" the tailings themselves) and this quantity must not be considered as part of the final effluent. Based on a mining rate of 10,000 tpd, this corresponds to about 3,732 m^3/d (Figure 20.4).

20.3.3 Preliminary Water Management Plan

The Property is mostly flat and badly drained (clayey to silty soils). However, in some areas, the land is hillier with elevation ranging from 270 masl near Lake Duparquet to 365 masl at the top of the highest hill in the easternmost section of the Property. Such hilly sections are also associated with better-drained surface deposits such as sand and/or sandy silt.

All mine infrastructure components are located within only one watershed, namely Lake Duparquet. Furthermore, almost 90% of all areas to be impacted by the implementation of the proposed infrastructure are located within a single sub-watershed, namely Creek #1, which drains all water coming from the eastern-, northern- and westernmost sections of the Property.

A small portion of the proposed tailing storage facility (TSF) is located outside that area, as well as the TSF Polishing Ponds, but all waters coming out of those facilities will be managed in such a way that it will be discharged with the final effluent (as defined by Québec's Directive 019 issued by the MDDELCC) in Creek #1. No water discharge is located outside that sub-watershed, and consequently, once treated to comply with both provincial and federal requirements, the final effluent will end up in Lake Duparquet as it is the case prior to site implementation.

Water from the TSF Polishing Pond will be recirculated to the mill. Make-up water required at the mill is estimated at 13,288 m^3/d (Figure 20.4). In order to guarantee that input during the first years of mill operation, the first construction phase will include the implementation of the Process Water Pond (or Main Storage Water Ponds; 280,000 m^3), i.e. prior to production at the mine so that enough water can be accumulated for use as process water. An additional input of 159 m^3/d of fresh water will also be required.

All water accumulated in the TSF Polishing Ponds that will not have been recirculated (reclaimed) to the mill will be discharged with the final effluent following treatment.

All surface water on the process facilities (mill and office) pad will be managed (Table 20.8; Figure 20.4). The surface water will flow to the peripheral drainage ditch which will discharge at the lowest point to be transfer to the treatment pond (TSF Pond). A segregation ditch will also be made to separate naturally flowing water and surface water that needs to be treated. The same management procedure will be applied to the adjacent ore stockpile. Mine water will be pumped to the TSF.

According to Directive 019, only one final effluent is to be considered (at 0.09 m^3/s or 8,079 m^3/d; Table 20.8; Figure 20.4). According to Directive 019, a final effluent is defined as an effluent which needs no further treatment before being discharged to the receiving environment in full compliance with all applicable guidelines and/or criteria. Equipment will be implemented to measure pH and flow. Also, the final effluent will be monitored as specified in Directive 019. The final effluent discharge criteria prescribed in Directive 019 consist of maximum limit values. The environmental discharge objectives (EDO) to be imposed by the MDDELCC may be more stringent than those specified under Directive 019 and the Metal Mining Effluent Regulations (MMER). The EDOs will be based on water volumes discharged as well as water quality, the minimum annual flow, and usages of the receiving aquatic environment (Creek #1 and, downstream, Lake Duparquet).

Additional hydrological data will be collected in 2014 to adequately assess any potential impact of effluent discharge on Creek #1 and Lake Duparquet and to develop relevant mitigation measures (erosion control, habitat enhancement, etc.). Such measures (ex. artificial wetlands, riprap, etc.) are already expected to be required immediately downstream of the final effluent discharge point and along the section running towards Lake Duparquet, immediately west of the existing Beattie tailings management area and south of provincial highway Route 388.

Four other water discharges were also considered: one at the overburden stockpile and one at each of the three waste rock stockpiles (Table 20.8). Since overburden is not considered as mining residue and waste rock is classified as “low risk”, in accordance with Directive 019, waters running off those piles will be collected by ditches towards a settling pond prior to discharge in the adjacent receiving environment. Provincial and federal best-practice measures will be implemented to adequately manage such discharge and limit any potential impact on downstream aquatic environments.

20.3.4 Final Effluent Water Treatment

A 300-000 m^3 polishing pond will be installed at the end of the Flotation TSF in order to control for TSS content (adequate retention time) and ensure compliance with provincial and federal guidelines/criteria.

A water treatment unit will be implemented at the end of the POX TSF to make sure its effluent comply with all applicable guidelines/criteria. However, it should be noted that both effluents (Flotation and POX) will be combined before discharge as one final effluent as per Directive 019 definition. To do so, a 25,000-m^3 pond will be installed at the end of the POX TSF to enable pumping of water to the Flotation TSF Polishing Pond. The final effluent will come out of the Flotation TSF Polishing Pond.

As described in Section 20.1.2 of this Report, flotation tailings are classified as not potentially acid-generating, not cyanide-containing and not leachable. Other than TSS content, which is to be controlled using a polishing pond, no other treatment is required for the Flotation TSF.

Since no kinetic test has yet been performed on POX tailings to refine the characterization of the POX TSF effluent, a CAPEX provision of $1M was considered for that treatment unit (including building and power) and an OPEX of $200,000 per year for the use of chemicals was also considered. POX tailings are considered as potentially acid-generating, cyanide-containing and not leachable (see Section 20.1.2 of this Report). The discharge rate at the end of the POX TSF is expected to be lower than 300 m^3/d.

The effluent of the Flotation TSF Polishing Pond is to be considered as the only final effluent according to Directive 019. The final effluent will comply with all provincial (Directive 019) and federal (CCME, MMER) criteria/guidelines

20.4 Relations with Stakeholders

The Town of Duparquet and two aboriginal communities, Abitibiwinni First Nation (Pikogan) in Québec and Wahgoshig First Nation in Ontario, have been identified as communities of interest. A socio-economic profile of these communities was completed by Roche in 2013. It addressed the following topics:

- The socioeconomic environment of Duparquet and the Abitibi-Ouest Regional County Municipality (RCM);
- The socioeconomic environment of the Abitibiwinni First Nation and of the Wahgoshig First Nation;
- Land-use planning;
- Land use by non-aboriginals;
- Land use by the Abitibiwinni First Nation and by the Wahgoshig First Nation
- Archaeological potential study;
- Recreational activities;
- Transport, energy-related, community and institutional infrastructure;
- Public services.

20.4.1 First Nations

The First Nation located the closest to the Duparquet Project area is the Algonquin Nation, which includes eleven communities among which two are in Ontario and nine in Québec. Six of those communities – Abitibiwinni, Eagle Village, Kitcisakik, Kitigan Zibi, Lac-Simon, Long Point and Wahgoshig – have joined to form the Algonquin Anishinabeg Nation Tribal Council, thus encompassing about 75% of all Algonquins. The Abitibiwinni (Pikogan, Québec) and Wahgoshig (Abitibi 70, Ontario) communities are considered as stakeholders for the Duparquet Project. In February 2013, those two communities signed an agreement which involves working together towards a common approach with regards to their rights, interests and land claims.

The Algonquin Anishinabeg Nation officially has no specific or general land claim registered with Aboriginal Affairs and Northern Development Canada. However, in 2011, it has publicly claimed a large part of lands which encompasses sections of both Québec and Ontario (about 650,000 km^2, from Sault-Ste-Marie to Trois-Rivières and from just south of James Bay to the St-Lawrence River). Consequently, Clifton Star has initiated communication with Québec's *Secretariat aux affaires autochtones* in order to better define how to communicate and engage with the tribal council and communities (Pikogan being the closest community to Duparquet, in the province of Québec).

It should also be noted that the Pikogan and Lac-Simon communities have entered into a special agreement with the provincial governement called *"Entente de principe sur la consultation et l'accommodement entre le gouvernement du Québec et le Conseil de la Première Nation Abitibiwinni et le Conseil de la Nation Anishinabe de Lac-Simon"*.

Although Algonquin communities in Québec have not signed any federal or provincial agreement or treaty, all three communities – Pikogan, Lac-Simon and Wahgoshig – have recently signed agreements with mining companies (Canada Lithium in 2012, Northern Gold Mining and Royal Nickel in 2013), therefore demonstrating their willingness towards harmonious relations with the mining industry.

It should finally be noted that an archaeological potential study was recently completed and confirmed that there are no areas of interest on the Property.

20.4.2 Non-Aboriginal Communities and Governmental Authorities

Since 2011, Clifton Star representatives have established sustainable relationships with several local, regional and provincial stakeholders. These include representatives from the Town of Duparquet, the Abitibi-Ouest RCM as well as local businesses, landowners and residents. As part of the future steps of project development, additional stakeholders – among others, the Town of Rouyn-Noranda, the *Conférence régionale des Élu(e)s de l'Abitibi-Témiscamingue*, the *Conseil régional en environnement de l'Abitibi-Témiscamingue*, *Action boréale de l'Abitibi-Témiscamingue*, the *Organisme de bassins versants du Témiscamingue/Abitibi-Jamésie*, the *Société des eaux souterraines en Abitibi-Témiscamingue*, the *Regroupement d'éducation populaire de l'Abitibi-Témiscamingue*, the *Agence de santé et de services sociaux de l'Abitibi-Témiscamingue*, the Abitibi Chamber of Commerce, ComaxAT, the *Centre local d'emploi Abitibi-Ouest* and the *Centre local de développement de Rouyn-Noranda* – will be contacted to inform them about the Project and to gather their concerns.

Finally, Clifton Star representatives have taken part in numerous meetings with the Department of Energy and Natural Resources (MERN), of Sustainable Development, Environment and the Fight to Climate Change (MDDELCC) and of Transportation (MTQ) to discuss mutual interests and to ensure that all involved parties could be satisfied. All parties are working towards common objectives, including the rehabilitation of the historical Beattie Gold Mine site as part of the development of the Duparquet Project, and have been fully cooperating with Clifton Star.

21. CAPITAL AND OPERATING COSTS

The PFS is based on capital pricing as of the last quarter in 2013. The PFS considered two possible processing scenarios: Pressure Oxidization (POX) process and Concentrate production process. The strategy of using the POX option generated the highest financial return, while the alternative resulted in lower capital and lower operating costs. Based on the current results, the POX process is favored and was selected as the base case for the PFS.

21.1 Capital Cost Estimate

The pre-production capital costs for the POX option are estimated at $394 million and sustaining capital is estimated at $118 million. The capital costs include various added contingencies depending on the sector. In the base case estimate, contingencies and indirect costs total $98.7 million of the pre-production costs and represent 26% of the costs. Indirect costs (owner's costs; engineering, procurement and construction management (EPCM); and detailed engineering) of 37% have been applied to the process plant and to other surface infrastructure. The average contingency for all environmental items is 20%.

The total capital expenditure of $512M for the Duparquet Project is broken down into five (5) cost components: mine production equipment; surface installation and equipment; processing facilities; tailings storage facilities; and environmental (Table 21.1).

The tailing storage facilities item in the table below includes the reclaim pumping station and pipeline. The remainder of the tailings dam infrastructure is included in the environmental pre-production and sustaining costs.

Table 21.1 – Breakdown of the capital cost

Description	Pre-production ($)	Sustaining ($)	Total cost ($)
Capitalized operating cost	*51,012,141*	-	*51,012,141*
Capitalized revenue	- *21,984,860*	-	- *21,984,860*
Mine production equipment	23,120,924	91,126,227	114,247,151
Surface installation and equipment	58,218,662	10,144,723	68,363,385
Processing Facilities	226,611,220	-	226,611,220
Tailings Storage Facilities	3,374,029	-	3,374,029
Environmental	53,707,038	16,712,074	70,419,112
Total	**394,059,154**	**117,983,024**	**512,042,179**

21.1.1 Capitalized Operating Cost

The capitalized operating cost includes all owners' cost for the pre-production period as well as the production cost for the startup of the operation, for a total of $51M.

21.1.2 Capitalized Revenue

During the pre-production period, it is anticipated that 15,005 ounces of gold and 4,690 ounces of silver will be produced, providing revenue of $22M. The pre-production revenue was capitalized.

21.1.3 Mine Production Equipment Cost

The capital expenditure for principal equipment will be financed by partnership with the supplier with a 25% down payment upon receipt of the equipment, and the remaining amount financed at 4.75%. The total capital cost for the production equipment is $114.2M, as presented in Table 21.2

Table 21.2 – Breakdown of mine production equipment capital cost

Mining Equipment	Pre-production ($)	Sustaining ($)	Total ($)
Production equipment			
Truck 785D (150t)	7,022,769	42,281,650	49,304,419
Hydraulic Shovel 6030FS	2,290,030	15,656,672	17,946,702
Loader 994H	3,935,079	1,756,365	5,691,444
Drill DR540 (6 ½")	1,076,303	480,392	1,556,695
Drill D55SP (8 ½")	909,031	5,860,843	6,769,874
Support equipment			
Grader 16M	596,683	1,625,169	2,221,852
Dozer D9T	521,481	3,565,303	4,086,784
Wheel Dozer 844H	811,061	1,307,670	2,118,731
Water Truck 76,000L (777G)	807,383	1,301,740	2,109,123
Excavator (rockbreaker) CAT 349	-	855,556	855,556
Excavator CAT 349	396,646	1,080,336	1,476,982
Wheel loader CAT 980K	357,863	308,420	666,283
Small loader (hole stemming)	117,686	189,745	307,431
Fuel truck CT660	289,699	249,673	539,372
Tow lowboy	287,334	463,268	750,602
Tow truck 777G	473,048	762,694	1,235,742
Pickup truck	549,185	3,758,568	4,307,753
Pit busses	52,352	221,164	273,516
Maintenance equipment			
Service truck CT660	242,689	661,006	903,695
Boom truck	109,729	176,915	286,644
Tool carrier	172,969	278,878	451,847
Contingency (10%)	2,101,902	8,284,203	10,386,105
Total	**23,120,922**	**91,126,230**	**114,247,152**

21.1.4 Surface Installation and Equipment

The surface installation and equipment cost estimate mainly represents expenses for new site infrastructure, such as the mill, garage, office, and principal and secondary equipment. No mining costs are included in the capital costs and neither is the cost of waste removal to start environmental work in pre-production years PP3 and PP4.

Table 21.3 – Breakdown of surface installation and equipment capital cost

Description	Pre-production ($)	Sustaining ($)	Total ($)
Electrical and Communication			
New 120 kV transmission line	7,240,000		7,240,000
Main substation	5,136,278		5,136,278
Site power distribution	9,897,259		9,897,259
Communication/IT system	2,107,604	100,000	2,207,604
Sub-Total	**24,381,141**	**100,000**	**24,481,141**
Infrastucture and equipment			
Site preparation	6,424,949	408,000	6,832,949
Maintenance shop and warehouse	10,173,153	168,120	10,341,273
Administration and services building/guard house	2,092,170	3,480,000	5,572,170
Explosive storage	571,043		571,043
Reagent storage/cold shed	512,038		512,038
Site fuel storage	534,985		534,985
Surface support equipment	1,346,803	2,893,000	4,239,803
Site roads	5,868,712		5,868,712
Green wall	-	531,048	531,048
Assay laboratory	1,861,001	697,000	2,558,001
Sub-Total	**29,384,854**	**8,177,168**	**37,562,022**
Water management			
Mine dewatering	517,151	1,867,555	2,384,706
Potable water and sewage treatment & distribution	1,239,657		1,239,657
Fresh water pumping	1,157,926		1,157,926
Fire protection	1,537,933		1,537,933
Sub-Total	**4,452,667**	**1,867,555**	**6,320,222**
Total	**58,218,662**	**10,144,723**	**68,363,385**

Note: Indirect cost and contingency are included in the presented cost.

21.1.5 Processing Facility

The new mill is the largest expense for the Project, totalling $226.61M. The following table describes the capital cost for the processing facility, including contingency and indirect cost.

Table 21.4 – Breakdown of processing facilities capital cost

Description	Pre-production ($)	Sustaining ($)	Total ($)
Crushing	8,266,943	-	8,266,943
Stockpile & Ore handling	13,097,901	-	13,097,901
Grinding	33,582,706	-	33,582,706
Flotation and regring circuit	16,550,339	-	16,550,339
Concentrate thickening	787,773	-	787,773
Concentrate pre leach	1,951,700	-	1,951,700
Pressure oxidation	36,538,143	-	36,538,143
CCD & neutralization	8,041,687	-	8,041,687
Lime boil	4,304,024	-	4,304,024
Cyanidation (CIP)	8,742,718	-	8,742,718
Float tails pre-leach thickener	2,667,732	-	2,667,732
Gold recovery	4,636,574	-	4,636,574
Finale tailing thickener & cyanide destruct	826,462	-	826,462
Reagents	1,230,621	-	1,230,621
Reagents (POX)	14,216,934	-	14,216,934
Water services	2,000,826	-	2,000,826
Water services (POX)	1,447,053	-	1,447,053
Air services	892,845	-	892,845
Air services (POX)	437,234	-	437,234
Process plant building	66,391,005	-	66,391,005
Total	**226,611,220**		**226,611,220**

21.1.6 Tailing storage facilities

The tailing storage facilities included the reclaim pumping station and pipeline for a total of $3.37M.

21.1.7 Environmental

The environmental capital costs include, all the preliminary work before operation such as treating existing contaminated soils, the building demolition of former Beattie Mine and house relocation or purchases.

Also included, are the costs related to the tailing management facility construction and all polishing ponds, the costs associated with the preparation of the waste, ore and overburden pile as well as the dam required to isolated the old tailing area with the lake.

Table 21.5 – Breakdown of environmental costs

Description	Pre-production ($)	Sustaining ($)	Total ($)
Preliminary work			
Existing contaminated soils	1,330,560	-	1,330,560
Building demolition of former Beattie mine	9,213,696	-	9,213,696
House relocation	1,400,000	-	1,400,000
Contingency (20%)	2,986,064		2,986,064
Sub-Total	**14,930,320**	**-**	**14,930,320**
Environment infrastructure			
Tailings pond (flottation)	14,794,434	13,619,368	28,413,802
Tailings pond (POX)	6,182,747	-	6,182,747
Process water pond	3,444,730	-	3,444,730
Polishing pond (flottation)	3,206,850	-	3,206,850
Polishing pond (POX)	1,245,598	-	1,245,598
Waste stockpile	1,129,624	-	1,129,624
Ore stockpile	148,827	-	148,827
Overburden stockpile	212,638	307,360	519,998
Process plant preparation	393,616	-	393,616
Pit drainage preparation	465,073	-	465,073
Waste dump dam	1,089,797	-	1,089,797
Contingency (20%)	6,462,786	2,785,346	9,248,132
Sub-Total	**38,776,718**	**16,712,074**	**55,488,792**
Total	**53,707,038**	**16,712,074**	**70,419,112**

21.1.8 Infrastructure and Process

A capital cost estimate for the Duparquet Project has been produced for two options. Option 1, the Concentrate option, consists of processing the ore to produce a flotation concentrate of gold for shipping to a smelter and treating flotation's tails by cyanidation for doré bars production. Option 2, the Pressure Oxidation (POX) option, uses flotation, pressurized oxidation and cyanidation of the flotation concentrate to produce doré bars on site.

There is an increase of $36.4M in direct capital costs between the Concentrate option and the POX option due to the additional processing equipment, building, and infrastructure required for the POX option. This also has an impact on the indirect capital costs, with an increase of $13.7M.

21.1.9 Scope of the Estimate

21.1.9.1 Scope definition

The scope of the Infrastructure and Process Facilities CAPEX covers all of the management, engineering, procurement, construction, commissioning, and start-up costs of the Duparquet Project's pre-production phase as well as sustaining capital costs incurred after the beginning of production.

21.1.9.2 Estimate classification

The estimate is classified as prefeasibility study level, which is defined by the AACE International (Association for the Advancement of Cost Engineering) as: *"[...] a comprehensive study of the viability of a mineral project that has advanced to a stage where the mining method, in the case of underground mining, or the pit configuration, in the case of an open pit, has been established and an effective method of mineral processing has been determined, and includes a financial analysis based on reasonable assumptions of technical, engineering, legal, operating, economic, social, and environmental factors and the evaluation of other relevant factors which are sufficient for a qualified person, acting reasonably, to determine if all or part of the mineral resource may be classified as a mineral reserve."* A prefeasibility study level cost estimate is the equivalent of the AACE Class 4 cost estimate.

The purpose of this prefeasibility study phase is to prepare a capital cost estimate with an accuracy of ±25%. In order to obtain this level of accuracy, the following have been done:

- Obtained the major equipment procurement cost based on suppliers' quotations;
- Obtained all-in construction labour rates from previous mining project studies and local contractors in the Abitibi-Témiscamingue area;
- Performed calculations of quantities to reflect the latest engineering documents issued;
- Confirmed unit prices based on previous mining project studies and local contractor quotes in the Abitibi-Témiscamingue area;

21.1.9.3 Estimate Presentation

It provides cost information for all sub-projects, area, disciplines, and activities/equipment.
The main contributors to the Infrastructure and Process Facilities CAPEX are Roche and Tenova-Bateman (for the POX option).
The estimate is divided into direct and indirect costs. The direct cost section of the estimate is divided into sub-projects, areas, disciplines, and cost items. Each sub-project is divided into areas representing different physical entities, be it a service, a building, a process, etc. Each area is divided by disciplines. Each discipline is responsible for evaluating the scope, quantities, and unit costs for each activity or equipment required for any given area.

21.1.10 Cost Information

For each individual cost item, cost data was entered for three categories: material, equipment, and installation/labour. Descriptions of the three categories are found below.

21.1.10.1 Material

"Material" includes all construction materials (concrete, steel, piping, electrical cables, etc.) and non-tangible items (earthworks).

21.1.10.2 Equipment

"Equipment" includes mechanical and HVAC equipment, platework as well as freight estimation.

21.1.10.3 Installation/Labour

All cost related to the man-hours required to install or perform a task is categorized as installation/labour. The purpose is to separate the installation cost, especially the required man-hours, from the material and equipment items in order to assess how many workers will be needed during the construction period. The cost of construction equipment (ex: crane) is estimated separately as an indirect cost.

Note that for some cost items, mostly for earthworks and concrete, the installation/labour cost is embedded in the material unit costs.

21.1.11 Design Basis - Reference Documents

The prefeasibility study level CAPEX is based on the engineering documents produced throughout the PFS. The main reference documents include, but are not limited to:

- Design criteria;
- Equipment list;
- Flowsheets, mass & water balance;
- Project location & site layout;
- General arrangements.

21.1.12 Units of Measure

International System (SI) units are used throughout the estimate. In some cases, imperial units might be used (ex: pipe diameter) but are converted to metric.

21.1.13 Currency Base Date and Exchange Rates

The base date of the cost estimate is November 1, 2013.
The estimate is expressed in Canadian dollars.
All duties and taxes are excluded from the capital cost
For reference, the currency conversions rates used during the preparation of the estimate were established by Clifton Star:

- 1 CAD = 0.95 USD (American dollar);
- 1 CAD = 0.74 EUR (Euro);
- 1 CAD = 0.94 AUD (Australian dollar);

For all materials and installation/labour, the cost information came from local vendors and/or contractors and was expressed in Canadian Dollars (CAD or C$). Most equipment quotations come from North America and are therefore in USD or CAD. Some cost sources come from Australia for the POX option and have been converted to CAD.

21.1.14 Direct Costs

Direct costs cover cost that is directly attributable to the cost item and/or activity. It covers equipment supply, material costs, and installation costs (labour, contractor's supervision and management costs, contractors' travelling and living allowances, contractors/suppliers administration and profits).

21.1.14.1 Mechanical - Equipment procurement

Based on the engineering documents produced for the PFS, an equipment list was created for all process equipment for both options (Concentrate and POX). Datasheets were prepared in order to request for quotations from equipment suppliers. All quotations received were budgetary. For minor equipment, historical data from previous similar projects and factorization of equipment size were used in order to complete the estimate. Allowances were used in some cases for small equipment when historical data was not available. The approximate breakdown of sourcing for the process equipment procurement is estimated as follows in Table 21.6.

Table 21.6 – Breakdown of equipment costs

Source	Percentage
Budget quotations	≈90%
Historical data	≈9%
Allowances	≈1%

The freight estimate is included in the direct equipment cost and corresponds to 5% of the direct equipment purchase cost.

21.1.14.2 Piping

Piping cost for the process facilities is divided into the crushing area, ore storage area and process plant. The costs for the crushing and ore storage areas are estimated based on an allocation for small piping, compressed air, etc. Cost for process plant piping is estimated with a factor of around 15% of the total process plant equipment cost. The cost of the tailings pipeline was estimated based on historical data from Roche. Diameter and quantities are based on process design criteria for diameter sizing and site layout for distance and topography calculations. The fresh water pipeline is assumed to be made of steel and is heat traced.

21.1.14.3 Electrical and instrumentation

Budget quotations from local suppliers were obtained for major electrical material. Unit prices for sections of MCCs and switchgears were established based on recent projects and used as per the electrical distribution design. Cable sizing and lengths were estimated based on the PFS general arrangements and were integrated to the load list to allow adequate estimate accuracy. Man-hours for installation of the equipment, services, grounding, cable trays, and cables were based on similar projects and from standard working methods and recognized estimating tools.

For the automation and instrumentation, a ratio was used to estimate the cost of material and installation.

21.1.14.4 HVAC - Services

The methodology for estimating the heating, ventilation, and air conditioning cost for the process plant is based on cost data from previous similar projects located in northern Québec. For the auxiliary buildings, the cost has been estimated based on the typical requirement based on the area of the building.

21.1.14.5 Construction labour rates

The all-in rate for the Duparquet Project is based on recent similar projects in the area. The all-in rate was validated by consulting the Québec Construction Association (ACQ) hourly rate publication for January 2013. The chosen all-in rate for the PFS CAPEX is an average of $122/hour for all trades. This is based on a 60-hour per week schedule, corresponding to 6 days a week at 10 hours per day (Monday to Saturday). It involves 40 hours per week at regular rate, and 20 hours per week double time.

No productivity factor was included in the all-in labour rates. The productivity factor is estimated in the number of man-hours.

No productivity factor has been applied to take into account the local conditions for construction work in Abitibi-Témiscamingue region.

No separate productivity factor was used for outside winter work. For this estimate, it is assumed that minimal work will take place outside during the worst winter months. Winter work usually results in a lower productivity in the range of 75%.

<u>All-in labour rates – Inclusions</u>

The all-in construction labor rate includes mobilization and demobilization of contractor's personnel, room and board, living allowances, transportation costs, safety PPE, contractor's indirect personnel (foreman, general foreman, superintendent), contractor's site supervision personnel, head office overhead, expenses, insurance, contractor's profit, consumables, and contractor's temporary facilities. It also includes the contractor's construction vehicles, small tools and consumables.

<u>All-in labour rates – Exclusions</u>

Excluded items are estimated separately in the number of man-hours and include daily pre-start safety meetings, weekly tool box meetings, safety induction sessions, special safety trainings, and other time consuming activities other than real installation hours.

Mobilization and demobilization of specialized equipment, office trailers and lunch rooms, crane rentals, project management complex and/or facilities, construction's guard house and security personnel as well as first aid station are all included in the Indirect Costs section of the estimate.

21.1.14.6 Unit prices

The unit prices for concrete and steel are commonly the most important costs related to infrastructure work. These unit prices can vary widely depending on the project's location and overall construction activities in the region. The major unit prices used for this study are presented in Table 21.7.

Table 21.7 - Major unit price summary

Earthworks	**Price**
Deforestation	\$2000/ha
Stump clearing & 200 mm soil stripping	\$10/m^3
Second class backfill material	\$5/m^3
Excavation second class	\$5/m^3
Drilling and rock blasting	\$5/t
Concrete	**Price**
Concrete (including form work & rebar)	\$1,100/m^3
Structural Steel	**Price**
Structural steel	\$4,950/t
Labor All-In Rates	**Price**
General (all trades)	\$122/hr

21.1.15 Indirect Costs

Indirect Costs cover costs that are required for completion of the installation but are not directly attributable to the cost item / activity.

21.1.15.1 Detailed Engineering, Procurement, Construction Management

For the purpose of this PFS, a percentage of 15% of Direct Costs is used to cover the cost of the detailed engineering, procurement, and construction management services. It also includes office operation expenses during construction:

- Detailed engineering – process plant, tailings pond, infrastructure;
- Vendor representative assistance;
- QA/QC consultant and other miscellaneous consultation;
- Project construction personnel salaries;
- Site supervision consultants;
- Health and safety: coordinators and supplies;
- Vehicle rental;
- Surveying support and equipment;
- Site office supplies and expenses;
- Site communication (phone, radio, cell phone, internet, network);
- Site electrical consumption;
- Site gasoline/fuel/oil;
- Site trash removal and sewage disposal;
- Site snow removal and road maintenance;
- Computers, hardware and software, office furniture.

21.1.15.2 Temporary services and site facilities

The temporary site facilities cost is estimated as a factor of 1.5% of direct costs. It includes the project management team's site installation requirements for project execution, such as:

- Project management team office complex (trailers) supply and set-up;
- Toilet facilities for construction;
- Vehicles;
- General site clean-up;
- Waste disposal;
- Site security guards;
- Computer, phone and communication system, and office supplies;
- Contingencies for temporary electrical distribution and maintenance for trailers and construction;
- Dismantling of facilities at the end of the contract.

21.1.15.3 Common site construction equipment

The common site construction equipment cost is estimated as a factor of 1.5% of direct costs and corresponds to the rental of specialized equipment such as cranes, forklifts, etc. for the duration of construction (24 months), including mobilization and demobilization.

21.1.15.4 Maintenance during construction

Site maintenance and mobile equipment maintenance cost during the construction period is estimated as a factor of 0.5% of direct costs.

21.1.15.5 Start-up and commissioning

The start-up and commissioning cost is estimated in the economic analysis. The cost is included in the Capitalized Operation Cost section of the expense schedule. It includes the processing equipment cold commissioning, as well as the cost for external contractor assistance at start-up.

21.1.15.6 Vendors' costs

This item covers the cost for the vendors' representative for:

- Erection assistance;
- Start-up and commissioning assistance;
- Training.

It is estimated at 2% of direct equipment purchase cost.

21.1.15.7 Construction insurance

A percentage of 0.5% of direct costs was used to cover for the cost of construction insurance.

21.1.15.8 Owner's Cost

The owner's cost is included in the financial analysis. It covers the owner's incurred costs for the pre-production period. Where costs incurred by the owner prior to the project approval date are to be capitalized on the project, they are not included in the PFS CAPEX estimate but are presented in Section 26.0 – Recommendations.

During the pre-production period, the owner's construction team is involved in all aspects of the Project. Items like insurance, permits and certifications, performance bonds, taxes & duties, land acquisition, pre-production salaries and benefits, training expenses, consultants, security, human resources, public relations, environmental follow-up, health & safety operations, etc. are all under the owner's direct responsibility. This involves a team and considerable cost associated with it. Owner's costs also include head office support for legal, accounting, engineering and other related support cost that will be charged under the Duparquet Project by the Clifton Star Head Office. Owner's costs are estimated in the economic analysis in Section 22 as part of Capitalized Operating Costs in the expense schedule.

21.1.15.9 Mill first load and two years of spare parts (part of working capital)

The mill first load includes the cost of the first fill of reagents' reservoirs, grinding media, lubricants, etc. Spare parts cost includes the cost of initial inventory of spares for the different equipment on site. These items are included in the Working Capital presented in the economic analysis in Section 22 of the Report. Therefore, these items are excluded from the Capital Cost estimate.

21.1.16 Contingency

Contingency is an amount of money allowed in an estimate for costs which, based on past experience, are likely to be encountered, but are difficult or impossible to identify at the time the estimate is prepared. It is an amount which is expected to be expended during the course of the Project. Contingency does not include scope changes, force majeure, labour strikes/wobbles or labour availability.

Contingency for the PFS is divided into three (3) categories: direct costs contingency; construction indirects contingency; and owner's costs contingency. A contingency of 15% has been applied to the construction indirects and owner's costs.

The direct costs contingency has been determined based on the level of confidence for different cost items / areas. For example, earthwork-related costs have mostly been assigned a 20% contingency, since the level of definition, geotechnical data, etc. was limited. Process equipment purchase costs coming from suppliers' quotations have been assigned a 10% contingency, since the level in confidence in the price is greater. Installation, freight, and all other cost items all have been looked into to determine an appropriate contingency. The result for direct cost contingencies for the Concentrate option and POX option are respectively 14.70% and 14.68%. Total contingencies for the Project's Concentrate option and POX option are respectively 14.75% and 14.73%.

In the POX option, the contingencies and indirect costs total $98.7 million of the pre-production costs and represent 26% of the costs.

21.1.17 Escalation

No escalation is included in the CAPEX.

21.1.18 Assumptions and Qualifications

The following items are assumptions and qualifications concerning the capital cost estimate:

- There will be no major delays in the Project such as those associated with environmental permitting;
- There is sufficient accommodation available in the Duparquet area for manual and non-manual workers during construction and operation as the cost of a camp is NOT included in the estimate;
- It will be possible to insure safety without incurring significant loss of efficiency;
- Pre-production work will take place over a period of around 24 months.

21.1.19 Exclusions

The following items are not included in the capital cost estimate:

21.1.19.1 Feasibility study

Cost incurred between the end of the PFS and the beginning of the detailed engineering was excluded from the scope of the PFS estimation. Recommendation of additional work required to bring the Project to the detailed engineering phase is treated in Section 26 – Recommendations. It includes but is not limited to additional testwork, drilling and feasibility study.

21.1.19.2 Labour

- Allowance for industrial disputes or lost time arising from industrial actions;
- Allowance for special incentives (schedule, safety, or others).

21.1.19.3 Environmental and community relations

- Asbestos, lead paint, and any other hazardous material removal;
- Cost for removal of sheet metal with lead paint;
- Allowance for future designation of hazardous classification areas;
- Provisions for the cost of remedial actions with respect to contaminated soil, lead contaminants, and archaeological historical findings;
- Environmental studies, permitting, and mitigation beyond the tabling of the Environmental and Social Impact Assessment (ESIA);
- Plant closure and rehabilitation costs (excluded from initial CAPEX, included in Sustaining Capital Expenditure).

21.1.19.4 Legal costs and taxes

- Legal costs (head office charges);
- Force majeure issues;
- Licence and royalty fees;
- All owner payable taxes;

- Permits / cost of permits.

21.1.19.5 Financing costs

- Owner's cost prior to project approval;
- Any requirements related to project financing;
- Financing fees;
- Working capital;
- Cost changes due to currency fluctuation;
- Sunk cost;
- Resettlement / relocation costs;
- Project interest and financing cost during construction;
- Other Owner's costs not described above and not included in the CAPEX indirect costs.

21.1.19.6 Operating and maintenance costs

- Operating and maintenance costs are provided separately in the OPEX;
- Any operational insurance such as business interruption insurance and machinery breakdown.

21.2 Operating Cost Estimate

21.2.1 Scope and Methodology

21.2.1.1 Scope

The scope of the Operating Cost Estimate (OPEX) covers all costs related to the operation of the Duparquet Project and includes mining, ore processing, tailings management, on-site water management, general and administration (G&A) fees, as well as infrastructure and services. The scope covers the Duparquet Project's yearly operation for a typical production year of 3.65 Mt of ore milled.

21.2.1.2 Methodology

The methodology used to estimate the operating cost consisted of using data from existing similar mining operations and quotations received from local contractors, as well as consulting reference publications. Roche, InnovExplo, and Tenova-Bateman's experience with estimating operating costs for mining projects also contributed to estimating parts of the operating costs when detailed information was limited.

21.2.1.3 Basis of estimate

The operating cost estimate for the processing cost is based on tonnage per year for a typical full production year. Other costs have been estimated on zero-based costing and have been built based on the following parameters:

- Annual tonnes milled 3,650,000 metric tons per year
- Electricity cost (Hydro-Québec Tarif L) $12.36 per subscribed kW per month
  3.04 ¢/kWh consumed
- Credit for 120 kV transformation 2.55 ¢/kWh
- After tax fuel cost: $0.99 /l
- After tax gasoline cost: $0.99 /l
- Fresh water consumption: $0.07 /$m^3$
- Municipal & school taxes: $1.27 per $100 evaluation
- Labour costs include a 30% fringe, plus percentages for overtime, production bonuses, general bonuses and holidays which were estimated based on experience for each worker.
- All costs presented in this section are in Canadian Dollars (CAD) per year and CAD per metric tonne.
- Marketing costs are excluded from the OPEX and treated in the Financial Model.

21.2.2 Cost Breakdown Structure

The OPEX cost breakdown for the PFS is divided into five (5) main categories: general and administration (G&A); processing; mining; environmental; and overburden removal. The G&A category include the costs of technical services and administration. Open pit mining costs include drilling, blasting, loading, hauling, auxiliary, and general mine maintenance. The processing category includes manpower, the cost to process ore from the pit, and the cost to process the old tailings. The environmental category includes manpower and departmental costs.

Table 21.8 – Summary of total operating costs

Description	Total cost estimate (production period)		Unit cost	
			($/t ore)	($/oz Au)
General and administration	$	95,457,201	2.46 $/t	56.72 $/t
Processing cost	$	608,136,366	15.66 $/t	361.35 $/t
Mining cost	$	707,899,014	18.23 $/t	420.63 $/t
Environmental monitoring	$	6,900,077	0.18 $/t	4.10 $/t
Overburden removal cost	$	15,966,057	0.41 $/t	9.49 $/t
Total	**$**	**1,434,358,715**	**36.94 $/t**	**852.28 $/t**

21.2.3 General & Administration

General and administration (G&A) operating costs cover all costs incurred that are not directly attributable to the mining and/or processing operations. It includes all the administration staff, contracts, general cost, municipal taxes and technical services. A summary of the total estimated annual G&A operating cost is presented in Table 21.9.

Table 21.9 – General and administration cost summary

Item		Total cost estimate (production period)	Unit cost estimate
Administration	Manpower	$ 25,721,068	0.66 $/tonne milled
	Departemental cost	$ 38,257,299	0.99 $/tonne milled
Technical services	Manpower	$ 28,632,728	0.74 $/tonne milled
	Departemental cost	$ 2,846,106	0.07 $/tonne milled
Total G & A cost		**$ 95,457,201**	**1.40 $/tonne milled**

21.2.3.1 Administration

The administration cost includes salaries and material required for the department. It also includes all the site insurance, most of the information technologies services required for the Project, all consultant costs, and the municipal taxes.

The total cost for the production period is estimated at $63.98M. This cost represents $1.62 per tonne milled.

21.2.3.2 Technical services

Technical services include two (2) departments: engineering and geology. Both services include manpower salaries and the cost of materials required for the department. The total technical services cost is estimated at $31.48M. This cost represents $0.80 per tonne milled.

21.2.4 Processing Operating Costs

The processing operating costs include all costs applicable to the operation of the processing facilities and tailings storage facilities. The scope of the processing operating costs includes all processing activities from the crushing of ore to the gold concentrate and/or gold doré products. It is comprised of the processing manpower, energy, fresh water, reagents, consumables, as well as other processing costs.

For a typical year at the design processing rate, the operating costs for both options of the process plant are summarized in Table 21.10. These are subdivided into the following components: manpower; energy (electrical power and fuel); fresh water; reagents and consumables; and other processing elements. These costs were derived from information provided by suppliers and from the Roche database, or were factored from similar operations.

Table 21.10- Process operating cost summary (OPEX)

Activity	CONC Option		POX Option	
	Annual Cost (CAD/y)	Cost per tonne milled (CAD/t)	Annual Cost (CAD/y)	Cost per tonne milled (CAD/t)
PROCESS				
Man power - process	6 096 141	1,67	6 304 501	1,73
Energy	7 103 574	1,95	9 020 782	2,47
Fresh water	11 155	0,003	11 155	0,003
Reagents	7 896 980	2,16	15 625 090	4.28
Consumables	21 059 647	5,77	25 031 500	6,86
Other processing	1 760 000	0,48	1 760 000	0,48
Total PROCESS OPEX Cost	**43 927 497**	**12.03**	**57 753 028**	**15.82**

A value of $16.63/t was used in the POX option economic scenario. This value includes an operating cost of $15.82 as presented in Table 21.10 plus $0.82/t for general costs related to mill activity. Since the tailings will not need crushing, a cost of $12.63/t was considered for that portion of the mineralized material.

The annual operating cost for the POX option is higher than the Concentrate option due to additional processing manpower required as well as additional electrical, reagent and consumable consumptions.

21.2.4.1 Processing facilities manpower operating costs

For the Concentrate option, the processing facilities operations and supervision group (72 employees) is composed of a mill superintendent, metallurgists, metallurgical technicians (wet lab), a chief chemist, lab technicians (assay lab), mill supervisors, equipment operators as well as a maintenance group.

For the POX option, the processing facilities operations and supervision group (74 employees) is composed of one (1) additional electrician and one (1) additional mechanic to compensate for the additional pieces of equipment.

21.2.4.2 Energy costs

Duparquet operations will be powered by the following four (4) sources of energy: electricity, diesel fuel, gasoline, and propane. Most fixed equipment will be powered by electricity. Generators, most mobile equipment and the elution process for the POX option will use diesel fuel. Gasoline will be kept for small pick-up trucks, small generators, and hand tools. Propane will be used for the elution process for the Concentrate option. The consumption of each energy source and the basis for operational cost evaluation is described below. Details are shown in Table 21.11.

Table 21.11 - Energy cost summary

Energy Cost Items	Annual Cost CONC Option ($/y)	Annual Cost POX Option ($/y)
Electrical power	6,716,903	8,634,319
Diesel fuel	314,325	334,983
Gasoline	51,480	51,480
Propane	20,867	0
Total Energy Costs	**7,103,574**	**9,020,782**

Electricity consumption

For practical reasons, the electrical power consumption for the entire site is based on the process plant load because it typically represents 95% of the energy consumption. Electricity consumption is based on connected and running power (kW) for the entire site. The total electrical power requirement is based on the equipment load list, which is derived from the mechanical equipment list.

Based on the electrical load list, the estimated electrical power consumption for the Duparquet Project is 21,460 kW for the Concentrate option and 27,590 kW for the POX option, both of which are above the 5,000 kW threshold that qualifies for Hydro-Québec's "Tarif L" program.

As of April 2013, the "Tarif L" program states that Hydro-Québec charges $12.36 per subscribed kilowatt agreed to with Hydro-Québec on a montly basis and 3.04¢ per kWh consumed.

Diesel consumption

Diesel cost per litre is established at $0.99/L and corresponds to the cost after tax credit/refund from the government plus transportation fee. Diesel consumption for mobile support equipment (pick-up trucks, small loader, etc.) has been estimated based on 6 mobile vehicles consuming 50 L/day per piece of equipment for 365 days a year. Diesel consumption for fixed equipment (emergency gensets, diesel heaters, etc.) has been estimated at 4,000 L/week. Diesel is used for carbon regeneration and melting furnace for the POX option.

Gasoline consumption

Gasoline cost per litre is established at $0.99/L and corresponds to the cost after tax credit/refund from the government plus transportation fee. Gasoline consumption allowance for potential gasoline-powered pick-up trucks, small generators, pumps, vibrating plates, etc. has been estimated at 1,000L/week.

Propane consumption

Based on supplier's quotation for the carbon regeneration kiln and melting furnace, propane consumption is close to 21,000 liters per year. Propane cost per litre is estimated at $1.00/L.

21.2.4.3 Fresh water

Fresh water will come from the old underground mine located southeast of the process facilities site. The cost for fresh water consumption comes from Québec's regulation regarding the water usage royalty[5]. The rate is $0.07/m^3 of fresh water used. Fresh water consumption is estimated at 20 m^3/hr. The anticipated annual cost for the water royalties is $11,155.

21.2.4.4 Reagents

The annual consumption of reagents has been based on pilot plant and laboratory testing done throughout the prefeasibility study. The quantities have been scaled up to reflect the full scale process plant mass and water balances. Reagent unit prices came from various manufacturers and reflect annual quantities required as well as actual market price. Table 21.12 details the cost of reagents.

[5] *Règlement sur la redevance exigible pour l'utilisation de l'eau* (L.R.Q., c. Q-2, a. 31, 46, 109.1 and 124.1, D. 1017-2010, a. 5)

Table 21.12 – Detailed reagent operating costs

Reagents	Annual Consumption CONC Option (tpy)	**Annual Cost** CONC Option **($/y)**	Annual Consumption POX Option (tpy)	**Annual Cost** POX Option **($/y)**
Flotation				
MIBC (frother)	99	281 218	112	318 370
Potassium Amyl Xanthate PAX (collector)	441	1 274 349	521	1 507 854
R208 (collector)	77	302 702	218	858 054
Neutralization				
Lime			3 127	625 326
Limestone			66 623	4 663 585
Lime Boil				
Lime			18 906	3 781 210
Leach/CIP				
Activated carbon			7	24 740
Lime			1056	211 258
Sodium cyanide			810	2 752 231
Leach/CIL				
Activated carbon	131	488 001		
Lime	1 900	379 900		
Sodium cyanide	98	351 185		
Carbon Stripping/Regeneration				
Caustic soda	147	136 322	58	54 158
HCl	20	5 268	336	88 006
Sodium cyanide			13	45 127
Thickening				
Flomin 912 (flocculant)	133	475 927	92	329 928
Cyanide Destruction				
Lime	2 620	524 000		
CuSO4	112	277 405	0.03	83
Sulphur dioxide	3 936	6 400 704	423	365 155
Total Reagent Costs		**7 896 980**		**15 625 090**

21.2.4.5 Consumables

Consumables are divided in four sub-groups: liners, grinding media, supplies, and lubricants. For the POX option, there is a monthly allowance for the oxygen plant operation service. The increased supplies cost for the Concentrate option represents the shipping supplies for the concentrate (bulk bags). Details are shown in Table 21.13.

Table 21.13- Summary of consumable costs

Consumables items	Annual Cost CONC Option ($/y)	Annual Cost POX Option ($/y)
Liners	1,890,500	1,890,500
Grinding Media	15,381,000	15,381,000
Supplies	3,388,147	2,500,000
Lubricant	400,000	400,000
O2 Plant - Monthly Allowance	0	4,860,000
Total Consumables Costs	**21,059,647**	**25,031,500**

21.2.4.6 Other processing costs

Other processing costs, such as mechanical, electrical, instrumentation and piping maintenance contracts given to outside contractors, as well as wet laboratory supplies, emergency genset rental and surface equipment maintenance costs, have been estimated based on Roche's experience. The emergency genset rental and operating cost was obtained through a quotation received from a supplier. Total annual other processing costs are shown in Table 21.14.

Table 21.14 – Details of other processing costs

Other Processing Cost Items	Annual Cost CONC & POX Option ($/y)
Mechanical Contractor Maintenance	360 000
Electrical & Inst. Contractor Maintenance	108 000
Piping Contractor Maintenance	108 000
Laboratory Supplies (Wet Lab)	60 000
Laboratory Supplies (Assay Lab)	180 000
Process - Emergency Genset Rental	894 000
Surface Equipment, Other, Misc…	50 000
Total Other Processing	**1 760 000**

21.2.5 Mining costs

The open pit mining cost is separated into six (6) categories: drilling; blasting; loading and hauling; auxiliary; general mine; and maintenance. The loading and hauling category has three (3) components: loading and hauling of the overburden; of the rock (waste and ore); and of the old tailings. Table 21.15 presents a summary of the open pit mining costs.

Table 21.15 – Open pit mining cost summary

Item		Total cost estimate (production period)	Unit cost estimate
Drilling		$ 72,241,639	0.23 $/tonne mined
Blasting		$ 150,630,026	0.47 $/tonne mined
Loading & Hauling	Overburden	$ 15,966,057	0.84 $/tonne overburden
	Rock (waste & ore)	$ 295,504,477	0.93 $/tonne mined
	Old tailings	$ 14,999,204	3.76 $/tonne tailings
Auxilliary		$ 98,425,837	0.31 $/tonne mined
General Mine		$ 22,535,033	0.07 $/tonne mined
Maintenance		$ 53,562,798	0.17 $/tonne mined
Total Operating mining cost (waste & ore)		**$ 692,899,810**	**2.17 $/tonne mined**
Total operating mining cost (overburden)		$ 15,966,057	0.84 $/tonne overburden
Total operating mining cost (old tailings)		$ 14,999,204	3.76 $/tonne tailings

21.2.5.1 Drilling

The drilling cost includes the salaries of the drillers, the drill helpers and the sharpeners. The cost also includes the cost of the drill operating costs and the consumable cost for equipment required to drill holes, such as bits, rods and hammers. Table 21.16 presents a summary of the drilling costs.

Table 21.16 – Drilling costs summary

	Total cost during production	Unit cost per tonne mined	Unit cost per tonne milled
Manpower salaries	26,567,937	0.08 $/t	0.68 $/t
Equipment operating cost	28,259,611	0.09 $/t	0.73 $/t
Consumable cost	17,414,091	0.05 $/t	0.45 $/t
Total	**72,241,639**	**0.23 $/t**	**1.86 $/t**

21.2.5.2 Blasting

The blasting cost includes manpower, the equipment operating cost, blasting consumables and aggregates for the stemming. All costs except for the aggregates were provided by Orica. The manpower cost is estimated at $100,320/month and includes all manpower required for blasting. The operating cost is estimated at $44,536/month and includes the equipment operating cost, the site cost and the MMU (Mobile Manufacturing Unit). Consumable costs are variable because they depend on the waste and rock that need to be blasted each year. Consumable costs include the cost of explosives, detonators, boosters, etc. Table 21.17 presents a summary of the blasting costs.

Table 21.17 – Blasting costs summary

	Total cost during production	Unit cost per tonne mined	Unit cost per tonne milled
Manpower salaries	12,038,400	0.04 $/t	0.31 $/t
Equipment operating cost	5,344,290	0.02 $/t	0.14 $/t
Consumable cost	133,070,407	0.42 $/t	3.43 $/t
Aggregates	176,929	0.001 $/t	0.005 $/t
Total	**150,630,026**	**0.47 $/t**	**3.88 $/t**

21.2.5.3 Loading and hauling

Loading and hauling cost are separated in three (3) categories: loading and hauling cost of the overburden, of the rock (ore and waste), and of the old tailings.

Overburden

The overburden removal cost includes the manpower salaries and the equipment operating cost. A total cost of $15.97M is estimated for the production period. This cost represents $0.84 per tonne of overburden.

Rock (ore and waste)

The loading and hauling cost includes manpower and the equipment operating cost. The manpower cost includes the salaries of the shovel operators, the loader operators, the truck operators, the mine helpers and the trainers. The equipment operating cost includes the operating costs of the trucks, the shovels and the loader. The loading and hauling costs are separated into two categories: loading and hauling the rock from the pit to the mill or stockpiles and loading and hauling the ore from the stockpile to the mill. Table 21.18 presents a summary for both categories.

Table 21.18 – Loading and hauling costs summary

Loading & hauling		Total cost during production	Unit cost per tonne mined	Unit cost per tonne milled
Rock and Ore from Pit to stockpile or mill	Manpower	63,766,848	0.20 $/t	1.86 $/t
	Equipment operating cost	226,561,265	0.71 $/t	6.60 $/t
Ore from stockpile to mill	Manpower	1,204,175	0.004 $/t	0.04 $/t
	Equipment operating cost	3,972,189	0.01 $/t	0.12 $/t
Average cost for loading & hauling		**295,504,477**	**0.93 $/t**	**8.60 $/t**

Tailings

The loading and hauling of tailings will be done by a contractor. A unit cost of $3.76 per tonne of tailings was provided by Fournier. The total cost to load and haul the old tailings from their current place to the mill is estimated at $14,999,204 for the production period.

21.2.6 Auxiliary

The auxiliary cost includes all costs related to haul road maintenance and the cost of equipment support. It includes the salaries of the operators of the equipment, and the equipment operating costs. The total cost is estimated at $98.43M for the production period. This cost represents $0.3 per tonne mined or $2.50 per tonne milled.

21.2.7 General mine

The general mine cost includes all supervision salaries related to the mining operation, departmental expenses, crane operation and rental, and aggregates for road maintenance. Table 21.19 presents the summary of the costs.

Table 21.19 – General mine costs summary

	Total cost during production	Unit cost per tonne mined	Unit cost per tonne milled
Manpower salaries	16,537,978	0.05 $/t	0.43 $/t
Departmental expenses	3,690,500	0.01 $/t	0.10 $/t
Aggregates cost	1,909,400	0.01 $/t	0.05 $/t
Total	**22,137,878**	**0.07 $/t**	**0.57 $/t**

21.2.8 Maintenance

The maintenance cost includes all salaries related to maintenance such as mechanics, welders, maintenance supervisors, etc. It also includes the departmental expenses and the equipment operating cost required for the maintenance employees. Table 21.20 presents the summary of the maintenance cost.

Table 21.20 – Maintenance costs summary

	Total cost during production	Unit cost per tonne mined	Unit cost per tonne milled
Manpower salaries	44,827,446	0.14 $/t	1.15 $/t
Equipment operating cost	3,027,532	0.01 $/t	0.08 $/t
Departemental expenses	5,707,820	0.02 $/t	0.15 $/t
Total	**53,562,798**	**0.17 $/t**	**1.38 $/t**

21.2.9 Environmental cost

The environmental cost includes all costs related to the monitoring of air quality, noise, water quality and ground water. It also includes manpower salaries and departmental expenses. The total cost during production is estimated at $6.9M and represents $0.02 per tonne mined (or $0.18/tonne milled).

21.2.10 Manpower operating costs

A total of 339 employees will be needed for the Duparquet Project. A summary of the total estimated annual manpower is presented in Table 16.11. Manpower salaries have been estimated based on existing mining operations in the Abitibi-Témiscamingue area and were benchmarked with the results of the Canadian Mine Salaries, Wages and Benefits 2012 Survey[6].

The Duparquet Project manpower work group has been divided into three (3) subgroups: mine employees, process facilities; employees and administration Staff. Table 21.21 summarizes the manpower costs for both options. Two (2) additional employees are required for the POX option to account for the additional maintenance workload resulting from additional pieces of equipment.

Table 21.21 - Manpower costs summary

Activity	Number of Employees CONC Option	Annual Cost (CAD/y)	Number of Employees POX Option	Annual Cost (C$/y)
Mine	212	21,784,752	212	21,784,752
Processing facilities	72	6 096 141	74	6,304,509
Administration	53	5,605,505	53	5,605,505
Total Man Power	**337**	**33,486,398**	**339**	**33,694,766**

21.2.11 Departmental cost

The departmental cost includes general costs that cover the following recurring items: site insurance, head office backcharge, public relations, environmental services, consultants, legal and accounting fees, buildings maintenance, training expenses, communication, summer students and grants, safety equipment, as well as other miscellaneous costs. These costs were attributed specifically to each department. Details are presented in Table 21.22.

[6] Canadian Mine Salaries, Wages & Benefits, 2012 Survey Results, compiled by Krista Noyes Salzer, Infomine USA, Inc.

Table 21.22 – Departmental cost

Items		Cost	
Office Supplies	General Material	240	$/person/year
Communication (Telephone, Internet, etc..)	Equipment Rental	12,000	$/year
	Communications Expenses	60,000	$/year
	Software Support	6,000	$/year
Admin Information Technologies (IT) Services	Equipment Rental	60,000	$/year
	Faxes/Photocopiers/Printers	6,000	$/year
	Network Costs	6,000	$/year
	Computers	12,000	$/year
	Server Hardware Costs	3,600	$/year
	Software & Licences	72,000	$/year
	Computers Supplies	360	$/person/year
Site Insurance	Liability	480,000	$/year
	Infrastructure	480,000	$/year
	Environment	480,000	$/year
Vehicle	Registration & Insurance	1,200	$/ truck/year
	Light Vehicle Maintenance	30,000	$/truck/year
Association Memberships	Professional Memberships	300	$/person/year
Public Relations	Community Relations	200,000	$/year
Recruiting	Recruiting	50,000	$/year
	Promotional Items	40,000	$/year
	External Sponsorship	25,000	$/year
Safety Equipment/Supplies and Related Costs	Personnal Protection Equipment	600	$/person/year
Worker's Social Activities	Recreational Activities	60,000	$/year
R&D, Summer Students, Grants, etc.		300,000	$/year
Legal & Accounting Fees	Legal	8,333	$/month
Consultants	Consultants - General	200,000	$/year
Travel & Seminars	Travelling	150	$/staff/month
	Airfare	100	$/staff/month
	Meals & Entertainment	100	$/staff/month
Training Expenses		1	% of salary
Infirmery	Supplies & Related Costs	36,000	$/month
Buildings	Maintenance & Supplies	360,000	$/year
Environmental Services & Supplies	Air Quality Monitoring Material & Equipment	10,000	$/year
	Analytical Costs - External Lab	25,000	$/year
	Noise Monitoring Reporting	15,000	$/year
	Water Quality - Off-site Analytical Costs	40,000	$/year
	Update to Site Water Balance - Consultant Input	200,000	$/year
	Lab Cost and Sampling (incl. equipment)	10,000	$/year
	Fish Habitats and Wetlands	10,000	$/year
	Annual Geotechnical Inspection & Reporting	20,000	$/year
	Decontamination of Contaminated Soil	30,000	$/year
	Shipping Hazardous Waste	15,000	$/year
	Materials	6,000	$/year
Miscellaneous (Small Supplies)	Administration	15,600	$/year
	Technical Services	7,200	$/year
	Environment	6,000	$/year
	Mine	24,000	$/year
	Maintenance	36,000	$/year
	Mill	12,000	$/year
Municipal tax		0.013	$/100$ value

22. ECONOMIC ANALYSIS

A pre-tax and after-tax cash flow projection has been generated from the life-of-mine (LOM) schedule according to the capital and operating cost estimates. It has been made in constant 2013 money terms and in Canadian currency unless stated otherwise, with no allowance for inflation or escalation. The net cash flow has been discounted for the purposes of calculating the net present value (NPV). A base discount rate of 5% per year has been selected as most likely to represent a low capital expense gold project in a mining-friendly environment. Future annual cash flow estimates are based on grade, gold recoveries and cost estimates, as previously discussed in this Report.

The PFS considered two possible processing scenarios. Cash flow models were created for both options. The strategy of using the Pressure Oxidization (“POX”) process generated the highest financial return and, as a result, the POX process is favoured in the PFS and presented in this section as the base case of the PFS.

The undiscounted pre-tax cash flow totals $493.19M over the 11-year mine life and the payback period is 4.3 years.

A summary of the base case cash flow model is given in Table 22.1. LOM totals for undiscounted and discounted cash flows are also provided. Table 22.1 shows that the pre-tax net present value of the project cash flow at a discount of 5% per year is evaluated at approximately $222M and a pre-tax internal rate of return of 15.11%. The average cash cost of production equates to US$775/oz gold.

Table 22.1 – Cash flow analysis summary

Parameters		Results
Gold Price		1,300 US$/oz
Foreign exchange rate		1.10 : 1.00 (CAN/USD)
Mineable reserves		35,2 Mt @ 1.56g/t Au; 1.89 g/t Ag
Old tailings		4,1 Mt @ 0.93 g/t Au; 2.40 g/t Ag
Recovered Gold	From mine	1.6 Moz
	From old tailings	0.1 Moz
	Total	1.7 Moz
Recovered Silver	From mine	1.9 Moz
	From old tailings	0.3 Moz
	Total	2.2 Moz
Average annual gold production (ounces):		173,000 oz (first 5 years) 158,000 oz (Average 11 year)
Total waste		291 Mt
Total OVB		23.4 Mt
Mine life (excluding 4 years of pre-production)		11 years
Daily mine production		10,000 tpd
Metal recovery Au	Mine	90.10%
	Old tailings	83.90%
Pre-production capital		394M$
Sustaining capital (excluding 24.5M$ for closure cost)		118M$
Average operating cost		36.94 C$/tonne milled
Average total Site Cash Cost (US$/ounce)		775 US$/oz Au
Average total All in Cost, Average (US$/ounce) LOM		1042 US$/oz Au
Net cashflow		493M$
Pre-tax NPV (5%)		222M$
Pre-tax IRR		15.11%
After-tax NPV (5%)		135M$
After-tax IRR		12.06%
Payback period		4.3 years

**All amounts in Canadian dollars unless stated otherwise.*

22.1 Gold Price Forecast

The long-term price used in this study is US$1300/oz, at an exchange rate of 1.1 CAD/USD. The gold price has been considered constant for the mine life of the Duparquet Pit, which is spread over eleven (11) years.

A selling cost of $3/oz was applied to the revenue.

22.2 Silver content

No mineral estimate was performed to determine the silver content of the Duparquet deposit because very few silver assays were available. At the request of Clifton Star, InnovExplo used a predetermined silver content to calculate the cash flow in the current PFS. The value was derived from the 2009 tailings sampling program, and the 2013 12 t composite bulk sampling program and the 238 individual samples, representing the “Master Holes” for the bulk sampling program that were analysed for silver. Details justifying the silver grade considered in the cashflow are presented in Section 24.

22.3 Royalties

No royalties affect the Duparquet Project.

22.4 Income taxes

Income taxes are calculated in accordance with the federal and provincial tax legislations relating to mining companies. The calculations were estimated by Lucie Chouinard of Raymond Chabot. The federal income tax rate is 15% and the combined provincial income tax rate is 11.9%.

22.5 Mining duties

Québec mining duties are calculated in accordance with Bill 55, which contains amendments to Québec’s *Mining Tax Act* and received its first reading in the Québec legislature on November 12, 2013. Under the new regime in the *Mining Tax Act*, mining operators in Québec will be required to pay the higher of a new minimum mining tax applied to the value of the ore at the mine shaft head and a progressive tax on excess profits. The new mining tax is introducing progressive mining tax rates ranging from 16% to 28% (replacing the single tax rate of 16%), and a minimum mining tax based on the mine-mouth output value is used.

The effective rate of this tax on mining profits will start at the existing 16% rate for mining companies with a profit margin of 35% or less, but rising to 17.8% for mining companies with a profit margin from 35% to 50%, and reaching as high as 22.9% for mining companies with a profit margin of more than 50%. The profit margin will be calculated on the operator’s mining profit divided by the total of the gross value of annual output for all the mines it operates. Therefore, the higher a mining corporation’s profit margin, the higher the mining tax.

22.6 Working Capital

No provision has been made for working capital, assuming this cost would be covered by financing fees.

22.7 Residual Value

A residual value of $15.9M was integrated into the cashflow model based on the estimated value of major equipment at the end of the mine life.

22.8 Financial Guarantee Bond

In the cash flow estimation, it is assumed that the closure cost will be a secured bond. The financial cost for the bond was estimated at 1.5% per year of the bond cost and was integrated in the cashflow model.

Table 22.2 – Pre-tax and after-tax cash flows for the POX option (base case study)

	Pre-production				Production											Total
	PP 1	PP 2	PP 3	PP 4	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	
PRODUCTION																
Tonnes from pit milled				400,000	2,900,000	2,900,000	2,900,000	2,900,000	2,900,000	3,275,400	3,650,000	3,650,000	3,650,000	3,650,000	2,463,029	35,238,429
Grade AU (g/t)				1.14	1.90	2.01	1.60	1.98	1.68	1.32	1.33	1.59	1.42	1.68	0.70	1.56
Mill recovery AU(%)				79.16%	90.1%	90.1%	90.1%	90.1%	90.1%	90.1%	90.1%	90.1%	90.1%	90.1%	90.1%	90.1%
Grade Ag (g/t)				1.38	2.29	2.43	1.94	2.39	2.03	1.59	1.61	1.92	1.71	2.03	0.85	1.89
Mill recovery Ag (%)				90.00%	90.00%	90.00%	90.00%	90.00%	90.00%	90.00%	90.00%	90.00%	90.00%	90.00%	90.00%	
Gold Produced (oz)				**11,607**	**159,604**	**169,114**	**134,783**	**166,312**	**141,054**	**125,005**	**140,702**	**168,114**	**149,781**	**177,808**	**50,214**	**1,594,097**
Silver Produced (oz)				**15,921**	**192,352**	**203,813**	**162,438**	**200,436**	**169,996**	**150,653**	**169,571**	**202,608**	**180,513**	**214,291**	**60,517**	**1,923,111**
Tonnes from tailing milled				135,450	750,000	750,000	750,000	750,000	750,000	239,150						4,124,600
Grade AU (g/t)				0.93	0.95	0.93	0.93	0.93	0.93	0.93						0.93
Mill recovery AU(%)				83.90%	83.90%	83.90%	83.90%	83.90%	83.90%	83.90%						83.90%
Grade Ag (g/t)				2.40	2.40	2.40	2.40	2.40	2.40	2.40						2.40
Mill recovery Ag (%)				83.90%	83.90%	83.90%	83.90%	83.90%	83.90%	83.90%						83.9%
Gold Produced (oz)				**3,398**	**19,219**	**18,815**	**18,815**	**18,815**	**18,815**	**5,999**						**103,875**
Silver Produced (oz)				**8,769**	**48,554**	**48,554**	**48,554**	**48,554**	**48,554**	**15,482**	**0**	**0**	**0**	**0**	**0**	**267,021**
TOTAL Gold Produced (oz)				**15,005**	**178,823**	**187,928**	**153,598**	**185,127**	**159,869**	**131,004**	**140,702**	**168,114**	**149,781**	**177,808**	**50,214**	**1,697,973**
Total Silver Produced (oz)				**24,690**	**240,906**	**252,367**	**210,992**	**248,990**	**218,550**	**166,136**	**169,571**	**202,608**	**180,513**	**214,291**	**60,517**	**2,190,133**
Tonnage assigned to production					**3,650,000**	**3,650,000**	**3,650,000**	**3,650,000**	**3,650,000**	**3,514,550**	**3,650,000**	**3,650,000**	**3,650,000**	**3,650,000**	**2,463,029**	**38,827,579**
Gold assigned to production (oz)					**178,823**	**187,928**	**153,598**	**185,127**	**159,869**	**131,004**	**140,702**	**168,114**	**149,781**	**177,808**	**50,214**	**1,682,968**
Silver assigned to production (oz)					**192,352**	**203,813**	**162,438**	**200,436**	**169,996**	**150,653**	**169,571**	**202,608**	**180,513**	**214,291**	**60,517**	**1,907,190**
Gold Price ($US/oz)	$1,300	$1,300	$1,300	$1,300	$1,300	$1,300	$1,300	$1,300	$1,300	$1,300	$1,300	$1,300	$1,300	$1,300	$1,300	$1,300
Silver Price ($US/oz)	$21	$21	$21	$21	$21	$21	$21	$21	$21	$21	$21	$21	$21	$21	$21	$21
Exchange rate (CAN/US)	1.100	1.100	1.100	1.100	1.100	1.100	1.100	1.100	1.100	1.100	1.100	1.100	1.100	1.100	1.100	1.100
Gold Price ($C/oz)	$1,430	$1,430	$1,430	$1,430	$1,430	$1,430	$1,430	$1,430	$1,430	$1,430	$1,430	$1,430	$1,430	$1,430	$1,430	$1,430
Silver Price ($C/oz)	$23	$23	$23	$23	$23	$23	$23	$23	$23	$23	$23	$23	$23	$23	$23	$23
Gross Revenue				**$22,029,873**	**$261,311,664**	**$274,597,848**	**$224,544,526**	**$270,512,763**	**$233,687,763**	**$191,193,464**	**$205,141,191**	**$245,108,257**	**$218,377,983**	**$259,241,861**	**$73,210,976**	**$2,478,958,170**
Mint (cost 3.00$ per oz)				$45,014	$536,470	$563,785	$460,794	$555,380	$479,607	$393,012	$422,105	$504,343	$449,342	$533,425	$150,641	$5,093,918
Capitalized revenue				$21,984,859.70												**-$21,984,860**
Net Revenue					**$260,775,194**	**$274,034,063**	**$224,083,732**	**$269,957,383**	**$233,208,156**	**$190,800,452**	**$204,719,086**	**$244,603,914**	**$217,928,641**	**$258,708,436**	**$73,060,335**	**$2,451,879,392**
OPERATING EXPENDITURES																
General and Administration	$573,226	$1,120,411	$2,556,419	$5,296,729	$9,264,409	$9,264,409	$9,264,409	$9,264,409	$9,264,409	$9,264,409	$9,264,409	$9,264,409	$9,264,409	$7,669,499	$4,408,020	$105,003,987
Processing Operating Cost	$0	$0	$183,615	$10,485,498	$56,289,953	$56,289,953	$56,289,953	$56,289,953	$56,289,953	$55,546,200	$57,999,953	$57,999,953	$57,999,953	$57,999,953	$39,140,589	$618,805,479
Mining Cost	$0	$2,408,716	$11,448,129	$12,538,561	$66,142,681	$79,064,156	$76,147,638	$76,973,802	$75,654,522	$73,219,246	$84,356,400	$62,183,474	$66,166,792	$44,263,644	$3,726,660	$734,294,421
Environmental Monitoring	$0	$5,400	$265,377	$771,377	$646,377	$646,377	$646,377	$646,377	$646,377	$646,377	$646,377	$646,377	$646,377	$646,377	$436,305	$7,942,232
Overburden Removal Cost	$0	$392,536	$2,356,916	$609,230	$5,016,727	$0	$1,848,318	$1,144,619	$1,074,139	$4,317,808	$37,739	$1,883,878	$505,489	$137,341	$0	$19,324,739
Capitalized operating costs	-$573,226	-$3,927,064	-$16,810,457	-$29,701,395												-$51,012,141
Total Operating Costs	**$0**	**$0**	**$0**	**$0**	**$137,360,147**	**$145,264,895**	**$144,196,695**	**$144,319,160**	**$142,929,400**	**$142,994,040**	**$152,304,879**	**$131,978,091**	**$134,583,021**	**$110,716,813**	**$47,711,573**	**$1,434,358,716**
Op. cost/tonne $C					$37.63	$39.80	$39.51	$39.54	$39.16	$40.69	$41.73	$36.16	$36.87	$30.33	$19.37	$36.94
Op. cost/oz $C					$768	$773	$939	$780	$894	$1,092	$1,082	$785	$899	$623	$950	$852.28
Op. cost/tonne $US					$34.21	$36.18	$35.91	$35.94	$35.60	$36.99	$37.93	$32.87	$33.52	$27.58	$17.61	$33.58
Op. cost/oz $US					$698	$703	$853	$709	$813	$992	$984	$714	$817	$566	$864	$774.80
Operating Cash Flow					**$123,415,046**	**$128,769,168**	**$79,887,037**	**$125,638,223**	**$90,278,756**	**$47,806,412**	**$52,414,207**	**$112,625,823**	**$83,345,620**	**$147,991,623**	**$25,348,762**	**$1,017,520,676**
CAPITAL EXPENDITURES																
Capitalized operating cost	$573,226	$3,927,064	$16,810,457	$29,701,395												$51,012,141
Capitalized revenue				-$21,984,860												-$21,984,860
Mine Production Equipment	$0	$4,623,043	$21,112,254	$13,831,727	$30,313,123	$19,898,635	$16,199,085	$14,512,159	$10,898,296	$1,956,849	$722,930	$1,144,021	$1,041,247	$287,079	$120,526	$136,660,974
Surface Installation and Equipment	$0	$724,000	$14,707,222	$26,341,340	$417,600	$417,600	$417,600	$417,600	$417,600	$417,600	$417,600	$417,600	$417,600	$418,600	$0	$45,949,562
Processing Facilities	$0	$3,653,814	$102,778,344	$120,179,062	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$226,611,220
Tailings Storage Facilities	$0	$0	$0	$3,374,029	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$3,374,029
Environmental	$703,200	$14,227,120	$38,776,718	$0	$5,081,497	$0	$0	$0	$0	$0	$11,630,576	$0	$0	$0	$0	$70,419,112
Total Capital Expenditures	**$1,276,426**	**$27,155,041**	**$194,184,994**	**$171,442,693**	**$35,812,221**	**$20,316,235**	**$16,616,685**	**$14,929,759**	**$11,315,896**	**$2,374,449**	**$12,771,106**	**$1,561,621**	**$1,458,847**	**$705,679**	**$120,526**	**$512,042,178**
Total Cost cost/oz $C					$968	$881	$1,047	$860	$965	$1,110	$1,173	$794	$908	$627	$953	$1,146
Total Cost cost/oz $US					$880	$801	$952	$782	$877	$1,009	$1,067	$722	$826	$570	$866	$1,042
Financial Guarantee Bond					$183,406	$275,110	$366,813	$366,813	$366,813	$366,813	$366,813	$366,813	$366,813	$366,813	$366,813	$3,759,832
Salvage Value															$15,929,714	$15,929,714
Closure Costs															$24,454,194	$24,454,194
Net Cash flow	**-$1,276,426**	**-$27,155,041**	**-$194,184,994**	**-$171,442,693**	**$87,419,419**	**$108,177,823**	**$62,903,539**	**$110,341,651**	**$78,596,047**	**$45,065,150**	**$39,276,287**	**$110,697,390**	**$81,519,960**	**$146,919,131**	**$16,336,943**	**$493,194,186**
Cumulative Cashflow Pre-Taxes	**-$1,276,426**	**-$28,431,467**	**-$222,616,462**	**-$394,059,155**	**-$306,639,736**	**-$198,461,913**	**-$135,558,374**	**-$25,216,722**	**$53,379,325**	**$98,444,475**	**$137,720,762**	**$248,418,152**	**$329,938,112**	**$476,857,243**	**$493,194,186**	
Estimated Mining and income taxes	**-$56,256**	**-$1,362,702**	**-$4,242,460**	**-$463,507**	**$6,652,797**	**$3,983,156**	**$1,463,777**	**$21,020,288**	**$3,288,739**	**$7,712,801**	**$7,709,874**	**$29,115,909**	**$29,379,885**	**$51,217,577**	**-$26,680**	**$155,393,200**
Cash Surplus After Taxes	**-$1,220,170**	**-$25,792,339**	**-$189,942,534**	**-$170,979,186**	**$80,766,622**	**$104,194,667**	**$61,439,762**	**$89,321,363**	**$75,307,309**	**$37,352,349**	**$31,566,413**	**$81,581,480**	**$52,140,074**	**$95,701,554**	**$16,363,624**	**$337,800,986**
Cumulative Cashflow After-Taxes	**-$1,220,170**	**-$27,012,509**	**-$216,955,043**	**-$387,934,230**	**-$307,167,608**	**-$202,972,941**	**-$141,533,179**	**-$52,211,816**	**$23,095,492**	**$60,447,841**	**$92,014,254**	**$173,595,735**	**$225,735,809**	**$321,437,362**	**$337,800,986**	

Pre-Tax NPV (5%)	**$222,199,164**
Pre-tax IRR	**15.11%**

After-Tax NPV (5%)	**$135,070,430**
After-tax IRR	**12.06%**

22.9 Sensitivity Analysis

Project risks can be identified in economic and non-economic terms. Key economics were examined by running cash flow sensitivities against:

- Operating cost
- Capital cost
- Revenue
- Gold price, exchange rate, mill grade and mill recovery

Sensitivity analyses were performed on the Project's pre-tax NPV (5%) and IRR, revenue, operating cost, and capital cost. Sensitivity calculations were performed on the Project's NPV and IRR, applying a range of variation (± 25%).

While project revenues are directly proportional to gold price, mill recovery and grade, a sensitivity analysis was only done on gold price.

22.9.1 Sensitivity Analysis Results

As illustrated in the figures below, the Duparquet Project is highly sensitive to changes in gold price. It is moderately sensitive to changes in OPEX and CAPEX.

Results from the sensitivity on the NPV at 5% are presented in Table 22.3 and illustrated in Figure 22.1. A 10% reduction in gold price, which corresponds to C$1,287/oz, reduces the NPV to $50.07M and drops the IRR from 15% to 8%.

To generate an IRR of 15% after-taxes and royalties requires a gold price of $C1,368/oz over the LOM.

Table 22.3 – Before-taxes - Sensitivity analysis of economical parameters, NPV at 5%

	-25%	-20%	-15%	-10%	-5%	Base Case scenario	5%	10%	15%	20%	25%
Revenue	-166.68	-88.91	-11.13	66.646	144.42	**222.2**	299.98	377.75	455.53	533.3	611.08
Opex	449.42	403.98	358.53	313.09	267.64	**222.2**	176.75	131.31	85.87	40.42	-5.02
Capex	326.71	305.81	284.91	264	243.1	**222.2**	201.3	180.39	159.49	138.59	117.69

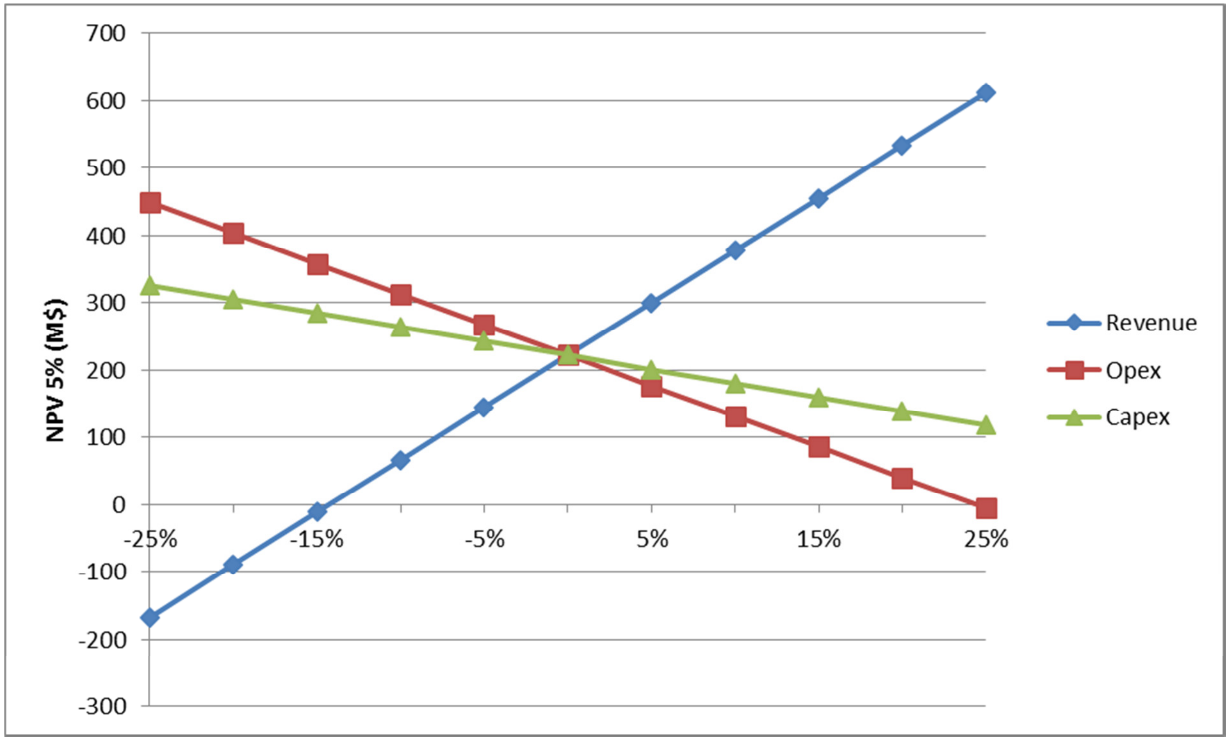

Figure 22.1 – Sensitivity analysis graph of economical parameters, NPV at 5%

Table 22.4 – Before-taxes - Sensitivity analysis of gold price parameter, NPV at 5%

	-25%	-20%	-15%	-10%	-5%	Base Case scenario	5%	10%	15%	20%	25%
Gold price (CAN$/oz)	1,073	1,144	1,216	1,287	1,359	**1430**	1,502	1,573	1,645	1,716	1,788
Gold Price	-191.81	-112.39	-31.93	50.07	134.39	**222.2**	315.25	416.17	528.86	659.05	815.09

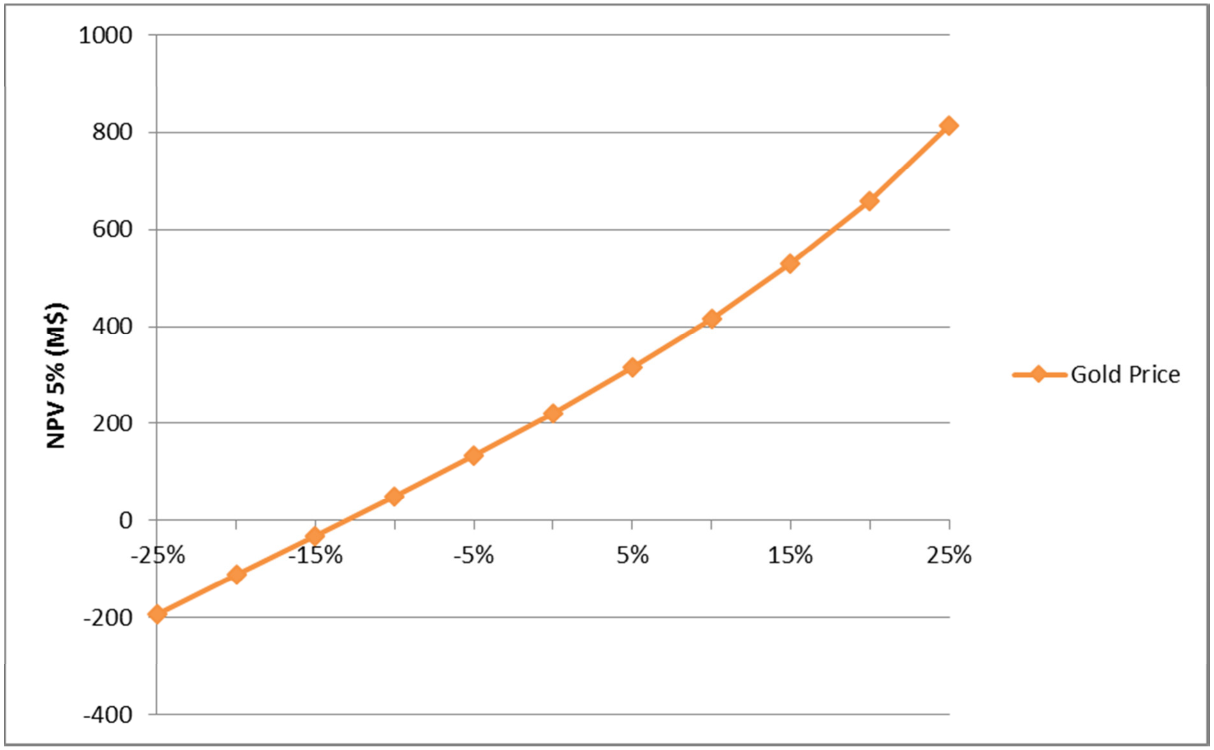

Figure 22.2 – Sensitivity diagrams of gold price parameter, NPV at 5%

Table 22.5 – Before-taxes - Sensitivity analysis of economical parameters, IRR

	-25%	-20%	-15%	-10%	-5%	Base Case scenario	5%	10%	15%	20%	25%
Revenue	-5%	0%	4%	8%	12%	**15%**	18%	21%	24%	27%	29%
Opex	24%	22%	20%	19%	17%	**15%**	13%	11%	9%	7%	5%
Capex	23%	21%	20%	18%	16%	**15%**	14%	13%	12%	11%	10%

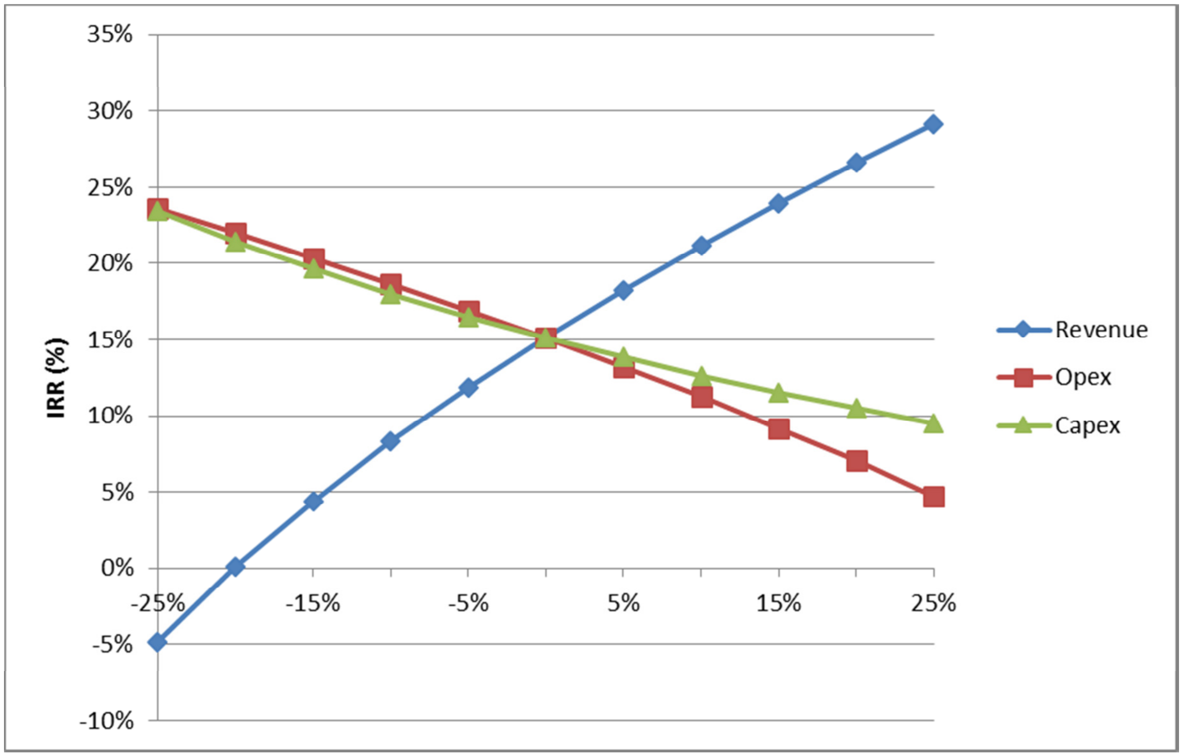

Figure 22.3 – Sensitivity diagrams of economical parameters, IRR

Table 22.5 – Before-taxes - Sensitivity analysis of gold price parameter, IRR

	-25%	-20%	-15%	-10%	-5%	Base Case scenario	5%	10%	15%	20%	25%
Gold price (CAN$/oz)	1,073	1,144	1,216	1,287	1,359	**1430**	1,502	1,573	1,645	1,716	1,788
Gold Price	-7%	-1%	3%	8%	11%	**15%**	19%	22%	26%	30%	33%

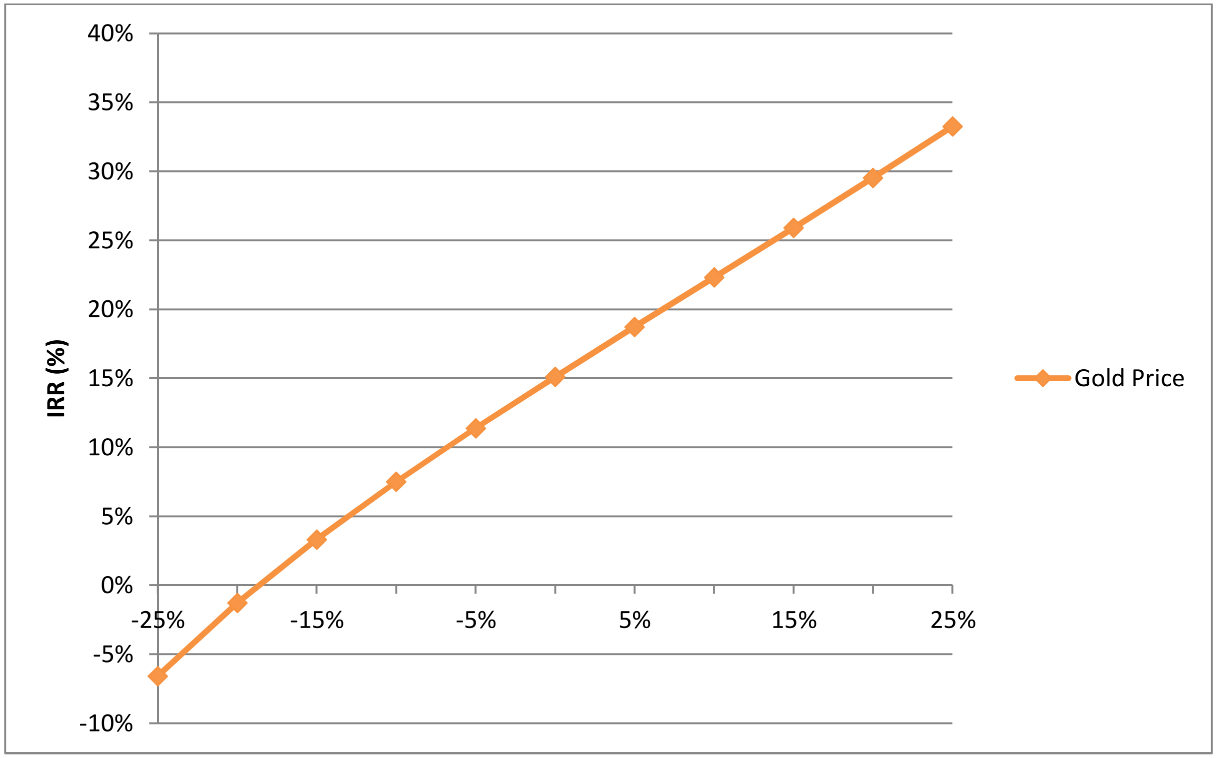

Figure 22.4 – Sensitivity diagrams of gold price parameter, IRR

23. ADJACENT PROPERTIES

23.1 Duquesne-Ottoman Property (Globex Mining Enterprises Inc.)

The Duquesne-Ottoman property is located to the east and adjacent to the Central Duparquet claims of the Duparquet Project (Fig. 23.1). The Duquesne-Ottoman property is comprised of 20 claims known as Duquesne West Block and 40 claims, known as Ottoman Fault Block (more commonly referred to as Ottoman), and covers an area of 928.6 hectares. All claims are held 100% by Duparquet Assets Inc (2012 MD&A of Xmet from SEDAR website).

On March 5, 2012, Xmet entered into a new Share Option Agreement with Globex Mining Enterprises Inc. and Geoconseil Jack Stoch Ltée, (collectively the “optionors”), which granted the Company an accelerated option to acquire either a 75% or 100% interest in Duparquet. On July 3, 2013, Globex Mining Enterprises Inc. announced by press release that the optioning of 50% interest in the Duquesne West property (Duquesne-Ottoman) to Xmet Inc. has been terminated due to market conditions.

The northwest part of the Duquesne-Ottoman property is located along the eastern extension of the east-west trending deformation corridor defined by the Beattie–Donchester–Central Duparquet fault system. The southeast portion of the property, which hosts most of the gold showings and zones, appears to be related to the Destor-Porcupine-Manneville deformation corridor or to another east-west trending deformation corridor at the south contact of the Duparquet Formation (Fig. 7.3).

On September 8, 2011 Xmet published the results of a new NI 43-101 compliant resource estimate for the Duquesne-Ottoman property. The Current Resource at Duquesne-Ottoman stands at 853,000 inferred ounces gold uncut at a grade of 6.36 g/t Au or 727,000 ounces gold at a grade of 5.42 g/t Au applying a gold top cut-off grade of 30 g/t. The resource estimate was carried out by Watts, Griffis McOuat Limited Consulting Geologists and Engineers of Toronto, Canada (Power-Fardy and Breede, 2011).

23.2 Brionor Property

On May 16, 2012, Xmet announced the signature of a purchase agreement with Brionor Resources Inc. to acquire twenty-four (24) contiguous mineral claims from Brionor (known as the “Pitt Gold Project”), which claims are immediately adjacent to the Duquesne-Ottoman Property. On April 30, 2013, Brionor announced by press release that the agreement with Xmet Inc. on Pitt Gold Project will not proceed.

As set forth in the technical report entitled “NI 43-101 Technical Report and Audit of the Preliminary Mineral Resource Estimate for the Pitt Gold Project, Duparquet Township, Abitibi Region, Québec, Canada” dated June 10, 2011, by William J. Lewis and Alan San Martin of Micon International Limited, the Pitt Gold Project is an NI 43-101 compliant gold resource with 600,000 t at 7.83 g/t equalling 151,000 oz Au at the Indicated level, and 476,000 t at 6.91 g/t equalling 106,000 oz at the Inferred level.

23.3 Duparquet Gold Property (Tres-Or Resources Ltd)

Tres-Or Resources Ltd owns the Duparquet Gold property, located northeast of the Central Duparquet property, and north and adjacent to the Duquesne-Ottoman property. According to Tres-Or Resources, the property is believed to host potential splay-fault extensions of mineralization associated with the Beattie and Donchester past-producing mines (Tres-Or website: http://www.tres-or.com/Québec-Duparquet). Tres-Or Resources Ltd has carried out a magnetic and electromagnetic survey on the property.

23.4 East Bay Property (Explor Resources Inc.)

The East Bay property is located in the central part of Duparquet Township, approximately 1 km west of the Town of Duparquet, Québec. The property lies on the Porcupine Destor Fault Zone. The mineralization on the property is exposed on Beattie Island in Lake Duparquet and is hosted within the Destor tholeiitic unit of the Blake River Group (website of Explor Resources). Mineralization is associated with interflow volcanic sediments between andesite flows and dioritic sills. The alteration consists of silicification, carbonization and considerable sulphide enrichment. Gold occurs with pyrite-chalcopyrite in shallow dipping quartz veins that crosscut the stratigraphy. Historical channel samples completed by Lacana Mining Corporation in 1982 include 0.81 oz/ton over 5 feet, 0.165 oz/ton over 6 feet and 0.10 oz/ton over 10 feet.

Explor has completed a study and a complete compilation of work executed in the past, followed by line cutting, magnetic survey and VLF to determine the localization of structural targets on the property. After analysing all of this information, in July 2013, the Corporation started an initial 4-hole drill program of 1,500 m on the property (October 31, 2013 MD&A of Explor Resources from SEDAR website). The four holes drilled in 2013 on this target (EXS-13-01 to EXS-13-04) were laid out according to the follow-up drill program proposed in 1988, through a tighter spacing. With an aggregate total of 879 m, all four drill holes encountered a sequence of highly sheared and altered felsic tuffs and quartz porphyries, with subordinate felsic and mafic volcanics and ultramafics. Stronger gold mineralization was cored in drill hole EXS-13-04, which returned 1.68 g/t Au over a 2.0 m interval (Explor Resources press release February 17, 2014). The host rock is rhyolite-looking, weakly pyritized (1-3% pyrite) and wedged between strongly sheared to mylonitic quartz-floored felsic tuffs and ultramafics.

23.5 Gold Potential of the Other Adjacent Properties

InnovExplo has been unable to verify the above information on adjacent properties near the Duparquet Project. The presence of significant mineralization on these adjacent properties is not necessarily indicative of similar mineralization on the Duparquet Project.

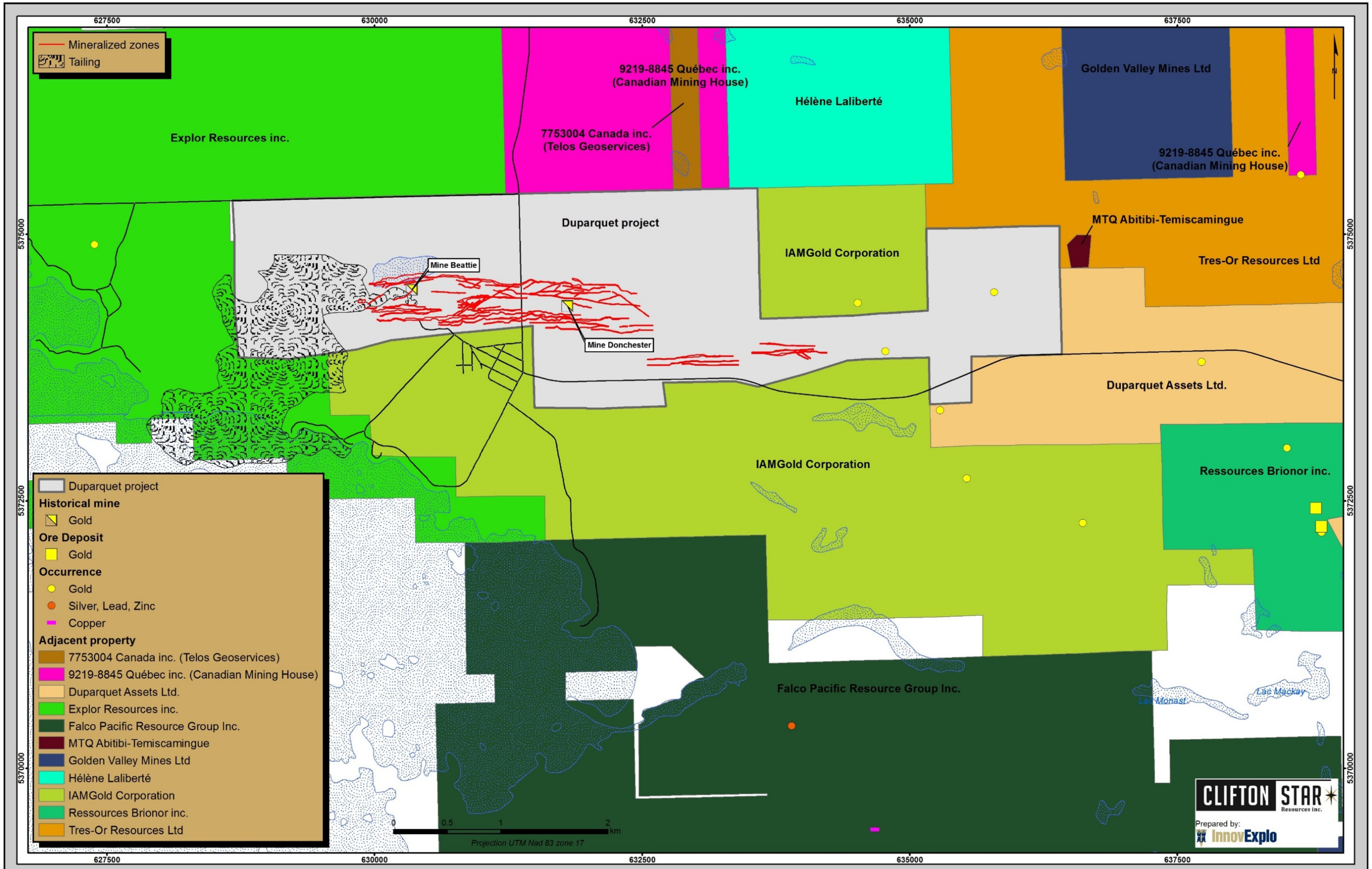

Figure 23.1 – Adjacent properties to the Duparquet Project

24. OTHER RELEVANT DATA AND INFORMATION

No mineral estimate was performed to determine the silver content in the Duparquet deposit because very few silver assays were available. At the request of Clifton Star, InnovExplo used a predetermined silver content to calculate the cash flow in the current Prefeasibility Study. The value was derived from the 2009 tailings sampling program, the 2013 12 t composite bulk sampling program and the 238 individual samples, representing the “Master Holes” for the bulk sampling program that were analysed for silver.

24.1 Silver assays results from Beattie mine tailings (Fillion, 2009)

A total of eight (8) samples from the tailings were assayed for silver in 2009 (Table 24.1). The average silver grade of all results was 2.4 g/t Ag. This grade was used to calculate the cash flow in the current Prefeasibility Study.

Table 24.1 – Silver assays results from Beattie mine tailings (Fillion, 2009)

Hole #	Composite	COV	Ag	As	Be	Cd	Cr+6	Cu	Hg	Pb	Pd	Pt	BPC
	#	mg/kg	mg/kg	mg/kg	mg/kg	mg/kg	mg/kg	mg/kg	mg/kg	mg/kg	mg/kg	mg/kg	mg/kg
Standard MDDEP, cat C		5 - 50	40	50		20		500	10	1000			10
L4-C	77051		3.7	1,233	0.2	0.416	<0.2	1,184	0.2	41.0	<.005	<.005	<0.01
BoisÉch8	77052		3.1	6,915	0.6	0.260	0.8	21.0	4.4	967.0	<.005	<.005	<0.01
L9-B	77053		3.3	846	0.2	0.495	<3.0	425.0	1.9	18.5	<.005	<.005	<0.01
L19-C	77054		1.4	1,778	0.2	0.164	<3.0	42.4	1.7	9.2	<.005	<.005	<0.01
L21-C	77055		1.0	1,231	0.3	0.179	<3.0	39.2	0.9	6.9	<.005	<.005	<0.01
L15-B	77056		2.7	923	0.2	0.138	<3.0	42.8	1.3	13.5	<.005	<.005	<0.01
L13-B	77057		2.4	929	0.1	0.212	<3.0	67.3	2.9	12.3	<.005	<.005	<0.01
L17-A	77058		1.4	819	0.2	0.133	<3.0	48.2	1.2	18.8	<.005	<.005	<0.01

24.2 Head analysis results of the 2013 bulk sample

Clifton Star sent approximately 12 tonnes of full HQ core of its 2013 composite bulk sample (see section 11.7) to SGS Canada Inc. The results yielded a head analysis of 1.84 g/t Au and 2.2 g/t Ag, and the Au/Ag ratio was 0.84 (Table 24.2). These values were used to calculate the cash flow in the current Prefeasibility Study.

Table 24.2 - Head analysis of the pilot plant feed sample obtained at the SGS Canada laboratory

Element		PP Feed	Element		PP Feed
Au	g/t	1.84, 1.83	S	%	1.16
Au*	g/t	1.84	$S^{=}$	%	1.15
Ag	g/t	2.2	C_T	%	1.60
As	%	0.055	CO_3	%	7.61
Hg	g/t	3.5	TCM	%	0.08
Te	g/t	<4			
ICP Scan					
Al	g/t	72900	Na	g/t	11400
Ba	g/t	1580	Ni	g/t	< 20
Be	g/t	1.7	P	g/t	1050
Bi	g/t	< 20	Pb	g/t	31
Ca	g/t	39800	Sb	g/t	< 20
Cd	g/t	< 2	Se	g/t	< 20
Co	g/t	16	Sn	g/t	< 20
Cr	g/t	42	Sr	g/t	533
Cu	g/t	29.3	Ti	g/t	3430
Fe	g/t	35400	Tl	g/t	< 30
K	g/t	51400	U	g/t	< 20
Li	g/t	< 5	V	g/t	126
Mg	g/t	7000	Y	g/t	20.1
Mn	g/t	1040	Zn	g/t	64
Mo	g/t	23			

*Average direct assay of PP Feed samples taken during the pilot plant campaign
TCM = total carbonaceous matter (C_T - C as CO_3)

24.3 Silver assay results from master holes of the 2013 bulk sample

The 2013 Clifton Star bulk sample comprises material from six (6) mineralized zones (Figure 11.3). This represents a better representative sample corresponding to the future mineralized zones that will be mined at the Duparquet Project and also represents the mineralized zones treated in the current PFS. A total of 238 pulp samples were assayed for silver. The silver grades are plotted against Au/Ag ratios on Figure 24.1.

The Au/Ag ratios on Figure 24.1 were taken from historical data. From the start of production in 1933 until the end of 1940, the mill at the Beattie Gold mine processed 3,921,281 short tons of ore and recovered 471,085 ounces of gold and 73,214 ounces of silver, for an average of 0.120 oz/t Au and 0.019 oz/t Ag (Dresser and Denis, 1949). This tonnage came from the North and A zones. The Au/Ag ratio was 6.32. According to the Department of Energy and Natural Resourcesof Québec (Lavergne, 1985), the Beattie mill treated 9,645,000 metric tonnes with an average grade of 4.01 g/t Au and 0.99 g/t Ag. The ore came from the Beattie mine and Donchester mine between 1933 and 1956. The Au/Ag ratio decreased to 4.05 during this time because the ore from the North Zone of the Donchester mine was added to the Beattie mill. This is also evident on Figure 24.1 where the North Zone of the Donchester mine (green diamonds; DDH D09-18) shows a higher silver content than that of the North Zone of the Beattie mine (beige diamonds; DDH BD12-15).

Furthermore, the fact that silver recovery was not optimal during these years must be taken into account. In reality, the historical Au/Ag ratio should probably be lower than that reported by Dresser and Denis (1949) and Lavergne (1985).

InnovExplo is of the opinion that results obtained for silver content from the tailings, the bulk sample and the Master Hole are reliable and valid.

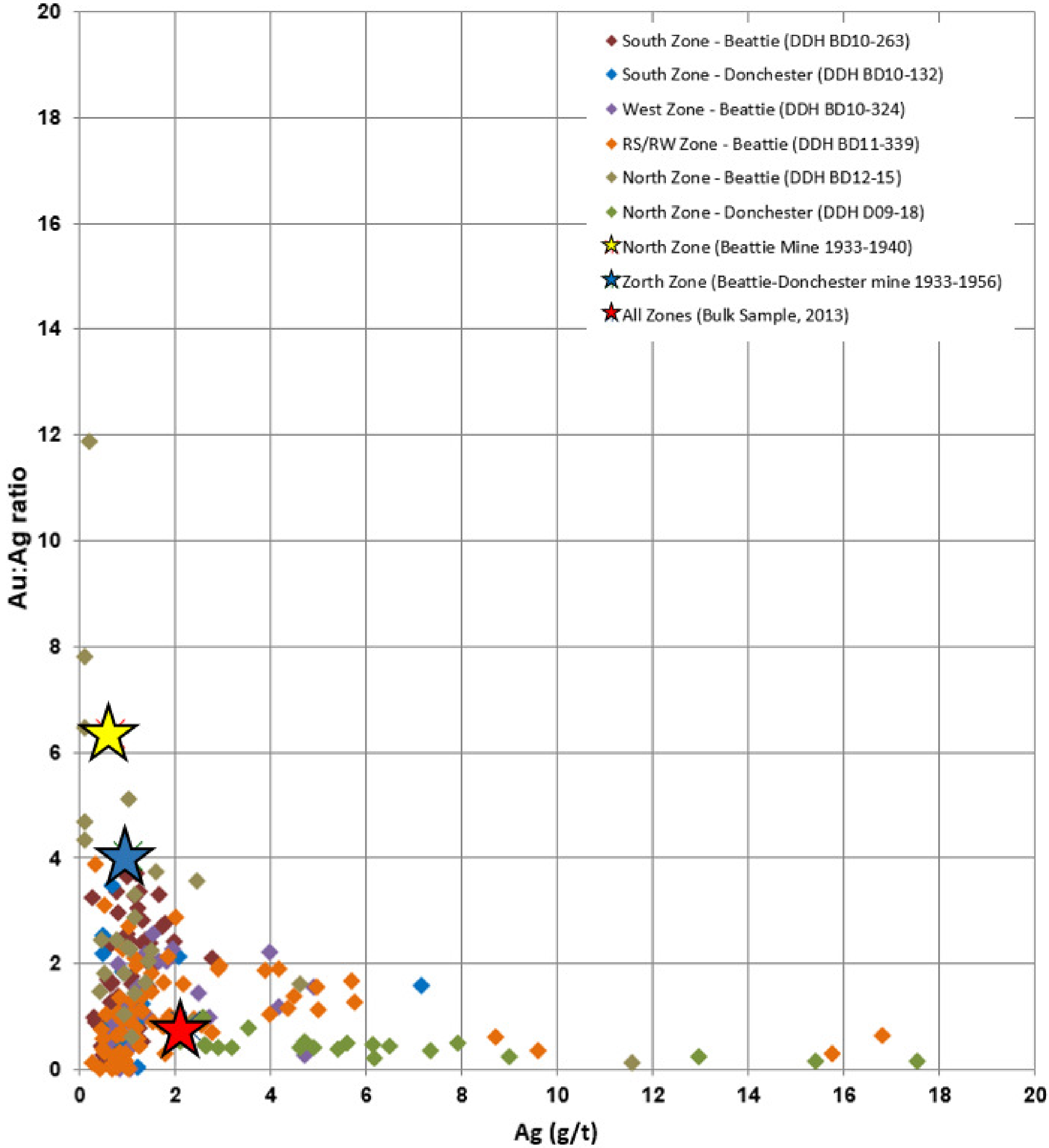

Figure 24.1 – Plot of silver grades (g/t Ag) vs. Au/Ag ratios from assay results from master holes of the 2013 bulk sample. The Au/Ag ratios from historical production from Beattie mine is also shown by yellow and blue stars. The red star corresponds to the Au/Ag ratio from the head analysis.

25. INTERPRETATION AND CONCLUSIONS

The objective of the mandate assigned to InnovExplo Inc., Roche Ltd. Consulting Group and Dresinger Consulting was to produce a Prefeasibility Study (PFS) for the Duparquet Project in accordance with National Instrument 43-101 and Form 43-101F1. The updated Mineral Resource Estimate on which the PFS is based was published in an earlier report titled “Technical Report and Mineral Resource Estimate Update for the Duparquet Project (according to Regulation 43-101 and Form 43-101F1)”, dated August 2, 2013 (Williamson et al., 2013b).

InnovExplo considers the present PFS (and Mineral Resource Estimate presented herein) to be reliable and thorough, and based on quality data, reasonable hypotheses and parameters compliant with NI 43-101 and CIM standards regarding mineral resource estimates.

Resources

Using new geological and analytical information available, InnovExplo updated the mineralized-zone wireframe model of the Duparquet Project. The following summarizes the changes and additions made to the mineralized material zone wireframe model (as presented in the August 2013 report by Williamson et al., 2013b).

1) The South Zone was redesigned in such a way that it now presents a series of ten (10) “en-echelon” subzones associated with the DFZ;
2) The West Zone has been remodelled into three (3) distinct lenses;
3) A new zone, CD S2, has been created at Central Duparquet;
4) Five (5) mineralized zones have been created at Dumico;
5) Thirty-four (34) new zones (secondary zones) have been created within the “inter-zone” mineralized envelope.

After conducting a detailed review of all pertinent information and the updated Mineral Resource Estimate (August 2013), InnovExplo concludes the following:

- In spite of the current drill spacing, the geological and grade continuities of the sixty (60) interpreted gold mineralized zones of the Duparquet Project were demonstrated.
- The definition drilling program and the updated mineralized-zone wireframe model successfully upgraded a significant part of the in-pit resources to an Indicated Resource category, which allowed the Duparquet Project to advance to a prefeasibility study stage.
- New drilling included in the update added to the overall Measured and Indicated resources, bringing it to a total of 3.11 Moz, a 29% increase over the previous estimate. The total Inferred Resource amounts to 1.44 Moz. In-Pit Measured and Indicated Resources have increased by 38% to 2.67 Moz.
- Not all of the Inferred Resources had been converted into Indicated Resources in the pit area, so it could have an impact on a future economic study.

- The new geological and structural model greatly enhances resource conversion to the Indicated category and would benefit considerably from detailed mapping of the numerous outcrops present on the Property.
- The potential is high for adding new resources along strike, at depth and along the down-dip extensions of the main and secondary zones, through additional diamond drilling.

Reserves

The reserves for the detailed pit design have been calculated in accordance with the definitions and guidelines adopted by the Canadian Institute of Mining, Metallurgy and Petroleum (CIM Standards on Mineral Resources and Reserves) in August 2000.

The effective date of the Mineral Reserve is March 26, 2014, the date on which the latest processing cost estimation was concluded.

The PFS is based on a Mineral Resource Estimate produced by InnovExplo in an earlier report prepared for the issuer titled “Technical Report and Mineral Resource Estimate Update for the Duparquet Project (according to Regulation 43-101 and Form 43-101F1)”, dated August 2, 2013.

The mineral resource block model developed by InnovExplo was imported into Whittle™ software from Dassault Systèmes GEOVIA (formerly Gemcom Software International). Design parameters, such as operating costs, mine recovery, dilution and gold price, were used to generate an optimal pit shell.

InnovExplo evaluated dilution and mine recovery by performing an analysis of mineable blocks on typical cross-sections at the mill cut-off grade. After a cross-sectional evaluation, average dilution was estimated at 10% at a grade of 0 g/t, and mine recovery was estimated at 95% assuming good blasting and dilution control practices.

To integrate dilution in Whittle, an initial 5% dilution factor was used when performing reblocking and merging the original small blocks (5x5x5) into grade bands (high and low). This feature defines a maximum number of parcels per reblock blocks. This action regroups the smaller blocks of 5x5x5 into larger parcels, by grade interval. In this evaluation, two parcels were considered. The resulting parcel includes blocks lower than the cut-off grade that are considered internal dilution. If the average grade of the parcel is higher than the cut-off grade, it is considered as ore and sent to the mill. This dilution was estimated at 5%. An additional 5% dilution factor was added to the Whittle parameters for a total dilution factor of 10%.

Iterations were performed to generate a pit design that fits the selected pit shell. To do so, a pit design, including a ramp and catch bench, is made in the selected pitshell. For the Duparquet Project, the initial pit wall angle was too steep to accommodate the ramp. The slope angle was smoothed until a ramp could fit in the Whittle pit shell. Many iterations were needed before finding an adequate pit wall angle.

After many iterations, pit slopes were set at 29° in the overburden, 45° on the north side of the pits, and 48° on the south side of the pit. The ramp was designed on the north side of the pit due to the constraints imposed by the golf course and houses on the south side and due to the location of the mill.

The concurrent prefeasibility work allowed InnovExplo and Roche Ltd Consulting Group ("Roche") to better quantify the operating costs. These costs were used for a final pit optimization in Whittle. These parameters are presented in Table 15.1.

Whittle parameters (Table 15.1)

Input parameters		Value	Provided by
Gold Price		1,417.5 $CAN/oz	InnovExplo
Gold selling cost		5 $/oz	InnovExplo
Dilution		*5 %	InnovExplo
Mining recovery		95 %	InnovExplo
Milling recovery		93.9 %	Roche
Overburden Cost		0.80 $/t	InnovExplo
Mining cost		1.93 $/t	InnovExplo
General & Administration		3.12 $/t	InnovExplo
Processing Cost		16.77 $/t	Roche
Environmental monitoring		0.20 $/t	Roche
Stockpile rehandling cost		0.88 $/t	InnovExplo
Pit slope	North	45 °	InnovExplo
	South	48 °	InnovExplo
	OVB	29 °	InnovExplo

Note: An additional 5% dilution factor is included in the reblocking for a total dilution factor of 10%.

The mining cost of $1.93/t represents the initial cost to mine the rock present at the surface of the open pit. An incremental hauling cost of $0.21 per kilometre of ramp was added, depending on pit depth.

The open pit production is supplemented by 4.1 Mt of available old tailings. The old tailings will be processed during the beginning of the Project at a rate of 750,000 tonnes per year.

The global In-pit and Tailings Proven and Probable Reserves total 39,363,000 tonnes at an average grade of 1.50 g/t (1,895,530 contained ounces of gold). The Whittle pit shell selected for this PFS generates 35,238,429 tonnes of ore, including dilution and losses. Another 4,124,600 tonnes from the old tailings completes the resources for a total of 39,363,029 tonnes of ore from which 0.5% is Proven Reserves and 99.5% is Probable Reserve. A cut-off grade of 0.45 g/t has been considered for the tailings material and 0.51 for the pit. The Mineral Reserve Estimate for the Duparquet Project is presented in Table 15.2.

Mineral Reserve Estimate (Table 15.2)

Reserves type	Parameters	Area		TOTAL
		Tailings	In-Pit 1	
	Cut-off (g/t)	> 0.45	> 0.51	
Proven	Tonnes (t)	19,600	175,100	194,700
	Grade (g/t)	2.06	1.31	1.38
	Au (Oz)	1,295	7,372	8,667
Probable	Tonnes (t)	4,105,000	35,063,400	39,168,400
	Grade (g/t)	0.93	1.56	1.50
	Au (Oz)	123,200	1,763,664	1,886,864
Proven + Probable	*Tonnes (t)*	*4,124,600*	*35,238,400*	39,363,000
	Grade (g/t)	*0.94*	*1.56*	**1.50**
	Au (Oz)	*124,495*	*1,771,035*	**1,895,530**

Mining

The Duparquet Project pit optimization for the present PFS generates 35,238,429 tonnes of ore. Another 4,124,600 tonnes from the old tailings complete the resource for a total of 39,363,029 tonnes of ore. The pit also generates 291,213,881 tonnes of waste and 23,398,085 tonnes of overburden resulting in an LOM strip ratio of 8.26 to 1. Taking into account the overburden, the LOM strip ratio is 8.92 to 1. The estimated LOM average grade is 1.50 g/t including the ore from the pit and the old tailings. A total of 1,682,968 ounces of gold would be recovered over the mine life. An average of 173,000 ounces per year would be recovered for the first 5 years, and an average of 158,000 ounces per year over the 11 years of production studied. A stockpile will be used to vary the cut-off grade in order to optimize project economics. The ore, waste and tailings production plan is presented in the table below on a yearly basis.

Production by year and stripping ratio (Table 16.5)

Year	Stripping		Mineralized material		Total production		Stripping ratio
	Overburden (tpy)	Waste (tpy)	From pit (tpy)	From old tailings (tpy)	Without overburden (tpy)	With overburden (tpy)	
PP1	-	-	-	-	-	-	
PP2	513,834	800,000	-	-	800,000	1,313,834	
PP3	3,467,354	5,398,400	-	-	5,398,400	8,865,754	
PP4	402,075	626,000	894,717	135,450	1,520,717	1,922,792	1.15
1	6,272,956	27,162,421	3,288,695	750,000	30,451,116	36,724,072	10.17
2	-	28,864,822	5,727,596	750,000	34,592,418	34,592,418	5.04
3	1,859,667	29,360,581	3,646,700	750,000	33,007,281	34,866,948	8.56
4	1,260,908	31,612,674	2,236,841	750,000	33,849,515	35,110,423	14.70
5	1,291,105	29,831,261	3,562,544	750,000	33,393,805	34,684,910	8.74
6	4,446,345	28,281,662	2,884,832	239,150	31,166,494	35,612,839	11.34
7	30,288	33,081,261	2,831,059	-	35,912,320	35,942,608	11.70
8	2,954,360	29,211,166	3,649,988	-	32,861,154	35,815,514	8.81
9	743,041	31,605,924	3,650,216	-	35,256,140	35,999,181	8.86
10	156,985	15,377,709	2,865,243	-	18,242,952	18,399,937	5.42
Total	**23,398,919**	**291,213,881**	**35,238,429**	**4,124,600**	**326,452,310**	**349,851,229**	**8.93**

The Duparquet Pit LOM was based on supplying the mill with 3,650,000 tonnes of ore per year. Initially, 4.1 Mt of tailings would be reserved to supplement the mill when necessary, but it was later decided to process the tailings at a rate of 750,000 tonnes per year right from the start in order to clean the tailings area to make room for the waste stockpile.

The Duparquet Pit LOM will be spread over 11 years, preceded by a 4-year pre-production period. This schedule will yield a yearly production of 3,650,000 tonnes. For the Project, three (3) 6030FS shovels have been selected, as well as one (1) 994H front-end loader, and it has been determined that 785D trucks will be used. A total of 14 trucks will be necessary during the production peak, which covers Years 2 to 7. Sanvik recommended two types of drill for the Project: one DR540 for the pre-shear drilling which will drill 140 mm holes, and four (4) D55SP drills for the production drilling which will drill 215 mm holes.

A total of 339 employees will be needed for the Duparquet Project. This assumes the operation will run 24 hours a day, 7 days a week, 52 weeks per year.

The working schedule for most yearly compensated employees will be a standard 40-hour week at 8 hours per day, 5 days per week, Monday to Friday. Some yearly compensated employees will be working 12-hour shifts, equivalent to 84 hours per week, as part of a two-week repeating schedule: the first week working 4 days followed by 3 days off, the second week working 3 days followed by 4 days off. The hourly workers will be working 12-hour shifts as part of the same two-week repeating schedule. Most activities require 24-hour per day operation, which is split into 4 shifts.

Processing

Bench scale and pilot plant metallurgical testwork programs have been carried out for the PFS. The PFS considered two processing options for the recovery of the gold from the Clifton Star Duparquet Project ores: the Concentrate option and the Pressure Oxidation (“POX”) option. Whereas gold doré would be produced on site with the POX option, the Concentrate option would produce a sulphide concentrate product for sale.

The Concentrate option would produce a sulphide rich concentrate having the highest gold content possible. Maximizing the gold grade in the concentrate has an impact on the flotation gold recovery, and justifies a CIL circuit to recover the remaining gold from the flotation tailings. For the POX option, the flotation circuit is operated to maximize gold recovery to flotation concentrate instead of maximizing the gold concentrate grade, as in the Concentrate option. With much less gold being rejected in the flotation tailings in the POX option, cyanidation of the tailings is currently not economically feasible in light of the CAPEX and OPEX associated with a tailings CIL circuit, as well as the price of gold used for the current study (US$1,300/oz).

The strategy of using the POX process generated the highest financial return on investment, while the alternative concentrate production process option resulted in lower capital and lower operating costs. Based on the current results, conventional

POX technology, using an autoclave, is the preferred method of oxidation for the Duparquet Project.
The processing plant is designed to process an average of 10,000 tpd of ore with an overall recovery of 90.1% for gold and 90% for silver.

The process flowsheet chosen includes a conventional open circuit primary crushing stage followed by two grinding stages. The flotation circuit includes a: rougher flotation stage, a rougher concentrate regrind stage and a cleaner flotation stage. After thickening, the flotation concentrate will pass through the pressure oxidation circuit to be oxidized followed by a carbon-in-pulp (CIP) circuit for cyanidation and carbon adsorption for gold recovery. The flotation tailings will be considered as final tailings. In the CIP circuit, gold will be cyanide leached and adsorbed onto the carbon and then recovered from the carbon by an elution circuit, followed by electrowinning, and doré smelting. Carbon regeneration has also been included.

Environment Permitting

Up to this date, environmental, geochemical and social baseline studies have not led to the identification of any fatal flaw that could seriously impact the future development of the Duparquet Project.

The overall Project is subject to environmental assessment provisions of Quebec's Environment Quality Act (EQA) and the Canadian Environmental Assessment Act (CEAA). The Environmental and Social Impact Assessment (ESIA) that is required pursuant to these Acts will be initiated as soon as a Project Notice and a Project Description will have been tabled with the provincial and federal authorities, respectively. A schedule for the environmental assessment and permitting of the Project has been developed. Environmental and social baseline studies were and will be conducted and reports either have been or will be prepared. Permitting requirements are also well-defined and have been considered in the project development plan.
A tailings and water management strategy has been defined at a prefeasibility design level.
A Mine Rehabilitation and Closure Plan will be prepared for the Project as part of the environmental permitting process. The Plan will describe measures planned to restore the Property as close as reasonably possible to its former use or condition, or to an alternate use or condition that is considered appropriate and acceptable by the Québec Department of Natural Resources (MERN). The Plan will outline measures to be taken for progressive rehabilitation, closure rehabilitation and post-closure monitoring and treatment. It will also help refine the evaluation of restoration costs completed as part of this Report.

Capital and Operating Costs

The pre-production capital cost for the POX option is estimated at $394 million and sustaining capital is estimated at $118 million. The capital costs include various added contingencies depending on the sector. For the base case (POX), contingencies and indirect costs total $98.7 million of the pre-production costs and represent 26% of the costs. Indirect costs (owner's costs; Engineering, Procurement and Construction Management (EPCM); and detailed engineering) of 37% have been applied on the process plant and to the other surface infrastructure. An average contingency for all environmental items is 20%.

The total estimated capital expenditure of $512M for the Duparquet Project consists of five (5) components (see table below): mining; surface installation and equipment; processing facilities; tailings storage facilities; and environmental.

The tailing storage facilities item in the table below includes the reclaim pumping station and pipeline. The remainder of the tailings dam infrastructure is included in the environmental pre-production and sustaining costs.

Breakdown of the capital cost (Table 21.1)

Description	Pre-production ($)	Sustaining ($)	Total cost ($)
Capitalized operating cost	*51,012,141*	-	*51,012,141*
Capitalized revenue	- *21,984,860*	-	- *21,984,860*
Mine production equipment	23,120,924	91,126,227	114,247,151
Surface installation and equipment	58,218,662	10,144,723	68,363,385
Processing Facilities	226,611,220	-	226,611,220
Tailings Storage Facilities	3,374,029	-	3,374,029
Environmental	53,707,038	16,712,074	70,419,112
Total	**394,059,154**	**117,983,024**	**512,042,179**

The OPEX cost breakdown for the PFS is divided into five (5) main categories: general and administration (G&A); processing; mining; environmental; and overburden removal. The G&A category includes the costs of technical services and administration. Open pit mining costs include drilling, blasting, loading, hauling, auxiliary, and general mine maintenance. The processing category includes manpower, the cost to process ore from the pit, and the cost to process the old tailings. The environmental category includes manpower and departmental costs.

Summary of total operating costs (Table 21.8)

Description	Total cost estimate (production period)	Unit cost ($/t ore)	Unit cost ($/oz Au)
General and administration	$ 95,457,201	2.46 $/t	56.72 $/t
Processing cost	$ 608,136,366	15.66 $/t	361.35 $/t
Mining cost	$ 707,899,014	18.23 $/t	420.63 $/t
Environmental monitoring	$ 6,900,077	0.18 $/t	4.10 $/t
Overburden removal cost	$ 15,966,057	0.41 $/t	9.49 $/t
Total	**$ 1,434,358,715**	**36.94 $/t**	**852.28 $/t**

Financial Analysis

Pre-tax and after-tax cash flow projections were generated from the LOM schedule according to the capital and operating cost estimates. It was done in constant 2013 money terms and in Canadian currency unless stated otherwise, with no allowance for inflation or escalation. The net cash flow has been discounted for the purposes of calculating NPV. A base discount rate of 5% per year has been selected as most likely to represent a low capital expense gold project in a mining-friendly environment. Future annual cash flow estimates are based on grade, gold recoveries and cost estimates.

The PFS considered two possible processing scenarios. Cash flow models were created for both options. The POX option generated the highest financial return and, as a result, the POX process is favoured and used as the base case of the present PFS.

The undiscounted pre-tax cash flow totals $493.19M over the 11-year mine life and the payback period is 4.3 years.

A summary of the base case cash flow model is given in the table below. LOM totals for undiscounted and discounted cash flows are also provided. The table shows that the pre-tax NPV of the project cash flow at a discount of 5% per year is evaluated at approximately $222M and a pre-tax internal rate of return of 15.11%. The average cash cost of production equates to US$775/oz gold.

Cash flow analysis summary (Table 22.1)

Parameters		Results
Gold Price		1,300 US$/oz
Foreign exchange rate		1.10 : 1.00 (CAN/USD)
Mineable reserves		35,2 Mt @ 1.56g/t Au; 1.89 g/t Ag
Old tailings		4,1 Mt @ 0.93 g/t Au; 2.40 g/t Ag
Recovered Gold	From mine	1.6 Moz
	From old tailings	0.1 Moz
	Total	1.7 Moz
Recovered Silver	From mine	1.9 Moz
	From old tailings	0.3 Moz
	Total	2.2 Moz
Average annual gold production (ounces):		173,000 oz (first 5 years) 158,000 oz (Average 11 year)
Total waste		291 Mt
Total OVB		23.4 Mt
Mine life (excluding 4 years of pre-production)		11 years
Daily mine production		10,000 tpd
Metal recovery Au	Mine	90.10%
	Old tailings	83.90%
Pre-production capital		394M$
Sustaining capital (excluding 24.5M$ for closure cost)		118M$
Average operating cost		36.94 C$/tonne milled
Average total Site Cash Cost (US$/ounce)		775 US$/oz Au
Average total All in Cost, Average (US$/ounce) LOM		1042 US$/oz Au
Net cashflow		493M$
Pre-tax NPV (5%)		222M$
Pre-tax IRR		15.11%
After-tax NPV (5%)		135M$
After-tax IRR		12.06%
Payback period		4.3 years

<u>Risk and opportunities</u>

Risk

- There is a risk that Hydro-Québec might not have the capacity to supply the Duparquet Project, depending on other power consumptions. Further study and discussion should be initiated with Hydro-Québec.

- There is a risk that the MERN will not allow Clifton Star to recover the portion of the Mineral Resources in the tailings that are outside the Property boundary. If this were to happen, the authors wish to point out that this very small portion of the total Mineral Resources would only have a minor impact and would not affect the potential viability of the Project.

- There is a risk that the CAPEX to develop the project will be higher than estimate. Not all recent projects have achieved the expected cost anticipated to reach production. To limit the risk, Clifton Star should put considerable effort into assembling a highly qualified and experienced team for project development.

- The foundations cost could be higher or lower depending on the results of future geotechnical studies.

- If development targets are not achieved during the preproduction and early production periods, the ramp-up schedule to full production may be compromised.

- There is a risk that the Québec Ministry of Transport will not allow heavy equipment to travel across Route 393, or will require a larger buffer zone between the highway and the pit edges.

- The social and economic effects in developing a by-pass for the portion of highway 393 that passes over the mineralized zone should be investigated.

- The risk related to metallurgical issues are that:
  - variations in mineralogy could cause fluctuations in mill recovery;
  - grinding characteristics may differ from test results;
  - thickening and filtration testwork may not be representative of the ore deposit;

- As with all mining projects in the Province of Québec or, more largely, in Eastern Canada and also elsewhere in the world, there is always a risk that the mining company may not be successful in obtaining a “social licence to operate”. In the present case however, the “social acceptability” risk is relatively mitigated in part because the Project is located in an area with a long history of mining development (when compared with other regions). Still, for any mining project located close to urban areas, there is always a risk of developing a project that is not acceptable for the local, regional or provincial communities.

Opportunity

- The project is located in located in the vicinity of the town of Duparquet and close to major center of Rouyn-Noranda and La Sarre. Both skilled and general workforces are readily available. Suppliers, contractors, consulting firms and competent workers are available locally.

- The inferred Mineral Resource blocks as well as all blocks having at least 0.01% of their volume contained within stope solids that fall within the pit limit have been treated as waste and have been assigned a zero grade. If these blocks are incorporated in the mine schedule, it should translate in a tonnage and grade increase.

- The additional 47 drill holes and channel samples that were completed after current resource cut-off date (May 6th, 2013) could be included in a new resource calculation and thus increase the current resource.

- Additional definition drilling could convert the current Inferred Resources into Indicated Resource and Reserves.

- Additional exploration drilling could expand the known mineralized zones along strike and at depth.

- Geotechnical drilling along the proposed pitwalls may allow steeper pitwalls and hence reduce the waste to ore strip ratio.

- The economic effects in developing a by-pass, for the portion of highway 393 that passes over the mineralized zone, that would permit mining a single larger pit, should be investigated.

- Future mining potential of the Duparquet Project, below the current pit shell, by an underground bulk mining method will need to be assessed and could provide an extension of the mine life.

- The province's mining industry is well established. The hydro-electricity is affordable and offers competitive advantage for companies operating in Quebec. Mining companies also enjoy relatively stable mining legislation.

- The historical Beattie Mine site surface area will be cleaned up and the current buildings demolished and removed prior to the development of the open-pit mining. The old tailing area will be reclaimed as part of the mine closure plan.

InnovExplo, Roche and Dresinger Consulting conclude that this PFS allows the Duparquet Project to advance to a feasibility study stage. InnovExplo believes that more advanced engineering work is necessary for the Duparquet Project in order to support a feasibility study and reduce the risks to the Project.

InnovExplo considers the present PFS to be reliable and thorough, and based on quality data, reasonable hypotheses and parameters compliant with NI 43-101 and CIM standards regarding mineral resource estimations.

26. RECOMMENDATIONS

Based on the results of the updated Mineral Resource Estimate and the positive outcome of the PFS, InnovExplo, Roche and Dreisinger Consulting recommend advancing the Duparquet Project to the next phase, which would consist of preparing a feasibility study.

Drilling

InnovExplo recommends further exploration drilling on the Duparquet Project to increase resources and the confidence level of the geological model. More specifically, the NE-SW striking secondary zones should be drilled to test their lateral and depth extensions. In the same way, detailed surface mapping and channel sampling is recommended to enhance the structural model.

A compilation of historical diamond drill holes is recommended on the Central Duparquet property, particularly those of the 1980s drilling programs that covered the current mineralized zones in the area.

Further definition drilling is recommended along strike and at depth to upgrade Inferred resources to the Indicated category in order to address the underground potential for all zones.

In order to advance to the feasibility stage, the reserve and resource calculations should be updated to include the results from the 47 holes that were received after the last cut-off date.

Mining

Additional geotechnical and hydrogeological studies should be undertaken to better define the pit wall slopes from those presented in this Report. In order to proceed to a feasibility level study, it is recommended that the proposed slope configurations be verified. This involves confirming the structural data over the proposed open pit footprint. Ideally, this would involve a geotechnical drilling program with a minimum of one (1) hole oriented perpendicular to each of the four pit walls (north, south, east and west).

Pit optimization should continue, and other scenarios should be investigated following an update of the resources according to previous recommendations:

- Investigation of a scenario that considers bypassing Route 393 and for which there is no road constraint;
- Investigation of a scenario that uses a contractor for overburden removal.

Metallurgy

Although intensive laboratory and pilot plant work has been conducted on representative samples and bulk samples, additional metallurgical testing will be required as the Project proceeds to the feasibility stage in order to firm up the design basis, further refine the ore processing flow sheet, and finalize the design of the mill facilities.

Recommendations for Additional Testwork:

- A variability testwork program including metallurgical testing on samples from various depths, various zones throughout the pit, and various sulphide contents. The program should also study the metallurgical behaviour for each lithology and for combined ore types.
- Grindability testwork including crusher work index, Bond ball mill grindability index, Bond rod mill grindability index, drop-weight test and abrasion index. These are required to better define the rock variability for each zone of the deposit;
- Testwork to further confirm the process and reagent selection and dosage for optimized recovery and grade;
- Additional testwork to confirm silver recovery;
- Mineralogical characterization and gold and silver deportment should also be verified;
- Further optimization tests on each unit of operation. An integrated pilot plant for the whole circuit should be operated for an extended period of time to confirm the results and also test other unit operations within the proposed flowsheet;
- Further work to investigate leaching and adsorption kinetics on lime boiled samples to determine the impact of liberated silver on precious metal leaching and adsorption kinetics to ensure the cyanidation extraction and recovery circuit configuration is designed more accurately;
- More testwork on the limestone that will be sourced for the Project, particularly in terms of optimal particle size, reaction kinetics and dosage rates.
- Further tests to accurately ascertain the rheology of the thickened stream.

Infrastructure

Condemnation drilling will be required for infrastructure locations, such as the waste rock dump, primary crusher, process plant, buildings, etc.

Geotechnical work such as geotechnical drilling, test pits, seismic analysis and stability analysis are recommended as the Project proceeds to the feasibility level in order to confirm the assumptions used for the tailings storage facility design and the waste piles. It will also confirm bedrock depth and capacity, and optimize civil design criteria related to the foundations for the plant and auxiliary facilities.

A study on borrow banks for suitable materials for use during construction of the various dykes, pads and roads, and for use as concrete aggregate should be undertaken during the feasibility stage to determine available quantities and distances from the various facilities.

Groundwater hydraulics should be verified at the feasibility stage to determine available flow rates for plant make-up water.

The current method for tailings disposal and rheology is not well known since the tailings rheology is not well understood. Two possibilities are presented:

- Compacted cyclone tailings
- Low plasticity tailings.

More study is required at the feasibility stage to determine the methods and to refine CAPEX estimations. The reduced material quantities resulting from better estimates of the bulk density should partly offset costs resulting from deposition methods and equipment.

It is recommended that negotiations with Hydro-Quebec be initiated to advance the work for installation of the power line.

Environment

In order to complete the surface water quality assessment program, it would be important to collect samples at different hydrological periods such as during spring freshet since this important hydrological period is usually different from the others. In addition, all waterbodies likely to be affected by the projected infrastructures should be comprehensively portrayed. Additional sediment quality sampling should also be done at different periods of the year in order to monitor changes in their quality.

As part of the future development of the Duparquet Project, an inventory of terrestrial habitats and wetlands will be conducted in order to list plant species present on the Property as well as those who have the potential to be present. Bird and terrestrial wildlife surveys should also be done for the subsequent realization of the project impact study.

Even if a preliminary fish habitat characterization was performed on the Property, only a specific fish field survey could confirm fish presence/absence in streams as well as which ones can be considered as fish habitats. This survey would also characterize streams that were not surveyed in August 2013 and should consequently be done as part of future project development steps.

Static leaching and acid-generation potential tests were completed using representative samples of waste rock, ore and tailings. However, kinetic tests will have to be performed to confirm that those mining residues are not leachable.
As part of the future steps of the Duparquet Project, a comprehensive hydrogeology assessment will have to be completed in order to adequately estimate the groundwater input in the water balance. Such assessment will include, among other things, mathematical modelling of the successive mine operation phases to estimate the daily flow rate required to keep the pit dry.

In order to ensure project permitting in a timely manner, Clifton Star should initiate as soon as possible the above environmental work and also cover all biophysical and socio-environmental elements typically included in an ESIA compliant with both provincial and federal requirements/guidelines. As outlined in sections 20.2.2 and 20.2.3 of this Report, Project Notice documents should be issued to initiate the environmental assessment and permitting process. It should also be noted that as part of the provincial permitting process, a Mine Rehabilitation and Closure Plan must be prepared and submitted for approval prior to obtain the general CA.

Finally, with regards to the social acceptability of the Project, it is highly recommended that Clifton Star continues informing and discussing with all stakeholders. Over recent years, representatives of Clifton Star have been present in the community and provided information on the Project to the representatives of the Town of Duparquet, the Abitibi-Ouest RCM, and governmental authorities. Such actions will have to be pursued and also extended to all other stakeholders identified in Section 20.4 of this Report.

InnovExplo, Roche and Dresinger Consulting have prepared a cost estimate for the recommended work program to serve as a guideline for the Project. The budget for the proposed program is presented in Table 26.1.

The estimated cost for the work program would amount to approximately $8.8M, and would include exploration, definition and condemnation drilling, metallurgical, geotechnical and hydrogeological testwork and a Feasibility Study final report. Table 26.1 presents the estimated costs for the phases of the recommended program.

Table 26.1 – Estimated costs for the recommended work program

Budget	Cost Estimate
Drilling	
Drilling and compilation of historical diamond drill holes.	$ 2,250,000
Resource update	$ 80,000
Subtotal:	**$ 2,330,000**
Mining	
Geotechnical study	$ 812,500
Subtotal:	**$ 812,500**
Metallurgical testwork budget	
Drilling and sample collection	$ 225,000
Sample preparation	$ 30,000
Variability testwork program	$ 120,000
Grindability testwork	$ 55,000
Testwork to confirm reagent selection and dosage (float)	$ 25,000
Additional testwork to confirm silver recovery	$ 30,000
Mineralogical characterization	$ 70,000
Integrated pilot plant of the whole circuit, operated for an extended period of time	$ 400,000
Investigation of leaching and adsorption kinetics on lime boiled samples	$ 30,000
Subtotal:	**$ 985,000**
Infrastructure work budget	
Condemnation drilling	$ 562,500
Geotechnical work	$ 150,000
Borrow pit assessment	$ 10,000
Subtotal:	**$ 722,500**
Environmental work budget	
Environmental baseline studies (surface water, sediment, fish habitat, air quality, noise level, hydrology, avifauna, wildlife, etc.)	$ 375,000
Hydrogeological study (ground water hydraulics)	$ 225,000
Geochemical characterization (kinetic tests)	$ 35,000
Social studies (economic spin-offs, landscape change analysis, traffic study, land use, risk assessment, etc.)	$ 200,000
Social acceptability program	$ 170,000
Environmental and social Impact assessment	$ 389,750
Mine rehabilitation and closure plan	$ 30,000
Subtotal:	**$ 1,424,750**
Feasibility Study Budget	**$ 2,500,000**
Total:	**$ 8,774,750**

InnovExplo is of the opinion that the recommended two-phase work program and proposed expenditures are appropriate and well thought out. InnovExplo believes that the proposed budget reasonably reflects the type and amount of the contemplated activities.

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APPENDIX I
UNITS, CONVERSION FACTORS, ABBREVIATIONS

Units

Units in this report are metric unless otherwise specified. Precious metal content is reported in gram of metal per metric ton (g/t Au or Ag) except otherwise stated. Tonnage figures are dry metric tons unless otherwise stated. Ounces are Troy ounces.

Unit Abbreviations used

°C	degrees Celsius	oz	troy ounces
ha	hectares	avdp	avoirdupois pound
g	grams	st	short ton
kg	kilograms	oz/t	ounces per short ton
µm	microns	t	metric ton (tonne)
mm	millimetres	Mt	millions of tonnes
cm	centimetres	t.milled	tonnes milled
m	metres	t.moved	tonnes moved
km	kilometres	tpd / tpy	metric tons per day /year
masl	metres above sea level	g/t	grams per metric ton
’ or ft	feet	ppb	parts per billion
cfm	cubic feet per minute	ppm	parts per million
m^3/min	cubic metres per minute	hr or h	hour
Mbs	megabytes per second	cps	counts per second
LOM	life-of-mine	hp	horsepower
$M	millions of dollars	kWh/t	kilowatt-hours per tonne
$ or C$ or CAD	Canadian dollars	kV/kVA	kilovolts/kilovolt-amps
US$ or USD	American dollars	kPa/MPa	kilo/mega pascals

Acronyms and Abbreviations – Section 20 (Environmental Studies)

AMD	Acid mine drainage
ANC	Acid neutralizing capacity
BAPE	Bureau d’audience publique sur l’environnement (Québec)
bgs	Below ground surface
CA	Certificate of authorization
EIA	Environmental impact assessment
EQA	Environmental Quality Act (Québec)
ESA-1	Environmental site assessment phase I
ESA-2	Environmental site assessment phase II
ESIA	Environmental and social impact assessment
HMR	Hazardous Materials Regulations
LPRR	Land Protection and Rehabilitation Regulation (Québec)
MPA	Maximum potential acid
MERN	Department of Energy and Natural Resources(Québec)
MDDELCC	Department of Sustainable Development, Environment and the Fight to Climate Change (Québec)
NP	Neutralization potential
RMD	Regulation respecting hazardous materials

Conversion factors for measurements

Imperial Unit	Multiplied by	Metric Unit
1 inch	25.4	mm
1 foot	0.305	m
1 acre	0.405	ha
1 ounce (troy)	31.103	g
1 pound (avdp)	0.454	kg
1 ton (short)	0.907	t
1 ounce (troy) / t (short)	34.286	g/t

APPENDIX II
IN-PIT MINERAL RESOURCE ESTIMATE
PER ZONE

Duparquet In-Pit

Clifton Star Resources Inc.

Mineral Resource Estimate - Final Results

Per Zone detail using a Cut-off grade of 0.45 g/t Au

SUMMARY

	MEASURED CATEGORY			INDICATED CATEGORY			TOTAL: MEASURED + INDICATED				INFERRED CATEGORY		
Zone	Tonnes t	Grade g/t Au	Au Oz	Tonnes t	Grade g/t Au	Au Oz	Tonnes t	Grade* g/t Au	Au Oz	Zone	Tonnes t	Grade g/t Au	Au Oz
NORTH	-	-	-	15,640,300	1.88	947,814	15,640,300	1.88	947,814	NORTH	456,900	2.06	30,276
SOUTH_01	-	-	-	1,304,700	1.09	45,759	1,304,700	1.09	45,759	SOUTH_01	400,100	0.91	11,763
SOUTH_02	110,400	1.56	5,547	3,546,600	1.41	161,045	3,657,000	1.42	166,593	SOUTH_02	410,800	1.24	16,358
SOUTH_03	25,100	0.98	793	2,198,600	1.08	76,632	2,223,700	1.08	77,425	SOUTH_03	669,000	1.06	22,813
SOUTH_04	-	-	-	2,837,700	1.17	106,493	2,837,700	1.17	106,493	SOUTH_04	2,150,800	1.09	75,570
SOUTH_05	-	-	-	1,118,600	1.21	43,682	1,118,600	1.21	43,682	SOUTH_05	408,900	0.90	11,808
SOUTH_06	-	-	-	4,001,600	1.37	176,292	4,001,600	1.37	176,292	SOUTH_06	31,400	1.30	1,310
SOUTH_07	-	-	-	714,900	1.68	38,601	714,900	1.68	38,601	SOUTH_07	377,400	1.21	14,666
SOUTH_08	-	-	-	475,900	1.06	16,199	475,900	1.06	16,199	SOUTH_08	779,400	1.99	49,921
SOUTH_09	-	-	-	27,400	1.04	918	27,400	1.04	918	SOUTH_09	376,800	0.72	8,676
SOUTH_10	-	-	-	35,700	0.64	730	35,700	0.64	730	SOUTH_10	105,500	1.38	4,686
3	-	-	-	1,694,400	1.11	60,612	1,694,400	1.11	60,612	3	1,840,500	1.46	86,252
3B	-	-	-	1,624,700	1.26	65,976	1,624,700	1.26	65,976	3B	1,032,700	1.94	64,458
WEST_01	-	-	-	7,323,900	1.67	393,249	7,323,900	1.67	393,249	WEST_01	-	-	-
WEST_02	-	-	-	1,662,100	2.13	114,086	1,662,100	2.13	114,086	WEST_02	-	-	-
WEST_03	-	-	-	265,200	1.70	14,463	265,200	1.70	14,463	WEST_03	-	-	-
RWRS	29,600	1.44	1,370	3,619,600	1.36	157,715	3,649,200	1.36	159,085	RWRS	400,100	2.21	28,479
6	-	-	-	702,100	1.81	40,874	702,100	1.81	40,874	6	68,900	1.17	2,601
CD	-	-	-	980,400	2.69	84,693	980,400	2.69	84,693	CD	377,000	2.52	30,530
CD_SOUTH	-	-	-	81,700	0.98	2,582	81,700	0.98	2,582	CD_SOUTH	25,600	0.84	689
CD_S2	-	-	-	102,700	2.30	7,589	102,700	2.30	7,589	CD_S2	95,400	1.39	4,252
O_01	-	-	-	295,300	1.88	17,883	295,300	1.88	17,883	O_01	376,100	1.41	17,027
O_02	-	-	-	82,400	1.31	3,465	82,400	1.31	3,465	O_02	123,400	0.71	2,835
O_05	-	-	-	64,200	1.02	2,104	64,200	1.02	2,104	O_05	17,700	1.26	719
O_06	-	-	-	81,600	0.69	1,820	81,600	0.69	1,820	O_06	39,100	0.65	813
O_07	-	-	-	20,200	0.64	418	20,200	0.64	418	O_07	19,300	0.56	344
O_08	-	-	-	53,500	0.88	1,515	53,500	0.88	1,515	O_08	-	-	-
O_09	-	-	-	48,300	0.71	1,104	48,300	0.71	1,104	O_09	34,600	0.67	744
O_10	-	-	-	37,300	1.16	1,396	37,300	1.16	1,396	O_10	800	0.82	21
O_11	-	-	-	62,600	0.93	1,879	62,600	0.93	1,879	O_11	100	0.50	2
O_12	-	-	-	127,800	1.39	5,716	127,800	1.39	5,716	O_12	-	-	-
O_13	-	-	-	25,600	1.03	848	25,600	1.03	848	O_13	2,600	1.04	87
O_14	-	-	-	6,900	0.56	125	6,900	0.56	125	O_14	1,000	0.54	17
O_15	-	-	-	157,900	1.08	5,468	157,900	1.08	5,468	O_15	34,800	0.74	831
O_16	-	-	-	60,300	0.92	1,775	60,300	0.92	1,775	O_16	78,900	0.80	2,027
O_17	-	-	-	161,000	1.05	5,458	161,000	1.05	5,458	O_17	-	-	-
O_18	-	-	-	82,100	1.09	2,879	82,100	1.09	2,879	O_18	8,100	0.98	255
O_19	-	-	-	29,500	0.63	599	29,500	0.63	599	O_19	-	-	-
O_20	-	-	-	187,600	1.35	8,154	187,600	1.35	8,154	O_20	49,500	1.64	2,608
O_24	-	-	-	198,200	1.02	6,478	198,200	1.02	6,478	O_24	173,100	1.11	6,202
O_25	-	-	-	174,000	1.25	6,972	174,000	1.25	6,972	O_25	54,700	0.97	1,711
O_27	-	-	-	135,500	1.09	4,735	135,500	1.09	4,735	O_27	191,100	2.76	16,947
O_28	-	-	-	162,700	0.95	4,965	162,700	0.95	4,965	O_28	26,000	1.10	923
O_29	-	-	-	159,000	0.86	4,413	159,000	0.86	4,413	O_29	39,600	1.28	1,624
O_30	-	-	-	26,800	0.70	604	26,800	0.70	604	O_30	100	0.80	3
O_31	-	-	-	70,700	0.79	1,801	70,700	0.79	1,801	O_31	6,200	0.67	134
O_33	-	-	-	43,100	0.57	787	43,100	0.57	787	O_33	200	0.48	3
O_38	-	-	-	15,700	0.67	337	15,700	0.67	337	O_38	8,500	1.58	433
O_40	-	-	-	58,100	0.84	1,563	58,100	0.84	1,563	O_40	65,100	1.32	2,760
O_41	-	-	-	0	-	-	-	-	-	O_41	-	-	-
O_42	-	-	-	113,000	0.85	3,080	113,000	0.85	3,080	O_42	75,400	1.06	2,577
O_46	-	-	-	143,600	0.94	4,337	143,600	0.94	4,337	O_46	61,200	0.84	1,658
O_48	-	-	-	0	-	-	-	-	-	O_48	-	-	-
O_49	-	-	-	83,100	0.80	2,136	83,100	0.80	2,136	O_49	14,200	0.66	302
O_50	-	-	-	52,500	0.90	1,527	52,500	0.90	1,527	O_50	124,300	6.35	25,366
DUM_01	-	-	-	0	-	-	-	-	-	DUM_01	84,200	0.78	2,116
DUM_02	-	-	-	91,800	1.47	4,352	91,800	1.47	4,352	DUM_02	87,700	0.65	1,842
DUM_03	-	-	-	0	-	-	-	-	-	DUM_03	428,500	2.25	30,958
DUM_04	-	-	-	0	-	-	-	-	-	DUM_04	82,700	0.54	1,431
DUM_05	-	-	-	0	-	-	-	-	-	DUM_05	20,500	0.64	423
ENVELOPE	-	-	-	-	-	-	-	-	-	ENVELOPE	11,755,535.27	0.85	319,778
TOTAL	165,100	1.45	7,710	53,070,700	1.56	2,666,697	53,235,800	1.05	2,674,408	TOTAL	24,092,035	1.18	910,627

* Weigthed average based on tonnage

- The Independent and Qualified Persons for the Mineral Resource Estimate, as defined by NI 43-101, are Kenneth Williamson, M.Sc., P.Geo and Karine Brousseau, Eng. under the supervision of Carl Pelletier, B.Sc., P.Geo. (InnovExplo Inc.), and the effective date of the estimate is June 26, 2013.
- Mineral Resources which are not Mineral Reserves, do not have demonstrated economic viability.
- Results are presented undiluted within Whittle-optimized pit shells. The estimate includes 60 gold-bearing zones and the envelope zone containing isolated gold intercepts.
- Resources were compiled at 0.35, 0.40, 0.45, 0.50, 0.55, 0.60, 0.65, 0.70, 0.80 and 0.90 g/t Au cut-off grades.
- Cut-off grades must be re-evaluated in light of prevailing market conditions (gold price, exchange rate and mining cost).
- A fixed density of 2.73 g/cm^3 was used in the mineralized zones and the envelope zone.
- A minimum true thickness of 3.0 m was applied, using the grade of the adjacent material when assayed, or a value of zero when not assayed.
- High grade capping was done on the raw data and established at 25.0 g/t Au for diamond drill hole assays and channel sample assays.
- Compositing was done on drill hole and channel sample sections falling within the mineralized zone solids (composite = 1 m).
- Resources were evaluated from drill hole and surface channel samples using an ID2 interpolation method in a two folders percent block model.
- The Measured category is defined by blocks having a volume of at least 25% within an envelope built at a distance of 10 m around existing Channels.
- The Indicated category is defined by the combination of blocks within a maximum distance of 15 m from existing stopes and blocks for which the average distance to drill hole composites is less than 30 m.
- Ounce (troy) = Metric tons x Grade / 31.10348. Calculations used metric units (metres, tonnes and g/t).
- The number of metric tons was rounded to the nearest hundred. Any discrepancies in the totals are due to rounding effects; rounding followed the recommendations in NI 43-101.
- InnovExplo is not aware of any known environmental, permitting, legal, title-related, taxation, socio-political or marketing issues or any other relevant issue that could materially affect the Mineral Resource Estimate.
- Input parameters used for MCoG estimation and Whittle pit design: Gold price: C$ 1,450; Gold selling cost: C$ 5.00; Mining costs: C$ 2.40; Processing cost: C$ 13.46; Transportation cost: C$ 0.25; Administration cost: C$ 4.18; Processing recovery: 93.9%; Mining recovery: 90.9%; Mining dilution: 10.0%; Overall pit slope: 52°.

APPENDIX III
UNDERGROUND
MINERAL RESOURCE ESTIMATE
ZONES DETAILS

Duparquet Underground

Clifton Star Resources Inc.

Mineral Resource Estimate - Final Results

Per Zone detail using a Cut-off grade of 2.00 g/t Au

SUMMARY

	INDICATED CATEGORY		
Zone	Tonnes t	Grade g/t Au	Au Oz
NORTH	2,442,000	2.66	208,638
SOUTH_01	400	2.46	32
SOUTH_02	500	2.14	34
SOUTH_03	400	2.44	31
SOUTH_04	161,300	2.57	13,314
SOUTH_05	10,800	2.40	835
SOUTH_06	292,300	3.50	32,891
SOUTH_07	101,700	2.75	8,979
SOUTH_08	131,700	3.09	13,094
SOUTH_09	68,200	3.01	6,607
SOUTH_10	300	2.10	20
3	30,000	2.79	2,688
3B	16,300	2.69	1,407
WEST_01	41,800	2.53	3,405
WEST_02	3,500	2.53	285
WEST_03	-	-	-
RWRS	123,900	2.98	11,868
6	5,800	2.27	424
CD	900	2.84	82
CD_SOUTH	1,200	2.61	101
CD_S2	2,500	2.45	197
O_01	-	-	-
O_02	2,400	2.15	166
O_05	-	-	-
O_06	-	-	-
O_07	-	-	-
O_08	-	-	-
O_09	400	2.12	27
O_10	800	2.28	59
O_11	18,000	4.72	2,730
O_12	-	-	-
O_13	-	-	-
O_14	-	-	-
O_15	-	-	-
O_16	-	-	-
O_17	-	-	-
O_18	3,800	2.44	298
O_19	-	-	-
O_20	-	2.22	-
O_24	-	-	-
O_25	-	-	-
O_27	-	-	-
O_28	-	-	-
O_29	-	-	-
O_30	21,500	2.72	1,877
O_31	-	-	-
O_33	1,200	2.33	90
O_38	500	2.09	34
O_40	-	-	-
O_41	35,700	3.47	3,988
O_42	-	-	-
O_46	200	2.18	14
O_48	-	-	-
O_49	-	-	-
O_50	-	-	-
DUM_01	-	-	-
DUM_02	400	2.39	31
DUM_03	-	-	-
DUM_04	-	-	-
DUM_05	-	-	-
ENVELOPE	-	-	-
TOTAL	3,520,400	2.78	314,247

	INFERRED CATEGORY		
Zone	Tonnes t	Grade g/t Au	Au Oz
NORTH	1,959,700	2.71	170,616
SOUTH_01	-	-	-
SOUTH_02	8,500	3.02	825
SOUTH_03	300	3.16	31
SOUTH_04	49,900	2.41	3,867
SOUTH_05	13,300	2.39	1,022
SOUTH_06	340,400	3.72	40,681
SOUTH_07	142,800	2.65	12,176
SOUTH_08	406,500	3.22	42,063
SOUTH_09	165,300	2.77	14,717
SOUTH_10	142,100	3.89	17,793
3	132,900	2.46	10,524
3B	122,000	3.71	14,556
WEST_01	-	-	-
WEST_02	-	-	-
WEST_03	-	-	-
RWRS	398,100	2.79	35,687
6	29,000	2.50	2,335
CD	610,500	3.26	63,901
CD_SOUTH	6,100	2.09	410
CD_S2	105,200	2.71	9,154
O_01	8,600	3.02	835
O_02	2,800	2.07	187
O_05	-	-	-
O_06	-	-	-
O_07	30,800	3.14	3,113
O_08	-	-	-
O_09	3,800	3.43	419
O_10	2,900	2.13	199
O_11	10,800	3.60	1,250
O_12	-	-	-
O_13	-	-	-
O_14	-	-	-
O_15	-	-	-
O_16	-	-	-
O_17	-	-	-
O_18	-	-	-
O_19	-	-	-
O_20	100	2.15	7
O_24	1,900	2.29	140
O_25	103,000	2.75	9,110
O_27	48,100	3.82	5,902
O_28	-	-	-
O_29	1,000	2.41	78
O_30	10,300	2.39	790
O_31	-	-	-
O_33	-	2.06	-
O_38	5,900	2.34	444
O_40	4,200	2.44	329
O_41	200	2.04	13
O_42	-	-	-
O_46	12,700	2.75	1,121
O_48	-	-	-
O_49	-	-	-
O_50	25,900	5.82	4,848
DUM_01	300	2.21	21
DUM_02	500	2.02	32
DUM_03	33,700	3.61	3,909
DUM_04	-	-	-
DUM_05	1,000	2.01	65
ENVELOPE	651,500	2.81	58,899
TOTAL	5,592,600	2.96	532,069

- The Independent and Qualified Persons for the Mineral Resource Estimate, as defined by NI 43-101, are Kenneth Williamson, M.Sc., P.Geo and Karine Brousseau, Eng. under the supervision of Carl Pelletier, B.Sc., P.Geo. (InnovExplo Inc.), and the effective date of the estimate is June 26, 2013.
- Mineral Resources which are not Mineral Reserves, do not have demonstrated economic viability.
- Results are presented undiluted, outside Whittle-optimized pit shells. The estimate includes 60 gold-bearing zones and the envelope containing isolated gold intercepts.
- Resources were compiled at cut-off grades of 1.5, 2.0, 2.5, 3.0, 3.5, 4.0 and 5.0 g/t Au.
- Cut-off grades must be re-evaluated in light of prevailing market conditions (gold price, exchange rate and mining cost).
- A fixed density of 2.73 g/cm^3 was used in the mineralized zones and in the envelope zone.
- A minimum true thickness of 3.0 m was applied, using the grade of the adjacent material when assayed, or a value of zero when not assayed.
- High grade capping was done on the raw data and established at 25.0 g/t Au for diamond drill hole assays and channel sample assays.
- Compositing was done on drill hole and channel samples sections falling within the mineralized zone solids (composite = 1 m).
- Resources were evaluated from drill hole and surface channel samples using an ID2 interpolation method in a two folders percent block model.
- The Indicated category is defined by the combination of blocks within a maximum distance of 15 m from existing stopes and blocks for which the average distance to drill hole composites is less than 30 m.
- Ounce (troy) = Metric tons x Grade / 31.10348. Calculations used metric units (metres, tonnes and g/t).
- The number of metric tons was rounded to the nearest hundred. Any discrepancies in the totals are due to rounding effects; rounding followed the recommendations in NI 43-101.
- InnovExplo is not aware of any known environmental, permitting, legal, title-related, taxation, socio-political or marketing issues or any other relevant issue that could materially affect the Mineral Resource Estimate.
- Parameters used for UCoG estimation: Gold price: C$ 1,450; Gold selling cost: C$ 5.00; Mining cost: C$ 58.00; Milling cost: C$ 13.46; Processing recovery: 93.9%; Mining dilution: 15.0%.

APPENDIX IV
TAILINGS
MINERAL RESOURCE ESTIMATE
ZONES DETAILS

Duparquet Tailings

Clifton Star Resources Inc.

Mineral Resource Estimate - Final Results

Per Zone detail using a Cut-off grade of 0.45 g/t Au

SUMMARY

	MEASURED CATEGORY			INDICATED CATEGORY			TOTAL: MEASURED + INDICATED		
Zone	Tonnes t	Grade g/t Au	Au Oz	Tonnes t	Grade g/t Au	Au Oz	Tonnes t	Grade* g/t Au	Au Oz
1	4 900	2.48	391	-	-	-	4 900	2.48	391
2	14 700	1.91	903	-	-	-	14 700	1.91	903
3	-	-	-	979 800	0.93	29 400	979 800	0.93	29 400
4	-	-	-	3 125 200	0.93	93 800	3 125 200	0.93	93 800
TOTAL	19 600	2.05	1 295	4 105 000	0.93	123 200	4 124 600	0.94	124 494

- The Independent and Qualified Persons for the Mineral Resource Estimate, as defined by NI 43-101, are Kenneth Williamson, M.Sc., P.Geo and Karine Brousseau, Eng. under the supervision of Carl Pelletier, B.Sc., P.Geo. (InnovExplo Inc.), and the effective date of the estimate is May 22, 2012.
- Mineral Resources which are not Mineral Reserves, do not have demonstrated economic viability.
- Results are presented undiluted and in situ. The estimate includes four (4) tailings zones.
- Tailings resources were compiled at cut-off grades of 0.35, 0.40, 0.45, 0.50, 0.55, 0.60, 0.65, 0.70, 0.80 and 0.9 g/t Au.
- Cut-off grades must be re-evaluated in light of prevailing market conditions (gold price, exchange rate and mining cost).
- A fixed density of 1.45 g/cm^3 was used in zones and waste.
- High grade capping was done on the raw data and established at 13.0 g/t Au for Zone 1, 3.5 g/t Au for Zone 2, 1.7 g/t Au for Zone 3 and 2.2 g/t Au for Zone 4.
- Compositing was done on drill hole sections falling within the mineralized zone solids (composite = 0.5 m).
- Resources were evaluated from drill hole samples using an ID2 interpolation method in a block model.
- The Measured and Indicated categories were defined based on the drill hole spacing (Measured: zones 1 and 2 = 30m x 30m grid; Indicated: zone 3 = 100m x 100m grid and zone 4 = 200m x 200m grid).
- Ounce (troy) = Metric tons x Grade / 31.10348. Calculations used metric units (metres, tonnes and g/t).
- The number of metric tons was rounded to the nearest hundred. Any discrepancies in the totals are due to rounding effects; rounding followed the recommendations in NI 43-101.
- InnovExplo is not aware of any known environmental, permitting, legal, title-related, taxation, socio-political or marketing issues or any other relevant issue that could materially affect the Mineral Resource Estimate.