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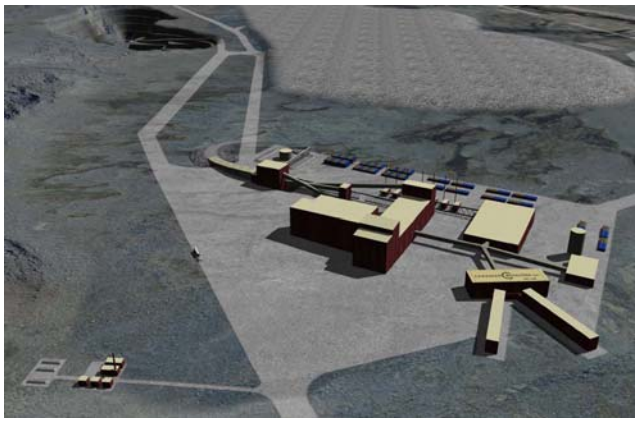
Mining and Metallurgy

CANADIAN ROYALTIES INC.

Raglan South Nickel Project
Technical Report



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Summary (Item 3)

General

This summary presents the results of a Bankable Feasibility Study (the "Study") undertaken by SNC-Lavalin Inc. (SLI) on behalf of Canadian Royalties Inc. (CRI) to evaluate the technical feasibility and economic viability of the Raglan South Nickel Project (RSNP).

Discounted cash flow modeling of the project yields a full equity base case internal rate of return (IRR) of 8.1% and a net present value (NPV) of CAD$0.85 million at a discount rate of 8%, both calculated after tax and in nominal terms. A construction escalation rate of 3.5% annually and an inflation rate of 2% during the mine operation were assumed.

Project Description

The project is located in Quebec's Ungava peninsula, approximately 1000 km north of Montreal. The project is approximately 120 km inland from the port of Deception Bay.

The project area has seen historical mining activity by the Asbestos Hill Corporation. Xstrata Nickel's currently active Raglan mine is located some 18 km from the RSNP project site; As such, infrastructures already in place facilitate the implementation of the Raglan South Nickel Project.

The project comprises the development of three open pit mines: The Expo mine, the Mesamax mine and the Ivakkak mine. The Ivakkak mine also has planned underground workings that will be developed once the open pit is exhausted.

At a site in close proximity to the Expo open pit a process plant will process ore from the pits at a nominal rate of 3,500 tonnes per day. A nickel sulphide and a copper sulphide concentrate will be produced and trucked to the port at Deception Bay for shipment.

A series of low-emissions, diesel fired, reciprocating generators will supply electrical power to the project. Waste heat will be recovered from these engines in order to increase the overall energy efficiency of the project.

Permanent accommodations are planned for 172 persons, including dining and recreation services.

A service complex will be provided to service the project's equipment fleet.

A rockfill dam will impound a reservoir to supply the project with fresh water. The reservoir capacity will provide for nine months of water consumption.

A waste rock and tailings co-disposal facility will be constructed adjacent to the Expo open pit and industrial complex.

Upgrades are planned to the existing Donaldson airport infrastructures.

A dedicated wharf and concentrate storage facility will be constructed at Deception Bay. Diesel and jet fuel handling and storage facilities will be provided.

Mining and Reserves

The Study considered only those resources in the indicated category converted to mineral reserves for the Mesamax, Expo and Ivakkak deposits.

Conventional open pit mining methods would be used to exploit 98% of the reserves of the project. Mesamax and Ivakkak ores would be blended with Expo ores for the first four years of the project. Reserves from the Ivakkak underground zones A and C would be mined in years 5 and 6 and also blended with ore from the Expo open pit.

Open pit designs incorporate allowances for appropriate access ramps, wall slope angles, catchment berms and minimum mining widths for the equipment selected. The following parameters were used in the pit design: US$6.00/lb nickel, US$1.50/lb copper, US$900/oz platinum and US$300/oz palladium.

Reserve determinations include allowances for dilution, which vary with each resource. Mining losses were assumed to be 5% (not included in the following table), and were factored into the mining plan and financial analysis. The average strip ratio for open pit mining is 3.37:1 over the life of mine, excluding the waste material extracted during the underground mine development.

The probable open pit reserve estimates for the three pits on the project are as follows:

	Ore	**Ni**	**Cu**	**Co**	**Au**	**Pt**	**Pd**	**Waste**	**Stripping Ratio**
	tonnes	%	%	%	g/t	g/t	g/t	**Tonne**	**(waste / Ore) t/t**
Mesamax	2,077,000	1.85	2.49	0.07	0.19	0.95	3.46	5,704,000	2.75
Expo	7,843,000	0.68	0.69	0.04	0.07	0.29	1.25	29,834,000	3.80
Ivakkak Pit	604,000	1.22	1.53	0.05	0.16	0.67	3.22	3,136,000	5.19
Ivakkak Underground	197,000	2.28	2.73	0.10	0.21	1.04	4.90		
Total	**10,721,000**	**0.97**	**1.13**	**0.05**	**0.10**	**0.45**	**1.86**	**38,674,000**	**3.67**

P&E Mining Consultants Inc. ("P&E") completed an open pit versus underground mining trade-off study for the Ivakkak deposit. As a result, a smaller open pit with a significantly reduced strip ratio was developed to exploit 75% of the Ivakkak reserves. P&E developed an underground mining scenario and preliminary plans.

Mining contractor Ross-Finlay 2000 Inc. estimated the underground mining costs, and this figure was incorporated in SNC-Lavalin's operating cost estimate.

Capital Cost Estimate

The current budget estimate reflects an accuracy range of – 5% to + 15%. All estimated costs are in Canadian, second quarter 2007 Dollars (CDN Dollars).

The capital costs for the project are estimated as follows:

Administration	
Airport Facilities	797,978
Buildings, Utilities and Infrastructures	69,241,546
Port Facilities (Wharf, warehouse, fuel and shiploading)	51,793,300
Power and cogeneration facilities	22,480,175
Subtotal	144,312,999
Mining	
Prestripping - Expo and Mesamax	14,087,165
Expo and Mesamax Mines	2,014,065
Expo Co-disposal Site (Golder)	2,569,423
Mining and Administrative Mobile Equipment Fleet	21,474,964
Subtotal	40,145,617
Milling	
Mineral Processing Facility (SLI, less tailings)	75,043,586
Paste tailings plant and pipeline (Golder)	8,158,934
Subtotal	83,202,520
Indirect Costs	
Construction Indirects	47,246,345
Contingency (10%)	39,025,756
Engineering, Procurement, Construction Management ("EPCM")	44,873,130
Owners Costs	39,463,632
Subtotal	170,608,864
Subtotal of Direct and Indirect Costs	438,270,000
Construction Escalation (3.5% annually)	27,430,000
Escalated Total Capital Costs	**465,700,000**

Operating Cost Estimate

The mine operating cost estimates are based on CRI owning all of the mining equipment. A subcontractor will execute the underground mining operations at Ivakkak. The operating cost estimates were developed on a total production of 10.72 M tonnes of ore over 8.5 years, at an average daily tonnage of 3,500 tonnes and a total of 38.7 M tonnes of waste rock for the mine life.

The average operating cost over the life of the project, including hauling, is $760/t of nickel concentrate.

Financial Analysis

The parameters used in the financial analysis were as follows:

- Construction Cost : CAD$465.7 M (escalated);
- Mine Life : 8.5 years;
- Nickel price: US$8 /lb in 2010, US$6.5 /lb in 2011 and US$6.0 /lb throughout life-of-mine (LOM, in 2007 USD);
- Copper price: US$2.25 /lb in 2010, US$1.9 /lb in 2011 and US$1.5 /lb throughout LOM (in 2007 USD);
- Total nickel content in Ni concentrate throughout LOM : 75,453 tonnes;
- Total copper content in Cu concentrate throughout LOM: 91,741 tonnes.

Analyses were carried out for a 100% equity financed project and the financial model does not include any financing up-front fee or equity underwriting fee.

Discounted cash flow modeling of the project yields a full equity base case IRR of 8.1 % and an NPV of CAD$0.85 million at a discount rate of 8%, both calculated after tax and in nominal terms and a construction escalation rate of 3.5% annually and an inflation rate of 2% during the mine operation were assumed.

Sensitivity analysis shows that the project IRR is mainly sensitive to variations in USD/CAD exchange rate, capital expenditure and Ni price.

Introduction and Terms of Reference (Item 4)

General

This Technical Report summarizes a Feasibility Study Report (the "Study") prepared for Canadian Royalties Inc. ("CRI") by SNC-Lavalin International ("SLI") according to the terms of a Contract dated August 16, 2006 between the two companies. The Study aims to assist CRI in its assessment of the technical feasibility and financial viability of its project to establish an open pit mining operation and a nickel, copper, palladium and platinum group elements (PGE) containing ore treatment facility for their Raglan South Nickel Project ("RSNP") situated in the northern most part of Quebec, Canada.

Maps showing the location of the project are presented in Figure 1 and 2 on the next pages.

Basis of the Study

The SLI study includes a review of studies performed to this date including test work undertaken since the completion of the conceptual study "Preliminary Economic Assessment" (July 2006) by P & E Mining Consultants (P&E). The scope of SLI services included the following:

- Mining plans preparation and mineral reserves estimation for the Mesamax, Expo and Mequillon deposits using the Mineral Resource Evaluations prepared by P&E;
- Determination of the basic criteria and the design of the crushing and material handling functions;
- Review of test work, studies and the existing documentation concerning the concentration process. Design of a mineral processing flowsheet based on the results of the metallurgical testing. Establishment of a material balance and selection of the proper equipment;
- Determination of suitable locations for the tailings and waste rock dumps;
- Definition of the infrastructure requirements;
- Port handling facility;
- Additional requirement to the existing airstrip;
- Elaboration of a project schedule that encompasses all the activities required for project execution;
- Estimation of the capital costs related to the construction of the required installations and to the purchase of the equipment and its erection;
- Estimation of the operating costs related to manpower, consumable supplies, reagents, energy, maintenance, etc.;

- Preparation of an economic evaluation and financial analysis for the project;
- Preparation of the final Study.

The following items were not the responsibility of SLI. The work has been coordinated by CRI and the results have been incorporated in the final Study:

- Mineral resources evaluation;
- Mine plans preparation and reserve evaluation for the Ivakkak deposit, open pit and underground;
- Condemnation drilling;
- Site surveying;
- Environmental assessment;
- Geotechnical investigations (for open pits and surface structures);
- Geochemical and acid base accounting test work;
- Design of the quay for the port at Deception Bay;
- Waste rock and tailings disposal.

Sources for the Study

The study by SLI was based upon data, design criteria and information developed by other consultants under contract to CRI and supplied to SLI by CRI for inclusion into the Study. SLI's mandate did not require that SLI validate said data, design criteria and information, however SLI has conducted a reasonable level of due diligence in accordance with SLI's standard duty of care as Engineer, in order to rely on the previous work of other consultants.

In particular, SLI made extensive use of:

- Technical Report by H. Thalenhorst (Strathcona) and T. Keast, May 29, 2003 on the South Trend Group of Properties, Nunavik, Quebec (the **"Strathcona Report"**). and Raglan South Nickel Project, Nunavik, Quebec Technical Report and Preliminary Economic Assessment on the Mequillon, Mesamax, Expo and Ivakkak Deposits (May 5, 2006 Revised as of July 24, 2006) by P & E Mining Consultants (P&E), P&E Report 113 (the **"P&E Report 1"**);
- Technical Report (2007) and Resource Estimate Update on the Ivakkak Ni-Cu-PGE Deposit, South Trend Property, Raglan South Nickel Project. (December 31, 2006 Signed March 22, 2007) by P & E Mining Consultants (P&E), P&E (the **"P&E Report 2"**);

- Technical Report and Resource Estimate Update on the Expo Ni-Cu-PGE Deposit Raglan South Nickel Project Nunavik, Quebec. (December 24, 2006 Signed February 26, 2007) by P & E Mining Consultants (P&E), P&E Report 129 (the **"P&E Report 3"**).

These reports, containing a description of the geology, the exploration to-date, the geological models and resource estimates of the four deposits, were electronically filed with SEDAR, the System for Electronic Document Analysis And Retrieval, www.sedar.com on April 2, 2007, March 22, 2007 and February 26, 2007 respectively. P&E Reports 1, 2 and 3 are collectively defined as the P&E Reports in this document.

Sections 2, 3 and 4 of the Study were based on corresponding sections of the P&E Reports and, as noted hereafter, a number of the items which would otherwise form parts of this Technical Report have already been filed with SEDAR.

Three representatives of SLI have visited the site; Mr. Simon Desjardins, P.Eng., PhD, a civil engineer, to evaluate the influence of the permafrost or infrastructure design, Mr. Michel Fitzgibbon, a cost estimator to review local conditions for cost estimating purposes, and Mr. Pierre Demers, P.Eng., a mining engineer, who acted as project sponsor on behalf of SLI.

Figure 1 Project Location

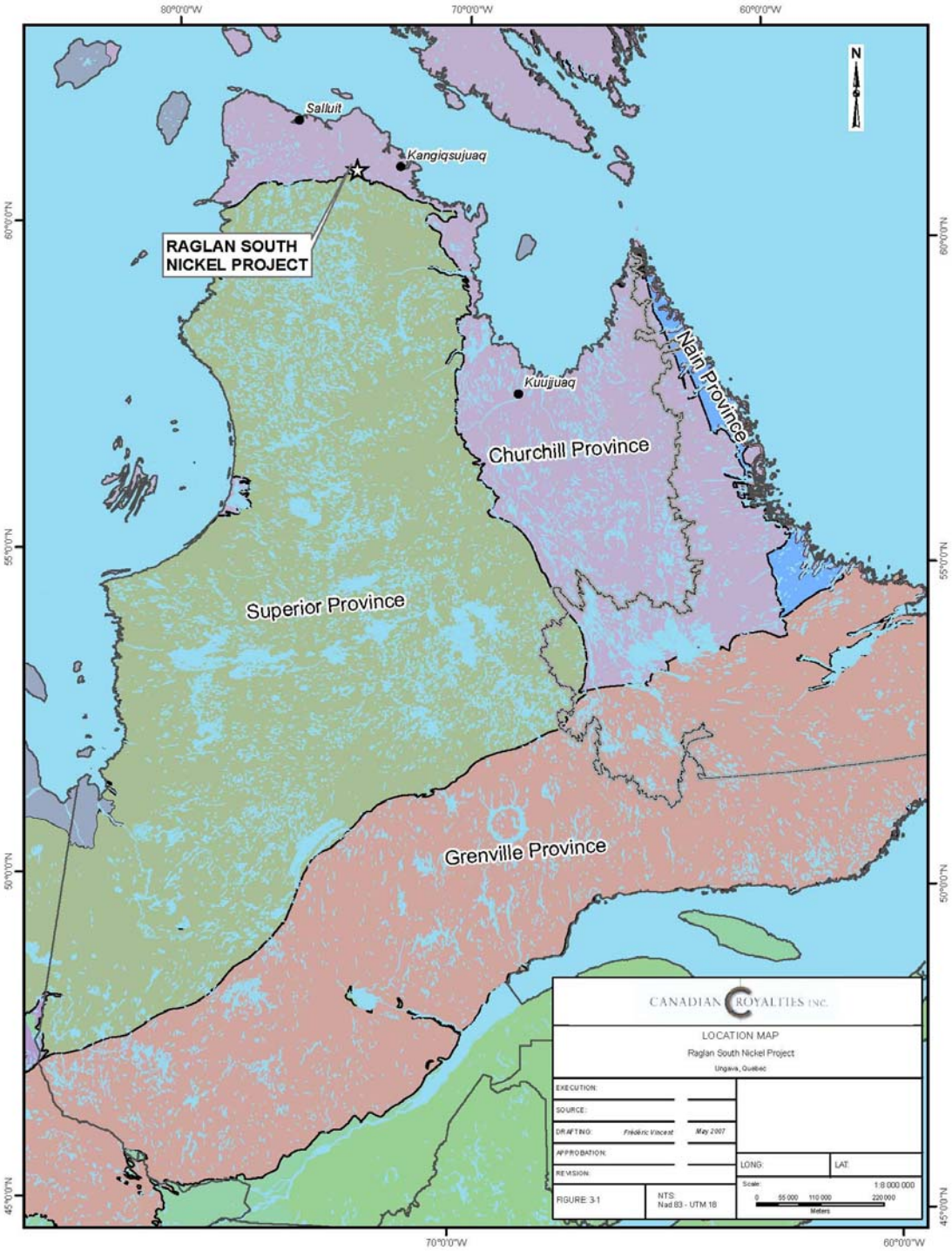
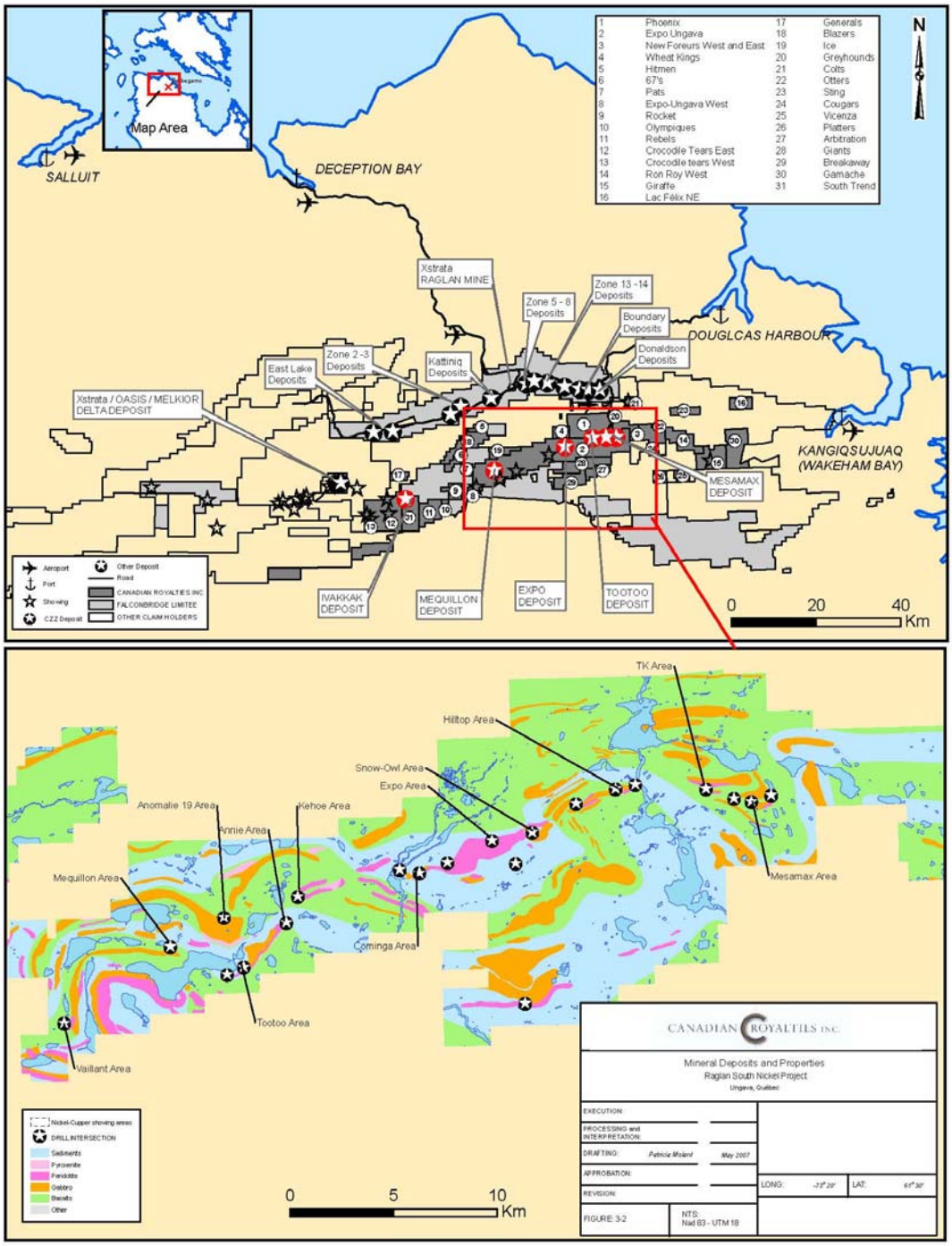


Figure 2: Raglan South Nickel Project Location Map



Reliance on Other Experts (Item 5)

The Study incorporates material from other consultants which was reviewed according to the methodology stated in Item 4, “Sources of the study”. The parties are independent and their technical work were not managed but only coordinated by SLI for inclusion into the Study. The recommended approaches, the detailed analyses and the precision of the data and calculations remain the total responsibility of each consultant. The following presents the respective contribution from each consultant:

- P & E Mining Consultants Inc. (P&E):
  - Prepared the geological models and block plans and performed the estimation of the updated mineral resources of the various deposits deposits (Mesamax, Expo, Ivakkak and Mequillon). The mineral resources are published in distinct Technical Reports for each pit and were transmitted to SLI for use in the preparation of the mining plans and reserves;
  - Prepared the open pit / underground mining tradeoff study for the Ivakkak deposit, and the open pit and underground reserves for the Ivakkak deposit including the pit configuration and the underground development with the operating costs details for the underground operation supported by a third party contractor quotation.
- Golder Associates Ltd. (Golder):
  - Geotechnical test work that supplied the input for the various slope angles for the open pit design and waste pile slopes;
  - Geochemical and acid base accounting test work that forms the design basis of the tailings management systems for design of the concentrator tailings disposal systems as well as for the mine waste rock disposal systems;
  - Design and estimation of the capital costs for the waste rock disposal facilities at the Mesamax and Ivakkak sites;
  - Design and estimation of the capital and operating costs for the waste rock and tailings co-disposal facilities at the Expo site;
  - Design and estimation of the capital and operating costs for the paste plant and pipeline system at the Expo site.
- GENIVAR:
  - Environmental and Social Impact Assessment (ESIA) for the Raglan South Nickel Project;
  - Design and estimation of the capital costs for the quay at the port site to be located at Deception Bay.

- SGS Lakefield Research Limited:
  - SGS Lakefield Research Limited carried out pilot plant and bench scale metallurgical test work on the ores that will be mined. These results were used to determine the design parameters for the concentrator as well as for the determination of the Net Smelter Returns (NSR) of the ore blocks in the resource and reserve estimates.

Property Description and Location (Item 6)

*Previously filed in the **P&E Reports**.*

Accessibility, Climate, Local Resources, Infrastructure, Physiography (Item 7)

*Previously filed in the **P&E Reports**.*

History (Item 8)

*Previously filed in the **P&E Reports**.*

Geological Setting (Item 9)

*Previously filed in the **P&E Reports**.*

Deposit Types (Item 10)

*Previously filed in the **P&E Reports**.*

Mineralization (Item 11)

*Previously filed in the **P&E Reports**.*

Exploration (Item 12)

*The text of this Section is taken from the corresponding sections of the **P&E Reports**.*

CRI completed exploration programs on the RSNP over six consecutive seasons commencing with the first program during the summer of 2001. Significant exploration programs at the Expo, Mesamax, Mequillon, Ivakkak deposits and other areas were performed at a combined cost of approximately CDN$38 million. An annual expenditure breakdown over the six-year period is provided in Table 1:

Table 1: Summary of Exploration Expenditures 2001 – 2006

Year	Exploration Expenditures
2001	$1,132,128
2002	$2,612,311
2003	$5,042,188
2004	$6,476,869
2005	$10,267,568
2006	$12,574,852
	Total: $38,105,916

2001 RSNP Exploration

During July, August and September of 2001, CRI completed an integrated exploration program on portions of its RSNP. The 2001 exploration program included geological mapping, prospecting, ground magnetometer surveys, horizontal loop electromagnetic (HLEM) surveys, surface pulse-EM surveys, down-hole pulse EM surveys, geochemical surveys, stream sediment surveys, diamond drilling, and the re-logging and assaying of historical drill core. Geological mapping and prospecting in conjunction with airborne and ground magnetometer surveys identified and delineated highly prospective ultramafic rocks hosting nickel-copper-PGE mineralization.

Some highlights of the 2001 program were the discovery of massive sulphides of the TK deposit and the confirmation of persistent, widespread PGE mineralization within the historic Expo deposit.

2002 RSNP Exploration

CRI's 2002 field exploration program focused on exploring areas deemed favourable for hosting Ni-Cu-PGE mineralization. Areas explored in 2002 included mineralized ultramafic bodies identified during previous exploration programs, areas identified by CRI during the 2001 exploration program, areas of documented sulphide occurrences within or adjacent to ultramafic bodies, and geophysical targets along the ultramafic trend with coincident surface mineralization. Diamond drilling by CRI in 2001 and 2002 at several areas intersected Ni-Cu-PGE values in disseminated, net textured and massive styles of sulphide mineralization. The highlight of the 2002 program was the discovery of wide intervals of massive sulphide mineralization at the Mesamax deposit.

2003 RSNP Exploration

The exploration in 2003 of the Expo northeast zone yielded some of the widest intersections of Ni-Cu-PGE mineralization discovered to-date at the Expo deposit. In total, 65 holes were drilled at the Expo deposit. At the Mesamax deposit, 39 additional infill and definition drill holes

confirmed the continuity of the deposit and extended the deposit to the east, west, and in some places at depth. Other diamond drilling was carried out at the Mequillon, Cominga and Tootoo areas with promising results. CRI conducted an extensive AeroTEM airborne electromagnetic survey in 2003, of 4,818 line kilometres on the central and eastern portion of the RSNP.

2004 RSNP Exploration

In 2004 CRI began undertaking pre-development studies in preparation for any future development of mining operations on the property. Mineralogical, petrological, metallurgical, mineralogical studies to determine PGE distribution, environmental baseline studies, and additional evaluation of the setting, origin, and emplacement of the Ni-Cu-PGE mineralization in the Expo and Mesamax deposits were initiated. In total, over 25,000 metres of new diamond drilling was completed in 2004, in part tracing the continuation of the Mequillon deposit well beyond its 400 metre length for at least 1,000 metres, better defining the Expo deposit, and completing pre-feasibility drilling on the Mesamax deposit.

In summary, the 2004 exploration work started the process of laying the groundwork for new or updated resource estimates on the three principal deposits, and CRI initiated metallurgical test work on the three principal deposits, in preparation for project scoping studies.

2005 RSNP Exploration

The 2005 exploration program on the RSNP included 30,829 metres of diamond drilling on the Mesamax and Ivakkak deposits, together with the collection of a 33 tonne representative bulk sample of large diameter drill core from the Mesamax deposit to be used as feed for pilot plant metallurgical testing (commenced in January 2006). A number of studies initiated in 2004 were completed in 2005.

Some highlights from the 2005 program are the following:

- Over 30,000 metres of diamond drilling was completed with five diamond drills. Efforts were directed primarily at expanding the Mequillon deposit, the systematic collection of the metallurgical bulk sample at Mesamax, and exploration drilling which in part led to the Ivakkak discovery (press release September 21, 2005);

- A 15 kilometre road from the vicinity of the Donaldson airport on Xstrata Nickel's Raglan mine site to the Mesamax deposit was constructed, allowing equipment access to the deposit area, and the ground transportation of personnel and supplies;

- A 33 tonne bulk sample of Mesamax ore was collected. The sample was transported to Douglas harbour by road, then by ship and truck to SGS Lakefield Research Limited (SGS Lakefield). Pilot plant metallurgical test work commenced in January, 2006;

- Environmental baseline field studies were conducted for the third season;

- CRI conducted a significant AeroTEM airborne electromagnetic survey in 2005, of 2,674 line kilometres on the western portion of the RSNP.

2006 RSNP Exploration

Between July and October 2006, CRI successfully completed its sixth consecutive exploration season on the RSNP. A large part of the drilling performed during 2006 was dedicated towards geotechnical tasks, condemnation drilling and infill-drilling necessary for the completion of the Study. However, CRI also completed some significant exploration drilling, the focus of which was the expansion of the Ivakkak deposit, the extension of the Mequillon deposit, the exploration of the Tootoo zone and exploration at depth near the Mesamax deposit.

The exploration program resulted in 24,659 metres of diamond drilling in 163 holes for the year ended December 31, 2006. Some highlights from the 2006 program are as follows:

- Preliminary Economic Assessment (PEA) completed during July, 2006;
- Drilling of 24,659 meters, up from a planned 15,000 meters;
- Bulk metallurgical test (Pilot Plant) completed in September, 2006;

Drilling (Item 13)

*The text of this Section is taken from the corresponding sections of the **P&E Reports**.*

CRI has expended over $38 million on exploration in the region since 2001, covering a broad range of exploration activities including diamond drilling of 941 holes totaling 119,821 metres. The bulk of the drilling was confined to relatively shallow depths (average hole depth 130 metres) as the mandate has been to explore for near-surface deposits.

Table 2 summarizes the exploration drilling activities of CRI during the period 2001 to 2006.

Table 2: Summary of Drilling Activity by CRI 2001 to 2006

Exploration Season	2001	2002	2003	2004	2005	2006
Total Program Cost ($ Millions)	$1.1	$2.6	$5	$6+	$10 +	$12+
Drilling metres	1,500 m	10,000 m	20,000 m	25,000 m	30,000 m	~24,600 m
No. of Holes	15	116	170	245	232	163

Drilling of the Mesamax Deposit

The Mesamax area contains two discrete, generally east-west trending ultramafic units, referred to as the north ultramafic and south ultramafic bodies. Both are hosted within a package of rocks comprised of basalt, gabbro and pelitic sediments. Although Ni-Cu-PGE mineralization has been outlined within both intrusive bodies, the southern ultramafic has received most of the recent exploration efforts and is host to the Mesamax deposit. CRI initiated a diamond drill program on the Mesamax deposit in 2002 in order to follow up on the favourable 2001 exploration results. A total of 41 holes were completed in 2002, for a total of 3,500 meters, defining the mineralization along a strike length of 250 metres

Subsequent grid drilling on a 25 and 50-metre pattern broadly outlined a near-surface massive sulphide lens with a heavy net textured upper portion. Mineralization occurs in subcrop below three to five metres of overburden and to date has been traced to vertical depth of 85 metres. During 2004 another 36 drill holes with a total length of 3,900 metres were drilled. An additional 26 holes were drilled in 2005 for a total length of 1,900 meters.

In addition to the drilling for the resource estimating, 14 holes were drilled in 2004 for metallurgical flotation test work. The drill program in 2005 consisted of 63 diamond drill holes, which were drilled exclusively to collect a 33 tonne bulk sample for metallurgical testing purposes in order to establish the basis for the design of a milling and concentration plant. In 2006, Canadian Royalties drilled 19 diamond drill holes and deepened two holes from the 2003 program. Three holes were also drilled for geotechnical engineering purposes, however, these holes were neither split nor sampled as they were required in their entirety for specific whole core testing procedures.

Table 3 summarizes the drilling completed on the Mesamax deposit by CRI during the period 2001-2006.

Table 3: Summary of CRI Drilling on the Mesamax Deposit 2001-2006

Year	Total Holes	Total Meterage
2001 - 2003	76 exploration holes	7,400 m
2004	27 exploration holes 14 metallurgy holes (oxide)	1,900 m (1,640 m)
2005	63 HQ metallurgical holes	33 tonne bulk sample
2006	19 holes and 2 extensions	3,438 m

Drilling of the Mequillon Deposit

The initial discovery of Mequillon net-textured mineralization was in 1957/58. During that period 5 holes were drilled. CRI drilled 6 holes in 2002 and 12 additional holes in 2003 for a total of 2,243 metres, covering a strike length of 500 metres on sections 100 metres apart. Given the encouraging results from the 2002-03 drilling a further 11,757 metres in 77 holes was completed in 2004.

A drilling program consisting of 5,081 metres in 19 holes was completed by CRI in late 2005. At present the mineralized zone at the Mequillon deposit is defined over a total length of 1.4 kilometres and to an approximate depth of 260 metres. The 2006 drilling program on the Mequillon deposit consisted of 2,827 metres in 9 holes over various portions of the deposit. Additional diamond drilling is planned for the future to further test the open easterly extension of the deposit.

The drill programs at Mequillon completed to the end of 2006 are summarized in Table 4.

Table 4: Summary of CRI Drilling on the Mequillon Deposit 2001-2005

Year	Company	Core Size	No. of Holes	Length (metres)
1958	Compagnie Minière de L'Ungava	EX	5	699
2002	CRI	BQ	6	573
2003	CRI	BQ	12	1,670
2004	CRI	BQ	77	11,757
2005	CRI	BQ	19	5,081
2006	CRI	BQ	9	2,827
TOTAL			**119**	**22,607**

Drilling of the Expo Deposit

CRI field activities at Expo commenced in 2001 with a comprehensive program of re-logging, re-sampling and re-assaying the historic drill core that was stored at the historic Expo camp site. The Expo deposit was originally discovered in 1967 by Expo Ungava Mines Ltd. As part of the general exploration program by CRI on the project, a substantial program of surface diamond drilling was completed in 2003 and 2004 to test the entire known area of mineralization at Expo.

With the completion of 111 diamond drill holes (11,578 metres of drilling) in 2003 and 2004, CRI better defined the series of mineralized zones that extend over a 1,000 metre length and which occur up to 120 metres in depth and across 150 metres in width.

At the end of 2004 the Expo database comprised 145 surface drill holes with a total length of more than 15,000 metres on 23 sections. These figures exclude the drill holes that were drilled in the general vicinity of the Expo deposit but which have no bearing on the resource estimate, and some of the old drill holes that were not used because they were either twinned by a CRI drill hole or had an unreliable location.

In 2005, 6,153 metres in 31 holes were drilled at Expo, and in 2006, 4,578 metres were drilled in 44 drill holes, along with eight geotechnical holes, four of which were included in the resource calculation.

All of the drill programs completed by CRI at Expo to the end of 2006 are summarized in Table 5.

During January, 2007 CRI made public a significant resource increase as reported by P & E Mining Consultants Inc. in a fully compliant Technical Report dated February 26, 2007 at the Expo deposit as a result of infill drilling performed during the 2005 and 2006 exploration seasons.

Table 5: Summary of CRI Diamond Drilling on the Expo Deposit

Year	Company	Core Size	Number of Holes	Length (m)	No. of Holes in Resource Estimate
1967	Expo Ungava	AXT	8	731	3
1968	Expo Ungava	AXT	35	3,281	22
1969	Amax	AXT	34	5,439	4
1997	High North	BQ	6	1,023	6
2003	CRI	BQ	67	6,747	67
2004	CRI	BQ	43	4,825	43
2005	CRI	BQ	31	6,153	31
2006	CRI	BQ	44	4,578	40
Total			**268**	**32,777**	**216**

Drilling of the Ivakkak Deposit

In 2005, CRI began an exploration program at the western extremity of the RSNP area, which led to the discovery of the Ivakkak deposit. Thirty-two delineation holes, for a total of 3,532 metres were drilled in 2005 with an average hole length being 107 metres. The drill pattern included multiple holes with a variety of inclinations from the same drill set-up at approximately 25 metre intervals over a strike length of 275 metres with the deepest intersection at a vertical depth of 135 metres. The diamond drilling program in 2006 consisted of 58 diamond drill holes (BQ size core) totalling 7,362 metres and were drilled between July 9, 2006 and September 10, 2006.

Sampling Method and Approach (Item 14)

*The text of this Section is taken from the corresponding section of the **P&E Reports**.*

Samples were obtained during the 2002 field season by two methods: grab samples were taken in the field to characterize the host rock and approximate metal accumulations in obviously mineralized material, and drill core was sampled to determine the metal concentrations in a quantitative way.

Grab Samples

Geological mapping, prospecting and collecting of individual rock samples (“grab” sampling) from outcrop or frost-heaved boulders was part of the daily geologic routine in the field. Sampling was focused on locating mineralization in outcrop or frost-heaved boulders, which offer an effective sampling medium of the sub-crop geology. Grab samples by their nature provide only a characterization of any mineralization found, and being representative is not expected.

Geological mapping and prospecting were completed on picketed grids, and sample locations were referenced from the established grid coordinates. During reconnaissance prospecting traverses off the grid areas, GPS units and aerial photographs were utilized to establish an accurate position. Rock samples 0.5-1.0 kg in size were hammered from outcrop and frost-heaved boulders. A unique sample number was assigned to each grab sample, with that unique sample number recorded on the sample bag and a corresponding sample tag placed in the bag.

Coordinates (grid and/or UTM) for each sample were recorded, as well as rock type, mineral content, and other relevant observations. A small rock sample from the same site was collected in a separate bag to be stored at the exploration camp for reference purposes. The sample sites were labelled in the field with a sample number recorded on flagging tape so that the site may be relocated if required.

Samples were sealed in sample bags with twist ties. The majority of rock samples were examined in camp by the geologists to ensure consistency in mapping, mineral identification and sulphide estimation. This information was entered on computer spreadsheets. Samples

were placed in shipping bags, tagged and sealed with plastic tie straps for shipping to the assay laboratory.

Drill Core Sampling

All core logging and core splitting was completed in a single core shack at the Expo camp site. Core was logged by CRI geological personnel. Discussions and observations by the core loggers ensured consistent unit identification. The distance between the depth markers added by the drill personnel was measured to check for misplaced markers and for lost core. All logging information was recorded directly into laptop computers. Core intervals identified for sampling were marked with wax crayons, with sample tags placed at the start of a sample interval.

The sample length in obvious mineralization was generally one metre, but individual samples were not allowed to cross lithologic contacts or abrupt changes in mineralization. Core was split in half using a hydraulic core splitter, with a sample tag placed in the bag and the bag sealed with staples. Individual bagged samples were placed in shipping bags. Where possible, contiguous sample tag series were used for core logging. Sample intervals were recorded on sample ticket books, and later recorded on the computerized drill logs.

The authors of the P&E reports were not aware of any drilling, sampling or recovery factors that would impact the reliability of the core samples. The even distribution of the sulphides in both the massive and disseminated sulphides sampled ensured that the samples were of high quality and representative of the material or mineralization being sampled.

Sample Preparation, Analysis and Security (Item 15)

*The text of this Section is taken from the corresponding section of the **P&E Reports**.*

Drill core from each of the CRI projects, i.e. Expo, Mesamax, Mequillon, and Ivakkak, was transported from the drill to a central core handling facility at the main Expo site. This procedure ensured that the core from each property was logged, sampled, packaged, and shipped in a uniform and consistent fashion. After the core was logged and marked for sampling by the project geologist it was split with individual half-core samples being placed in shipping bags (approximately 25 kg/bag). These bags were then sealed with an electrical tie and the bags then taped. The sample shipments were kept either at the local camp or the central (Expo) camp until time for fixed wing transport to the south. On the flight day the samples would be moved to the Donaldson airport and later loaded onto CRI's chartered aircraft for transport to Val-d'Or. Generally all samples were offloaded at the Val-d'Or airport and taken by truck directly to the ALS Chemex lab facility.

ALS Chemex is a reputable laboratory who provides international analytical services to the mining and mineral exploration industry. ALS Chemex is registered under ISO 9001:2000 quality standard. At the time of delivery, the lab would acknowledge receipt of the sample

shipment. Samples were prepared in Val-d'Or and shipped via bonded air carrier to the ALS Chemex facility in Vancouver, B.C. for analysis.

The analytical procedure employed for Ni, Cu, Co was by fusion-ICP-AES. Sample decomposition was accomplished by sodium peroxide fusion and samples were analyzed by inductively coupled plasma - atomic absorption spectroscopy. Detection limits for the metals were 0.002 % for Co and 0.005% for Cu and Ni. The analytical procedure employed for Au, Pt and Pd was by fire assay on a 30 gram aliquot with an ICP-AES finish. The detection limit for all three metals was 0.03 ppm.

Quality Control / Quality Assurance

CRI instituted a quality control program using uncertified reference material standards. Two uncertified reference material standards were inserted in this program in order to monitor the various metals.

The standard used for massive sulphides was prepared at SGS-Lakefield and named CR-CS01-MS. The standard was prepared as follows: The sample as received required drying and grinding. Drying was done in an oven at 105° Celsius and the grinding was performed in a rod mill. A portion of the ground sample was verified to be 98% passing -200 mesh. The material was then "tumble-homogenized" for 24 hours. The homogenized sample was then rotary split and packaged into plastic bottles of 1 kg aliquots. Each bottle was sub-sampled and a composite sample prepared. The composite sample was split into 18, 100-g sub-samples. Six of the sub-samples were submitted for analysis at SGS Lakefield, Acme Analytical Laboratories Ltd. (Acme) and ALS Chemex. Assay results of the material from all three labs (a total of 18 x 3 = 54 samples) were used to characterize the reference material.

A second standard of the less massive (referred to as *net-textured*) material was prepared for CRI by SGS-Lakefield. The standard, CR-CS02-NT was prepared in the same manner as the massive sulphide standard. A total of 54 samples, (18 per lab) were used in the sample characterization. A second version named CR-CS03-NTv2, which was lower grade, was prepared at a later date by CDN Resource Labs Limited of Vancouver, with the round robin assaying being coordinated by CDN Resource Labs. Due to the inability to accurately reproduce the values for the net textured standard, only 20 values were used in calculating the mean for Ni, Cu and Co, and 30 values were used in calculating the mean for Au, Pt and Pd. Only the values obtained from SGS Lakefield and ALS Chemex were used while the values from Acme and Assayers were rejected.

In addition to the QC samples inserted in the field by CRI, ALS Chemex prepared and analyzed their own lab duplicates and inserted their own internal reference standards and blanks. A complete set of the ALS Chemex lab QC data files was obtained and verified by Strathcona for the Expo, Mesamax, and Mequillon deposits and by P&E for the Ivakkak deposit. There were no abnormalities detected in either the procedures or the results.

For further information on sampling and analysis the reader is referred to the individual 43-101 reports prepared for each of the foregoing properties and available at SEDAR under Canadian Royalties Inc. It is P&E's opinion that the sample preparation, security, and analytical procedures were satisfactory. P&E is also confident that the integrity of the samples is uncompromised, given the security measures employed and the sample handling protocols in use at the assay laboratories.

Data Verification (Item 16)

The verification of data input for the resources estimation performed by P&E and which has been certified by P&E in their technical reports are filed electronically on Sedar. SLI has conducted a reasonable level of due diligence in accordance with SLI's standard duty of care as Engineer, in order to rely on the previous work of other consultants.

All open pit designs and mineral reserves derived from the mineral resources are based on a computer generated block models of the deposits developed by P&E using Gems software, while the pit geometry and mine plan were also designed using the Whittle and Gems software packages from Gemcom. In the process of performing the reserve calculations, SLI made random verifications to compare the ore tonnage and grade calculation vs declared resource tonnages for each deposit. All comparisons were equivalent.

*The text of the following Section is taken from the corresponding section of the **P&E Reports**.*

Drill hole Location Survey Data

As part of the general CRI field program on the RSNP, J. L. Corriveau & Associates Inc. ("Corriveau") of Val-d'Or was engaged by CRI starting in 2003 to provide survey control for the precise locations of all field grids and diamond drill holes. For each of the three deposits, Expo, Mequillon, Mesamax, the collars of all surface drill holes were surveyed by Corriveau, in 2003-04, using a differential GPS method. The surveys are considered accurate to within about 0.1 metre in all three dimensions. Drill-hole deviation was monitored with a combination of acid dip tests and Reflex down-hole dip and azimuth measurements. The holes drilled in 2005 and 2006 were also surveyed by Corriveau.

Core Loss

It has been noted and reported by Strathcona in their various reports that the core recovery at all the projects has been virtually 100%, including the mineralized intervals. Historical core recoveries, as observed during the re-sampling program undertaken by CRI in 2001 on the Expo property, were described by Keast and Thalenhorst (2005) as being comparable to that experienced by CRI in their own drilling. The core recovery of the historical holes although not quantified is estimated by Strathcona at around 95%. Given the overall excellent recoveries and the even distribution of the minerals of economic interest in the core, there is no reason to

believe that core recovery has in any way influenced the reliability of the assay database for any of the projects (Keast and Thalenhorst, 2003, 2005).

Bulk Density

The numerous specific gravity determinations available for the various rock types at the RSNP are summarized in Table 6, which also presents the bulk density figures chosen for the various resource estimates.

Table 6: Summary of Specific Gravity and Derived Bulk Density Figures

	Specific Gravity Results			Bulk Density
	Number	Mean	Std Deviation	tonnes per m^3
Mesamax Property				
Sulphides				
Massive (fresh)	66	4.60	0.23	4.5
Massive (oxidized)	24	4.39	0.10	4.0
Net-Textured	43	3.26	0.12	3.2
Host Rocks	51	2.99	0.09	3.0
Volcanics	51	2.99	0.09	3.0
Sediments	5	2.91	0.13	2.9
Gabbro	24	3.05	0.10	3.0
Pyroxenite	23	3.07	0.10	3.1
Mequillon Property				
Sulphides				
Disseminated	9	3.00	0.07	3.0
Net-Textured	8	3.11	0.09	3.1
Host Rocks				
Volcanics	1	3.02		3.0
Unmineralized Ultramafic	12	2.98	0.11	3.0
Expo Property				
Sulphides				
Disseminated	47	2.94	0.11	2.95
Net-Textured	47	3.15	0.12	3.1
Massive	17	4.49	0.38	4.4
Vein Mineralization	3	4.00	0.83	3.7
Host Rocks				
Sediments	20	2.76	0.07	2.8
Overburden	None			2.4
Ivakkak Property				
Sulphides				
Massive	32	4.37		4.37
Net-Textured	38	3.22		3.22

Assay Database Verification

Where historical assay data were available (such as at Expo) they were originally entered by hand from paper copies of drill logs. The historical drill data at Expo was verified and used in the resource estimate database while data from the eight historical holes at Mequillon were not used in the resource estimate nor was any historical data included in the Ivakkak or Mesamax databases.

The historical drill logs for those projects previously drilled were found to be concise and adequate. The information present in the original logs was used in subsequent work, and for the geologic model underlying the resource estimate. The project assay databases are now in good

condition and can be used with confidence (Thalenhorst, 2004 and Keast and Thalenhorst, 2003, 2005).

ALS Chemex Internal Quality Control

ALS Chemex made available the results of their internal quality assurance/quality control (QA/QC) data generated in the autumn of 2002 during the assaying of samples originating from the RSNP. The ALS Chemex internal QA/QC measures included the addition of standards into the sample stream at the rate of one in twenty (5%) overall. In addition, “procedure blanks” were added at the rate of one in forty, and a like proportion was selected for duplicate assay, inserted into the same tray of 20 samples that also contained the original sample. The duplicate assays characterize the precision of the assay process.

Given the decision to use ALS Chemex as the main assay lab and considering that CRI did not have its own external standards in place this early in the project, a thorough review of the assay results using the internal standards employed by ALS Chemex was undertaken.

Overall, the review revealed that the precision for the base metals was excellent, while the precious metals show a degree of “nuggety” behaviour that makes some of the individual assays unreliable, particularly for gold. This is due to the nature of the mineralization and not to the Chemex assay procedures. Thalenhorst and Keast concluded that the assay data produced by Chemex was precise and the drill hole database for the deposits based on the assay data is accurate.

Precision and Accuracy of the Historical Data

The drill data on the Mequillon, Mesamax and Ivakkak deposits has been derived from drilling carried out by CRI as early as 2002. No historical data has been incorporated into the databases for these projects and thus all of the data collected has been the subject of careful QA/QC procedures that have been analyzed by both CRI and independently by Strathcona. The Expo deposit, however, has had considerable historical drilling that has been re-sampled and included in the project database. Little is known about the sample preparation and QA/QC protocols used in the past and the historical nickel and copper results have been interpreted as being anomalously high (Thalenhorst, 2005). Strathcona has therefore factored the old assay data, to the extent that they remain in the current resource assay database, by factors of 89% for nickel and 84% for copper. The new, lower figures were used for the current resource estimate, replacing the old assay data in the database.

The historical drill assay data at Expo did not include any analysis for cobalt. To rectify this, Strathcona ran a regression analysis of the CRI-generated nickel and cobalt values for the Expo deposit obtaining a very tight correlation. Attempts to develop reliable predictors for the precious metals from either the nickel or copper values did not meet with success, and no gold, platinum or palladium values are thus available for that part of the historical assay database that has not been re-assayed.

Residual Cobalt Values

Thalenhorst (2004) investigated the correlation of the nickel and cobalt assays. This work showed that the regression line between the two data sets intercepts the cobalt axis at around 50 to 60 ppm. The correlation between the two elements is excellent at 97%, a reflection of the substitution of cobalt for nickel in the mineral pentlandite.

Statistics of External QA/QC

In 2003 and 2004, CRI staff added a total of 1,042 standards and blanks to the 14,200 drill core samples submitted for assay from the RSNP (Expo, Mesamax and Mequillon). There were 171 massive sulphide standards, 401 net-textured sulphide standards and 470 field blanks. Strathcona (2003, 2004, 2005) and CRI continuous monitored the QA/QC assay procedures.

The flotation testwork undertaken by SGS Lakefield on a variety of samples from the Mesamax, Expo and Mequillon deposits has given additional data with which to gauge the accuracy of the assay database. Table 7 compares the grades predicted for nickel, copper, platinum and palladium from the core assays with those obtained by SGS Lakefield. There was no systematic assaying at SGS Lakefield for cobalt or gold during the testwork.

Table 7: Comparison of Core Assays and Flotation Testwork Grades (SGS Lakefield)

	Flotation Testwork Head Grades					Core Assay Results			
Material	**Ni (%)**	**Cu (%)**	**Pt (g/t)**	**Pd (g/t)**	**No. of Tests**	**Ni (%)**	**Cu (%)**	**Pt (g/t)**	**Pd (g/t)**
Sulphides									
Massive	3.8	3.4	1.4	6.1	11	3.8	3.4	1.2	5.8
Mesamax	3.6	5.2	1.8	2.6	9	3.7	5.3	1.5	2.2
Net-Textured Mesamax	0.9	1.1	0.7	2.9	11	0.9	1.1	0.6	2.6
	0.8	1.4	0.6	4.7	15	0.8	1.0	0.4	3.3
Net-Textured Mequillon	0.9	1.2	0.8	3.4	4	0.8	1.2	0.7	3.3
Net-Textured	0.8	0.7	0.3	1.5	13	0.8	0.7	0.3	1.4
Expo	0.7	0.9	0.4	1.8	14	0.8	0.9	0.4	1.8
Total/Average	**1.33**	**1.61**	**0.7**	**2.7**	**70**	**1.36**	**1.57**	**0.6**	**2.4**

The expectation that the nickel and copper values in the massive sulphides might be under-reported in the drill core is not supported, as the drill hole and flotation test head values for the massive sulphide composite are indistinguishable.

Conclusions

The results of the assay verification measures undertaken by CRI for the assay databases of the Expo, Mesamax, Mequillon and Ivakkak deposits indicate no major shortcomings. Only a small amount of the original historical assay data survives into the current databases; mainly at the Expo property. Check sampling, check assaying and twin-hole drilling has shown the historical assays for copper and nickel to be high, and appropriate factors have been applied to the surviving data. Three-quarters of the samples used for the resource estimates originated in

the years 2003, 2004 and 2005 and was subjected to a comprehensive QA/QC program.There is acceptable correlation between the ALS Chemex assay data on the QA/QC sulphide standards employed by CRI and the associated flotation testwork head grades produced by SGS Lakefield.

It is the opinion of the independent consultants (Strathcona and P&E) that the databases for the Expo, Mesamax, Mequillon and Ivakkak deposits, have levels of both precision and accuracy that are satisfactory for the estimation of feasibility-level mineral resources.

Adjacent Properties (Item 17)

*The text of this Section is taken from the corresponding section of the **P&E Reports**.*

The reader is cautioned that the qualified person from P&E who prepared the various technical reports from which this sub-section has been taken, has been unable to verify the information concerning adjacent properties and it is emphasized that the information in this section is not necessarily indicative of the mineralization on the property that is the subject of this technical report.

Mineralization at the nearby Raglan mining complex now operated by Xstrata Nickel. is associated with nine separate peridotite flows, which make up an overall ultramafic body, that has been identified over a distance of 55 kilometres. Deposits along this ultramafic body include from east to west the Donaldson, Boundary, West Boundary, 13-14, 5-8, Katinniq, East Lake, 2-3 Zone and Cross Lake deposits. The thickness of the sulphide lenses varies from a few metres to a few tens of metres, and the strike length can vary from tens of metres to 200 metres. Production began in April 1998 at Katinniq, which consists of over 20 discrete lenses of massive and disseminated sulphide, which vary in size from 10,000 tonnes to 1.4 million tonnes (Falconbridge Limited, Website May 2003). The lenses extend along an ultramafic horizon 1,400 metres in strike length, which dips to the northwest at 45° to 50°. The mineralized horizon has been traced to a depth of 600+ metres and is open in that direction. Proven and probable reserves for the Raglan operation currently stand at 14.7 million tonnes grading 2.80% nickel and 0.78% copper, plus measured and indicated resources of 14.7 million tonnes grading 3.12% nickel and 0.86% copper (derived from Xstrata Nickel Ore Reserves and Mineral Resources, March 6th, 2007).

In addition to the Expo deposit, the Delta deposit has historically been known on the South Raglan Trend. The Delta deposit, in which Falconbridge owns 51%, Oasis Diamond Exploration Inc. 34% and Melkior Resources Inc. 15%, is reported to have a “mineral inventory” of 0.8 million tonnes at 3.1% nickel, 1.1% copper, 1.0 g/t platinum, 1.6 g/t palladium and 0.2 g/t gold as determined by Falconbridge (Melkior Annual Report 2002). This resource estimate is not NI 43-101 compliant.

Mineral Processing and Metallurgical Testing (Item 18)

Mineral Processing

Metallurgical testing of samples from the RSNP deposits began in 2004, when SGS Lakefield conducted preliminary testing on Mesamax samples and Expo net-textured samples in order to provide projections for two types of flowsheets.

In 2005, testwork was conducted on the Mesamax, Mequillon and Expo deposits including grindability investigations, flotation development, tailings characterizations and concentrate characterizations.

Major findings were that:

- Expo ore is dictating the fineness of primary grinding;
- Copper floated preferentially for Mesamax. It was not as obvious with Expo ore.

Mineralogical Study

At the end of 2005 CRI prepared a 33 t bulk sample in order to perform a pilot plant test at SGS Lakefield.

The following summarizes the body of work completed on CRI's RSNP Mesamax Ni-Cu-PGE deposit. The scope of the program encompassed comminution studies, laboratory and pilot plant flotation, concentrate and tailings characterization, and determination of the filtration characteristics. The following tables 8 to 10 show respectively the analytical analysis from the pilot plant; the results from locked cycle tests and the estimated flotation metallurgy by pit.

The main objectives of the program were:

- To prepare an overall composite from 33 tonnes of HQ core considered to be representative of the Mesamax deposit;
- To study the breakage and grindability characteristics of the composite and generate design data for milling;
- To confirm and optimize flotation conditions on a pilot plant feed sample in the laboratory with the focus on reagent conditions, and flowsheet configuration;
- To process the remainder of 33 tonnes of ore in a flotation pilot plant using conditions as set in the laboratory, confirming the established flowsheet and further optimizing conditions as required;
- To provide complete metallurgical balances and summaries describing the overall performance of ore processing;

- To provide circuit design data to be used by an engineering company to design a plant;
- To evaluate the effect of process water recirculation on pilot plant performance and environmental considerations; and
- To provide detailed characterization of tailings (solid-liquid separation design data, environmental) and concentrate characterization (solid-liquid separation testing, self-heating testing, detailed analytical analysis) from the pilot plant products.

Table 8: Analytical Analysis – Pilot Plant

Ore Type	Product	Wt %	Assays, %, g/t Cu	Ni	Pt	Pd	% Distribution Cu	Ni	Pt	Pd
Mesamax Massive	Comb Concentrate	31.5	16.6	9.34	2.87	6.2	99.2	81.4	59.7	93.0
	Cu Clnr Concentrate	11.9	32.1	0.76	1.91	5.8	72.8	2.5	15.1	33.2
	Ni Bulk Concentrate	19.6	7.13	14.6	3.46	6.4	26.5	78.9	44.7	59.8
	Combined Bulk Training	68.5	0.058	0.98	0.89	0.21	0.8	18.6	40.3	7.0
	Head (calc.)	100.0	5.19	3.61	1.51	2.10	100.0	100.0	100.0	100.0
Overall Net (mesamax/ Mequillon)	Comb Concentrate	8.9	11.3	7.66	4.66	31.8	96.6	73.0	62.3	85.7
	Cu Clnr Concentrate	3.0	29.0	0.97	3.69	68.5	84.3	3.0	17.3	64.1
	Ni Bulk Concentrate	5.9	2.16	11.2	5.16	12.4	12.3	70.0	45.0	21.6
	Combined Bulk Training	91.2	0.037	0.29	0.28	0.50	3.4	27.0	37.7	14.3
	Head (calc.)	100.0	1.03	0.94	0.66	3.19	100.0	100.0	100.0	100.0
EXPO	Comb Concentrate	7.9	9.56	7.23	3.51	16.6	83.9	77.1	74.7	85.5
	Cu Clnr Concentrate	2.1	26.9	0.67	5.61	33.7	64.3	1.9	32.6	47.4
	Ni Bulk Concentrate	5.8	3.05	9.73	2.72	10.2	19.5	75.2	42.1	38.1
	Combined Bulk Training	92.1	0.16	0.18	0.10	0.24	16.1	22.9	25.3	14.5
	Head (calc.)	100.0	0.90	0.74	0.37	1.53	100.0	100.0	100.0	100.0

Table 9: Results from locked Cycle Tests

Test No.	Comp	Sizing K_{00} µm	Objectives	Product	Wt %	Assays, %, g/t Cu	Ni	Pt	Pd	Cp	Pn	Po	NSG	% Distribution Cu	Ni	Pt	Pd	Cp	Pn	Po	NSG
MQN-LCT1	MUNC 'fresh' dc	Pri: 56	-Initial locked-cycle test with similar conditions to MN-LCT5 -sulphite	Comb Conc	8.9	12.3	7.16	5.95	31.2	35.7	20.2	23.2	20.8	98.0	76.8	72.1	87.9	98.0	82.1	13.0	2.4
		Cu: n/a		Cu Clnr Conc	3.4	28.7	0.75	5.28	62.2	83.2	2.08	5.62	9.13	87.7	3.1	24.6	67.3	87.7	3.2	1.2	0.4
				Ni Bulk Conc	5.5	2.10	11.2	6.37	11.9	6.09	31.5	34.3	28.1	10.3	73.7	47.5	20.5	10.3	78.8	11.8	2.0
		Cu/Ni: 18		Cu/Ni Tailing (calc)	15.5	0.013	0.64	0.33	0.52	0.037	1.35	41.6	57.0	0.2	11.9	6.9	2.5	0.2	9.5	40.6	11.2
				Bulk Rougher Scav Tail	75.6	0.028	0.12	0.20	0.40	0.080	0.24	9.72	90.0	1.9	11.3	21.0	9.6	1.9	8.4	46.4	86.4
				Combined Bulk Tailing	91.1	0.025	0.21	0.22	0.42	0.073	0.43	15.1	84.4	2.0	23.2	27.9	12.1	2.0	17.9	87.0	97.6
				Head (calc.)	100.0	1.12	0.83	0.73	3.16	3.24	2.19	15.9	78.7	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
MQN-LCT1	MUNC 'fresh' dc	Pri: 56	-Initial locked-cycle test with similar conditions to MN-LCT5 -sulphite	Comb Conc	7.7	14.5	8.04			42.1	22.8	16.5	18.6	96.6	71.8			96.6	76.8	8.0	1.8
		Cu: n/a		Cu Clnr Conc	3.0	31.4	0.65			91.0	1.89	-2.55	9.70	80.8	2.2			80.8	2.5	-0.5	0.4
				Ni Bulk Conc	4.7	3.89	12.7			11.3	36.0	28.5	24.3	15.8	69.5			15.8	74.3	8.4	1.5
		Cu/Ni: 18		Cu/Ni Tailing (calc)	11.4	0.084	1.21			0.24	2.97	44.5	52.9	0.8	16.0			0.8	14.8	31.8	7.7
				Bulk Rougher Scav Tail	80.9	0.036	0.13			0.11	0.24	11.8	87.8	2.6	12.3			2.6	8.4	60.2	90.5
				Combined Bulk Tailing	92.3	0.042	0.26			0.12	0.57	15.9	83.5	3.4	28.2			3.4	23.2	92.0	98.2
				Head (calc.)	100.0	1.15	0.85			3.32	2.24	15.9	78.5	100.0	100.0			100.0	100.0	100.0	100.0

Final estimated average flotation metallurgy by deposit is summarized in Table 10.

Table 10: Estimated Flotation Metallurgy by Pit

	Grade						Distribution (%)					
	%Cu	%Ni	%Co	Pt(g/t)	Pd(g/t)	Au(g/t)	Cu	Ni	Co	Pt	Pd	Au
Mesamax Pit												
Head	2.44	1.80	0.07	0.30	3.06	0.19	100	100	100	100	100	100
Cu Conc	29.8	0.8	0.02	2.04	36.7	2.54	86.9	2.3	1.4	12.9	53.9	70.4
Ni bulk Conc	1.1	13.4	0.46	3.35	11.6	0.30	9.9	76.2	61.1	40.2	32.3	13.9
Expo Pit												
Head	0.64	0.63	0.03	0.27	1.16	0.07	100	100	100	100	100	100
Cu Conc	24.10	1.22	0.06	3.3	34.20	2.64	64.88	3.60	2.73	20.52	47.07	55.41
Ni bulk Conc	1.84	6.85	0.34	1.8	6.58	0.28	17.49	71.45	57.54	40.61	31.94	20.91
Mequillon Lake Pit												
Head	0.60	0.43	0.02	0.40	1.52	0.12	100	100	100	100	100	100
Cu Conc	28.7	0.75	0.04	5.28	62.2	9.55	87.7	3.09	2.7	24.6	67.3	88.6
Ni bulk Conc	2.1	11.2	0.54	6.37	11.9	0.35	10.3	73.7	58.5	47.5	20.5	5.2
Ivakkak Pit												
Head	1.55	1.19	0.05	0.62	3.06	0.15	100	100	100	100	100	100
Cu Conc	30.5	0.97	0.04	2.28	37.00	1.88	76.10	3.09	2.57	14.5	57.0	53.9
Ni bulk Conc	3.51	10.3	0.45	2.68	9.89	0.54	19.8	73.5	71.4	38.5	34.3	34.7
Blend Mesamax/Expo (40%/60%)												
Head	1.50	1.38	0.05	0.59	2.34	0.14	100	100	100	100	100	100
Cu Conc	26.1	1.02	0.02	1.91	21.1	1.96	84.8	3.5	1.8	15.7	43.8	66.1
Ni bulk Conc	1.25	10.3	0.45	2.69	8.82	0.25	8.4	74.0	60.1	45.4	37.7	13.9

The CRI has two other deposits, Mequillon and Ivakkak that needed additional testing for the purpose of the Study.

The Mequillon net-textured ore has been tested to determine its flotation response. There was no grindability testing done on the Mequillon ore at the time of the BFS. Flotation results from the Mequillon deposit have indicated that its response was very similar to the net-textured component of the Mesamax deposit.

The Ivakkak ore is comprised of two distinct zones, a massive sulphide zone grading 2.55% Ni and 2.89% Cu and a net-textured zone grading 0.44% Ni at 0.9% Cu. The Ivakkak zones were tested in early 2007.

The purpose of the additional testing program was to determine the Ball Bond work indices and Rod Bond work indices and to confirm the individual responses of the Mequillon and combined Ivakkak ore and to test the response to blending ore from each deposit.

Two samples were submitted for bench-scale testing. These were a Mequillon sample (Eastern and deeper part of the deposit), and an Ivakkak ore sample. Approximately 150-kg of Mequillon ore and approximately 150-kg of the Ivakkak ore were required to complete the test program. Ore was received at SGS Lakefield as a coarse minimum of ½" or ¼" drill core with minimum dimension of ½".

Each of the ore samples were subjected to the following sample preparation:

- Ore received was inventoried and weighed;
- The sample was stage-crushed to ½", blended and 25-kg was riffled out for grindability testwork;
- The 25-kg grindability sample was split with 15-kg dedicated for Bond rod mill work index testing and the remainder stage-crushed to 6 mesh for Bond ball mill work index testing;
- The remainder of the sample was stage-crushed to -10 mesh and blended. Approximately 60-kg was riffled out and rotary split into 2-kg test charges and a head sample for assay;
- A fraction of the remainder was combined with the other sample in a ratio set by SLI as an overall representative of blended Mequillon-Ivakkak ore sample. It was anticipated that this blend will be 4 parts Mequillon to 1 part Ivakkak. The blended ore was rotary split into 2-kg test charges and a head sample for assay. Sufficient components of each of the ores must be used to make up 40-kg of blended ore;
- All test charges and bulk reject material was stored in a freezer to minimize opportunities for oxidation;
- Head samples were submitted for Cu, Ni, S, Pt, Pd, Ni(sulf).

The test for gold, platinum and palladium were assayed by fusion fire assays, ICP OES and the cobalt, copper and nickel were assayed by pyrosulfate fusion, XRF.

Metallurgical Test Work

Crushing:

It was recognized that the ore contained an asbestos-form tremolite. Necessary dust collection and related safety precautions were taken throughout the sample preparation processes. Various samples were submitted for an assessment of asbestos mineral content. Three accredited independent laboratories quantified the Mesamax pilot plant feed ore as containing in the range of 0.5% - 1% tremolite.

Grinding:

Grindability testing on the massive component of pilot plant ore determined the ore to be soft in terms of impact breakage, abrasion breakage and hardness. The Bond ball mill work index was 9.0 kWh/t. The net-textured component was variable in grindability response, but was on average hard with respect to impact and abrasion breakage. Bond ball mill work indices ranged from 10.5 – 13.7 kWh/t and averaged 12.8 kWh/t. Pilot plant feed ore was determined to be soft overall with a Bond ball mill work index of 10.5 kWh/t. The direct measurement from the pilot plant averaged 9.3 kWh/t.

For the additional test, the two samples were submitted for Bond rod mill work index testing (RWI) and Bond ball mill work index testing (BWI).

The Bond RWI test is performed according to the original Bond procedure. It requires 15 kgs of minus ½" material that is preferably prepared at the testing facility. The Bond RWI has been widely used for rod mill (or primary ball mill) sizing.

The Bond BWI test is performed according the original Bond procedure. It requires 10 kgs of minus 6-mesh material that is preferably prepared at the testing facility. The Bond BWI has been widely used for mill sizing, but is also utilized in computer simulation, and variability testing. The closing mesh size for these tests will be 150 mesh, consistent with previously tested CRI samples.

In order to calibrate the laboratory mill against flotation product size, an allowance was given for a single grind and particle size analysis on each of the samples. This will establish the grinding time required to achieve the desired product size of 80% passing 75 microns, and is part of mill selection criteria (see results in tables 6-1 and 6-2).

Flotation:

Bench-scale flotation was completed on pilot plant feed ore prior to pilot plant operations. A flowsheet identified in previous test programs was adopted. The optimum pH range in the bulk

rougher and scavenger stages for copper and nickel metallurgy is on the lower side of pH 10. Pilot plant collector dosages should be set to achieve approximately 20% mass pull to the bulk rougher concentrate and a further 15 – 20% mass pull in the bulk scavenger circuit to ensure that sufficiently high grade Ni concentrates are met at maximum recovery.

A pilot plant was set up to treat 33 tonnes of Mesamax ore considered to be representative of the deposit. The pilot plant test was completed over 13 day-long runs and one extended run lasting more than 50 hours. The pilot plant metallurgy was refined and optimized. Over the day-long runs recycle process water was gradually introduced into the operations. The recycle rate of the process water was determined to be 91% under stable operation.

Additional tests investigated the separate response of the Mequillon and the Ivakkak ore samples to the developed flowsheet to gauge the appropriate reagent conditions and the amenability of the established flowsheet. The flotation testing also investigated the response of a blended Ivakkak-Mequillon ore.

Six tests were carried out on the Mequillon ore. It was anticipated that the provided ore sample would respond in a similar manner to previously tested Mequillon samples. Three of these tests were rougher kinetics tests and the remainder were batch cleaner tests.

Six rougher tests and six batch cleaner tests were performed on Ivakkak ore. The testing focused on optimization of rougher collector and non-sulphide gangue depressant, appropriate mass recovery split between the bulk rougher and bulk scavenger, and pyrrhotite depressant dosage in the Cu/Ni circuit. A confirmatory locked cycle test was conducted under optimized conditions.

Testing on a blend of Ivakkak-Mequillon ore was initiated once the above testing was completed. One rougher test and one batch cleaner test were performed to confirm the anticipated response. One locked cycle tests was completed on blended ore to assess concentrate grade-recovery relationship in a continuous environment.

Products were analyzed for Cu, Ni, and S. Batch cleaner tests generated no more than 10 products each that were assayed for Cu, Ni, and S. Locked cycle tests generated no more than 40 products. Products were assayed for Cu, Ni, Pt, Pd. Selected products were additionally assayed for Pt and Pd. Various particle size analyses were conducted on selected test products.

Tailings Desulphidation:

The purpose of the testing program was to evaluate potential techniques to desulphidize tailings from RSNP ore in order to obtain non acid-generating tailings.

Flotation testing has been performed in order to explore sulphide recovery from the process tailings. Tests have explored the roles of collector, CMC, alkalinity, and promoters in the recovery kinetics of the sulphides present. It was expected that successful recovery of sulphides

would require neutral to slightly acidic pulp pH, moderate amounts of a strong collector such as potassium-amyl-xanthate (PAX), and carboxy-methyl-cellulose (CMC) to control floatable magnesium silicates.

Up to 90% of sulphide could have been removed, but there was no guarantee that the treated tailings would not generate acid. Golder's geochemical analysis with potentially acid generating waste rock that contained a small amount of sulphide, showed that there was a technical risk related to this concept. Desulphidation also meant that two different tailings ponds would be required which would be more complex to manage. The tailings desulphidation was therefore not further considered.

Magnetic Separation Testing:

In addition to flotation testing, the potential of magnetic separation to enhance sulphide recovery and concentrate grades was explored. Magnetic separation in several stages was carried out to test different concentrate stages. High-intensity magnetic separation is used to recover magnetic pyrrhotite.

Two tests were carried out on Ivakkak ore, using the first Ni-Cu cleaner feed which showed promising potential for rejecting up to 80% of the pyrrhotite in one and 60% in the second test.

Mineral Resource and Mineral Reserves Estimates (Item 19)

Mineral Resources

*The text of this Mineral Resource sub-section is based on that of the corresponding section of the **P&E Reports**.*

CRI provided P&E with Microsoft Excel files, drill logs and assay certificates for the RSNP deposits and cross sections for the deposits were developed. A Gemcon database of drill hole data was also provided by CRI to P&E. The database was validated in Gemcon and only minor corrections were made where required. Domain boundaries were determined from lithology, structure and Ni, Cu and PGE boundary interpretations from visual inspection of the drill hole sections. Four domains were developed named Massive Sulphide, Net-textured, Vein and Ultramafic. Rock codes used for the resource models were derived from the mineralized domain solids. In addition rock codes for Air, Waste Rock and Overburden were used.

Length weighted composites were generated for the drillhole data that fell within the constraints of the above mentioned domains. Grade capping was investigated on the raw assay values in the combined domains to ensure that the possible influence of erratic high values did not bias the database.

Variography was not carried out on the constrained domain composites within the four domains in the deposit models. The mineralized domains exhibited good sectional continuity but due to erratic distributed populations, did not yield discernable variograms.

The average bulk densities for each deposit were based on samples tested in ALS Chemex laboratories, and were calculated to be as follows: 4.5 t/m^3 for Massive Sulphide ore, 3.1 t/m^3 for Net-textured ore, 2.9 t/m^3 for Waste Rock and 2.0 t/m^3 for Overburden.

The resource models were divided into 3D block model frameworks. The block models have blocks which were 5 m in the X-direction, 5 m in the Y direction and 5 m in the Z direction. Separate block models were created for rock type, density, percent, Ni, Cu, Co, Au, Pt and Pd. The percent block model was set up to accurately represent the volume and subsequent tonnage that was occupied by each block inside each constraining domain. The Ni, Cu, Co, Au, Pt and Pd composites were extracted from the Microsoft Access database composite table into separate files for each mineralized zone. Inverse distance squared ($1/d^2$) grade interpolation was utilized.

The resource classification of all interpolated grade blocks were determined from the Ni interpolations for indicated and inferred resources due to Ni being the dominant revenue producing element in the Net Smelter Return (NSR) calculation. All blocks coded by the first interpolation pass were coded as indicated while the remaining blocks coded on the second interpolation pass as inferred.

The resource estimate was derived from applying a NSR cut-off grade to the block models. Tables 12 – 15 show the Global Mineralization Estimates (which are the mineralized resources within the block models upon which the Whittle optimization programs have been performed).

Table 12: Mesamax Global Mineralization Estimate

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INDICATED								
NSR Cut-Off $C/t	TONNES	Ni %	Cu%	Co%	Au g/t	Pt g/t	Pd g/t	NSR $C/t
$100	1,857,933	2.135	2.880	0.086	0.226	1.061	3.910	$316.89
$95	1,903,119	2.099	2.835	0.085	0.222	1.047	3.874	$311.68
$90	1,941,552	2.069	2.798	0.084	0.219	1.035	3.843	$307.34
$85	1,990,555	2.034	2.752	0.082	0.216	1.021	3.804	$301.92
$80	2,029,871	2.005	2.716	0.081	0.213	1.010	3.772	$297.67
$75	2,060,913	1.984	2.689	0.081	0.211	1.002	3.752	$294.36
$70	2,084,727	1.968	2.668	0.080	0.209	0.995	3.732	$291.83
$65	2,105,386	1.954	2.651	0.080	0.207	0.989	3.715	$289.63
$60	2,120,707	1.943	2.638	0.079	0.206	0.985	3.702	$287.99
$55	2,138,183	1.932	2.623	0.079	0.205	0.980	3.688	$286.11
$50	2,149,515	1.926	2.614	0.079	0.204	0.977	3.679	$284.88
$45	2,160,933	1.919	2.605	0.078	0.204	0.974	3.670	$283.62
$40	2,173,740	1.911	2.594	0.078	0.203	0.971	3.659	$282.20
$35	2,188,940	1.901	2.581	0.078	0.202	0.967	3.646	$280.50
$30	2,200,550	1.894	2.572	0.077	0.201	0.964	3.636	$279.19
$25	2,209,585	1.888	2.565	0.077	0.200	0.962	3.627	$278.17
$20	2,215,705	1.884	2.560	0.077	0.200	0.960	3.621	$277.46
$15	2,219,133	1.882	2.557	0.077	0.200	0.959	3.618	$277.06
$10	2,222,109	1.880	2.554	0.077	0.199	0.958	3.614	$276.70
$0.01	2,228,679	1.875	2.548	0.077	0.199	0.956	3.607	$275.91

INFERRED								
NSR Cut-Off $C/t	TONNES	Ni %	Cu%	Co%	Au g/t	Pt g/t	Pd g/t	NSR $C/t
$100	20,647	1.239	1.619	0.060	0.131	0.527	3.055	$193.64
$95	21,004	1.225	1.615	0.059	0.130	0.526	3.038	$191.99
$90	21,004	1.225	1.615	0.059	0.130	0.526	3.038	$191.99
$85	21,004	1.225	1.615	0.059	0.130	0.526	3.038	$191.99
$80	21,004	1.225	1.615	0.059	0.130	0.526	3.038	$191.99
$75	21,004	1.225	1.615	0.059	0.130	0.526	3.038	$191.99
$70	21,004	1.225	1.615	0.059	0.130	0.526	3.038	$191.99
$65	21,004	1.225	1.615	0.059	0.130	0.526	3.038	$191.99
$60	21,004	1.225	1.615	0.059	0.130	0.526	3.038	$191.99
$55	21,004	1.225	1.615	0.059	0.130	0.526	3.038	$191.99
$50	21,004	1.225	1.615	0.059	0.130	0.526	3.038	$191.99
$45	21,004	1.225	1.615	0.059	0.130	0.526	3.038	$191.99
$40	21,004	1.225	1.615	0.059	0.130	0.526	3.038	$191.99
$35	21,052	1.224	1.612	0.059	0.130	0.526	3.035	$191.63
$30	21,572	1.211	1.589	0.059	0.128	0.522	3.000	$187.79
$25	21,592	1.211	1.588	0.058	0.128	0.522	2.999	$187.65
$20	21,592	1.211	1.588	0.058	0.128	0.522	2.999	$187.65
$15	21,592	1.211	1.588	0.058	0.128	0.522	2.999	$187.65
$10	21,592	1.211	1.588	0.058	0.128	0.522	2.999	$187.65
$0.01	21,613	1.210	1.587	0.058	0.128	0.522	2.997	$187.47

Table 13: Expo Global Mineralization Estimate

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INDICATED								
NSR Cut-Off $C/t	TONNES	Ni %	Cu%	Co%	Au g/t	Pt g/t	Pd g/t	NSR $C/t
$100	2,858,334	1.288	1.204	0.063	0.091	0.490	1.954	$173.81
$95	3,200,196	1.225	1.152	0.060	0.091	0.471	1.897	$165.64
$90	3,611,741	1.161	1.100	0.057	0.090	0.451	1.834	$157.31
$85	4,032,296	1.105	1.053	0.055	0.089	0.433	1.779	$150.04
$80	4,484,226	1.053	1.011	0.053	0.087	0.417	1.726	$143.22
$75	5,008,233	1.002	0.969	0.051	0.086	0.400	1.669	$136.33
$70	5,605,843	0.951	0.927	0.048	0.084	0.383	1.610	$129.52
$65	6,285,502	0.901	0.888	0.046	0.082	0.367	1.549	$122.81
$60	7,049,190	0.853	0.848	0.044	0.080	0.351	1.487	$116.27
$55	7,930,712	0.807	0.808	0.042	0.077	0.334	1.422	$109.73
$50	8,847,431	0.765	0.771	0.040	0.075	0.318	1.359	$103.80
$45	9,822,197	0.726	0.735	0.039	0.072	0.303	1.299	$98.20
$40	10,901,498	0.688	0.700	0.037	0.070	0.288	1.237	$92.68
$35	12,183,534	0.649	0.662	0.035	0.067	0.271	1.169	$86.87
$30	13,901,527	0.604	0.617	0.033	0.063	0.252	1.090	$80.12
$25	16,609,408	0.548	0.557	0.031	0.057	0.227	0.988	$71.51
$20	20,579,327	0.487	0.491	0.028	0.051	0.200	0.872	$62.04
$15	24,826,319	0.438	0.436	0.026	0.046	0.178	0.777	$54.42
$10	28,108,910	0.408	0.402	0.025	0.043	0.164	0.716	$49.54
$0.01	29,644,489	0.393	0.387	0.024	0.041	0.158	0.690	$47.36

INFERRED								
NSR Cut-Off $C/t	TONNES	Ni %	Cu%	Co%	Au g/t	Pt g/t	Pd g/t	NSR $C/t
$100	136,826	1.213	1.223	0.060	0.074	0.420	1.711	$158.97
$95	149,862	1.171	1.189	0.059	0.075	0.409	1.670	$153.59
$90	172,379	1.107	1.133	0.056	0.076	0.391	1.606	$145.57
$85	186,018	1.071	1.098	0.055	0.076	0.384	1.576	$141.27
$80	204,303	1.029	1.051	0.054	0.075	0.375	1.550	$135.99
$75	228,500	0.978	1.016	0.051	0.074	0.362	1.491	$129.74
$70	261,731	0.930	0.956	0.050	0.071	0.350	1.441	$122.48
$65	301,316	0.877	0.907	0.048	0.068	0.334	1.372	$115.26
$60	366,554	0.805	0.847	0.045	0.065	0.312	1.286	$105.84
$55	464,690	0.727	0.776	0.042	0.060	0.286	1.193	$95.57
$50	683,757	0.619	0.684	0.037	0.053	0.258	1.054	$81.70
$45	1,177,039	0.507	0.593	0.032	0.046	0.231	0.917	$67.29
$40	1,883,351	0.436	0.534	0.028	0.041	0.217	0.817	$57.89
$35	2,824,187	0.388	0.490	0.025	0.038	0.204	0.745	$51.11
$30	3,964,731	0.353	0.445	0.023	0.035	0.185	0.688	$45.65
$25	6,062,750	0.318	0.384	0.021	0.032	0.159	0.614	$39.29
$20	10,180,176	0.280	0.319	0.019	0.030	0.132	0.526	$32.37
$15	14,017,935	0.259	0.281	0.018	0.028	0.116	0.472	$28.35
$10	16,049,401	0.248	0.264	0.017	0.027	0.110	0.446	$26.37
$0.01	17,155,748	0.241	0.254	0.017	0.027	0.106	0.431	$25.15

Table 14: Mequillon Global Mineralization Estimate

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INDICATED								
NSR Cut-Off $C/t	TONNES	Ni %	Cu%	Co%	Au g/t	Pt g/t	Pd g/t	NSR $C/t
$100	2,928,290	0.771	1.017	0.036	0.214	0.751	2.826	$132.48
$95	3,309,111	0.747	0.996	0.035	0.209	0.729	2.747	$128.46
$90	3,705,160	0.725	0.978	0.035	0.205	0.708	2.670	$124.60
$85	4,109,537	0.704	0.957	0.034	0.201	0.688	2.599	$120.95
$80	4,517,536	0.684	0.940	0.033	0.196	0.667	2.525	$117.48
$75	4,971,062	0.664	0.922	0.032	0.191	0.645	2.448	$113.83
$70	5,450,049	0.643	0.904	0.031	0.185	0.623	2.370	$110.19
$65	5,927,167	0.625	0.883	0.031	0.180	0.602	2.296	$106.75
$60	6,453,836	0.606	0.860	0.030	0.174	0.581	2.220	$103.14
$55	7,063,062	0.585	0.834	0.029	0.167	0.559	2.138	$99.20
$50	7,883,127	0.560	0.800	0.028	0.159	0.532	2.035	$94.32
$45	9,703,827	0.516	0.732	0.026	0.145	0.484	1.850	$85.48
$40	12,733,881	0.466	0.650	0.024	0.130	0.430	1.636	$75.20
$35	16,511,717	0.424	0.581	0.022	0.117	0.384	1.455	$66.56
$30	21,001,390	0.388	0.521	0.021	0.107	0.346	1.303	$59.27
$25	25,094,736	0.363	0.480	0.020	0.099	0.319	1.198	$54.10
$20	27,907,573	0.347	0.454	0.019	0.095	0.302	1.134	$50.94
$15	29,233,457	0.340	0.442	0.019	0.093	0.294	1.103	$49.45
$10	29,706,133	0.337	0.438	0.019	0.092	0.291	1.091	$48.87
$0.01	30,000,566	0.335	0.434	0.019	0.091	0.289	1.083	$48.45

INFERRED								
NSR Cut-Off $C/t	TONNES	Ni %	Cu%	Co%	Au g/t	Pt g/t	Pd g/t	NSR $C/t
$100	184,366	0.810	1.058	0.037	0.204	0.688	2.295	$133.32
$95	199,927	0.792	1.047	0.036	0.198	0.677	2.241	$130.54
$90	217,212	0.776	1.028	0.036	0.191	0.667	2.158	$127.53
$85	235,940	0.758	1.010	0.035	0.186	0.656	2.075	$124.35
$80	257,443	0.739	0.990	0.034	0.180	0.641	1.984	$120.86
$75	276,088	0.723	0.980	0.034	0.176	0.626	1.898	$117.93
$70	316,574	0.693	0.942	0.032	0.166	0.596	1.752	$112.08
$65	345,213	0.674	0.918	0.031	0.160	0.573	1.669	$108.41
$60	373,176	0.656	0.895	0.031	0.153	0.553	1.609	$104.97
$55	392,687	0.642	0.880	0.030	0.150	0.540	1.572	$102.61
$50	429,069	0.618	0.853	0.029	0.145	0.519	1.500	$98.39
$45	458,924	0.600	0.832	0.028	0.141	0.502	1.445	$95.07
$40	655,813	0.511	0.695	0.025	0.120	0.430	1.277	$79.05
$35	1,259,369	0.402	0.521	0.021	0.095	0.339	1.078	$59.10
$30	1,814,804	0.359	0.452	0.019	0.085	0.301	0.968	$50.97
$25	2,398,323	0.329	0.403	0.018	0.078	0.273	0.890	$45.25
$20	2,926,839	0.308	0.371	0.017	0.075	0.253	0.829	$41.21
$15	3,092,116	0.301	0.360	0.017	0.073	0.247	0.809	$39.96
$10	3,152,026	0.298	0.356	0.017	0.073	0.244	0.801	$39.46
$0.01	3,159,811	0.298	0.356	0.017	0.073	0.244	0.799	$39.37

Table 15: Ivakkak Global Mineralization Estimate

APR 29/07

INDICATED

NSR Cut-Off $C/t	TONNES	Ni %	Cu%	Co%	Au g/t	Pt g/t	Pd g/t	NSR $C/t
$100	751,068	1.653	2.063	0.070	0.189	0.831	4.075	$228.56
$95	777,036	1.624	2.035	0.069	0.186	0.818	4.014	$224.18
$90	805,688	1.594	2.005	0.068	0.184	0.806	3.954	$219.50
$85	830,999	1.568	1.979	0.067	0.182	0.795	3.906	$215.48
$80	861,015	1.538	1.950	0.066	0.180	0.782	3.849	$210.85
$75	891,290	1.509	1.919	0.065	0.177	0.770	3.792	$206.34
$70	921,280	1.482	1.890	0.064	0.175	0.758	3.737	$201.99
$65	951,799	1.455	1.862	0.062	0.172	0.747	3.684	$197.67
$60	980,926	1.430	1.835	0.062	0.170	0.736	3.633	$193.66
$55	1,009,527	1.406	1.810	0.061	0.168	0.726	3.584	$189.80
$50	1,038,735	1.383	1.784	0.060	0.166	0.716	3.536	$185.94
$45	1,068,728	1.359	1.757	0.059	0.163	0.705	3.485	$182.06
$40	1,099,047	1.336	1.732	0.058	0.161	0.695	3.435	$178.22
$35	1,131,203	1.313	1.705	0.057	0.159	0.685	3.385	$174.21
$30	1,164,507	1.290	1.680	0.056	0.157	0.675	3.334	$170.17
$25	1,192,274	1.272	1.660	0.055	0.155	0.667	3.295	$166.85
$20	1,215,013	1.258	1.644	0.055	0.154	0.660	3.264	$164.15
$15	1,233,667	1.246	1.631	0.054	0.152	0.655	3.239	$161.93
$10	1,251,137	1.236	1.620	0.054	0.151	0.650	3.217	$159.84
$0.01	1,272,187	1.223	1.607	0.054	0.150	0.645	3.191	$157.30

INFERRED

NSR Cut-Off $C/t	TONNES	Ni %	Cu%	Co%	Au g/t	Pt g/t	Pd g/t	NSR $C/t
$100	133,121	1.616	1.611	0.068	0.141	0.786	3.827	$204.61
$95	140,974	1.570	1.578	0.066	0.139	0.768	3.754	$198.67
$90	152,071	1.510	1.536	0.064	0.136	0.744	3.653	$190.93
$85	166,844	1.439	1.487	0.061	0.131	0.716	3.522	$181.75
$80	176,682	1.397	1.462	0.059	0.130	0.698	3.448	$176.21
$75	191,696	1.340	1.423	0.057	0.126	0.675	3.343	$168.47
$70	204,947	1.297	1.392	0.055	0.124	0.656	3.266	$162.26
$65	217,170	1.259	1.366	0.054	0.122	0.641	3.195	$156.92
$60	228,686	1.229	1.345	0.053	0.121	0.628	3.138	$152.17
$55	241,994	1.194	1.320	0.052	0.119	0.613	3.070	$146.95
$50	253,070	1.166	1.298	0.051	0.117	0.601	3.009	$142.81
$45	264,700	1.140	1.277	0.050	0.116	0.590	2.958	$138.61
$40	276,477	1.114	1.258	0.049	0.115	0.579	2.905	$134.52
$35	288,507	1.091	1.239	0.048	0.113	0.569	2.854	$130.47
$30	297,180	1.076	1.228	0.047	0.112	0.563	2.827	$127.61
$25	305,741	1.062	1.218	0.047	0.111	0.557	2.798	$124.81
$20	314,150	1.050	1.209	0.046	0.110	0.552	2.776	$122.07
$15	321,579	1.039	1.200	0.046	0.110	0.547	2.751	$119.66
$10	327,609	1.031	1.194	0.045	0.109	0.544	2.737	$117.69
$0.01	336,267	1.019	1.185	0.045	0.108	0.539	2.715	$114.81

Mineral Reserves

The in-situ mineral resources of the Expo, Mesamax, Mequillon, and Ivakkak deposits were estimated by P&E and are described above. Mining of the Expo, Mesamax and Ivakkak deposits will be carried out utilizing conventional open pit mining methods. The Mequillon deposit is not economically recoverable under the current study parameters (US$6.00 nickel, base case). Initially four open pit designs and mineral reserves derived from the indicated mineral resources which were based on a computer generated block model of the deposits developed by P&E using the Gems software, while the pit geometry and mine plan were designed using the Whittle and Gems software packages from Gemcom. Each of the four deposit block models was assessed for optimal pit design using the Whittle 4.0 software. The result of the optimizing process is a pit shell which will maximize the net present value of the resources according to the economic parameters that were used.

All four block models used selected small mining units (SMU) dimensions of 5 metres by 5 metres by 5 metres. Densities used are 4.5 t/m3 for massive ore, 3.1 t/m3 for net textured ore, 2.9 t/m3 for waste rock, and 2.0 t/m3 for overburden. Each SMU has its mass computed from the sum of all parcels of each material making it up, plus the product of the metal grades. This enables the model to compute the cost, revenue and the net income for each SMU in Whittle. Table 16 shows the costs for each area not including the cost adjustment for the pit depth ($0.022 increment per 10 m bench).

Table 16: Estimated Operating Costs for Each Pit

	Cost/t				
Area	**Mining**	Ore Transport to Mill	Processing Review	G&A	**Total Operating Cost**
Mesamax	**$ 4.19**	$ 1.94	$ 29.52	$ 22.66	**$ 54.12**
Expo	**$ 3.36**	$ -	$ 24.97	$ 17.36	**$ 42.33**
Ivakkak	**$ 3.36**	$ 7.67	$ 24.97	$ 17.36	**$ 50.00**
Mequillon	**$ 3.36**	$ 3.49	$ 24.97	$ 17.36	**$ 45.82**

The mill recoverable nickel, copper, platinum and palladium were taken into account to compute the net smelter return (NSR) per block in Whittle. No credits were given to gold and cobalt. Each metal has a corresponding NSR factor per grade unit. The NSR value is a combination of the four payable metals' contribution to gross mine income at the prices of US$6.00/lb for nickel, US$1.50/lb for copper, US$900/oz for platinum and US$300/oz for palladium. The NSR value accounts for mill recoveries, concentrate handling and freight costs, smelter costs and recoveries, including refinery and metal selling costs for both the nickel concentrate and the copper concentrate.

The methodology used in the preparation of the resource model included some dilution and mining recovery factors into the grade block model prior to developing the pit geometry for the Study in Whittle. Therefore, no additional mining dilution or mining recovery factors were used in

Whittle. Golder provided recommendations for pit wall slope angles in the form of average inter-ramp slopes. Each final pit design has a working bench height of 10 metres. The final bench height and the inter-ramp slope angles used for the pit design are consistent with recommendations from Golder. Overall, Golder slope angles range from approximately 49º to 55º.

The total estimated diluted reserves contained within the three pit designs, including the underground portion of the Ivakkak deposit and excluding assumed ore losses, is 10,721,000 tonnes which are summarized in Table 17. SLI estimated the reserves for the Mesamax and Expo deposits, while P&E estimated the reserves for the Ivakkak Pit and Ivakkak Underground. The total amount of waste within the same pit perimeters is 38,674,000 tonnes (excluding the waste development required for the underground portion of Ivakkak), for an average open pit stripping ratio of 3.67:1 for all three deposits.

Table 17: Total Estimated Diluted Payable Reserves

	Ore Tonnes	Ni %	Cu %	Co %	Au g/t	Pt g/t	Pd g/t	Waste Tonne	Stripping Ratio (waste / Ore) t/t
Mesamax	2,077,000	1.85	2.49	0.07	0.19	0.95	3.46	5,704,000	2.75
Expo	7,843,000	0.68	0.69	0.04	0.07	0.29	1.25	29,834,000	3.80
Ivakkak Pit	604,000	1.22	1.53	0.05	0.16	0.67	3.22	3,136,000	5.19
Ivakkak Underground	197,000	2.28	2.73	0.10	0.21	1.04	4.90		
Total	**10,721,000**	**0.97**	**1.13**	**0.05**	**0.10**	**0.45**	**1.86**	**38,674,000**	**3.67**

Although no dilution was added in the Whittle optimizations, for the estimate of the reserves of each of the pits, an estimate of expected dilution was made depending on the geological structure of the mineral resources. For the Expo reserves an average of 6.6% was estimated with the grades of the low grade portion (0.22% Ni and 0.20% Cu). The dilution for the Mesamax pit was estimated at 13% at 0% metal grades. For the open pit portion of the Ivakkak deposit an estimated dilution of 15% was added while for the underground portion P&E estimated 30% dilution, both at 0% metal grades.

The Mequillon deposit did not present any mineable ore according to the present parameters used for the NSR calculation.

Relevant Data and Information (Item 20)

SLI is not aware of any other relevant information related to the resources that are not described in the P&E reports or this Study which would change any conclusion or recommendations.

Interpretation and Conclusion (Item 21)

SLI just completed a Bankable Feasibility Study, Table 18 shows the results for the base case and three other scenarios.

Table 18: Summary of Financial Analysis Results

	Base Case	Scenario 1	Scenario 2	Scenario 3
Ni Price Long term (2011 and beyond, in USD/lb)	$6.00 /lb	$8.00 /lb	$10.00 /lb	$15.00 /lb
Cu Price Long Term (2011 and beyond, in USD/lb)	$1.50 /lb	$1.75 /lb	$2.00 /lb	$2.50 /lb
NPV after tax (discounted at 8%, in M$ CAD)	$0.85 M	$115.6 M	$282.8 M	$674.0 M
IRR Project after tax	8.1 %	15.3 %	24.0 %	39.6 %
IRR Project before tax	11.3 %	20.0 %	30.5 %	49.6 %
Payback period (years of operation)	2.9	2.4	1.9	1.4

Recommendations (Item 22)

The mining reserves were evaluated at 10.7 million tonnes based on mineral resource estimation in compliance with National Instrument 43-101 (NI 43-101) produced by P&E Mining Consultants Inc. and is sufficient for more than 8.5 years of operation at a production rate of 3,500 tonnes of ore per day. The level of accuracy of the capital cost estimate is -5%, +10%, and for the operating cost estimates: ± 15%. The capital cost estimates include a 10% contingency.

The total project duration for the construction phase is 24 months. This assumes that the environmental permits are obtained by April 2008. Environmental permitting is one of the major risks on the schedule that has to be managed. The worst case scenario is a 6 month delay with permit grant which would push operation start up from April 2010 to April 2011.

From a study of the schedule, the following recommendations are made:

1. Close follow-up on the environmental permitting work to obtain all mining, development, construction and environmental permits in time;

2. Complete detailed topographic mapping to add coverage for the Ivakkak and port zones of the project area;

3. Initiate detailed engineering activities as soon as possible:
   - For the module shell design for quick start fabrication early 2008;
   - Ball mill procurement initiate early 2007 due to long delivery;
   - Equipment tendering and awarded during third quarter 2007.
4. Prepare a condemnation drilling campaign in all disposal areas for waste and tailing and at the plant site area;
5. As recommended in the Expo Resource Technical Report by P&E, there are many gaps within the Expo deposit due to an absence of drilling which should be drilled to acquire information in these areas. This could have the impact of transferring a quantity of waste into ore, thus increasing the amount of ore tonnage while reducing the waste tonnage, thus improving the economics of the Expo mine;
6. Evaluate prior to detailed engineering the feasibility of producing a combined Ni/Cu concentrate. This would lower significantly capital costs for the mill and concentrate storage building at Deception Bay. Furthermore, the concentrate shipping logistics would also be simplified. However, this strategy might have a negative impact on revenues. This could be mitigated by having an offtake agreement in a smelter capable of processing a combined Ni/Cu concentrate;
7. Evaluate the integration into the flowsheet of a magnetic separation stage. This would increase concentrate grade, consequently reduce concentrate volume. This could translate in a reduction of the required storage volumes at the mill and at Deception Bay, with a corresponding reduction in capital costs associated with these elements. Furthermore, the concentrate self ignition potential would be reduced, and smelter treatment charges would be reduced.

References (Item 23)

Appendix M - OT - External Reports

Ot-0045 External Report - Golder - Conceptual Design Tailings & Waste Rock Disposal Facilities

Ot-0046 External Report - Golder - Preliminary Slope Design Recommendations

Ot-0047 External Report - Golder - Static Test Results for Waste Rock and Overburden

Ot-0048 External Report - Strathcona Mineral Services - Technical Report

Ot-0049 External Report - GENIVAR - Environmental and Social Impact Assessment

Ot-0050	External Report - P&E - Technical Report & Preliminary Economic Assessment
Ot-0051	External Report - Nesse Tech - Technical Report Self-Heating Test
Ot-0052	External Report - SGS Lakefield - Metallurgical Testing
Ot-0053	External Report - P&E - Technical Report - Expo Update
Ot-0054	External Report - P&E - Technical Report - Ivakkak Update
Ot-0055	External Report - GENIVAR - Overburden Thickness
Ot-0055A	External Report – P&E – Technical Memorandum – Mesamax Update
Ot-0055B	External Report - P&E - Internal Memorandum - Ivakkak Trade-Off Assessment Summary

Date and Signature Page (Item 24)

SNC-LAVALIN INC.

{SIGNED AND SEALED}

Pierre Demers, P. Eng.

{SIGNED AND SEALED}

Martial Côté, P. Eng.

{SIGNED AND SEALED}

Richard Lemieux, P. Eng.

{SIGNED AND SEALED}

Daniel Dufort, P. Eng.

{SIGNED AND SEALED}

Philippe Poirier, P. Eng.

{SIGNED AND SEALED}

Stéphane Rivard, P. Eng.

P&E MINING CONSULTANTS

{SIGNED AND SEALED}

Eugène Puritch, P. Eng., President

P. J. LAFLEUR GÉO-CONSEIL INC.

{SIGNED AND SEALED}

P. J. Lafleur, P. Eng.

DATED this 19th of July, 2007

QUALIFICATIONS CERTIFICATE

I, Pierre Demers, Professional Engineer, do certify that:

- I am employed as Project Manager for SNC-Lavalin Inc., located at 455, René Lévesque Boulevard West, Montréal, Québec, Postal Code H2Z 1Z3 (Tel. 514-393-1000).
- This Certificate applies to the Technical Report titled "Raglan South Nickel Project –Technical Report" dated June 29, 2007, as amended of July 19, 2007, completed by SNC-Lavalin for Canadian Royalties Inc.
- I graduated from the Laval University in Québec City in 1969 and hold a B.Sc. as Mine Engineer.
- I have worked as a mining engineer in the minerals industry for 38 years since my graduation from Laval University of Québec City. I worked as a mining engineer for John-Manville at the Jeffrey Mine from 1969 to 1972. From 1972 to 1981, I was a Technical Representative for C.I.L. From 1981 to 1997, I worked for Kilborn and Met-Chem-Pellemon. From 1997 to now, I am Director Project Development for SNC-Lavalin.
- I am a member of the Order of Professional Engineers of Québec (OIQ # 20123) and also a member of the Canadian Institute of Mines and Metallurgy.
- I have read the definition of ''qualified person'' set out in National Instrument 43-101 (NI 43-101) and certify by reason of my education, association 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.
- I did visit the Raglan South Nickel project site in June 2006 for the purpose of this Technical Report.
- I am overall responsible as Director Project Development for the Bankable Feasibility Study as well as the Technical Report titled "Raglan South Nickel Project –Technical Report" dated June 29, 2007, as amended of July 19, 2007.
- I have not had prior involvement with the properties that are the subject of the Technical Report.
- 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 or the omission to disclose which makes the Technical Report misleading.
- I am an independent consultant in the sense set out in section 1.4 of NI 43-101.
- I have not received, nor do I expect to receive directly or indirectly any interest in any form for the Raglan South project, or any property or project from Canadian Royalties Inc.
- As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
- I have read NI 43-101 and Form 43-101 F1, and the Technical Report has been prepared in compliance with that instrument and form 43-101 F1.

Prepared in Montreal, this July 19th, 2007.

{SIGNED AND SEALED}

Pierre Demers, P. Eng.

QUALIFICATIONS CERTIFICATE

I, Martial Côté, Professional Engineer, do certify that:

- I am employed as Project Manager for SNC-Lavalin Inc., located at 455, René Lévesque Boulevard West, Montréal, Québec, Postal Code H2Z 1Z3 (Tel. 514-393-1000).
- This Certificate applies to the Technical Report titled "Raglan South Nickel Project –Technical Report" dated June 29, 2007, as amended of July 2007, completed by SNC-Lavalin for Canadian Royalties Inc.
- I graduated from the École Polytechnique de Montréal in 1974 and hold a B.Sc. as Mine Engineer.
- I have worked as a mining engineer in the minerals industry for 33 years since my graduation from École Polytechnique de Montréal. I have experience in iron ore, coal, gold, base metals and industrial minerals as well. I worked for La Compagnie Minière Québec Cartier (1974-83), Met-Chem Canada Inc. (1984-2004), Breton Banville & Associates (2004-06) and from 2006 to now, I am Project Manager for SNC-Lavalin.
- I am a member of the Order of Professional Engineers of Québec (OIQ # 25686) and also a member of the Canadian Institute of Mines and Metallurgy.
- I have read the definition of ''qualified person'' set out in National Instrument 43-101 (NI 43-101) and certify by reason of my education, association 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.
- I have not visited the Raglan South Nickel project site.
- I am responsible as Project Manager and as Mine Engineer for the Bankable Feasibility Study as well as the Technical Report titled "Raglan South Nickel Project –Technical Report" dated June 29, 2007, as amended of July 19, 2007.
- I have not had prior involvement with the properties that are the subject of the Technical Report.
- 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 or the omission to disclose which makes the Technical Report misleading.
- I am an independent consultant in the sense set out in section 1.4 of NI 43-101.
- I have not received, nor do I expect to receive directly or indirectly any interest in any form for the Raglan South project, or any property or project from Canadian Royalties Inc.
- As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
- I have read NI 43-101 and Form 43-101 F1, and the Technical Report has been prepared in compliance with that instrument and form 43-101 F1.

Prepared in Montreal, this July 19th, 2007.

{SIGNED AND SEALED}

Martial Côté, P. Eng.

QUALIFICATIONS CERTIFICATE

I, Eugene J. Puritch, P. Eng., residing at 44 Turtlecreek Blvd., Brampton, Ontario, L6W 3X7, do hereby certify that:

- I am an independent mining consultant contracted by Canadian Royalties Inc.
- This Certificate applies to the Technical Report titled "Raglan South Nickel Project –Technical Report" dated June 29, 2007, as amended of July 19, 2007, completed by SNC-Lavalin for Canadian Royalties Inc.
- I am a graduate of The Haileybury School of Mines, with a Technologist Diploma in Mining, as well as obtaining an additional year of undergraduate education in Mine Engineering at Queen's University. In addition I have also met the Professional Engineers of Ontario Academic Requirement Committee's Examination requirement for Bachelor's Degree in Engineering Equivalency.
- I am a mining consultant currently licensed by the Professional Engineers of Ontario (License No. 100014010) and registered with the Ontario Association of Certified Engineering Technicians and Technologists as a Senior Engineering Technologist. I am also a member of the National and Toronto CIM.
- I have practiced my profession continuously since 1978. My summarized career experience is as follows:

- Mining Technologist - H.B.M.&S. and Inco Ltd.	1978-1980
- Open Pit Mine Engineer – Cassiar Asbestos/Brinco Ltd	1981-1983
- Pit Engineer/Drill & Blast Supervisor – Detour Lake Mine	1984-1986
- Self-Employed Mining Consultant – Timmins Area	1987-1988
- Mine Designer/Resource Estimator – Dynatec/CMD/Bharti	1989-1995
- Self-Employed Mining Consultant/Resource-Reserve Estimator	1995-2004
- President – P & E Mining Consultants Inc.	2004-Present

- 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.
- I contributed to portions of Items 6 to 17 and 19 of the Technical Report titled "Raglan South Nickel Project –Technical Report" dated June 29, 2007, as amended of July 19, 2007.
- I have had prior involvement with the property that is the subject of the Technical Report in that I participated in the following three NI-43-101 reports; "Revised Raglan South Nickel Project, Nunavik, Quebec, Technical Report and Preliminary Economic Assessment on the Mequillon, Mesamax, Expo and Ivakkak Deposits" dated July 27, 2006 and "Technical Report and Resource Estimate Update on the Expo Ni-Cu-PGE Deposit, Raglan South Nickel Project, Nunavik, Quebec" dated February 26, 2007 and "Technical Report (2007) and Resource Estimate Update on the Ivakkak Ni-Cu-PGE Deposit, South Trend Property, Raglan South Nickel Project" dated March 22, 2007.
- As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
- I am independent of the issuer applying the test in Section 1.4 of NI 43-101.
- I have read NI 43-101 and Form 43-101F1 and the Report has been prepared in compliance therewith.
- I visited the South Raglan Group of properties, including the Expo Property from August 1 to 4, 2005 and from August 17 to 20, 2006.

Dated this 19th Day of July, 2007

{SIGNED AND SEALED}

Eugene J. Puritch, P. Eng.

QUALIFICATIONS CERTIFICATE

I, Richard Lemieux, professional engineer do hereby certify that:

- I am an Engineer for SNC-Lavalin Inc., located at 455, René Lévesque Boulevard West, Montreal, Québec, Postal Code H2Z 1Z3 (Tel. 514-393-1000).
- This Certificate applies to the Technical Report titled "Raglan South Nickel Project –Technical Report" dated June 29, 2007, as amended of July 19, 2007, completed by SNC-Lavalin for Canadian Royalties Inc.
- I graduated from the Laval University in 1984 and hold a B.Sc.A in Chemical Engineering.
- I graduated from Sherbrooke University in 1988 and hold a M.Sc.A in Chemical Engineering.
- I have worked as a Chemical engineer in the chemical/industrial industry for 20 years since my graduation from Sherbrooke University.
- I worked for SNC-Lavalin (1996-2002 and 2005-2007) as Process engineer and Project manager in charge of feasibility studies and other technical studies for industrial projects, for Suez-Degrémont (2002-2005) as national technical director, Sanexen (1992-1993) as project manager, Cartier Engineering(1991-1992 and 1993-1995) as process engineer, Fournier Steelwork (1988-1991) as process engineer. .
- I am a member of the Order of Professional Engineers of Québec and am designated as a Chemical Engineer.
- I have read the definition of ''qualified person'' set out in National Instrument 43-101 (NI 43-101) and certify by reason of my education, association 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.
- I have not visited the Raglan South Nickel project site.
- I am responsible for engineering infrastructure of the Technical Report, titled "Raglan South Nickel Project –Technical Report" dated June 29, 2007, as amended of July 19, 2007".
- I have not had prior involvement with the properties that are the subject of the Technical Report.
- 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 or the omission to disclose which makes the Technical Report misleading.
- I am independent of Canadian Royalties Inc. applying all of the tests set out in section 1.4 of NI 43-101.
- I have not received, nor do I expect to receive directly or indirectly any interest in any form for the Raglan South project, or any property or project from Canadian Royalties Inc..
- As of the date of this certificate, to the best of my knowledge, information and belief, the Technical report contains all scientific and technical information that is required to be disclosed to make the technical Report not misleading;
- I have read NI 43-101 and Form 43-101 F1, and the Technical Report has been prepared in compliance with that instrument and form 43-101 F1.

Prepared in Montreal, this July 19th day of, 2007.

{SIGNED AND SEALED}

Richard Lemieux, P. Eng.

QUALIFICATIONS CERTIFICATE

I, Daniel Dufort, professional engineer do hereby certify that:

- I am an Engineer for SNC-Lavalin Inc., located at 455, René Lévesque Boulevard West, Montreal, Québec, Postal Code H2Z 1Z3 (Tel. 514-393-1000).
- This Certificate applies to the Technical Report titled "Raglan South Nickel Project –Technical Report" dated June 29, 2007, as amended of July 19, 2007, completed by SNC-Lavalin for Canadian Royalties Inc.
- I graduated from the Ecole Polytechnique de Montreal in 1979 and hold a BSca in Mining Engineering.
- I have worked as a mining engineer in the minerals industry for 28 years since my graduation from Ecole Polytechnique de Montreal.
- I worked for Noranda and Lac Minerals between 1979 and 1985. From 1985 to 1997, I worked for an explosive company, former CIL, in various positions such as: supervisor, technical representative, area manager. From 1997 to 1999, I was the mine manager at Kiena Mines for the mining company Mc Watters. From 1999 to 2006, I was vice-president for Orica Explosives. From 2006 to now, I am Project Manager for SNC-Lavalin in charge of feasibility studies and other technical studies for mining projects.
- I am a member of the Order of Professional Engineers of Québec and am designated as a Mining Engineer.
- I have read the definition of ''qualified person'' set out in National Instrument 43-101 (NI 43-101) and certify by reason of my education, association 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.
- I have not visited the Raglan South Nickel project site.
- I am responsible for Item port, airport and the page set-up of the technical report, titled "Raglan South Nickel Project –Technical Report" dated June 29, 2007, as amended of July 19, 2007.
- I have not had prior involvement with the properties that are the subject of the Technical Report.
- 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 or the omission to disclose which makes the Technical Report misleading.
- I am independent of Canadian Royalties Inc. applying all of the tests set out in section 1.4 of NI 43-101.
- I have not received, nor do I expect to receive directly or indirectly any interest in any form for the Raglan South project, or any property or project from Canadian Royalties Inc..
- As of the date of this certificate, to the best of my knowledge, information and belief, the Technical report contains all scientific and technical information that is required to be disclosed to make the technical Report not misleading;
- I have read NI 43-101 and Form 43-101 F1, and the Technical Report has been prepared in compliance with that instrument and form 43-101 F1.

Prepared in Montreal, this July 19th day of, 2007.

{SIGNED AND SEALED}

Daniel Dufort, P. Eng.

QUALIFICATIONS CERTIFICATE

I, Philippe Poirier, professional engineer in the Province of Quebec, do hereby certify that:

- I am employed as Project Manager, Financial Assessment, for SNC-Lavalin Inc., located at 455, René Lévesque Boulevard West, Montréal, Québec, Postal Code H2Z 1Z3 (Tel. 514-393-1000).
- This Certificate applies to the Technical Report titled "Raglan South Nickel Project –Technical Report" dated June 29, 2007, as amended of July 19, 2007, completed by SNC-Lavalin for Canadian Royalties Inc.
- I graduated from Universite Laval, Quebec City, in 1991 and hold a B.A.Sc. in Mining/Mineral Processing Engineering. I completed a M.A.Sc. (University of British Columbia) in 1995 and a MBA (HEC, Montreal) in 2003.
- I worked in mining operations for Aur Resources from 1993 to 1994 and for Barrick Gold Corporation from 1994 to 1999. I joined SNC-Lavalin Inc in 1999 as a project manager, where I completed several environmental and economical studies for different mining companies. Since 2006, I joined SNC-Lavalin Capital, where I am involved in financial assessment of different projects, mainly in the mining sector.
- I am a member of the Order of Professional Engineers of Québec (OIQ # 111091).
- I have read the definition of ''qualified person'' set out in National Instrument 43-101 (NI 43-101) and certify by reason of my education, association 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.
- I am responsible for Section 25.14 (Financial Analysis) of the Technical Report titled "Raglan South Nickel Project –Technical Report" dated June 29, 2007, as amended of July 19, 2007.
- I have not visited the Raglan South Nickel project site.
- I have not had prior involvement with the properties that are the subject of the Technical Report.
- 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 or the omission to disclose which makes the Technical Report misleading.
- I am an independent consultant in the sense set out in section 1.4 of NI 43-101.
- I have not received, nor do I expect to receive directly or indirectly any interest in any form for the Raglan South project, or any property or project from Canadian Royalties Inc.
- As of the date of this certificate, to the best of my knowledge, information and belief, the section of the Technical report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the technical Report not misleading;
- I have read NI 43-101 and Form 43-101 F1, and the section of the Technical Report for which I am responsible has been prepared in compliance with that instrument and form 43-101 F1.

Prepared in Montreal, this July 19th, 2007.

{SIGNED AND SEALED}

Philippe Poirier, Eng.

QUALIFICATIONS CERTIFICATE

I, Stéphane Rivard, Professional Engineer, do certify that:

- I am employed as Process Engineer for SNC-Lavalin Inc., located at 455, René Lévesque Boulevard West, Montréal, Québec, Postal Code H2Z 1Z3 (Tel. 514-393-1000).
- This Certificate applies to the Technical Report titled "Raglan South Nickel Project –Technical Report" dated June 29, 2007, as amended of July 19, 2007, completed by SNC-Lavalin for Canadian Royalties Inc.
- I graduated from Laval University in 1994 and hold a B.Sc. in Metallurgical and Material Science Engineering.
- I have worked as a metallurgical engineer in the minerals and metallurgical industries for 13 years since my graduation from LAVAL University. I have experience in base metals, copper,zinc and lead hydrometallurgical Processes and copper Pyrometallurgical processes as well. I worked for Cambior at Bouchard-Hébert mine from 1994 to 1997 as a production metallurgist. From 1997 to 2002, I have worked for Noranda Inc. in various positions such as: Mill Senior Metallurgist and as Black Belt at the Horne site and at their Copper Refinery in Montreal-East. From 2002 to 2006, I have worked for Nexans Canada in their Montreal copper rod production plant as a Process Engineer. From 2006 to now, I am a Process Engineer for SNC-LAVALIN in the Mining & Metallurgical department
- I am a member of the Order of Professional Engineers of Québec (OIQ # 118538).
- I have read the definition of ''qualified person'' set out in National Instrument 43-101 (NI 43-101) and certify by reason of my education, association 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.
- I have not visited the Raglan South Nickel project site.
- I am responsible for the process design, mill lay-out and for Section 6 and 7 of the Bankable Feasibility Study Report titled Raglan South Nickel Project –Technical Report" dated June 29, 2007, as amended of July 19, 2007.
- I have not had prior involvement with the properties that are the subject of the Technical Report.
- 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 or the omission to disclose which makes the Technical Report misleading.
- I am an independent consultant in the sense set out in section 1.4 of NI 43-101.
- I have not received, nor do I expect to receive directly or indirectly any interest in any form for the Raglan South project, or any property or project from Canadian Royalties Inc.
- As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
- I have read NI 43-101 and Form 43-101 F1, and the Technical Report has been prepared in compliance with that instrument and form 43-101 F1.
- Prepared in Montreal, this July 19th, 2007.

{SIGNED AND SEALED}

Stéphane Rivard, Eng.

QUALIFICATIONS CERTIFICATE

I, Pierre Jean Lafleur, Professional Engineer, do certify that:

- I am president of P.J. Lafleur Géo-Conseil Inc., a Corporation managing my professional services to SNC-Lavalin Inc., located at 455, René Lévesque Boulevard West, Montreal, Québec, Postal Code H2Z 1Z3 (Tel. 514-393-1000).
- This Certificate applies to the Technical Report titled "Raglan South Nickel Project –Technical Report" dated June 29, 2007, as amended of July 19, 2007, completed by SNC-Lavalin for Canadian Royalties Inc.
- I graduated from École Polytechnique de Montréal (B. ENG.) in Geology in 1976.
- I have practice my profession in exploration, geology and mining for more than 30 years, and I have experience in gold, base metals and industrial minerals as well. I have worked for Consolidated Goldfields (1980-81), Falconbridge (1981-84), Audrey Resources (1985-1993). I have been a consulting P.Eng. since 1987. I have worked in Canada and abroad. I have specialised in computer modeling of mineral resources and mine planning. I am also a Senior Business Associate of Gemcom Software International Inc.
- I am a member of the Order of Professional Engineers of Québec (OIQ # 39862) and also a member of the Canadian Institute of Mines and Metallurgy.
- I have read the definition of ''qualified person'' set out in National Instrument 43-101 (NI 43-101) and certify by reason of my education, association 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.
- I have not visited the Raglan South Nickel project site.
- I am responsible in part for section 5 (Mining Reserves) of the Bankable Feasibility Study as well as the Technical Report titled "Raglan South Nickel Project –Technical Report" dated June 29, 2007, as amended of July 19, 2007.
- I have not had prior involvement with the properties that are the subject of the Technical Report.
- 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 or the omission to disclose which makes the Technical Report misleading.
- I am an independent consultant in the sense set out in section 1.4 of NI 43-101.
- I have not received, nor do I expect to receive directly or indirectly any interest in any form for the Raglan South project, or any property or project from Canadian Royalties Inc.
- As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
- I have read NI 43-101 and Form 43-101 F1, and the Technical Report has been prepared in compliance with that instrument and form 43-101 F1.

Prepared in Ste-Therese, this July 19th, 2007.

{SIGNED AND SEALED}

Pierre-Jean Lafleur, P. Eng.

Additional Requirements for Technical Reports on Development Properties and Production Properties (Item 25)

25.1 Mining Method

Mining of the Expo, Mesamax and the upper part of the Ivakkak deposits will be carried out utilizing conventional open pit drill, blast, load and haul methods. The lower portion of the Ivakkak deposit will be mined with underground mining methods. The underground concept was developed by P&E. Ore will be hauled to the primary crusher located at the mill near Expo. Waste rock will be hauled and deposited on dumps located within close proximity of the pit. In accordance with environmental regulations and the proposed mining schedule, any potentially acid generating (PAG) rock will be disposed of in a manner designed to meet environmental regulations. Initially, production will be concentrated in a zone where the stripping ratio is low for each pit.

The mobile equipment size selected for the mine was based on the proposed mining bench height. The fleet size is based on equipment cycle time, material movement schedules and estimated auxiliary equipment requirements. Equipment types were selected to allow mining of ore and waste materials and provide mine auxiliary equipment support including the requirement for road maintenance, dump maintenance and snow removal. Where possible, equipment types were standardized across the project. The mine plans and equipment fleet are amended in accordance with operating experience gained at other properties.

A single final ramp is proposed for the transportation of both ore and waste rock. The minimizing of haul distances of the waste rock to the waste dump and the ore to the crusher was considered when determining the exit point of the road from the pits. A grade of 10% has been used in the design of the ramp.

The width of the final ramps is 20 metres for a truck 5 metres wide. A 2 metre drainage ditch as well as a 3 metre safety berm will be required leaving the remaining 15 metres (3 times the truck width) available for the trucks.

Table 19 shows the projected mine production schedule.

Table 19: Mine Production Schedule

	Mesamax			Expo			Ivakkak			Ivakkak Underground	Total Mined		
	Ore	Waste	Stripping Ratio	Ore	Waste	Stripping Ratio	Ore	Waste	Stripping Ratio	Ore	Ore	Waste	Ore + Waste
Year	(kt)	(kt)	t/t	(kt)	(kt)	t/t	(kt)	(kt)	t/t	(kt)	(kt)	(kt)	(kt)
Pre-Prod	0	1,170	0	0	4,000	0	0	0	0	0	0	5,170	5,170
1	650	444	0.68	260	2,300	8.85	0	0	0	0	910	2,744	3,654
2	913	2,299	2.52	365	190	0.52	0	1,050	0	0	1,278	3,539	4,817
3	514	1,791	3.48	365	526	1.44	398	1,450	3.64	0	1,277	3,767	5,044
4	0	0	0	1,072	1,805	1.68	206	636	3.09	0	1,278	2,441	3,719
5	0	0	0	1,147	4,248	3.70	0	0	0	131	1,278	4,248	5,526
6	0	0	0	1,211	4,278	3.53	0	0	0	66	1,277	4,278	5,555
7	0	0	0	1,278	4,130	3.23	0	0	0	0	1,278	4,130	5,408
8	0	0	0	1,277	4,016	3.14	0	0	0	0	1,277	4,016	5,293
9	0	0	0	868	4,341	5.00	0	0	0	0	868	4,341	5,209
Total	**2,077**	**5,704**	**2.75**	**7,843**	**29,834**	**3.80**	**604**	**3,136**	**5.19**	**197**	**10,721**	**38,674**	**49,395**

Expo Open Pit

The Expo deposit consists of a series of shallow dipping to sub-horizontal, high grade, massive sulphide zones surrounded by net textured and disseminated zones of lower grade.

The designed Expo pit varies in width from 230-340 metres, is 830 metres long and up to 115 metres deep. The main ramp in the north wall descends at a 10% gradient from elevation 535 metres to the lowest bench at elevation 420 metres.

During the pre-stripping period, waste rock that is non-acid generating could be used for dam and road construction. The waste dump area will not exceed 30 metres in height and the containment dyke will have a placement slope of 4:1 (14.0 degrees). Approximate pile dimensions are 2,000 metres long and up to 800 metres wide.

Mesamax Open Pit

The designed Mesamax pit is approximately 190 metres wide, 350 metres long and 110 metres deep. The main ramp descends at a -10%. Payable material will be hauled 10 kilometres to the Expo processing area.

Waste rock will be stored in a pile in close proximity to the pit. The pile will not exceed 20 metres in height and will have a placement slope of 4:1 (14.0 degrees). The pile is situated in such a manner that drainage can be contained and treated prior to release to the environment.

Ivakkak Open Pit and Underground

The designed Ivakkak pit is located approximately 40 kilometres to the west of the Expo deposit and has dimensions of approximately 175 metres wide, 285 metres long and 70 metres deep. The main ramp in the north wall descends at a 10% slope

Waste rock will be stored in a pile within close proximity to the pit. The pile height will not exceed 20 metres and will have a placement slope of 1:1 (45 degrees). The pile is situated in such a manner that drainage can be contained and treated prior to release to the environment.

Concentrate Production and Recovery

Table 11 shows the total concentrate tonnage for the mine life and also the recovery.

Table 11: Concentrate Tonnage and Recovery

Year of Production	Nickel			Copper		
	Concentrate Tonnage	Concentrate Grade	Recoveries	Concentrate Tonnage	Concentrate Grade	Recoveries
1	89 581	13.0 %	74.0 %	65 773	29.6 %	84.8 %
2	114 238	12.9 %	74.0 %	67 018	29.4 %	82.1 %
3	85 516	12.7 %	73.8 %	50 200	29.3 %	66.7 %
4	74 502	10.8 %	71.8 %	30 025	27.6 %	66.0 %
5	73 178	10.8 %	71.7 %	30 563	27.6 %	65.5 %
6	64 207	10.7 %	71.6 %	25 231	27.2 %	64.9 %
7	57 125	10.2 %	71.5 %	21 675	26.6 %	64.9 %
8	56 274	10.2 %	71.5 %	19 157	26.6 %	64.9 %
9	38 229	10.2 %	71.5 %	13 014	26.6 %	64.9 %

The nickel recovery represents the recovery of nickel in nickel concentrate and the copper recovery represents the recovery of copper in copper concentrate.

The daily tonnage throughout the mine life will be 3,500 t/day.

Equipment Selection

All equipment will be diesel powered. Drilling and blasting will be required on all ore and waste rock. It is planned to use conventional in-the-hole hammer drill rigs. Blasting will be with emulsion explosives and a down-hole delay initiation system. Front-end-loaders will be used to load haulage trucks of 63 t capacity. Tractor-trailers with a rock dump body of 48 tonnes capacity will be used to transport the ore from Ivakkak to the process plant and a concentrate ejector body of 45 tonnes capacity to transport the concentrate to the port. A standardized fleet of loaders of 6,9 m^3 capacity were selected. Bulldozers, graders and a fleet of service equipment will support the production equipment. The mine will be in operation 365 days per year on 2 shifts of 11 hours. Due to the severe climate, it is assumed that the operation will stop for an equivalent of 21 days per year. The table below shows the mine equipment required for the entire project.

Production equipment requirements were determined from first principle calculations of productivities necessary to accomplish the scheduled production during the time available. These calculations rely on manufacturers' productivity guidelines, and industry standards and experience.

Tables 20 and 21 show the mining equipment and support equipment.

Table 20: Mining Equipment

Equipment	Pre-Prod	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
Mine Loaders 6.9 m^3	2	2	2	2	2	2	2	2	2	2
Trucks 63 tonne - Mine	5	5	5	5	4	5	5	5	5	5
Semi-Trailer – Ore Haulage				6	3	2	2			
Mine Loaders – Rehandling Ivakkak				1	1	1	1			
Drill (165 mm dia.)	2	2	2	2	2	2	2	2	2	2
Dozer (D8 type)	1	1	1	1	1	1	1	1	1	1
Mobile Rock Breaker	1	1	1	1	1	1	1	1	1	1

Table 21: Mine Support Equipment

Equipment	Pre-Prod	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
Lube-Fuel Truck	1	1	1	1	1	1	1	1	1	1
Tire Handler Loader	1	1	1	1	1	1	1	1	1	1
Mobile Shop Truck	1	1	1	1	1	1	1	1	1	1
Boom truck (25 t)	1	1	1	1	1	1	1	1	1	1
Bus (16 seats)	1	1	1	1	1	1	1	1	1	1

Equipment	Pre-Prod	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
Mine Superintendent Pickup	1	1	1	1	1	1	1	1	1	1
Mine Shift Supervisor Pickup	2	2	2	2	2	2	2	2	2	2
Blasters Pickup	1	1	1	1	1	1	1	1	1	1
Engineering Pickup	2	2	2	2	2	2	2	2	2	2
Maintenance Pickup	1	1	1	1	1	1	1	1	1	1

25.2 Waste and Tailings Disposal

The waste rock and mill tailings disposal concepts were developed by Golder Associates ("Golder"). Capital and operating cost estimates were also developed by Golder. The costs estimates were integrated into SLI's capital and operating cost estimates.

Key Design Parameters

The design drivers for the waste disposal facilities include:

- Climatic conditions;
- Climate change and the consideration that continuous permafrost may eventually disappear from the project area. Reliance on permafrost can not be assumed for long term management of water, lixiviation of metals, or impermeability;
- Long cold winters (workability of concepts in cold weather);
- Chemical stability and consideration for the highly reactive geochemistry of the tailings and some of the waste rock;
- Water management and drainage;
- Minimizing the footprint area of the tailings disposal facility;
- Production schedule;
- Relative locations of the facilities with respect to the mill and mines;
- Wind erosion of exposed tailings must be minimized;
- Physical stability of the structures must be ensured;
- Decommissioning and closure; and
- Optimization of capital costs and operating costs to achieve the performance specification.

Waste Rock and Tailings Disposal Facility Design

Tailings from the Expo mill will be deposited into the Expo waste rock and tailings co-disposal facility. The Expo co-disposal facility will comprise four cells, which will be constructed in stages. The staged construction will match the supply of waste rock from the Expo pit, and it will also allow progressive closure of the cells which limit wind erosion of the deposited tailings and minimize the potential for oxidation of tailings. Mining of the Expo pit will be completed in Year 9 of operations; after that, the tailings from further potential mining production will be deposited into the mined-out Expo pit and the waste disposal facility will be closed out.

The disposal cells will be formed by constructing perimeter containment dykes using compacted waste rock from the Expo pit. The dykes will have 3:1 (H:V) side slopes and their crests and sideslopes will be covered with a 60 mil high density polyethylene (HDPE) geomembrane liner to limit infiltration. The liner will also extend under the bottom surface of cells 1, 2 and 4 (i.e. the cells which will contain tailings). A bedding layer of sand will be provided beneath the liners.

Tailings will be placed in the form of a high density, non-segregating paste, which will be produced using a deep cone thickener (DCT). With an estimated solids content of 72%, the thickened tailings paste should release minimal bleed water upon placement. This will limit the amount of ice which will be trapped in the placed tailings. Solids content will be refined in detailed engineering and paste testwork.

For closure, 60 mil linear low density polyethylene (LLDPE) geomembrane covers will be installed over the facility, thereby limiting oxygen and water penetration into the tailings, which is expected to effectively control (prevent) ARD. A bedding layer of sand or tailings will be placed beneath the geomembrane cover, and a sand cover layer will be placed above it. Finally, a layer of non-PAG rock will be placed on top of the cover to protect it. The covers will be placed progressively as the individual cells are closed.

The Mesamax and Ivakkak waste rock dumps will be regraded for closure. The side slopes of the waste dumps will be regraded to a slope of 3:1 (H:V) or flatter by using a dozer to remove the placement benches. The top surface of the dumps will be regraded to form a smooth, convex surface which will drain towards the perimeter. After regrading, low permeability covers will be installed over the top surface and side slopes of the Mesamax waste rock dump after mining is completed. Geochemical weathering tests conducted to date support this time lag before the placement of ARD-control measures. The covers will minimize infiltration water into the rock pile and as a consequence, will decrease the quantity of leachate draining at the toes of the piles. No cover will be placed on the Ivakkak waste rock dump because it is not expected to generate acid or leach metals in quantities that would affect water use at the site. The seepage collection ditches will be maintained post-closure, and these will conduct both seepage and waste dump runoff into the 3 individual water collection ponds.

Dewatering of the open pits will cease at the end of mining and the pits will be allowed to flood back naturally in response to runoff and groundwater inflow. It is expected that each of the pits will eventually overflow at the low point on their pit rim. In each case, the pit overflow will be directed through ditches into their respective water collection ponds. The water quality in the water collection ponds will be monitored monthly during ice-free periods for a minimum of 5 years after closure. If ongoing acid generation occurs on exposed pit faces above the final water level, it would be possible as a contingency to broadcast lime slurry into the pit water to raise the pH and to reduce the levels of dissolved metals.

25.3 Service Facilities

Development of the CRI RSNP will require infrastructures involving various maintenance services and utility systems at the mine/mill site as well as a port development at Deception Bay for the ocean shipment of concentrates and re-supply of goods and materials for the operations.

For the mine-mill operations, the following facilities and services will be provided:

- Verification of the existing 100 km long public road network from Deception Bay to the start of the Expo-Katinniq access road. A 5 km road section bypassing Xstrata Nickel's Raglan Mine Katinniq complex would be constructed;
- 18 km access road development from the Katinniq - Donaldson road to the Expo industrial site;
- Various service roads, including open pit access roads, service roads for sedimentation ponds, access to the Expo co-disposal area, and access to the explosives magazines;
- Accommodation complex with 172 single rooms, dining facilities and recreational facilities;
- A services complex with equipment maintenance and warehousing facilities, male/female mine dry, assay laboratory, infirmary and administration offices;
- Reciprocating diesel fired power plant to supply electrical power to the site. The power plant will be equipped with a heat recovery system for heating buildings and domestic hot water, and for process use;
- Various communications equipment and systems will be installed to support mine/mill operations, administration and recreation facilities;
- Potable water for the mine/mill site supplied from the reservoir impounded by a dam built approximately 5 km west of the Expo industrial site;
- Process water supplied from the tailings basin using a reclaim barge pump unit;

- Diesel fuel storage facilities consisting of six tanks at the port site with a total storage volume of 12 ML, one fuel tank at the Expo industrial complex with a capacity of 5 ML and two 55,000 litre day tanks at the power generation facility. One jet fuel tank of 1 ML also be installed at the port, with a 50,000 litre tank to be located at the airport;
- Rotary disc reactor type sewage treatment plant installed at the Expo industrial site;
- Mobile equipment to maintain the site roads, remove snow and support maintenance, concentrator and port site operations as well as off-highway duty tractor-trailer units to deliver concentrates from the mill to the concentrate storage facility at the port site, and fuel and general cargo from the port to the Expo industrial site;
- A mobile crushing and screening aggregate plant, and a concrete batch plant will be mobilized for construction and left on site following startup for use during operations;
- A mill with a capacity of 3,500 tpd will be built on site;
- A crushing plant including a jaw crusher, screens, cone crushers and conveyors.

From the port site, concentrates will be shipped to processing facilities via marine bulk carriers. The port site facilities will include fuel storage, concentrate handling and storage, shipping dock and shiploader.

Maintenance Facility

The service area complex will provide facilities for maintenance and servicing of all mine and plant mobile equipment, warehousing and laboratory. The service complex will be connected to the concentrator and the accommodation complex by means of utilidors. It will cover a total area of 4,330 m^2.

Administrative Complex

The administrative complex will be built with prefabricated modular construction and will be located on the same side as the fire truck and ambulance garage.

It will have a total area of 760 m^2 divided as follows:

- 220 m^2 for showers, lockers and toilet area;
- 540 m^2 for offices.

Materials Warehousing

Unheated storage space totaling approximately 640 m^2 will be provided in the warehouse building within the mill site. The building will be used for the storage of building materials, supplies and equipment during the construction phase of the project and for unheated storage

requirements during operations. The building will include a room for the storage of lube oil, which will be supplied in lube cubes. The buildings will be “fold-away” type with 3 metre (10 ft) wide sections and will include all components required for a complete and finished installation.

Permanent Personnel Accommodations

The accommodation complex will be designed to meet the sleeping, hygiene, dining and recreational requirements of 172 persons. The surroundings will provide a level of comfort intended to optimize individual productivity and minimize the adverse effects of being separated from home and family for extended periods of time. Interior design and colour, as well as the selection of furnishings, fittings and fixtures, will be given high priority.

The complex is envisaged to have six accommodation wings in a one-storey configuration connected through a central hub, plus a service core building. A utilidor will connect the central hub to the service core. Another utilidor at the west end of the service core will provide access to the service complex and the concentrator building.

Accommodation Wings and Central Hub

Shower/water closets will be shared centrally within each wing and storey. For every 29 rooms, there will be 6 showers/water closets and wash basins. Each floor of each wing will include a centrally located mechanical/electrical room, janitor’s closet, small lounge and exit stairwell at the opposite end of the wing from the central hub.

The sizing of the accommodation wings was based on the number of rooms required at any time for 172 persons, including dedicated rooms and those shared by rotating personnel, as determined in consultation with CRI. The central hub, which collects the traffic from the six accommodation wings, will include a lobby complete with pay telephones; a baggage room; a lounge; electrical, data and janitor’s rooms; a housekeeping area; and a laundry room. The accommodation wings and central hub will be constructed over a 1.2 metre crawl space with concrete perimeter frost walls. The building will be made up of factory-assembled, prefabricated, modular wood-frame units.

Mill Complex

The mill complex will contain all the equipment needed to process nickel and copper ore into a concentrate product. It will be positioned in the middle of the whole complex.

The mill building will have an area of 3,700 m^2. It will be erected on a concrete slab with piles underneath. Utilidors will be connected from the mill to: administration building, crusher complex, power station, garage and further away, accommodations.

Crusher Complex

The crusher complex will serve to crush ore rock to the right size to feed the mill.

The crusher complex will be installed within an insulated flexible structure. The ore trucks will dump on a grizzly (rock breaker included). An apron feeder will move the ore up to the portable jaw crusher (crusher mounted on wheels, Strongco type). The crushed ore will move from one cone breaker to the other via conveyors and screens. The final product will be dumped outside via a conveyor and will be ready to enter the milling process. The grizzly and the truck dump will be outside the flexible structure.

Flotation Circuit

The flotation circuit's function is to recover nickel and copper from the finely ground ore and separate them into a nickel-copper concentrate stream, a copper concentrate stream [GS8] and a tailings stream.

The flotation circuit separates the grinding circuit discharge into the two sulphide concentrate streams and a main tailings stream. The slurry is first fed to the bulk circuit where the fast floating copper is recovered by the rougher and sent toward the copper circuit to produce a final high grade copper concentrate, while the bulk scavenger recover the nickel and is sent to the Ni-Cu circuit where a nickel concentrate grading from 8 to 12% Ni is produced. The final nickel concentrate is combined with the Ni-Cu concentrate with the copper cleaner tails, which has a nickel content of about 10% to 16% Ni. Both products combined result in a final nickel concentrate grading from 10% to 13% Ni.

Infrastructures and Supply

There will be three main roads for the entire mining complex:

- One existing road from Deception Bay port to the main complex (mill, accommodations, etc.) via Katinniq with a distance of over 118 km;
- One road between Expo deposit and Mesamax deposit (12 kms);
- One road between the Expo site and the Ivakkak deposit passing near Mequillon deposit and over the fresh water dam at Expo (40 km long total).

Generators installed at the mine site will supply all of the energy needed for the complex. As energy is an important factor on the overall costs, the generators will be equipped with heat recovery systems designed to recover waste heat and distribute it to the different areas within the complex.

Puvurnituq River Bridge

To cross the Puvurnituq River at the selected point, two box structures are required. The crossing on the principal arm of the river is approximately 102 metres wide; the secondary crossing on the smaller arm of the river is approximately 90 metres wide, for a total length of 192 metres.

The box structure to cross the Puvirnituq River at the esker location would consist of 8 culverts (bridge plates), 7.9 metre span each, 2 metre height and 10 metre width. Each culvert would require two concrete block foundations placed within the river bed at a time of year when flow is lowest. The voids between the culverts would be filled with granular material harvested from a nearby esker.

Airport

CRI will share the airstrip already in place at Xstrata Nickel's Raglan mine Donaldson airport.

In addition to the existing facilities, the following infrastructures will be provided:

- An expansion to the existing service apron to the gravel airstrip of approximately 100 metres by 75 metres, to accommodate CRI owned facilities;
- Small modular building will be provided at the airstrip to house the de-icing system, a 100 kW generator and any electrical equipment requiring a controlled environment. The building will be heated with electric heaters;
- An air and cargo terminal will be built near the airstrip to accommodate CRI employees for departures and arrivals;
- An aircraft refuelling system will be provided. Two tanks will be installed adjacent to the airstrip. The tanks will be constructed in accordance with applicable regulations of federal airports. The tanks will be double walled with a capacity of 50,000 litres and will contain the hose, meters, and safety features to safely fuel the aircraft; and
- A steel framed fabric covered shelter will be provided to house aircraft service and logistics vehicles.

Port Facilities

The port facilities will be constructed on the southwest shore of Deception Bay, approximately 1 km south of the existing Xstrata Nickel port facilities, and will include the following:

- Concentrate storage building;
- Concentrate conveyor and weighing system;

- A telescopic shiploader providing loading to the centre of ship hold, for both winter and summer operating seasons;
- A wharf consisting of three steel sheet pile cells;
- Fire water pumping station at the wharf;
- Diesel fuel and jet fuel off-loading, storage tank farm and truck-loading facilities;
- General site services, including electrical distribution, potable water, fire protection and sewage septic tank;
- Port administration, accommodation and power generation facilities.

The wharf will be at the north of the storage building and as close as possible in order to minimize the length of the transfer conveyor to the shiploader. The wharf will be sized to accommodate concentrate ships and petroleum ships ranging in size from 10,000 dwt to 25,000 dwt. The staging yard for containers will be at the north of the wharf.

Concentrate Receiving and Storage Building

The concentrate receiving and storage building will be a pre-engineered, conventional steel frame, metal-clad, uninsulated structure.

The building will be 50.5 metres wide x 88 metres long and will be located on the south side of the port site. The main structural members of the building will be supported on concrete foundations seated on rock and anchored to the rock. The floor slab beneath the concentrate stockpiles will be reinforced concrete seated on a bench cut into the hillside.

Two types of concentrate will be stockpiled in the concentrate storage building, in the following tonnages:

- Copper concentrates . . . . . 11,000 t
- Nickel concentrates . . . . . 16,000 t

Wharf

The proposed quay will comprise three steel cells covered by concrete. Design of the quay was the responsibility of GENIVAR.

Each cell will have a radius of 12.5 metres and the accosting face will be 54.3 metres long in 13.5 metres of water at low tide.

Three steel ladders going lower than the low tide will be installed on the quay and, 6 mooring anchors near the vertical face and 2 mooring anchors on the shore will complete the installation.

The total area will be 3,780 m^2 and the quay will be connected to the main road by an 8 metre wide gravel road.

25.4 All Utilities

HVAC Systems

The heating system will be a hot water/glycol loop supplied from the primary loop in the concentrator. Circulating secondary loops will be provided for space heating, ventilation make-up air heating, potable water heating and under-floor heating in the service garage and shops. The welding bays will be equipped with point-of-use extraction hose and fan systems as well as a general exhaust from the bay. Ventilation rates will follow ACGIH recommendations. Vehicle point-of-use extraction will be provided where vehicles engines may be required to run while being serviced.

Domestic Hot Water Storage and Heating System

A centralized domestic hot-water storage and heating system will be provided. Domestic hot water will be heated to 60ºC by low-temperature hot water/glycol. Hot water will be stored in insulated hot-water tanks sized to suit the total number of fixture units, daily requirements for shift change demand and a maximum recovery period of six hours. The system will include a pumped recirculation circuit to minimize the cold draw time at each fixture. In remote areas or areas of small demand, hot water will be provided by individual, electrically heated hot-water heaters.

Sanitary Drainage Systems

The sanitary sewer piping system will carry all non-processed waste from sanitary fixture units and floor drains. All sanitary sewer systems will include a trap venting system. All sanitary drainage will be collected in lift stations and pumped by sewage-grinding type duplex pumps to the accommodation complex sewage lift station.

Storm Drainage Systems

Roof drainage will be collected from all roofs and piped to a common discharge location. The design basis is 60 mm (2.36") of rainfall in one hour based on a 10-year return, as advised by the Canadian Meteorological Centre.

General Fire Protection Systems

The proposed fire protection systems are based on the understanding that the building structures and equipment will be non-combustible, and that a minimal amount of combustible furniture and equipment will be used.

Electrical Services

Electrical Power Supply

Electrical power will be supplied to the service complex from the main 4.16 kV switchgear in the power house. A 5 kV feeder cable will be routed on cable trays through the process plant and the service complex building to the primary side of the unit substation in the electrical room.

Power Supply

Previous studies for arctic area performed by SLI found that the cost of a high-voltage overhead power line from any power plant using heavy fuel at Deception Bay, distant from the site, was economically unfeasible. It was determined that a stand-alone, base generation power plant with diesel-fuelled combustion engines would be the most cost-effective source of power for plant operations. This approach remains the best alternative for the current project development plan.

Main Plant (Expo Site)

The estimated maximum demand load for ore handling and processing, process auxiliaries, plant utilities and building services is 10.6 MW and the average estimated demand is 9.0 MW.

The system requires 5 units running continuously at 70% of nameplate rating.

A total of 6 base diesel generators, 5 operating and 1 standby will be provided to supply electric power to the site facilities.

Port

The estimated maximum demand load for port facilities is 1.2 MW and the estimated average demand is 0.8 MW.

Emergency Electrical System

The emergency system will be comprised of two generators of 1000 kW (1250 kVA) each installed in a prefabricated steel building and connected to the network.

It will supply power for heating and lighting purposes for the various buildings.

25.5 Heat Generation, Recovery and Distribution

Heat Recovery

The diesel generators will be equipped with heat recovery systems designed to recover waste heat from the generator exhaust gases, jacket water cooling, and lube and oil cooling systems. The waste heat from all three sources will be combined and utilized as the primary medium for

building and domestic hot water heating for the concentrator plant, power plant, service complex and accommodation complex.

Under normal conditions, the heat recovery system will be adequate for all building and make-up air heating requirements, with surplus heat available for process heating. To ensure full heating capacity under all circumstances, two packaged, low-temperature, oil-fired boilers will be installed at site as a standby system.

Heat Generation

A heating circuit consisting of one oil-fired boiler located in the emergency service building will make up any shortfall in heating from the diesel generator heat recovery system.

Heat Distribution

The heat distribution system will comprise primary and secondary heating loops circulating water/glycol as the heating medium. The primary heating loop will consist of the two boilers, circulating pumps and an expansion tank operating at a temperature of 93ºC supply and 71ºC return. The return temperature is designed to maintain the minimum temperature required at the hot water boilers.

25.6 Water Supply and Treatment

Fresh Water Requirements

Average fresh water requirements are evaluated at 80 m^3/h when the concentrator reaches its design capacity. The maximum demand within the concentrator building has been estimated at 100 m^3/h.

Upon mine start-up, no recycled water will be received from the mill, so 100% of above-mentioned water requirements will come from fresh water circuit.

Gradually, during the first two years of operation, the quantity of process water recirculated from the tailings pond will increase to reach 520 m^3/h or an average of 54%, and a maximum of 865 m^3/h or 90% in summer.

Selection of the fresh water source is based on its capacity to supply the total required quantity as well as on the quality of raw water and on its nearness to the concentrator site.

Fresh Water Dam

Fresh water will be supplied to the project by an artificial reservoir created by the construction of a rockfill dam on an unnamed tributary of the Puvirnituq river, approximately 5 km west of the Expo site. Water will be pumped from the reservoir to the process plant via an aboveground pipeline.

The dam will also serve as a mine haul road for the Ivakkak open pit. A bridge will be provided to cross the dam spillway.

Hydrology

The watershed area upstream of the proposed reservoir is 153 km^2. This watershed will provide a calculated annual flow of at least 37,000,000 m^3 of water. Public domain climatologic data and hydrologic data gathered by GENIVAR Inc. in the 2004, 2005 and 2006 field seasons were used to calculate this estimate.

The latest observed thaw was in July 2004, but thaw usually begins in June.

The reservoir impounded by the dam will fill very rapidly at the onset of spring thaw. Water shall then overflow through the spillway for the rest of the summer season.

The dam design will allow for the discharge of a minimum flow of 0.2 m^3/s downstream of the dam from the onset of spring thaw, in June or July, to the start of the winter freeze-up, in late September or October, at which time flow in the natural stream essentially ceases. An allowance shall be made in the reservoir live capacity to sustain this water flow.

An allowance for an ice cover thickness of 2.0 meters was made in calculating the live reservoir capacity.

No allowance will be made for evaporation of water from the reservoir, as it is assumed that the summer flow into the reservoir shall largely exceed evaporative losses. Furthermore, once an ice cover has formed on the reservoir, no more evaporation can take place.

Dam Construction

Construction of the dam will take place after spring thaw and summer rains.

A cofferdam(s), two (2) to three (3) metres high will be constructed from local fill material upstream of the propose fresh water dam location. Water will be pumped as required to provide a dry working area. A small cofferdam may be required downstream.

Construction of the dam will take place on the natural river bottom. Key trench will be excavated to bedrock at the upstream toe of the dam to grout and anchor the geomembrane.

The volume of material required is estimated to be 75,000 m^3 – 100,000 m^3 including riprap.

The crest of the dam will be covered with 0.3 metres of road surfacing aggregate, for an approximate volume of aggregate 1275 m^3.

Fresh Water Pipeline

A 6 km long main is required between the water dam pumping station and the fresh water tank.

There is a 40 metre rise between the average level of the water dam and the maximum level of the tank. In order to reduce dynamic head losses and obtain a pressure of operation for the pumps of about 690 kPA (100 psig), the maximum pipeline velocity was limited to 1.5 m/s. To meet this requirement, the inside diameter has been set at 200 mm. This will help reduce the occurrence of transient flow causing water hammers upon sudden interruption of pumps or sudden valve closure.

Even with the limitation of the maximum flow velocity, a line protection system has to be installed. This system consists of a 5.2 m^3 hydropneumatic tank located at the pumping station and an air inlet/outlet gate installed at the high point of the line, about 600 metres from the concentrator.

The pipeline will consist of a HDPE (DR11) of 200 mm diameter insulated with urecon (50 mm) with a final poly coating of 1.27 mm thickness.

The pipe will be installed above ground over its entire length alongside the new road in order to allow maintenance if required. The pipe will be insulated and the water will be heated using heat losses from the diesel pump exhaust heat exchanger. Drain valves will be installed at topographic low points to facilitate draining of the system for maintenance, or to prevent freezing.

Process Water Supply

The plant shall further be designed to process ore from the Expo open pit at a rate of 3,500 t/day, on a 365 days/year basis. As the plant is designed for 90% availability, the corresponding instantaneous production rate shall be 3,900 t/day.

Depending on the tonnage processed, fresh water input to the process plant can vary from 25 m^3/h to 65 m^3/h. annual water requirements for the process plant vary correspondingly from 225,000 m3 to 536,000 m^3.

Fresh Water Tank and Controls

A 1000 m^3 tank will be erected near the emergency building. This tank will provide an approximate 1 hour fresh water reserve (445 m^3) for process requirements and some 455 m^3 for fire protection.

The proposed tank will have a diameter of 12 metres and will be 9.8 metres high. It will be made of steel, and will be insulated.

Drinking Water Supply

The main camp will accommodate a peak maximum of 350 people during part of the construction period, while only 135 people will live there during normal mine operation. The water requirements are evaluated at 450 L/person/day during construction as well as during normal operation. In total, anticipated water consumption will be 160 m3/d during construction (Phase 0) and 60 m^3/d during normal operation of the mine phase. Hourly peaks are evaluated at 200 L/min and 150 L/min respectively.

The main camp potable water pumping station will be equipped with a triplex overpressure system, fled by an outdoor steel insulated tank complete with corrosion-proof inside layer approved for potable water. This inside layer approved for potable water. This tank will contain the operation reserve evaluated at 1,000 m^3 as well as the fire protection reserve.

Domestic Waste Water

The wastewater treatment system is designed to treat the permanent camp and operations complex wastewater. The system will be located between the mill and main camp, 200 metres north of the camp.

The total population on site will be 135 people during normal mining operations, with a peak of 325 people during construction. The wastewater treatment system will be designed to treat 31 m^3/d on a permanent basis, and will be able to bear a peak of 43 m^3/d, which corresponds to a unit flow of 190 L/pers/d during construction.

Mill Waste Water

The proposed water treatment plant is located within the southwest end of the concentrator building, adjacent to the tailing thickener. The plant will filter and treat water from the following sources and then discharge the treated water to the effluent:

- Plant site collecting pond;
- Tailings thickener overflow;
- Concentrate thickener overflows and concentrate filtrate from filters.

25.7 Internal and External Communication Systems

A complete communications system will be installed at the mine-mill complex site, at the port, at the different open pits and along the mine haul roads.

The existing exploration communications system which services the area from Mesamax to Ivakkak will be upgraded as necessary, which provides Internet protocol (IP). VHF and voice

over IP (VOIP) communications to the remote installations. Main satellite connections will be installed at the Expo industrial site and the port.

Vehicles will be equipped with GPS and VHF radios. Vehicles which travel the road to the port will be equipped with satellite telephones.

Fibre optics will be installed at the mine-mill complex for data processing.

25.8 Fuel Supply

Two types of fuel will be on site: diesel fuel for generators and mobile equipment and diesel jet fuel for planes landing at Donaldson airport.

All fuel will come via vessels at the Deception Bay port.

Diesel fuel will be unloaded via the vessel pumping and unloading system to the six tanks (2 ML each).

Fuel Delivery System

The power plant will be fuelled by arctic distillate stored in one main bulk tank near the concentrator building. The tank will hold 5 ML, for a total of 90 days' supply. Fuel from the storage tank will pass through an automatic cleaning system before transfer to one or two tanks of 55,000 L. Clean fuel from the day tank will be pumped into a continuously re-circulating fuel supply/return header piping system and be tapped off for each generator. An outdoor heat exchanger will be provided to extract heat from the fuel recirculated back from the gensets. A separate clean fuel feeder line will supply the smaller day tanks provided as part of the emergency diesel units.

25.9 Schedules and Construction Logistics

Strategy Overview

As the RSNP is located in Quebec's Ungava peninsula and subject to short shipping seasons, schedule slippage for engineering, procurement or construction could delay the entire project for a year. To address this constraint, modularization of the concentrator and of all possible infrastructures has been selected to produce concentrate in April 2010.

This strategy implies leasing and operating a prefabrication site in southern Québec, having all modules shipped by sea to Deception Bay and then hauled to the site on roads especially built or modified for that purpose. A supplier responsible for the transportation of comparable modules 1997 between Quebec City, Deception Bay and the Raglan mill site (owned by Xstrata Nickel) provided input and quotations for the Study. The schedule is included.

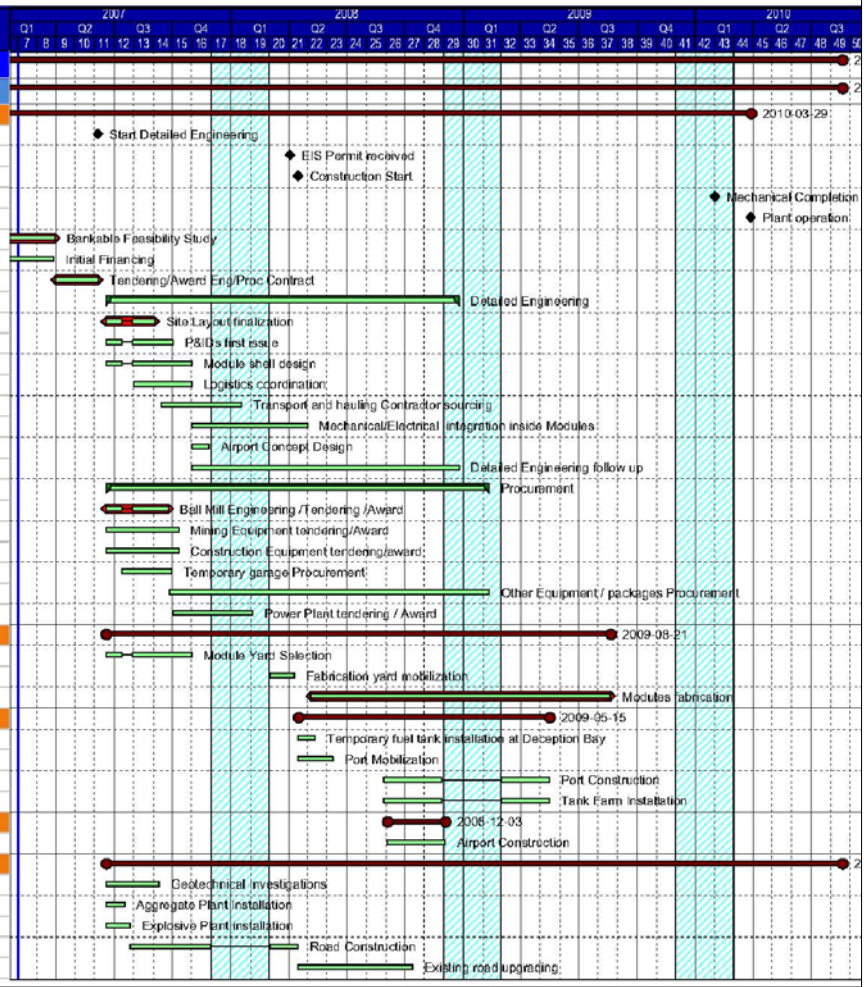
SNC·LAVALIN

Canadian Royalties Inc.

Raglan South Bankable Feasibility Study

SNC-LAVALIN PO 017387-Ms

Activity ID	Activity Name	Start	Finish
Total		2006-08-29 A	2010-08-20
017387-Ms	**Raglan South Bankable Feasibility Study**	2006-08-29 A	2010-08-20
017387-Ms-0	**General**	2006-08-29 A	2010-03-29
A1310	Start Detailed Engineering	2007-06-05	
A1150	EIS Permit received	2008-04-04	
A1040	Construction Start	2008-04-15	
A1130	Mechanical Completion	2010-02-01	
A1140	Plant operation	2010-03-29	
A1000	Bankable Feasibility Study	2006-08-29 A	2007-03-30
A1010	Initial Financing	2006-10-10 A	2007-03-28
A1290	Tendering/Award Eng/Proc Contract	2007-04-02	2007-06-05
A1020	Detailed Engineering	2007-06-18	2008-12-25
A1230	Site Layout finalization	2007-06-18*	2007-09-03
A1240	P&IDs first issue	2007-06-18	2007-10-02
A1250	Module shell design	2007-06-18	2007-11-01
A1260	Logistics coordination	2007-08-02	2007-11-01
A1270	Transport and hauling Contractor sourcing	2007-09-13	2008-01-17
A1280	Mechanical/Electrical integration inside Modules	2007-11-01	2008-05-01
A1520	Airport Concept Design	2007-11-01	2007-11-28
A1300	Detailed Engineering follow up	2007-11-01	2008-12-25
A1030	Procurement	2007-06-18	2009-02-10
A1160	Ball Mill Engineering /Tendering /Award	2007-06-18	2007-09-25
A1480	Mining Equipment tendering/Award	2007-06-18	2007-10-12
A1610	Construction Equipment tendering/award	2007-06-18	2007-10-12
A1580	Temporary garage Procurement	2007-07-13	2007-10-01
A1620	Other Equipment / packages Procurement	2007-09-26	2009-02-10
A1460	Power Plant tendering / Award	2007-10-03	2008-02-05
017387-Ms-8	**Southern Quebec**	2007-06-18	2009-08-21
A1400	Module Yard Selection	2007-06-18	2007-11-01
A1410	Fabrication yard mobilization	2008-03-03*	2008-04-10
A1100	Modules fabrication	2008-05-05*	2009-08-21
017387-Ms-7	**Deception Bay (Port)**	2008-04-15	2009-05-15
A1360	Temporary fuel tank installation at Deception Bay	2008-04-15	2008-05-13
A1370	Port Mobilization	2008-04-15	2008-06-10
A1210	Port Construction	2008-08-28	2009-05-15
A1350	Tank Farm Installation	2008-08-28	2009-05-15
017387-Ms-5	**Berbegamo (Airport)**	2008-09-03	2008-12-03
A1200	Airport Construction	2008-09-03	2008-12-03
017387-Ms-3	**Expo-Ungava**	2007-06-18	2010-08-20
A1490	Geotechnical Investigations	2007-06-18	2007-09-10
A1190	Aggregate Plant Installation	2007-06-19*	2007-07-17
A1510	Explosive Plant installation	2007-06-19	2007-07-27
A1180	Road Construction	2007-07-27	2008-04-15
A1220	Existing road upgrading	2008-04-15	2008-10-14



Hammock — Procurement
Critical Work — Milestone
Non critical work — Summary
Engineering — % Complete

File Name:017387-Ms Baseline: 017387-Ms - B2
Layout: 017387 Raglan South WBS
Filter: TASK filter: 017387 BFS.

Project Start: 2006-08-29
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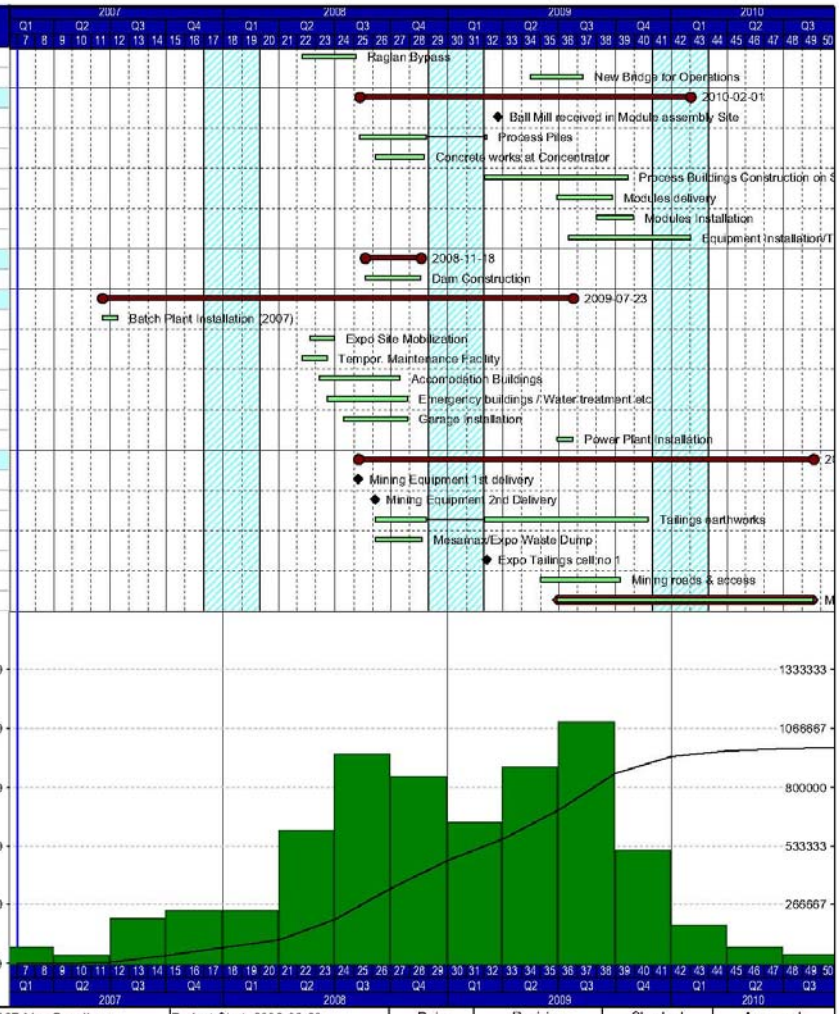
Date	Revision	Checked	Approved
2007-03-01	For planning meeting	YT	
2007-05-28	Revised for Report	YT	MC

Activity ID	Activity Name	Start	Finish
A1390	Raglan Bypass	2008-05-09	2008-08-04
A1380	New Bridge for Operations	2009-05-15	2009-08-10
017387-Ms-3-1	**Metallurgical Process / Concentrator**	2008-08-11	2010-02-01
A1170	Ball Mill received in Module assembly Site		2009-03-24
A1070	Process Piles	2008-08-11	2009-03-06
A1600	Concrete works at Concentrator	2008-09-05	2008-11-24
A1080	Process Buildings Construction on Site	2009-03-02	2009-10-22
A1110	Modules delivery	2009-06-29	2009-09-25
A1120	Modules Installation	2009-09-01	2009-10-30
A1090	Equipment Installation/Tie-in	2009-07-17	2010-02-01
017387-Ms-3-2	**Utilities**	2008-08-19	2008-11-18
A1440	Dam Construction	2008-08-19	2008-11-18
017387-Ms-3-3	**Infrastructures**	2007-06-19	2009-07-23
A1590	Batch Plant Installation (2007)	2007-06-19	2007-07-13
A1420	Expo Site Mobilization	2008-05-22	2008-07-01
A1570	Tempor. Maintenance Facility	2008-05-09	2008-06-18
A1060	Accomodation Buildings	2008-06-05	2008-10-15
A1430	Emergency buildings / Water treatment etc	2008-06-19	2008-10-28
A1560	Garage Installation	2008-07-15	2008-10-28
A1450	Power Plant Installation	2009-06-29	2009-07-23
017387-Ms-3-4	**Geology / Mining**	2008-08-08	2010-08-20
A1470	Mining Equipment 1st delivery	2008-08-08	
A1500	Mining Equipment 2nd Delivery	2008-09-04	
A1330	Tailings earthworks	2008-09-04	2009-11-24
A1540	Mesamax/Expo Waste Dump	2008-09-04	2008-11-21
A1530	Expo Tailings cell no 1		2009-03-05
A1340	Mining roads & access	2009-06-01*	2009-10-08
A1320	Mining preparation	2009-06-29	2010-08-20



Hammock
Critical Work
Non critical work
Engineering
Procurement
Milestone
Summary
% Complete

File Name:017387-Ms Baseline: 017387-Ms - B2

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Date	Revision	Checked	Approved
2007-03-01	For planning meeting	YT	
2007-05-28	Revised for Report	YT	MC

The detailed engineering contract has to be awarded as soon as possible (summer 2007) to allow concentrator modules to be ready for shipment during the summer of 2009. Only by doing so is it feasible to assemble the modules on site and to complete all tie-ins during the following months. This brings the northern construction efforts in winter time when all equipment is in place and protection against the weather is provided. Thus, the plant would begin production before or by the spring of 2010.

Engineering

Since the shipping season is short, it is mandatory to have the modules ready to leave the fabrication site in early June 2009. Typical module construction will take 6 to 10 months. Assuming 20 modules, a 15 to 18 months period is necessary to complete modules fabrication. Before stating the construction period, 3 months are needed for steel and miscellaneous procurement activities. An additional 4 to 6 months of detailed engineering are required to allow a competitive tendering process that will last 2 to 3 months.

The total time required is 24 months. Detailed engineering must be initiated at the latest in June 2007 to meet the June 2009 shipping target.

Modularisation Concept vs Equipment Selection

The modularisation concept has been addressed from the beginning of the project design phase. Modules design was developed in a standardized way allowing enough flexibility to fit a wide variety of equipment and substructures. The dimensions must be suitable for barging and hauling with standard heavy hauling equipment.

Furthermore, allowances must be made in the structural design for sea shipment and road haulage stresses. Long lead equipment, such as ball mills, must be assessed and ordered in the very first weeks of detailed engineering. Ball mill lead time may be as long as 18 months after award, and delivery has to be early enough to allow sufficient time for mechanical installation inside the module. Modules design will also depend on vendor drawings and engineering will be delayed if major equipment vendor data is not available in due time. Obviously, any major change would inevitably lead to the addition of a full year to the project schedule.

Construction

Construction activities can be split in two principle areas: the southern module construction yard, and the site construction itself.

For the module construction site, it is mandatory to work on a two shift basis to gain time and to make more room for the workers inside modules which will facilitate manoeuvres and decrease the risk of accidents.

Construction at site may be divided into three parts:

- Early works;
- Infrastructures;
- Process.

25.10 Environmental Considerations

GENIVAR executed the Environmental and Social Impact Assessment study for the RSNP. This section of SLI's Feasibility Study report is adapted from GENIVAR final report and also from the Executive Summary of the Environmental and Social Impact Assessment, also prepared by GENIVAR.

25.11 Operating Cost Estimate

Some important assumptions were used to develop the operating costs. These assumptions are presented hereafter under subtitles corresponding to the cost breakdown.

General

Operations will begin on April 1st 2010 and terminate in September 2018. Table 22 summarizes the operating cost and sustaining capital cost estimates for the RSNP.

The G&A operating costs includes salaries, accommodations, energy costs related to this section. Mining operating costs include manpower, equipment, fuel related to the production part. Concentrator operating costs include also manpower, equipment, mill building and a split for the energy costs.

Sustaining capital is required during the operation of the project to cover the investment for either additional or replacement of ageing equipment, for capital components of tailing and waste rock disposal facilities such as geotechnical liners, the road development to the Ivakkak deposit, the civil works around that open pit and the underground workings development once the pit has been depleted.

Table 22: Summary of Estimated Operating Cost and Sustaining Capital

	Year	2010	2011	2012	2013	2014	2015	2016	2017	2018
		1 (9 Month)	2	3	4	5	6	7	8	9
Quantity										
Tonne Ore	Kt	910	1,278	1,278	1,278	1,278	1,278	1,278	1,278	868
Tonne Waste	Kt	2,744	3,539	3,767	2,441	4,248	4,278	4,130	4,016	4,341
Total Tonnes	**Kt**	**3,654**	**4,816**	**5,045**	**3,718**	**5,526**	**5,555**	**5,407**	**5,294**	**5,209**
Stripping Ratio	Kt / Kt	3.01	2.77	2.95	1.91	3.33	3.35	3.23	3.14	5.00

	Year	2010 1 (9 Month)	2011 2	2012 3	2013 4	2014 5	2015 6	2016 7	2017 8	2018 9
Operating Cost										
G&A	K$	16,478	23,508	23,654	23,129	23,162	23,110	22,860	22,860	15,540
Mine	K$	10,535	14,713	19,473	34,384	38,165	24,055	15,249	15,155	12,317
Concentrator	K$	20,276	31,616	31,146	30,270	30,535	30,311	30,604	30,338	18,710
Total Operating Cost	**K$**	**47,290**	**69,837**	**74,273**	**87,783**	**91,862**	**77,475**	**68,714**	**68,353**	**46,567**
Sustaining Capital										
Mobile Equipment	K$	0$	1,040$	4,121$	1,286$	110$	220$	2,402$	0$	0$
Others	K$	8,718$	1,302$	327$	3,681$	860$	1,523$	6,322$	1,095$	10,661$
Concentrator	K$	0$	0$	0$	0$	0$	0$	0$	0$	0$

Manpower - Labour

The salaries are based on an 11 hour day, with a rotation schedule of 2 weeks in and 2 weeks off. The employee benefits were estimated at 33% of the base salaries.

A list of the manpower requirements for a typical year in provided in Table 23.

Table 23: Manpower Requirements – Typical Year

Operation	Off-site	Rotation 1	Rotation 2	Total
General Administration	8	23	22	53
Mine	0	41	42	83
Concentrator	0	51	50	101
Hauling	0	11	12	23
Grand Total	**8**	**126**	**126**	**260**

Mobile Equipment

Both actual and historical costs from other operations of a similar capacity are used to estimate the RSNP operating costs. Other sources used for cost estimation included vendor quotes, commercial database such as Western Mine Engineering Cost Estimating Guide and SLI's experience in similar projects.

Electricity and Energy

The arctic fuel is delivered at Deception Bay in two shipments of 10,000,000 litres at 0.745$/L. An extra tax of 0.124$/L is added to this price for public road transportation between the port and the Expo site. Fuel prices are based on the 2006 average port of Montreal rack price, plus quoted maritime shipping costs to Deception Bay. Electricity cost is based on operation of CAT-3612 generators, with maintenance cost estimates provided by the supplier, and the fuel prices above.

Concentrate Hauling

The cost for the port operation includes: concentrate transportation to the port, stockpiling, loading into the ship, unloading from the ship, transportation to the Expo site, road access and port maintenance and the energy and cost to maintain the port infrastructure. This cost is divided in two parts to obtain separately the cost for the transportation of concentrate from Expo and loading into the ship. A factor of 60% of the total port operating cost is applied to concentrate hauling, and the remaining 40% is applied to general and administration costs.

25.12 Cost Estimating Methodology

A list of the estimated direct and indirect costs for the entire project is provided in Table 19.

Quantities were developed for bulk products such as earthwork, concrete work, structural steel, piping bulk, electrical bulks and instruments.

In-house pricing based on current projects for which SLI is acting as construction manager were used to develop this estimate for the majority of the bulk materials, in addition to several R. S. Means Construction Cost Index manuals.

Labour basis for the piping estimate was based from “Estimator’s Piping Man-Hours Manual” by John S. Page, published by Gulf Professional.

For electricity, instrumentation and control, the National Electrical Association Manual of Labor Units – 2003-2004 edition (NECA) was used as reference.

The current budget estimate reflects an accuracy range of – 5% to + 15% and is prepared to support full project funding requests, and become the first of the project phase “control estimates” against which all actual costs and resources will be monitored for variations to the budget funding for the RSNP.

All estimated costs are in Canadian; second quarter 2007 Dollars (CDN Dollars).

This estimate is structured and based on the philosophy that the contracts will be awarded to qualified contractors on a lump-sum basis.

This data was evaluated in order to assist in labour productivity adjustments and the development of composite labour rates. Construction work is based on a 40 hours workweek for the components pre-assembled off site and on 70 hours per week for the activities to be accomplished on site.

The unit rates were used to estimate the construction costs of the following:

- Earthwork;
- Concrete Work;

- Structural Steel Work;
- Equipment Erection;
- Piping Work – fabricated and field, and associated items;
- Electrical Work.

In-house pricing was used to develop the estimate for piping bulk, electrical and instrumentation bulk.

The hourly construction labour rate is developed from the industrial sector of the collective agreement dedicated to the Construction in Quebec starting May 01, 2007 and from the proper rate per classification unit established by "La Commission de la Santé et de la Sécurité du Travail du Québec".

Table 24: Direct and Indirect Field Cost

Direct Costs:	Cdn$ (2007)
Administration	
Airport Facilities	797,978
Buildings, Utilities and Infrastructures	69,241,546
Port Facilities (Wharf, warehouse, fuel and shiploading)	51,793,300
Power and cogeneration facilities	22,480,175
Subtotal	144,312,999
Mining	
Prestripping - Expo and Mesamax	14,087,165
Expo and Mesamax Mines	2,014,065
Expo Co-disposal Site (Golder)	2,569,423
Mining and Administrative Mobile Equipment Fleet	21,474,964
Subtotal	40,145,617
Milling	
Mineral Processing Facility (SLI, less tailings)	75,043,586
Paste tailings plant and pipeline (Golder)	8,158,934
Subtotal	83,202,520
Indirect Costs	
Construction Indirects	47,246,345
Contingency (10%)	39,025,757
Engineering, Procurement, Construction Management (“EPCM”)	44,873,130
Owners Costs	39,463,632
Subtotal	170,608,864
Total of Direct and Indirect Costs	438,270,000

Productivity factors were applied and they account for labour requirements as compared to SLI standards and anticipated construction duration, labour requirement, site conditions, project location, etc.

Currency Exchange Rates Used for the Estimate

For proposal pricing purposes, a currency exchange rate has been assumed at **1 CAD = 0.89 USD**. This exchange rate is based upon a 12 month trailing average calculation.

Contracting Philosophy

The Client philosophy is:

1. To execute all principle earthworks by the company's mining team. Installation of pilings for mill foundations and the quay, grouting and the installation of geotechnical liners will be subcontracted to specialist firms;

2. To supervise all the construction work done by others;

3. To modularize the plant as much as possible. Fabrication of the modules will be shipped by boat, transported from the port to the site by a specialized company and assembled in place.

Allowance

An allowance of 7.5% has been considered on quantities for take-off accuracy, shrinkage and standard minimum lot purchases including wastage and overbuy.

Contingency

The contingency is estimated at 10% of the total direct and indirect costs.

The contingency is an allowance for undefined items of work that would be incurred within the defined scope of work covered by the estimate that cannot be explicitly foreseen or described at the time of the estimate.

Material Handling and Shipping Costs

All purchased bulk material shipping and handling costs are established based on the actual weight and/or volume of items. The shipping handling cost includes the cost of packing, shipping, loading, unloading and delivery to the job site.

A budgetary quotation was obtained for concentrator modules rigging, shipping and handling cost.

Bonds and Insurances

This estimate does not include an amount for bank guarantees or bonds to be provided by the contractors. Construction wrap-up insurance covering is estimated in the direct cost for the contractors' portion, and in the owner's cost for the remaining portion.

25.13 Closure

Cost estimation for post-closure rehabilitation of the site was based on experience gained from similar operations. No bids were obtained from demolition contractors; considering that the rehabilitation will likely take place 10 or 15 years from now, it was considered that factored estimates based on similar work would provide a satisfactory estimate for the purpose of the study. Cost estimations have an accuracy level of ± 50 % related to the provided information and the primary development grade assigned to this work.

Table 25: Cost Breakdown for Closure and Rehabilitation

	Description	Cost ($)
A	Buildings and surface infrastructures dismantlement	13,330 000 $
B	Management of residual and dangerous materials resulting from demolition	800,000 $
C	Contaminated soils	2,250,000 $
D	Post-closure environmental monitoring	500,000 $
E	Engineering, Management, Inspection	2,180,000 $
F	TOTAL (A+B+C+D+E):	**19,060,000 $**

25.14 Financial Analysis

Summary

Discounted cash flow modelling of the project yields a full equity base case internal rate of return (IRR) of 8.1 % and a net present value of CAD$0.85 million at a discount rate of 8%, both calculated after tax and in nominal terms and assuming an annual inflation rate of 2% during the mine operation. The primary project variables are outlined below.

- Construction Cost : 465.7 M (escalated in CAD$)
- Mine Life : 8.5 years
- Nickel price : $8 USD/lb in 2010, $6.5 USD/lb in 2011 and $6.0 USD/lb throughout life-of-mine (in 2007 USD)

- Copper price : $2.25 USD/lb in 2010, $1.9 USD/lb in 2011 and $1.5 USD/lb throughout life-of-mine (in 2007 USD)
- Total Nickel content in Ni concentrate throughout life-of-mine (LOM) : 75,453 tonnes
- Total Copper content in Cu concentrate throughout LOM : 91,741 tonnes

Methodology

The financial model covers 8 months of development, 2 years of construction (March 2008 to March 2010) and 8.5 years of production.

All capital and operating cost estimates were expressed in second quarter 2007 Canadian dollars.

Summary of Input Data

The financial analysis was carried out using the following assumptions for the base case:

1. All amounts are expressed in Canadian dollars, except the metal prices which are expressed in US dollars;
2. Project IRR is estimated using the discounted cash flow methodology;
3. The total project life is 10.5 years, i.e. 8.5 years of operation after a 2 year construction period;
4. Provincial and federal income tax, Quebec capital tax and Quebec Mining Rights were considered in the calculation of the project IRR on an after tax basis;
5. The project will be 100% equity financed;
6. An annual inflation rate of 2% was considered during operation while a 3.5% annual escalation was considered during the construction period;
7. A 0.89 USD/CAD exchange rate, which corresponds to the average exchange rate over the past 12 months, was used over the project life.

Working capital has been established as follows:

1. 90 days of sales for receivables;
2. 30 days of expenses for payables;

3. $8.2 M CAD for the inventory (mainly first fill of fuel, reagents, etc.);

4. Minimum cash of $500,000.

The detailed assumptions are as follows:

Revenues

The market outlook for metal prices was provided by CRI and is based on the Global Commodities Review published by Morgan Stanley on January 30, 2007. The following table presents the metal prices (in 2007 USD) considered in the financial model for the base case.

Table 26: Base Case Assumed Metal Prices (in 2007 USD)

Metal	Unit price	2010	2011	2012 and Beyond
Ni	USD/lb	$8.00	$6.50	$6.00
Cu	USD/lb	$2.25	$1.90	$1.50
Co	USD/lb	$15.0	$15.0	$15.0
Au	USD/oz troy	$500	$500	$500
Pt	USD/oz troy	$1000	$1000	$1000
Pd	USD/oz troy	$300	$300	$300

In addition to the base case scenario, three alternative scenarios were considered by CRI for nickel and copper prices. The future metal prices of those alternative scenarios were established by CRI and are provided only to demonstrate the sensitivity of the Project IRR to metal prices variations, as the future nickel and copper prices of those scenarios are not currently supported by a market outlook study.

- **Scenario 1** is based on a nickel price of $8.00 USD/lb and a copper price of $1.75 USD/lb throughout the project life.

- **Scenario 2** is based on a nickel price of $10.00 USD/lb and a copper price of $2.00 USD/lb throughout the project life.

- **Scenario 3** is based on a nickel price of $15.00 USD/lb and a copper price of $2.50 USD/lb throughout the project life.

All other metal prices remained unchanged from the base case scenarios.

At this stage, CRI does not have an off-take agreement or a smelting contract for its concentrates. The Net Smelter Return (NSR) was calculated using payable metals and smelting/refining charges typical for copper concentrates, and indicative terms negotiated by CRI for nickel concentrates. The actual revenues realized from the sale of nickel and copper concentrates may differ from the revenue predictions of this model, depending of the smelting/refining contract(s) negotiated for the project.

Expenses

The operating costs, which vary on an annual basis, can be summarized as follows (presented in 2007 CAD):

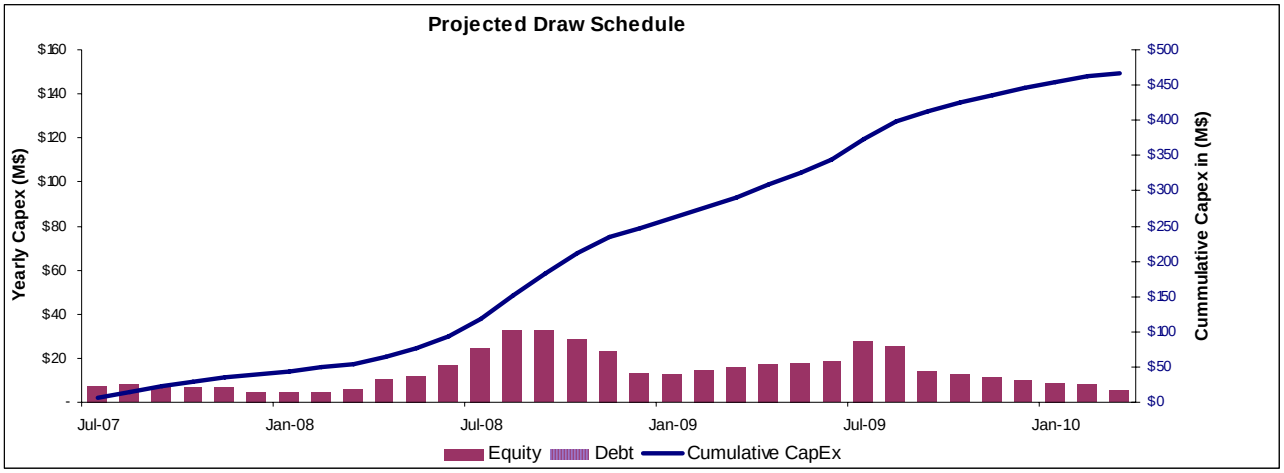
G&A	: **$18.12**/ metric ton of ore
Mining cost	: **$16.88**/ metric ton of ore
Milling cost	: **$23.55**/ metric ton of ore

Capital Expenditures

The construction cost of the project is estimated to be approximately $438.3 million in second quarter 2007 dollars (not escalated), of which $232.1 M are direct construction costs and $206.2 M are indirect costs (engineering and procurement, construction management, pre-stripping of open pits, construction equipment, etc). The capital cost is spent during the development period from July 2007 to February 2008 and then during the 2 year construction period from March 2008 to March 2010. Operations are scheduled to commence in April 2010. The construction cost has been escalated in the financial model, using an annual inflation rate of 3.5%, for a total capital cost of $465.7 millions. The inflation rate takes into account the construction labour and the materials inflation rates.

The capital cost does not include any financing fees. The projected draw schedule during the development and construction period is shown on Figure 3.

Figure 3: Projected Draw Schedule during Construction



The level of capital expenditures has been escalated during the construction period, using an annual inflation rate of 3.5%.

Sustaining capital expenditures, which will begin in 2010, were estimated at a total of $43.7 million (in 2007 dollars) throughout the project life. Approximately half of the sustaining capital relates to the tailings and waste rock management.

Fiscal and Balance Sheet Conditions

For the purpose of the financial model, the following assumptions were used :

1. Corporate Income Tax: The corporate income tax rate of mining takes into account the provincial and federal income tax rates. The capital cost allowance (CCA) was set at 25% on a declining balance basis for all capital expenditures. The Accelerated Capital Cost Allowance was used to provide for an additional depreciation allowance of up to 100% of asset cost. For accounting purposes, capital expenditures were depreciated on a straight-line basis over the project life;

2. Mining Rights: The Mining Rights were calculated for the province of Quebec. The RSNP is eligible for a northern mine allowance, in addition to the processing allowance. Since the project life is relatively short, it is anticipated that the project will not pay any mining rights during the life of the project;

3. Provincial Capital Tax: The 0.29% provincial capital tax was included in the financial model.

Financing

Analyses were carried out for a 100% equity financed project and the financial model does not include any financing up-front fee or equity underwriting fee.

Results

The Project Internal Rate of Return (IRR) is the rate at which the future cash flow profile must be discounted to achieve a net present value today of zero. The financial model developed for the RSNP yields a full equity Project IRR of 8.1 % after tax in nominal terms for the base case. When discounted at 8%, the Net Present Value (NPV) of the project is $ 0.85 M.

The project IRR and NPV, both after taxes as well as the payback period, are presented in Table 27 for the base case and for scenarios 1, 2 and 3.

Table 27: Summary of Financial Analysis Results

	Base Case	Scenario 1	Scenario 2	Scenario 3
Ni Price Long term (2011 and beyond, in USD/lb)	$6.00 /lb	$8.00 /lb	$10.00 /lb	$15.00 /lb
Cu Price Long Term (2011 and beyond, in USD/lb)	$1.50 /lb	$1.75 /lb	$2.00 /lb	$2.50 /lb
NPV after tax (discounted at 8%, in M$ CAD)	$0.85 M	$115.6 M	$282.8 M	$674.0 M
IRR Project after tax	8.1 %	15.3 %	24.0 %	39.6 %
IRR Project before tax	11.3 %	20.0 %	30.5 %	49.6 %
Payback period (years of operation)	2.9	2.4	1.9	1.4

Overall Cash Flow

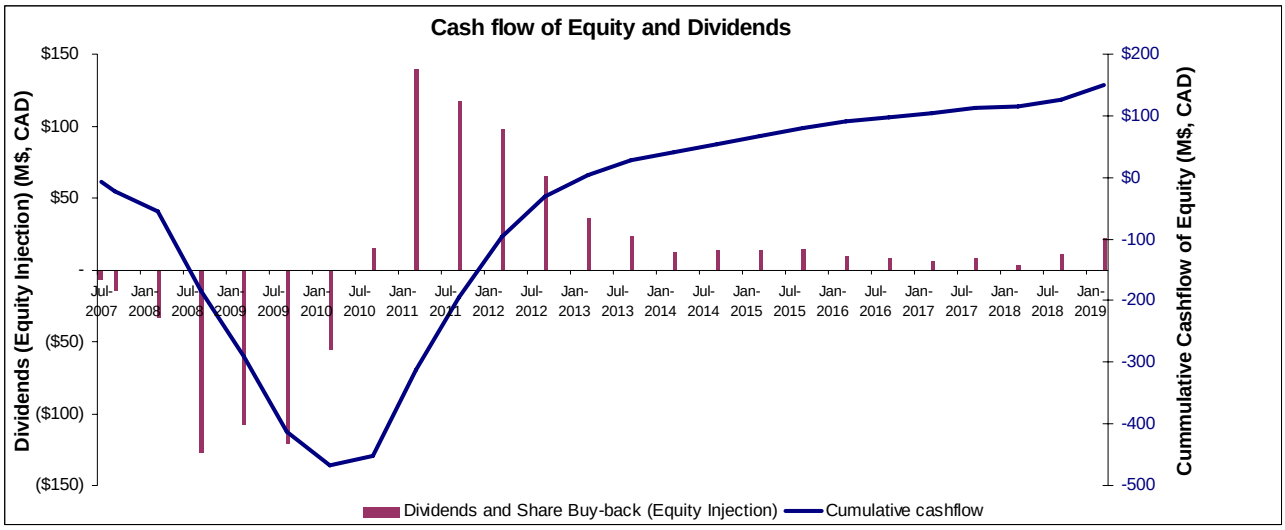
Table 28 presents the revenue and disbursements of the project on a dollar/metric ton of ore basis for the base case, for selected years.

Table 28: Cash-Flow of Disbursements and Revenues on a $/t Basis

Year	2011	2012	2014	2016	2018
Cost per m.t. of ore					
Operating cost ($/m.t. of ore)	58.11	60.36	81.11	65.91	66.26
Sustaining capital ($/m.t. of ore)	3.34	2.44	1.75	6.46	11.34
Taxes ($/m.t. of ore)	1.07	1.01	9.81	6.35	2.51
Variation in WC ($/m.t. of ore)	(6.98)	(8.55)	(0.63)	(2.07)	(7.72)
Div/Share Buy-back ($/m.t. of ore)	203.99	168.03	20.15	14.32	11.84
Total cost ($/m.t. of ore)	259.53	223.28	112.19	90.98	84.24
Revenue ($/m.t. of ore)	259.53	223.28	112.19	90.98	84.24

Figure 4 shows the Equity injection during the development and construction period and the dividends and share buy-back generated during the life-of-mine.

Figure 4: Cash flow of Equity Injection and Dividends/Share Buy-back



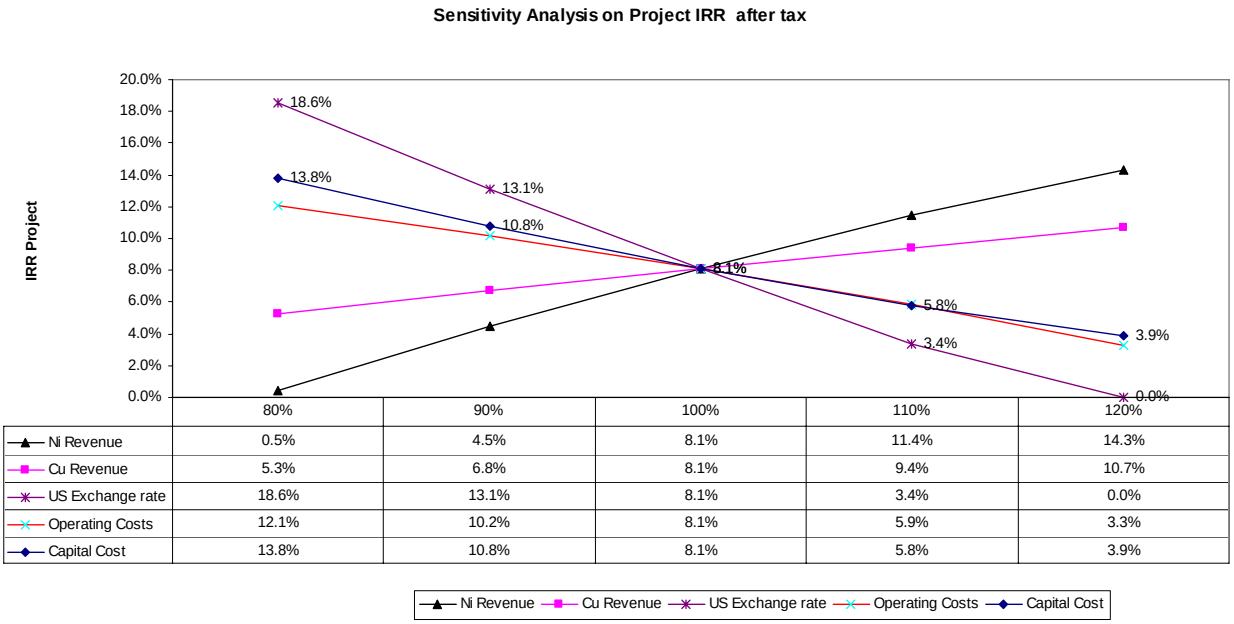
Sensitivity Analysis

The sensitivity of the project IRR (after tax) to changes in the key variables is shown in Figure 5, for +/- 10% and 20% changes from the base case assumptions in key parameters. The selected parameters were:

1. Capital expenditure and Operating costs;
2. Copper and Nickel prices;
3. USD/CAD exchange rate.

The sensitivity analysis shows that the project IRR is mainly sensitive to variations in USD/CAD exchange rate, capital expenditure and Ni price.

Figure 5: Sensitivity of Unleveraged Project IRR to Key Parameters – Base Case



	80%	90%	100%	110%	120%
Ni Revenue	0.5%	4.5%	8.1%	11.4%	14.3%
Cu Revenue	5.3%	6.8%	8.1%	9.4%	10.7%
US Exchange rate	18.6%	13.1%	8.1%	3.4%	0.0%
Operating Costs	12.1%	10.2%	8.1%	5.9%	3.3%
Capital Cost	13.8%	10.8%	8.1%	5.8%	3.9%

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