Hard Creek Nickel Corporation
Turnagain Nickel Project, British Columbia
NI 43-101 Technical Report on Preliminary Assessment

Prepared by: Greg Kulla, P.Geo.
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Effective Date: 25 September 2007

Project No.: 155195

IMPORTANT NOTICE

This report was prepared as a National Instrument 43-101 Technical Report for Hard Creek Nickel Corporation (HCNC) by AMEC Americas Limited (AMEC). The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in AMEC's HCNC services, based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by HCNC subject to the terms and conditions of its contract with AMEC. This contract permits HCNC to file this report as a Technical Report with Canadian Securities Regulatory Authorities pursuant to National Instrument 43-101, *Standards of Disclosure for Mineral Projects.* Except for the purposes legislated under provincial securities laws, any other uses of this report by any third party are at that party's sole risk.

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I, Gregory Kenneth Kulla, P. Geo., am a Professional Geoscientist employed as a Principal Geologist of AMEC Mining and Metals Consulting Services and residing at 9756 Crown Crescent in the City of Surrey in the Province of British Columbia.

I am a member of the Association of Professional Engineers and Geoscientists of British Columbia (APEGBC). I graduated from the University of British Columbia with a Bachelors of Science in Geology degree in 1988.

I have practiced my profession continuously since 1988 and have been involved in precious and base metal disseminated sulphide deposit assessments in Canada, United States, Australia, Mexico, Chile, Peru and India.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43-101.

This certificate applies to the Technical Report entitled “Turnagain Nickel Project, British Columbia, NI 43-101 Technical Report on Preliminary Assessment” that has an effective date of 25 September 2007.

I visited the Turnagain Nickel Property from June 18 to 22 June 2007.

I am responsible for the preparation of Sections 2 to 15, 17, 18, and portions of Sections 1, 20 to 23 of the Technical Report.

I am independent of Hard Creek Nickel Corporation Ltd, as independence is described by Section 1.4 of NI 43-101.

I have been involved with the Turnagain Nickel Project from 2006 to 2007 in the areas of geology, database validation, and resource estimation for the Preliminary Assessment.

I have read NI 43-101, and this report has been prepared in compliance with that instrument and form.

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.

Dated 11 January 2008.

“signed and sealed”

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TO: Alberta Securities Commission
British Columbia Securities Commission

Re: Hard Creek Nickel Corporation – Press Release Dated 10 December 2007 Entitled "Preliminary Assessment for Turnagain Nickel Project Shows Positive Economics with Potential 29 Year Mine Life"

I, Greg Kulla, P. Geo., consent to the public filing of Sections 1 to 15, 17, 18, 20 to 23 of the Technical Report entitled "Turnagain Nickel Project, British Columbia, NI 43-101 Technical Report on Preliminary Assessment" with an effective date of 25 September 2007 (the "Technical Report").

I consent to extracts from, or a summary of, Sections 1 to 15, 17, 18, 20 to 23 of the Technical Report being cited in the Hard Creek Nickel Corporation press release dated 10 December 2007 and entitled "Preliminary Assessment for Turnagain Nickel Project Shows Positive Economics with Potential 29 Year Mine Life".

I confirm that I have read the press release and that it fairly and accurately represents the information in the Technical Report that supports the disclosure.

Dated 11 January 2008.

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I, Gerrit Vos, P. Eng., am currently employed as a Principal Mining Engineer with AMEC Americas Limited.

This certificate applies to the Technical Report entitled "Turnagain Nickel Project, British Columbia, NI 43-101 Technical Report on Preliminary Assessment" that has an effective date of 25 September 2007.

I am a Professional Engineer in the province of British Columbia. I graduated from the Technical University of Delft, Holland, with a Masters of Science degree in Mining Technology, in September 1978.

I have practiced my profession for 29 years. Since 1978 I have continually been involved in mining projects and mining designs for different mining companies in Surinam, Liberia, Namibia, Brazil and Canada.

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

I visited the Turnagain Nickel Property between 18 and 20 June 2007.

I am responsible for the preparation of Section 19, and portions of Sections 1, 20 to 23 of the Technical Report.

I am independent of Hard Creek Nickel Corporation Ltd, as independence is described by Section 1.4 of NI 43-101.

I have been involved with the Turnagain Nickel Project from March 2007 to December 2007 in the area of Mining for the Preliminary Assessment.

I have read NI 43-101, and this report has been prepared in compliance with that instrument and form.

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.

Dated 11 January 2008.

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TO: Alberta Securities Commission
British Columbia Securities Commission

Re: Hard Creek Nickel Corporation – Press Release Dated 10 December 2007 Entitled "Preliminary Assessment for Turnagain Nickel Project Shows Positive Economics with Potential 29 Year Mine Life"

I, Gerrit Vos, P. Eng., consent to the public filing of Sections 1, 19, 20 to 23 of the Technical Report entitled "Turnagain Nickel Project, British Columbia, NI 43-101 Technical Report on Preliminary Assessment" with effective date of 25 September 2007 (the "Technical Report").

I consent to extracts from, or a summary of, Sections 1, 19, 20 to 23 of the Technical Report being cited in the Hard Creek Nickel Corporation press release dated 10 December 2007 and entitled "Preliminary Assessment for Turnagain Nickel Project Shows Positive Economics with Potential 29 Year Mine Life".

I confirm that I have read the press release and that it fairly and accurately represents the information in the Technical Report that supports the disclosure.

Dated 11 January 2008.

AMEC Americas Limited

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I, Tony Lipiec, P. Eng., am a Senior Process Engineer for AMEC Americas Limited.

This certificate applies to the Technical Report entitled "Turnagain Nickel Project, British Columbia, NI 43-101 Technical Report on Preliminary Assessment" that has an effective date of 25 September 2007.

I am a Professional Engineer in the province of British Columbia. I graduated from the University of British Columbia with a B.A.Sc. degree in Mining & Mineral Process Engineering, in 1985.

I have practiced my profession for 22 years, and have previously been involved with metallurgical design and process engineering for base metal and disseminated sulphide projects in Canada and Indonesia.

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

I visited the Turnagain Nickel Property from 13 to 15 September 2006.

I am responsible for the preparation of Section 16, and portions of Sections 1, 20 to 23 of the Technical Report.

I am independent of Hard Creek Nickel Corporation Ltd, as independence is described by Section 1.4 of NI 43-101.

I have been involved with the Turnagain Nickel Project from February 2006 to December 2007 in the area of metallurgy for the Preliminary Assessment. I have previously contributed to a Technical Report on the property, entitled:

AMEC, 2006: Turnagain Project, Preliminary Assessment of Turnagain Property: Technical Report prepared for Hard Creek Nickel Corporation by AMEC Americas Ltd., effective date 1 June 2006.

I have read NI 43-101, and this report has been prepared in compliance with that instrument and form.

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.

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TO: Alberta Securities Commission
British Columbia Securities Commission

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I, Tony Lipiec, P. Eng., consent to the public filing of Sections 1, 16, 20 to 23 of the Technical Report entitled "Turnagain Nickel Project, British Columbia, NI 43-101 Technical Report on Preliminary Assessment" with an effective date of 25 September 2007 (the "Technical Report").

I consent to extracts from, or a summary of, Sections 1, 16, 20 to 23 of the Technical Report being cited in the Hard Creek Nickel Corporation press release dated 10 December 2007 and entitled "Preliminary Assessment for Turnagain Nickel Project Shows Positive Economics with Potential 29 Year Mine Life".

I confirm that I have read the press release and that it fairly and accurately represents the information in the Technical Report that supports the disclosure.

Dated 11 January 2008.

AMEC Americas Limited

"signed and sealed"

Tony Lipiec, P. Eng.
Senior Process Engineer
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1.0 SUMMARY

Hard Creek Nickel Corporation (HCNC) holds a 100% interest in the Turnagain nickel property situated 70 kilometres east of Dease Lake in northern British Columbia (BC). The property consists of two legacy 4-post mineral claims, twenty 2-post claims and fifty-two cell mineral claims covering an area of approximately 29,370 hectares (293 square kilometres) centred on Turnagain River. Primary access to the property is by helicopter from Dease Lake. An airstrip at the property was lengthened to 900 metres in 2007 and is used to service the camp. A road from Dease Lake is used to convey heavy equipment to the project area but is not suitable for regular traffic.

This report, prepared at the request of HCNC, is based on discussions with company personnel, records of recent exploratory work provided by the company, information readily available in the public domain, previous technical reports prepared by N.C. Carter on the 2003 and 2004 exploration programs dated 21 April 2004 and 15 June 2005, and metallurgical testwork conducted by Process Research Associates (PRA) from 2004 to 2006. This Preliminary Assessment (PA) also relies on a Scoping Report and a PA prepared by AMEC in June 2006 (AMEC, 2006). Two of the authors, Greg Kulla and Gerrit Vos, conducted a site inspection of the property between 18 and 22 June 2007.

The first mineral claims in the area of the current Turnagain property were staked in 1956. Exploratory work since 1966, carried out by a number of operators including HCNC involved geological mapping, geophysical and geochemical surveys and more than 58,797 m of diamond drilling in 248 holes by 25 September 2007. HCNC will have completed an additional 16,700 m of drilling in 23 holes by the end of the 2007 drill season.

Nickel mineralization on the Turnagain property is associated with a zoned, Alaskan-type ultramafic body within Palaeozoic metasedimentary and lesser metavolcanic rocks adjacent to the faulted terrane boundary between the North America cratonic margin and accreted Quesnel terrane. The Turnagain ultramafic body hosting mineralization is elongate in a north westerly direction and measures 8 x 3.5 km. The mineral resources discussed in this report lie within the Horsetrail, Duffy and Hatzl zones and cover an area approximately 2.2 km long and 0.8 km wide in the southeast area of the Turnagain ultramafic body. Disseminated nickel and iron sulphides, predominantly pentlandite and pyrrhotite are preferentially hosted by wehrlite and dunite and serpentinized equivalents. Massive, semi-massive and sulphide matrix breccias have been noted in several surface showings and over restricted intervals in drill core within the Horsetrail zone.

Sulphide mineralization encountered in drill holes consists of between 1% and 5% disseminated blebs which locally coalesce to form net-textured sulphides. Most of the holes drilled contain nickel values of ≥0.10% over entire hole lengths, within which are intervals of 10 to several hundred metres of enhanced nickel values of greater than 0.20%. Nickel is the principal commodity of interest; copper and cobalt values average 0.05% and less than 0.02% respectively. Combined platinum and palladium values are generally less than 100 parts per billion.

Analytical studies indicate that between 60% and 90% of the nickel values are present in the form of sulphide minerals with the remainder occurring in crystal lattices of the silicate mineral olivine. Only the sulphide minerals can be considered recoverable into a saleable product. This is a critical issue for deposits of this type, because of the need to obtain accurate estimates of the nickel that can be recovered by established metallurgical methods. Therefore this study estimates resources, cut-off grades and mined grades in terms of nickel sulphide. The nickel sulphide determination has been performed by an ammonium-citrate–hydrogen-peroxide analysis (AC-Ni).

The ammonium citrate hydrogen peroxide is a partial extraction method believed to be selective at dissolving nickel from sulphide mineral species while leaving nickel in silicates undissolved. Partial digestions have poorer precision than total digestions, and poorer precision of assay results can conceal bias. In this case, based upon comparison with metallurgical testwork, the uncertainties of the partial digestion for estimating sulphide nickel are less than those obtained using a combination of rock type and total nickel and therefore the partial digestion approach provides a more reliable estimate of "recoverable" nickel for resource modelling purposes.

The AC-Ni analysis method could partially leach nickel from some silicate minerals which would result in an overestimation of recoverable nickel from low sulphide bearing rock types such as dunite. Careful petrographic work is recommended to establish whether any silicate nickel is being reported as part of the AC-Ni analysis. As a precautionary step the resources are reported in terms of NiS, a calculated value determined according to the following formulae:

- NiS = AC-Ni if Sulphur ≥0.2%
- NiS = 0 if Sulphur <0.2%

This precaution may cause an underestimation of the grade of the nickel resource if the sulphur assays have a low bias and if silicate nickel is not leached. The underestimation could be large enough to materially impact the resource estimation grade, but limits the possibility that an overestimation of the nickel resource grade will occur.

Total Digestion nickel (TD-Ni), AC-Co, Mg, and S were also interpolated in the resource model.

HCNC have used CANMET standard reference materials to support their AC-Ni nickel analyses. The CANMET certification is from 1974 and was not completed through a multi-lab round robin program. HCNC have initiated a new multi-lab round robin for the CANMET standards and for standards prepared with core from the property. Results for this round robin program are pending. The results of this round robin program should be examined to determine the affect on the confidence of the AC-Ni analyses.

The 2007 mineral resource estimate as at 25 September 2007 is based on a revised and expanded geologic interpretation of the Horsetrail zone and peripheral area. A significant difference between this estimate and previous estimates is the use of geological domains to constrain grade interpolation in the model. Previous models, which used grade shells without the benefit of a lithologic model, showed conditional bias and local excessive smoothing. The use of lithologic domains in this model has decreased both conditional bias and smoothing relative to previous models.

Modelling consisted of grade interpolation by Ordinary Kriging (OK) in three zones. The Horsetrail and Northwest zones were interpolated as one zone and the Duffy and the Hatzl zones were treated as separate zones. The search strategy employed ellipsoids with long axes increasing from 80 m to 200 m to 400 m. A minimum of 3 composites from at least 2 drill holes was required to interpolate a block grade.

Using a 0.10% NiS cut-off, the Turnagain Nickel deposit has an estimated open-pit resource as at 25 September 2007 of 489 Mt of Measured and Indicated Resources at 0.163% NiS and an additional 560 Mt of Inferred Resources at 0.152% NiS (Table 1-1). The resource tonnage is extremely sensitive to the cut-off grade. If the cut-off is increased from 0.10% to 0.16% NiS, the Measured and Indicated tonnage decreases by 50% and the contained metal decreases by 40%. These resources are based on the assumption of concentrate production on site with off-site smelting and refining.

In accordance with CIM best practice guidelines, namely the reporting of mineral resources which have a reasonable prospect of economic extraction, the classified resources of this model were constrained in a Lerchs-Grossman optimized pit using Whittle software. The parameters considered when constraining the resources are listed in Table 1-2.

This resource estimate was used in a PA of the Turnagain nickel deposit. The PA is preliminary in nature, includes Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PA will be realized. The results depend on inputs that are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here.

The PA addressed initial pit resources and design, evaluated two processing methods, reviewed tailings and waste considerations, reviewed ancillary and infrastructure requirements, and proposed a project execution plan. Through this work it was concluded that hydrometallurgical processing of concentrate on site was more economically viable than the saleable concentrate option.

Table 1-1: Resource Estimate, Effective Date 25 September 2007

NiS Cut-off (%)	Tonnage (kt)	NIS (%)	AC_NI (%)	NI_TOT (%)	AC_CO (%)
Measured					
>0.20	20,709	0.252	0.258	0.300	0.014
>0.18	27,056	0.237	0.244	0.287	0.013
>0.16	34,858	0.222	0.230	0.274	0.013
>0.14	43,417	0.208	0.218	0.263	0.012
>0.12	52,099	0.195	0.207	0.255	0.012
>0.10	59,464	0.184	0.199	0.250	0.011
Indicated					
>0.20	68,714	0.229	0.237	0.274	0.014
>0.18	127,918	0.211	0.220	0.256	0.013
>0.16	196,928	0.195	0.206	0.244	0.013
>0.14	277,330	0.181	0.194	0.233	0.012
>0.12	361,855	0.169	0.183	0.224	0.012
>0.10	429,688	0.160	0.175	0.218	0.012
Total Measured & Indicated					
>0.20	89,423	0.234	0.242	0.280	0.014
>0.18	154,974	0.216	0.224	0.261	0.013
>0.16	231,786	0.199	0.209	0.248	0.013
>0.14	320,747	0.184	0.197	0.237	0.012
>0.12	413,954	0.172	0.186	0.228	0.012
>0.10	489,152	0.163	0.178	0.222	0.012
Inferred					
>0.20	36,351	0.223	0.225	0.253	0.014
>0.18	116,654	0.200	0.203	0.238	0.013
>0.16	220,231	0.186	0.190	0.228	0.012
>0.14	329,470	0.174	0.181	0.219	0.012
>0.12	448,321	0.162	0.172	0.210	0.011
>0.10	560,052	0.152	0.164	0.204	0.011

Note: Mineral resources are not mineral reserves and do not have demonstrated economic viability

Table 1-2: Parameter Assumptions for Lerchs-Grossman Optimized Pit Constraining Resources As Reported in Table 1-1

Items	Assumptions
Mining Method	Best case open-pit
Mining Cost	CDN$1.40/t
Recoveries	Variable (average 75%)
Process Method	On-site concentrate; off-site smelting and refining
Process Cost and General and Administrative Costs	CDN$5.08/tonne processed
Nickel Metal Price	US$8.25/lb
Treatment Costs/Refining Costs	US$3.12/lb

The contained tonnes and grades resulting from the assessment assuming the hydrometallurgical option are 517 Mt at 0.160% NiS or 0.218% TD-Ni (Table 1-3). This resource was constrained in a Lerchs-Grossman optimized pit with a 100m setback from the Turnagain River, and includes 170 Mt of material in the Inferred category. The parameters considered when constraining the resources are listed in Table 1-4.

Table 1-3: Tonnes and Grades Contained in Ultimate Pit Shell

NiS >0.10% , undiluted in pit resources in ultimate pit shell	Mill Feed (kt)	AC_NI (%)	NIS (%)	NI_TOT (%)	AC_CO (%)
Measured	56,611	0.203	0.187	0.252	0.012
Indicated	290,871	0.176	0.164	0.222	0.010
Measured + Indicated	347,482	0.181	0.167	0.227	0.010
Inferred	169,941	0.155	0.145	0.199	0.010

Note: Mineral resources are not mineral reserves and do not have demonstrated economic viability

Table 1-4: Inputs to L-G Shells

Items	Unit		Assumptions
Mining Cost	CDN$/t mined	1.40	
Process + G&A Costs	CDN$/t milled	7.94	
Ni Price	CDN$/lb	7.89	based on US$7.50/lb
Selling Cost	CDN$/lb	0.00	all transport costs included in G&A
Exchange Rate	US$/CDN$	0.95	
Nickel Recovery	Average %	65	

The Turnagain mineralization is planned to be processed through an on-site concentrator and hydrometallurgical process facility that will produce nickel, cobalt and copper precipitation products. The nominal milling rate will be 50,000 t/d. The process plan is as follows. Run-of-mine (ROM) open pit mineralization will be crushed in a gyratory crusher. The crushed mineralization will be processed by means of a fine crushing circuit in combination with ball mill grinding, followed by rougher flotation, conventional cleaning, regrind, column cleaner flotation, and dewatering, to produce a nickel-cobalt-copper concentrate to be stored in holding tanks. The concentrate will be finely reground and then pressure oxidized in an autoclave. Leaching of the copper, cobalt and nickel will follow with precipitation of copper, cobalt and nickel occurring sequentially.

Three products are planned to be produced; copper sulphide, nickel hydroxide, and cobalt hydroxide. These products are expected to be trucked to the Port of Stewart for shipment to smelters. The truck haulage of concentrate is assumed to be contracted out. Tailings from the process will be impounded in a tailings pond; water will be reclaimed from the tailings pond and re-used in the process.

Saleable product is expected to be paid for on the basis of 85% for nickel contained in nickel hydroxide and 80% for cobalt in cobalt hydroxide and copper in copper sulphide.

It should be noted that there are a variety of issues related to the production of saleable products by concentration and subsequent hydrometallurgical processing that represent a risk to the project and will have to be addressed in future work.

Mining optimization studies were performed, considering the proximity of the Turnagain River, on Measured, Indicated and Inferred Resources within the Main Zone, with two process method options in mind (saleable concentrate vs. hydrometallurgical process).

The hydrometallurgical process was considered to be the optimal process method, based on these studies. AMEC optimized the Main deposit with nickel sulphide and cobalt in order to quantify the additional potential when considering revenue from cobalt. When compared to using only NiS, the additional tonnage was approximately 20 Mt. As this is a relatively small tonnage, and only preliminary recovery information for cobalt was available, it was decided to limit all further optimizations to NiS only.

A preliminary overall pit slope angle, including ramps, of 45° was used in mine planning as the rock is expected to be generally competent. Two pits were designed, which join towards the surface and have several common benches. The bench advance is reasonable and varies from 4 to 12 per year, to be mined by two loading units and a spare wheel loader. The mine plan considers a minimum mining width of 50 m, a total mine life of 29 years, and a production rate of 35 Mt/a. Maximum haul distance from pit to plant was 1 km. The area between the crusher and plant location and the proposed tailings storage facility was Indicated for the waste dump and low-grade stockpile locations.

Pit 47 of the 91 pits run was considered to be the base case, with a revenue factor of 0.76. In order to assess the impact of using Inferred material in this PA, AMEC undertook an additional optimization process using the same set of parameters, allowing only material classified as Measured or Indicated to be considered as mill feed. The mill feed tonnage is about 40% less when not considering the Inferred material.

The results of the economic analysis that follow represent forward-looking information as defined under Canadian securities law. Forward-looking information in this analysis includes, but is not limited to, statements regarding future mining and mineral processing plans, rates and amounts of metal production, operating and capital costs, tax and royalty terms, smelter and refinery terms, the ability to finance the project, and metal price forecasts.

The total estimated capital cost to design and build the Turnagain Project described in this report is CDN$1,381 million. Total operating costs are estimated at CDN$9.43 per tonne milled. Sustaining capital for the project over 29 years is CDN$173.5 million. Costs are summarized in Table 1-5.

Table 1-5: Summary, Capital and Operating Costs

Area	Cost (CDN$M)
Capital Cost Type	
Direct Costs	
Mining	116.3
Site Development	66.2
Main Process Facilities	390.9
Hydromet Plant & Reagent Services	106.0
Ancillary Buildings & Facilities	66.7
Tailings Facility	44.0
Total Direct Costs	792.3
Indirect Costs	
Owner's Cost	79.2
Construction Indirects	64.7
Engineering, Procurement & Construction Management	95.1
Construction Camp & Catering	32.4
Capital Spares	22.0
Freight	38.5
Start-up & Commissioning Allowance	6.5
Total Indirect Costs	338.4
Contingency	249.9
Total Capital Cost Estimate	**1,380.6**
Area	**Cost (CDN$/t)**
Operating Cost Type	
General & Administration (G&A)	
G&A Labour	0.14
Direct	0.29
Total G&A	0.43
Mining	1.40
Processing	
Process Labour	0.59
Consumables	5.59
Power	1.37
Miscellaneous	0.06
Total Process	7.60
Total Operating Cost Estimate	**9.43**

The financial base case assumed metal prices of US$7.50/lb for Ni, US$11.00/lb for Co, US$1.40/lb for Cu, and an exchange rate of US$0.95 = CDN$1.00. The base case assumed a net smelter return (NSR) of CDN$18.52/t and operating costs of CDN$9.43/t, for a margin of CDN$5.47/t. Using a 100% equity basis, the internal rate of return (IRR) was estimated at 12.2%, and payback in 6.4 years. A summary of the base case is included as Table 1-6.

Table 1-6: Base Case

Item		
Base Case Assumptions	Nickel Price	US $ 7.50/lb
	Cobalt Price	US $11.00/lb
	Copper Price	US $ 1.40/lb
Resources at 0.10% NiS Cut-Off	Measured and Indicated	489 Mt @ 0.163 % NiS and 0.012% Co
	Inferred	560 Mt @ 0.152 % NiS and 0.011% Co
Mining/Production	Strip Ratio	0.44:1
	Annual Throughput	18 Mt
	Daily Production Rate	50,000 t
	Overall Mineralization Milled	516.6Mt @ 0.160 % NiS and 0.011% Co
	Metallurgical Recoveries	73.6% Ni Overall Recovery
		66.5% Co Overall Recovery
	Average Annual Production	20,397 t Ni in nickel hydroxide
		1,301 t Co in cobalt hydroxide
Capital Cost		CDN$1.38
Operating Cost		CDN$9.43
Life of mine		29 years
Payback		6.4 years

Cumulative cash flow was estimated at CDN$2,828 million. The base case net present value (NPV) ranged from CDN$954 million at NPV 5% discount to CDN$-172 million at NPV 15% discount. NPV was also estimated for a range of cases at different metal prices (Table 1-7).

Table 1-7: NPV Range

Parameter	BaseCase	2	3	4	5	6	7	8	9
Commodity Prices (US$/lb)									
Nickel Prices	7.50	5.25	6.00	6.75	8.25	9.00	9.75	12.00	14.00
Cobalt Prices	11.00	7.70	8.80	9.90	12.10	13.20	14.30	34.00	14.00
Copper Prices	1.40	0.98	1.12	1.26	1.54	1.68	1.82	3.15	3.00
NPV									
%IRR	12.2	N.A.	4.8	8.8	15.3	18.3	21.1	31.8	34.9
Cumulative net cash flow ($M)	2,828	(80)	889	1,859	3,797	4,767	5,736	9,905	10,819
5% discount (CDN$M)	954	(511)	(22)	466	1,443	1,931	2,419	4,484	4,991
8% discount (CDN$M)	422	(629)	(279)	72	773	1,123	1,474	2,945	3,324
10% discount (CDN$M)	187	(679)	(390)	(102)	476	764	1,053	2,258	2,578
12% discount (CDN$M)	13	(713)	(471)	(229)	255	497	738	1,745	2,018
15% discount (CDN$M)	(172)	(744)	(553)	(362)	19	210	401	1,191	1,412

Some of the key technical risks include, but are not limited to:

- changes in government and changes in regulations affecting the ability to permit and operate a mining operation; discussions with stakeholders that may be impacted by any proposed mining operation are at a very early stage.
- the fact that mineral resources are estimates based on limited sampling data, interpretation of geology and assumptions applied that may change with increased exploration, development and mining.
- proximity to the Turnagain River, which may require a deeper setback than 100 m, negatively impacting resources.
- actual mining and metallurgical recoveries that may be achieved; there are a variety of issues related to the production of saleable products by concentration and subsequent hydrometallurgical processing that represent a risk to the project and will have to be addressed in future work.
- variations in operating and capital costs
- provision of power
- future metal prices may change from those used in the economic model; the project is very sensitive to metal prices.

A three-discipline approach, involving geology, mining and metallurgy is recommended for future work on the Turnagain project. The programs total CDN$1.08 million and are estimated to take about 6 to 9 months to complete. Once the data from these programs are available, have been evaluated, and indicate that additional work is warranted, HCNC should consider completing a pre-feasibility level study on the project.

2.0 INTRODUCTION

Hard Creek Nickel Corporation (HCNC) requested that AMEC Americas Limited (AMEC) provide an independent preliminary assessment (PA) on the Turnagain Project (the Project), located in northern British Columbia (BC).

The current work by AMEC entailed the preparation of resource estimate and a PA culminating in the preparation of a Technical Report as defined in National Instrument (NI) 43-101 and in compliance with Form 43-101F1 (the "Technical Report").

The resource estimate reported within this technical report includes results of the 2007 drill program which were available as of 1 October 2007 and constrains grade estimation by lithology as compared to the previous resource estimates constrained by grade shells (Simpson, 2006; 2007). The mineral resource estimate was prepared by Mr. Guillermo Pareja, an AMEC senior geostatistician and resource estimator, under the direction of Mr. Greg Kulla, P. Geo, Principal Geologist, AMEC, in conformance with the CIM Mineral Resource and Mineral Reserve definitions referred to in NI 43-101, Standards of Disclosure for Mineral Projects.

The Qualified Persons responsible for the preparation of this technical report on the PA are:

- Greg Kulla, P.Geo., Principal Geologist, AMEC Vancouver office
- Gerrit Vos, P.Eng., Principal Mining Engineer, AMEC Vancouver office
- Ignacy (Tony) Lipiec, P.Eng., Senior Process Engineer, AMEC Vancouver office.

2.1 Terms of Reference

AMEC is independent of HCNC and the Turnagain Property as the term "independence" is defined in NI 43-101.

In preparing this report, AMEC relied on geological maps, reports and miscellaneous technical data listed in the References section at the conclusion of this report.

A Technical Report was previously prepared by AMEC in June 2006 (AMEC, 2006) to present the results of a PA on the Turnagain deposits. This report is on file on the SEDAR website (www.sedar.com) and is entitled:

> *AMEC, 2006: Turnagain Project, Preliminary Assessment of Turnagain Property: Technical Report prepared for Hard Creek Nickel Corporation by AMEC Americas Ltd., effective date June 1, 2006.*

A portion of the background information and technical data for AMEC's current review of the Project were obtained from the aforementioned report. Additional background information and technical data was sourced from two technical reports filed by HCNC (Simpson, 2006 and Simpson, 2007), also on file on the SEDAR website, and entitled:

Simpson, R., 2006: Technical Report and Mineral Resource Estimate, Turnagain Nickel Project, Turnagain River Area, Liard Mining Division, British Columbia: Technical Report prepared for Hard Creek Nickel Corporation, effective date April 13, 2006.

Simpson, R., 2007: Mineral Resource Update, Turnagain Nickel Project, Turnagain River Area, Liard Mining Division, British Columbia: Technical Report prepared for Hard Creek Nickel Corporation, effective date March 29, 2007.

and from background information and technical data provided by HCNC.

Mr. Kulla completed a site visit to the Turnagain Project on 18 to 22 June 2007. The purpose of the visit was to review the geology and mineralization encountered on surface and in the drill holes completed to date related to the Northwest, Horsetrail, Duffy and Hatzl zones. Several peripheral nickel and PGE showings were not examined at during the visit. Drilling, sampling, quality assurance/quality control (QA/QC), sample preparation and analytical protocols and procedures, and database structure were briefly reviewed. AMEC's review has used CIM "Mineral Exploration Best Practice Guidelines" as a guideline to follow. During the site visit half-core samples from the Northwest, Horsetrail, and Hatzl zones were collected by AMEC and submitted to Acme Analytical Laboratories, in Vancouver, for preparation and analysis. The main intent of analysing these samples was to confirm the general range of nickel grades reported in previous exploration on the Project. Additional discussion, analytical methods and analytical results of these samples is provided in Section 14 of this Report. A site visit was also undertaken by Mr. Kulla in October 2005.

Mr. Vos visited the site from 18 to 20 June 2007. During this visit, Mr. Vos inspected the planned mine site, the potential waste dump and stockpile area and potential plant site. Core samples were also inspected to assess fracturing within the core for potential effects on slope stability and blasting.

Mr Lipiec visited the site from 13 to 15 September 2006. During this visit, drill core was inspected. In addition, a visit was paid to potential mill sites, tailings and waste disposal sites, and a flyover from the main highway to the exploration camp was undertaken.

The Effective Date of the Technical Report is 25 September 2007, which represents the cut-off date for the analytical data used in the preparation of this report.

All measurement units used in this report are metric, and currency is expressed in Canadian dollars unless stated otherwise.

3.0 RELIANCE ON OTHER EXPERTS

3.1 Introduction

The AMEC Qualified Persons (QPs), authors of this Technical Report, state that they are qualified persons for those areas as identified in the "Certificate of Qualified Person" attached to this report. The authors have relied, and believe there is a reasonable basis for this reliance, upon the following reports, which provided information regarding mineral rights, surface rights, permitting, and environmental issues in sections of this Technical Report as noted below.

3.2 Mineral Tenure

AMEC QPs have not reviewed the mineral tenure, nor independently verified the legal status or ownership of the Project area or underlying property agreements. AMEC has relied upon HCNC experts for this information through the following document.

- Fasken, Martineau, Doumoulin, LLP, 2007: Hard Creek Nickel Corporation – Turnagain Nickel Project – Title Opinion: letter to Hard Creek Nickel Corporation dated 9 November 2007 (Sections 4.1 and 4.2 of this report)
- Froc, N., 2007: Claim Status Info: email to B. Lee, AMEC, dated 13 November 2007 (Sections 4.1 and 4.2 of this report).
- Froc, N., 2007: Turnagain Project Mineral Claim Tenure List: Excel spreadsheet supplied to B. Lee, AMEC, dated 13 November 2007 (Sections 4.1 and 4.2 of this report)
- Froc, N., 2007: Amended Mines Act Permit MX-1-205: Copy of letter from Jill Pardoe, BC Inspector of Mines, dated 11 June 2007 (Sections 4.1 and 4.2 of this report).

3.3 Surface Rights, Access and Permitting

AMEC QPs have relied on information regarding Surface Rights, Access and Permits, including the status of the granting of surface rights by the Province of British Columbia for land designated for future potential mining, milling, dumps and tailings impoundments and have relied upon opinions and data supplied by HCNC representatives as follows:

- Froc, N., 2007: Claim Status Info: email to B. Lee, AMEC, dated 13 November 2007 (Section 4.2 of this report).

3.4 Environmental and Socio-Economics

AMEC QPs have relied upon the environmental status for the Project through the opinions of experts retained by HCNC, through the following document:

- Froc, N., 2007: Claim Status Info: email to B. Lee, AMEC, dated 13 November 2007 (Section 4.2 of this report)

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Turnagain Project is located in British Columbia, Canada, about 1,350 km north-northwest of Vancouver and 70 km east of the township of Dease Lake (Figure 4-1). The Project is centred at approximately latitude and longitude 58°30'N, 128°45'E or UTM NAD83 Zone 9 coordinates 6,481,000 N and 508,000 E. Elevations within the Project area range between 1,000 and 1,800 masl.

4.2 Property Description

The Turnagain nickel property consists of two legacy 4-post mineral claims, twenty-seven 2-post claims and fifty-two cell mineral claims, all of which are contiguous and are situated in the Liard Mining Division on NTS map sheets 104I/06E, 07E&W, 10W, 11E.

The mineral claims collectively cover an area of approximately 29,370 ha (293 km^2); see Table 4-1 and Figure 4-2.

Staking History

Mineral claims that were located in 1996 by J. Schussler and E. Hatzl were subsequently optioned to Bren-Mar Resources Limited, a predecessor company of Canadian Metals Exploration Limited and HCNC. The original option agreement gave Bren-Mar Resources the right to earn a 100% interest in the mineral claims in exchange for the issuance of 200,000 shares and incurring property expenditures of CDN$1 million within five years of acquisition. The 100% interest has been earned subject to a 4% NSR on possible future production from the mineral claim 511330. HCNC retains the right to purchase all or part of this royalty for CDN$1 million per each 1% of the royalty.

On 28 November 2002, HCNC entered into an agreement with John Schussler and Ernie Hatzl to acquire an additional 34 mineral claims, adjacent to the Turnagain Property, Liard Mining Division, British Columbia, in exchange for an aggregate total of 100,000 common shares.

Between November 2003 and March 2005, additional claims were staked, enlarging the Turnagain property from 3,700 ha to approximately 27,500 ha.

The 14 “Drift” and “Dinah” cell mineral claims, situated southeast of Turnagain River, were acquired early in 2005 by way of the BC Ministry of Energy and Mines “online” map selection process.

Figure 4-1: Project Location Plan

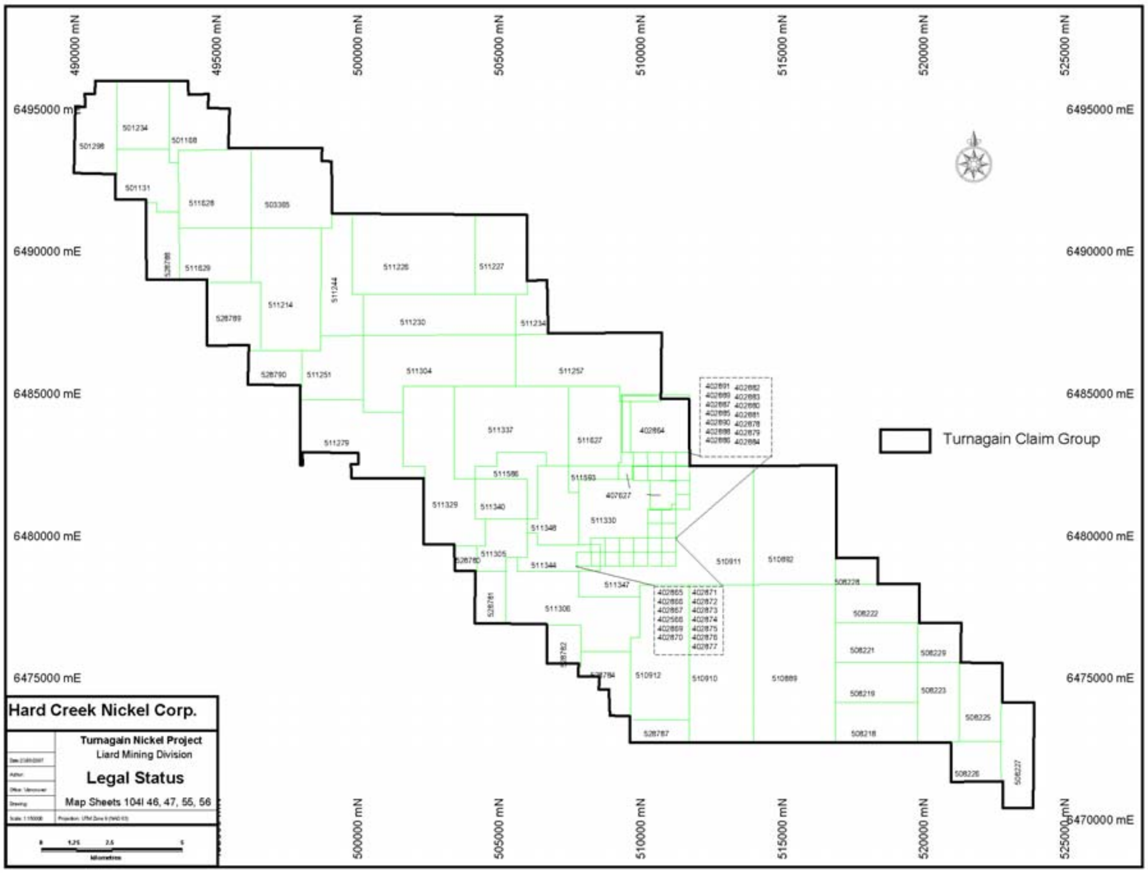
Note: Figure sourced from Hard Creek Nickel Corporation

Table 4-1: Claims Details

Tenure No.	Tenure Type	Claim Name Recorded	Expiry Date
402864	Mineral	BEAR 1	26/05/2017
402865	Mineral	BEAR 2	26/05/2017
402866	Mineral	BEAR 3	26/05/2017
402867	Mineral	BEAR 4	26/05/2017
402868	Mineral	BEAR 5	26/05/2017
402869	Mineral	BEAR 6	26/05/2017
402870	Mineral	BEAR 7	26/05/2017
402871	Mineral	BEAR 8	26/05/2017
402872	Mineral	BEAR 9	26/05/2017
402873	Mineral	BEAR 10	26/05/2017
402874	Mineral	BEAR 11	26/05/2017
402875	Mineral	BEAR 12	26/05/2017
402876	Mineral	BEAR 13	26/05/2017
402877	Mineral	BEAR 14	26/05/2017
402878	Mineral	BEAR 15	26/05/2017
402879	Mineral	BEAR 16	26/05/2017
402880	Mineral	BEAR 17	26/05/2017
402881	Mineral	BEAR 18	26/05/2017
402882	Mineral	BEAR 19	26/05/2017
402883	Mineral	BEAR 20	26/05/2017
402884	Mineral	BEAR 21	26/05/2017
402885	Mineral	BEAR 22	26/05/2017
402886	Mineral	BEAR 23	26/05/2017
402887	Mineral	BEAR 24	26/05/2017
402888	Mineral	BEAR 25	26/05/2017
402889	Mineral	BEAR 26	26/05/2017
402890	Mineral	BEAR 27	26/05/2017
402891	Mineral	BEAR 28	26/05/2017
407627	Mineral	PUP 4	01/01/2017
501131	Mineral	Drift 1	12/01/2013
501168	Mineral	Drift 2	12/01/2013
501234	Mineral	Drift 3	12/01/2013
501298	Mineral	Drift 4	12/01/2013
503365	Mineral	-	18/02/2013
508218	Mineral	Dinah 1	03/03/2010
508219	Mineral	Dinah 2	03/03/2010
508221	Mineral	Dinah 3	03/03/2010
508222	Mineral	Dinah 4	03/03/2010
508223	Mineral	Dinah 5	03/03/2010
508225	Mineral	Dinah 6	03/03/2010
508226	Mineral	Dinah 7	03/03/2010
508227	Mineral	Dinah 8	03/03/2010

Tenure No.	Tenure Type	Claim Name Recorded	Expiry Date
508228	Mineral	Dinah 9	03/03/2010
508229	Mineral	Dinah 10	03/03/2010
510889	Mineral	-	07/04/2011
510892	Mineral	-	03/03/2010
510910	Mineral	-	07/04/2012
510911	Mineral	-	07/04/2012
510912	Mineral	-	07/04/2012
511214	Mineral	-	18/02/2012
511226	Mineral	-	18/02/2010
511227	Mineral	-	17/02/2009
511230	Mineral	-	17/02/2010
511234	Mineral	-	16/02/2009
511244	Mineral	-	18/02/2012
511251	Mineral	-	17/02/2012
511257	Mineral	-	17/02/2011
511279	Mineral	-	17/02/2012
511304	Mineral	-	17/02/2012
511305	Mineral	-	27/09/2012
511306	Mineral	-	19/02/2014
511329	Mineral	-	27/09/2012
511330	Mineral	-	01/12/2016
511337	Mineral	-	01/12/2016
511340	Mineral	-	01/12/2016
511344	Mineral	-	19/02/2017
511347	Mineral	-	07/04/2015
511348	Mineral	-	01/12/2016
511586	Mineral	-	01/01/2017
511593	Mineral	-	01/01/2017
511627	Mineral	-	01/12/2016
511628	Mineral	-	18/02/2012
511629	Mineral	-	18/02/2012
528780	Mineral	T1	23/02/2012
528781	Mineral	T2	23/02/2012
528782	Mineral	T3	23/02/2012
528784	Mineral	T4	23/02/2012
528787	Mineral	T5	23/02/2012
528788	Mineral	T6	23/02/2012
528789	Mineral	T7	23/02/2012
528790	Mineral	T8	23/02/2012

Figure 4-2: Tenure Plan



Note: Figure from Hard Creek Nickel Corporation

Twenty-nine of the original 4-post mineral claims (now termed legacy claims) northwest of Turnagain River were converted to cell mineral claims in April 2006. This conversion process ensured greater security of mineral title by effectively eliminating the possibility of internal and external fractions within or adjacent to the various mineral claims. Accumulated assessment work credits were also retained under the conversion system.

One 4-post claim and twenty-seven 2-post claims, located adjacent to and partially within the central part of the property holdings but outside of the prospective ultramafic rocks, were the subject of a legal dispute between HCNC and Mr. Weise. On 10 July 2006 the Supreme Court of British Columbia ordered that these claims be transferred to HCNC. The transfer has been completed and the claims have been included in the Turnagain property. Mr. Weise subsequently filed a Notice of Appeal of the Order that was dismissed by the British Columbia Court of Appeal on 30 April 2007.

Mineral Title in British Columbia

Mineral claims in BC are of two types. Cell mineral claims are established by electronically selecting the desired land on government claim maps, where the available land is displayed as a grid pattern of open cells, each of approximately 450 to 500 ha. Payment of the required recording fees is also conducted electronically. This process for claim staking has been in effect since January 2005, and is now the only way to stake claims in BC. Prior to January 2005, claims were staked by walking the perimeter of the desired ground and erecting and marking posts at prescribed intervals. Claims staked before January 2005, remain valid and may be converted into cell claims.

Cell mineral claims may be kept in good standing by incurring assessment work or by paying cash-in-lieu of assessment work in the amount of CDN$4.00/ha/a during the first 3 years following the location of the mineral claims. This amount is increased to CDN$8.00/ha in the fourth and succeeding years. Claims staked before January 2005 may be kept in good standing by incurring assessment work or by paying cash-in-lieu of assessment work in the amount of CDN$100 per mineral claim unit per year during the first 3 years following the location of the mineral claim. This amount increases to CDN$200 per mineral claim unit in the fourth and succeeding years.

All claims within the Turnagain Project are in good standing with first required assessment work or “cash in lieu payment” on any claim due 16 February 2009. Assessment work required on any tenure is CDN$8/ha with an attached government filing fee of 5%. Work can be applied forward for up to 10 years.

4.3 Surface Rights

Exploration work on mineral properties in BC requires the filing of a Notice of Work and Reclamation with the Ministry of Energy and Mines. The issuance of a permit facilitating such work may involve the posting of a reclamation bond.

Securities to the value of CDN$187,900 were held against the 2007 work program. The work program for 2007 was granted under licence MX-1-505.

4.4 Surveying and Permits

The Project is situated within Crown Land in BC. AMEC is not aware of any other permits that are in place, or may have been applied for to advance the Project or whether there are any outstanding issues with permits that may be held by other parties that may affect future activities on the Project.

4.5 Environmental and Socio-Economic Issues

Environmental studies within the property area have been ongoing since 2003. These include hydrological measurements on tributary creeks and the collection of surface and ground water quality data, wildlife observations and determination of fish species and the maintenance of weather data. Multi-element analyses of soil samples have provided useful information regarding background concentrations of major and trace elements.

4.5.1 Current Studies

Collection and analysis of hydrological and meteorological site data is ongoing. Four continuous stream gauging stations and an automated meteorological station are being calibrated and downloaded at regular intervals. Fisheries studies have been started at the site, with a focus on the Hard Creek Valley.

AMEC notes that, to date, discussions with stakeholders that may be impacted by any proposed mining operation are at a very early stage. Such discussions will be necessary as the Project is advanced, in order to assess any socio-economic impacts.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

The nearest airport to the Project is at Dease Lake, 70 km to the west of the Project. Dease Lake has scheduled airline service by Northern Thunderbird Air three times a week in the summer and twice weekly in the winter.

A 700 m dirt airstrip immediately adjacent the HCNC exploration camp, constructed by Falconbridge in 1967, was extended to 900 m by HCNC in 2007. Exploration in 2007 was supported by a Sustut Air Inc. Cessna Grand Caravan via Smithers and by Pacific Western helicopters via Dease Lake.

Secondary roads extending easterly from Dease Lake have been used by large, articulated 4-wheel drive vehicles to convey equipment to the Turnagain property. This road is not suitable for regular vehicle traffic.

5.2 Climate

The Turnagain Property is located in the Stikine Ranges of the Cassiar Mountains. The Cassiar Ranges have a northern continental climate. Winters are long and cold; summers are cool and short. Rainfall and snow depths are moderate. The ground freezes deeply for a large part of the year and discontinuous permafrost can occur on some northern slopes and in peat lands.

Dease Lake, the closest urban centre, experiences winter climates from mid-October to April. Winter temperatures generally average -25°C with a three to five week period where temperatures can fall to approximately -45°C. Most snowfall occurs during November to February. Throughout an average winter, approximately 220 cm of snow will fall with approximately 100 cm on the ground during mid-winter. Summer temperatures are moderate with average temperatures ranging from 13°C to 22°C from May to September. Average rainfall during this period ranges from 25 to 60 mm per month.

5.3 Local Resources

An exploration camp built on the property in April 2003 is capable of accommodating approximately 30 people, and consists of 17 wall tents, 3 trailers and drill core storage facilities. Power is provided by an on-site diesel generator.

On-site communications include satellite telephone, facsimile and Internet connections.

There are approximately 32 km of unpaved roads and trails on the property, constructed from the late 1960s to the present.

5.4 Infrastructure

Dease Lake (population 650) offers some supplies and services. The communities of Terrace (population 12,000) and Smithers (population 5,500), 700 and 500 km to the south respectively, offer the best range of supplies and services which can be trucked to Dease Lake via Highway 37. The closest deep water port is the bulk terminal at Stewart. There is no rail link within the Cassiar district, although there is a rail bed between Dease Lake and Takla Landing to the south.

At present, the Cassiar district is not serviced by the provincial electricity grid. The 3MW Hluey Lakes Hydro Project produces electricity for Dease Lake. There is a region of high geothermal energy potential extending southward from Mount Edziza Park which has not been extensively explored. Coal deposits at Coal River and Tuya River north of Telegraph Creek contain low-grade thermal coal, which have potential to generate electricity. There is a significant deposit of coal in the Cassiar district at Mount Klappen east of Bob Quinn along the Spatsizi Park boundary.

5.5 Physiography

The data on Physiography of the Stikine region is taken from the Integrated Land Management Bureau (2007).

Between Dease Lake and the Property, topography comprises mountains and wide river valleys of the Stikine Ranges. Ridges, plateaus and summits lower than 1,800 m are rounded while higher summits are rugged. Valley bottoms are 1,000 to 1,350 m elevation while the highest peak, King Mountain, about 15 km south of the Turnagain property is 2,425 m in elevation. Plateau surfaces are at roughly 1,500 m.

The valley bottoms and lower elevation slopes are covered with glacial drift. Esker and drumlin formations are numerous and extensive. The ranges are characterised by the occurrence of flat-topped tuyas, steep-sided volcanoes, which were erupted on the plateau surface under the ice sheet during the Pleistocene glaciations.

Boreal white spruce and lodgepole pine forest occur on valley bottoms, where they are interspersed with wetlands. At higher elevations the boreal forest gives way to sub-alpine fir and scrub birch in open forests and woodlands. In areas of cold-air ponding, and in upper elevation exposed areas, the forest gives way to sub-alpine shrub and grassland and scrub vegetation. Above tree line, alpine shrub-land, heath and tundra occur. Bedrock is reasonably well exposed in the areas above tree line and along drainages.

Several species of large mammal, including grizzly bear, black bear, wolf, moose, caribou, mountain goat and sheep can be found in the Cassiar Mountains. Bird species noted in the mountains include gyrfalcon, golden eagle, willow ptarmigan, least sandpiper, red-necked phalarope, snow bunting and Smith's longspur.

The Turnagain Property straddles the Turnagain River near where it joins Hard Creek. The Turnagain property covers north, west, and east-facing slopes northwest and southeast of Turnagain River and alpine terrain above tree line. Elevations range from about 1,000 masl along Turnagain River in the central claims area to 1,800 m at an unnamed summit in the central property area.

6.0 HISTORY

The description of the property exploration history is based on work by Nixon (1998) and Baldys et al., (2006).

Nickel and copper sulphides were first recognized in rusty weathering exposures at the Discovery zone on the Turnagain River in about 1956. Falconbridge Nickel Mines acquired the property in 1966, and during the period 1966–1973 completed an airborne geophysical survey, ground geophysical surveys, geological mapping, geochemical surveys, and 28 wide-spaced diamond holes (2,895 m). The work identified a number of sulphide “showings”. The exploration program tested many of the mineralised outcrops by “packsack” drilling; the Discovery outcrop was not successfully drilled.

During the early 1970s, adjacent claims were investigated with a geochemical survey by Union Miniere Exploration and Mining Corporation Ltd. (UMEX). Once the Falconbridge and UMEX claims expired, a number of the showings were re-staked and tested with short, small diameter core holes by an unnamed party. Three EX-sized core holes, totalling 55.5 m, were drilled on the west bank of the Turnagain River in 1977. No significant intersections were reported and the collars have not been located. In 1979, a single drill hole (17 m) was drilled by S. Bridcut near the east bank of the Turnagain River and intersected unmineralized quartz diorite.

The commodity focus for exploration shifted to platinum group elements (PGEs) in the mid-1980s. A geochemical survey for PGEs was conducted for Equinox Resources Ltd. in 1986, and Bridcut re-sampled the Falconbridge core in 1988.

In 1996, Bren-Mar Resources Limited (Bren-Mar) optioned the Cub claims from J. Schussler and E. Hatzl. From 1996 to 1998, Bren-Mar completed an airborne magnetic survey over 45 km^2 (400 line-km of survey), 19 diamond drill holes (3,889 m), geological prospecting and sampling, and down-hole pulse electromagnetic surveys in 4 of the 1997 to 1998 drill holes and preliminary metallurgical testwork on drill core composite samples.

The company changed its name to Canadian Metals Exploration Limited, and resumed exploration in 2002 with an induced polarization (IP) and ground magnetic survey followed by 1,687 m of diamond drilling in seven holes. Drilling continued in 2003, with 23 holes (including deepening of 1 of the 2002 drill holes) completed for 8,769 m. Additional exploration included geological mapping, prospecting and bedrock, stream sediment and soil sampling.

In 2004 the company changed its name to Hard Creek Nickel Corporation and recommenced work on the property, and to the end of 2007, has completed geological mapping, bedrock, stream sediment and soil sampling, surface, borehole and airborne geophysical surveys, mineralogical, metallurgical and analytical studies, and 172 diamond drill holes for 41,502 m of drilling.

In 2006 Hard Creek reported a measured and indicated resource estimate inside a 0.2% sulphide nickel grade shell. Only the sulphide minerals were considered recoverable into a saleable product and therefore the 2006 resource estimate was reported in terms of sulphide nickel. Sulphide nickel was determined using ammonium citrate hydrogen peroxide partial extraction procedure. The estimate was completed by Geosim Service Inc. (Geosim) of Vancouver (Simpson, 2006).

Later in 2006, Hard Creek reported results of the first PA on the Project. A key assumption of the PA was that a 0.10% sulphide nickel analysis cut-off was economically reasonable for the Project. This cut-off was determined as a result of parameters selected for pit optimization. Resources in the PA were reported in terms of sulphide and total nickel.

In 2007 Hard Creek reported a new measured and indicated resource estimate in terms of sulphide and total nickel inside a 0.10% sulphide nickel grade shell. This estimate was completed by Geosim, and resulted in a significant increase in the tonnes of the deposit (Simpson, 2007).

Resource estimates reported in 2006 and March 2007 were constrained using sulphide nickel grade shells. The restriction on grade shells was appropriate given that no geological domains had been defined at that time.

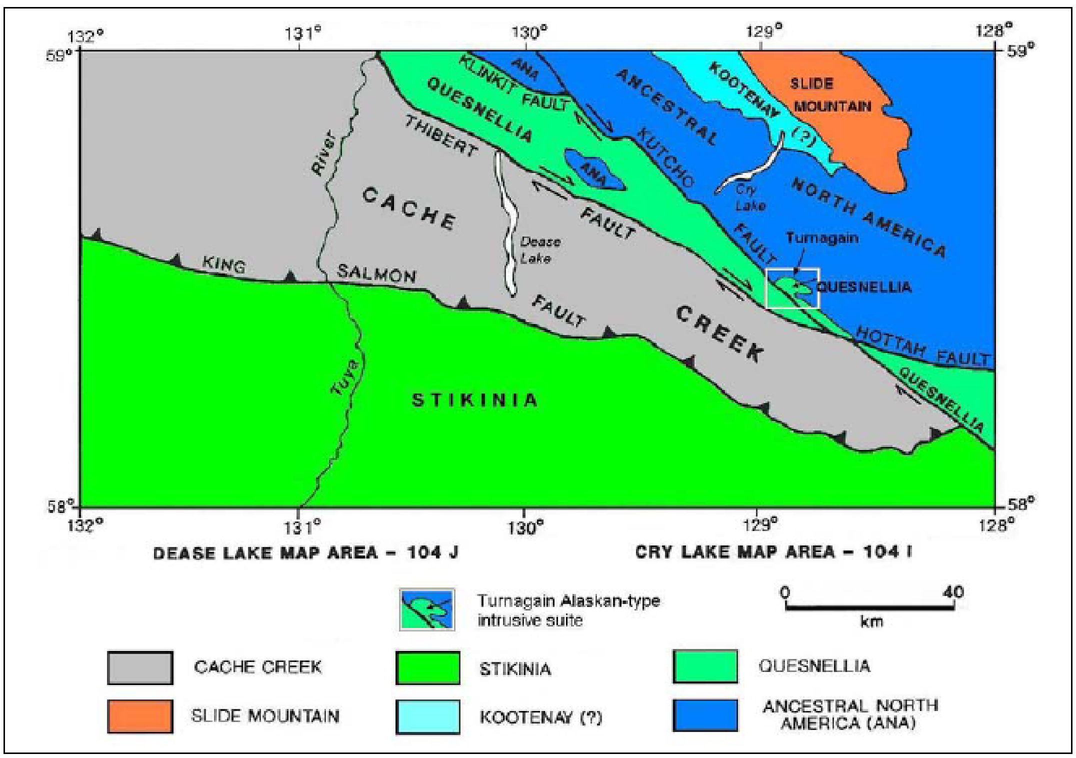
7.0 GEOLOGICAL SETTING

7.1 Regional Geology

The regional geology of the Turnagain Property has been previously described by Nixon (1997; 1998), Scheel et al, (2005), and in Technical Reports by Simpson (2006, 2007) and AMEC (2006). Geological understanding of the region and the setting of the deposit is continually being refined with additional information from drilling and exploration programs. The regional description presented in this report is based on work by Scheel (2007), Scheel et al., (2005) and Nixon (1998).

The Property is situated within the Turnagain ultramafic complex, which may be hosted within either the Yukon-Tanana terrane or the Quesnel terrane. The Turnagain complex is fault-bounded, has dimensions of about 3.5 x 8 km and lies to the north of two major fault systems, the Kutcho and Thibert–Hottah Faults (Figure 7-1) although neither of the fault system is exposed in the property.

Figure 7-1: Regional Structural Setting, Turnagain Property



Note: Figure from Scheel et al., 2005

The western, northern and eastern margins of the complex abut rocks attributed to the Lower Ordovician Road River Formation and the Mississippian Earn Group (Figures 7-2 and 7-3). The Road River and Earn Group rocks comprise graphitic phyllite, intercalated with lesser quartz-rich and calc-silicate tuff layers, which can be strongly pyritic and graphitic near the Turnagain complex. Metamorphism in the phyllites regionally reaches greenschist facies. No contact hornfelsing is mapped adjacent to the northern or eastern contacts with the Turnagain complex.

To the south of the Turnagain complex is a series of undated sedimentary, possibly volcaniclastic, rocks that may represent rocks of the Lay Range assemblage of the Quesnel terrane (Figures 7-2 and 7-3). On the south side of the Kutcho Fault, dioritic to granodioritic rocks of the early Jurassic Eaglehead Pluton crop out.

The regional setting and method of emplacement of the Turnagain complex is still being resolved. Gabrielse (1998) postulates that the Turnagain complex intrudes rocks of the miogeoclinal margin of ancestral North America, indicating that a supra-subduction setting was operational at the cratonic margin at the time of emplacement. An alternative view (Scheel et al., 2005 and Nixon 1998) places the Turnagain complex within an imbricated set of rocks that was thrust eastward onto the margin of the North American craton.

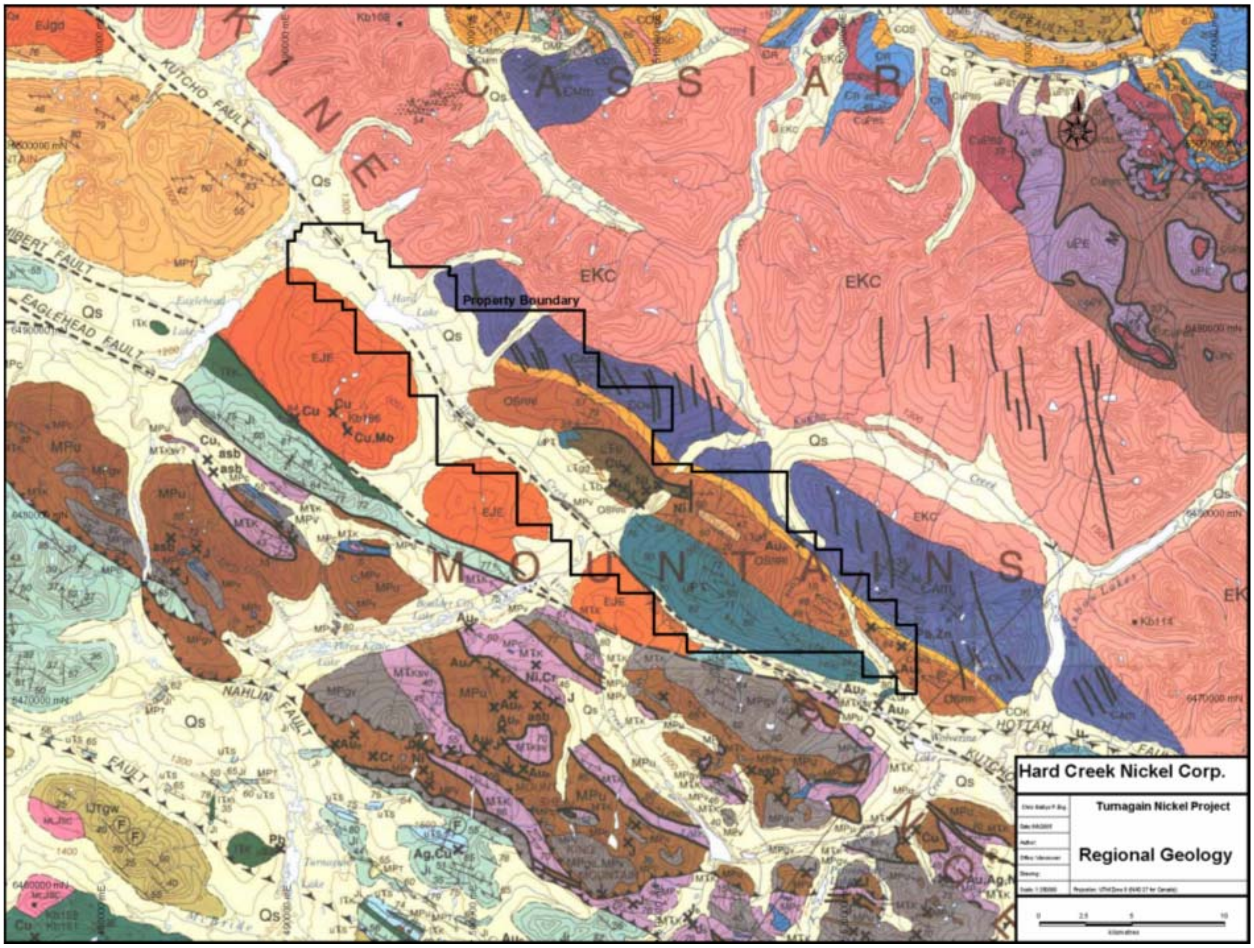
7.2 Property Geology

The Turnagain complex comprises a central core of dunite with bounding units of wehrlite, olivine clinopyroxenite, clinopyroxenite, representing crystal cumulate sequences, hornblende clinopyroxenite and hornblendite (Figure 7-4). No orthopyroxene is present. The complex is elongate and broadly conformable to the northwesterly-trending regional structural grain.

The ultramafic rocks are generally fresh to mildly serpentinized; however, more intense serpentinization and talc-carbonate alteration are common along faults and restricted zones within the complex. The central part of the ultramafic body is intruded by granodiorite to diorite, and hornblende–plagioclase porphyry dikes and sills.

Primary layering in clinopyroxene-rich cumulates, reflecting variations in the modal abundance of olivine and pyroxene, is visible in outcrop. The layering has moderate to steep dips and is truncated by the faulted eastern boundary of the complex. Despite localized zones of well developed layering, way up criteria are inconclusive and the internal structure of the Turnagain complex is poorly understood (Nixon, 1998).

Figure 7-2: Regional Geology, Turnagain Property

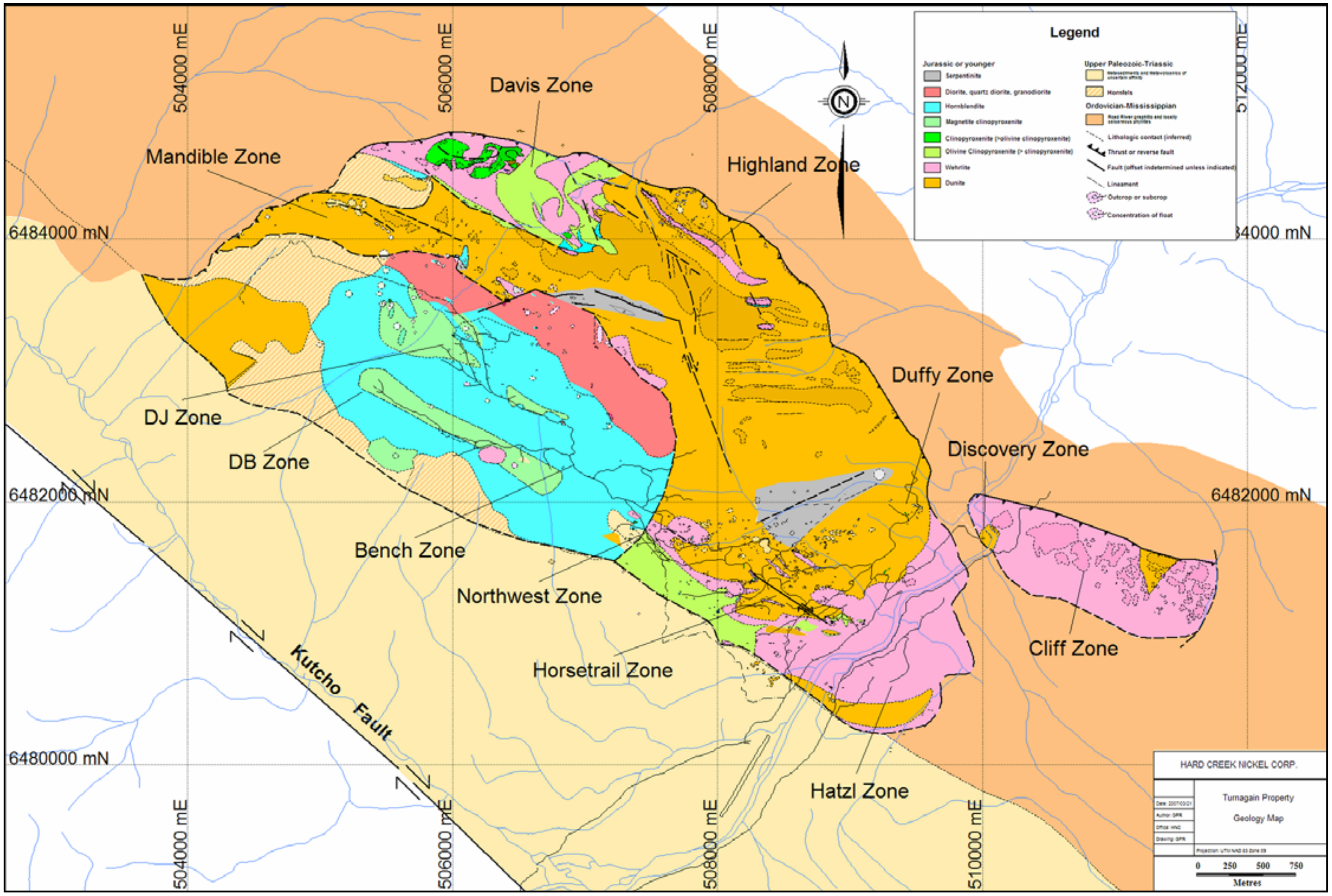


Note: Figure from Hard Creek Nickel Corporation

Figure 7-3: Geological Legend to Accompany Figure 7-2

Note: Figure from Hard Creek Nickel Corporation

Figure 7-4: Property Geology Plan



Note: Figure from Hard Creek Nickel Corporation

The following description of lithologies is modified from Scheel et al. (2005).

Dunite

Dunite is primarily found in the eastern and central portions of the complex. It is mainly composed of cumulus olivine, minor amounts of chromite and pyroxene, and trace amounts of primary phlogopite. Dunite commonly hosts grains of poikilitic green diopside, either as discrete, centimetre-scale crystals or elongate aggregations. The latter are interpreted to be small dikes resulting from the escape of trapped liquid.

Millimetre- to centimetre-scale layering in the dunite core is evident locally where concentrations of chromite crystals have accumulated. These chromitite horizons are discontinuous and commonly remobilized and intruded by thin dunite dikes.

Serpentinization volumes are highly variable but generally are no more than about 10% of the rock mass by volume. Serpentinized volumes are higher in the resource area. Large amounts of secondary magnetite are found where serpentinization is pervasive. Some dunite that is proximal to massive sulphide mineralization is commonly altered to grey tremolite.

Contacts between wehrlite and dunite are sharp to gradational over short distances, represented by a slight change in the size and modal abundance of pyroxene, and may reflect magmatic layering.

Wehrlite

Two different wehrlite types have been identified. On the west side of the Turnagain River, the wehrlite is mainly composed of cumulus olivine with a sizable proportion of intercumulus clinopyroxene and minor amounts of cumulus pyroxene. On the east side of the river, and in the far northwest of the intrusion, cumulus clinopyroxene reaches approximately 40% by volume of the rock mass.

Olivine Clinopyroxenite and Clinopyroxenite

These rock types occur in the northwestern part of the intrusion and commonly comprise around 85 volume percent cumulus pyroxene and smaller amounts of cumulus olivine. Depending on location within the complex, the clinopyroxenites can be either an original magma differentiate or an intrusive; in the northwestern portion of the complex, they appear to be related to the original magma, further to the east, they are brecciated and intrusive in nature. Pegmatitic clinopyroxenite dykes are commonly found adjacent to the cumulate clinopyroxenite.

Hornblende Clinopyroxenite and Clinopyroxenite

These rock types are restricted to an area co-incident with a Cu–Pt–Pd soil anomaly near the southwestern margin of the complex. They are very poorly exposed and their relationships to other units in the Turnagain complex are not well constrained.

Magmatic Hornblendite and Hornblende Clinopyroxenite

These rock types are generally found in the southwestern portion of the complex. They contain amphibole crystals that typically range from less than 1 cm to up to 3 cm in length. The crystals appear to be cumulus, but in some cases they replace pyroxene. Most hornblende-bearing ultramafic rocks in the Turnagain complex are associated with large amounts of magnetite that is interpreted to be cumulus in origin.

Hornblende Diorite

A 1,700 x 300 m elongate hornblende diorite to granodiorite body intrudes hornblendite and dunite in the central part of the intrusive suite and is offset by an east-northeast-striking fault. Narrow porphyritic granitic dykes, about 1 to 2 m wide and clearly post-mineral, were noted cutting wehrlites and clinopyroxenites in drill core (Simpson, 2007).

Metasediments

Numerous inliers, xenoliths and small inclusions of hornfelsed, calc-silicate metasedimentary rocks, similar to those seen marginal to the ultramafic intrusion, are present within the ultramafic intrusive rocks.

7.3 Geology of the Resource Area

The resource area is comprised of the Northwest, Horsetrail, Duffy, and the Hatzl zones. The mineralized zones are discussed in more detail in Section 9 of this report. The geology in the resource area is interpreted to trend west-northwest and to dip steeply north and south. There is a broad transition north to south from dunite/wehrlite and green dunite, to olivine clinopyroxenite, and clinopyroxenite. Contacts within the resource area are gradational and abrupt. This ultramafic package is juxtaposed against metasediments south of a west-northwest trending, northeast-dipping, listric fault.

While geological cross sections have been used to guide exploration and to develop grade shells at the Turnagain Project, a comprehensive set of lithology sections across the resource area had not previously been produced due to the gradational contacts and complex geology.

In September 2007, Hard Creek initiated a program to improve the understanding of the lithology of the resource area. Eighty-five distinct lithological descriptions from drill holes in this area were simplified to 21 rock types and 10 lithological domains as summarized in Table 7-1.

The alteration products of serpentinite, serpentinized dunite and serpentinized wehrlite are treated as a lithological domain due to their relative abundance, perceived continuity, and relatively elevated nickel grades.

The drill holes were composited using the 10 lithological domains on 15 m intervals down-hole and the composites were interpolated by nearest-neighbour (NN) methods into a block model with 25 m x 25 m x 15 m blocks. A search ellipse elongate in the east-west direction was used to reflect the general east west trend in the local geology. The lithology domains as interpolated into the block model were simplified into polygons on 51 north-south sections spaced 50 m apart. The current interpretation of the resource area geology is generally reflected in the NN interpretation which produced relatively continuous, west-northwest trending, and steeply dipping zones.

Figures 7-5 and 7-6 show the nearest-neighbour interpolation of lithology domains in plan and cross-section. The gap in blocks shown in Figure 7-5 represent the course of the Turnagain River which separates the Horsetrail, Duffy and Northwest zones to the north and west from the Hatzl zone to the southeast.

The nearest-neighbour interpretation of geology is considered to reasonably reflect the current interpretation of the resource area geology and for the first time shows the relative continuity of the lithological units in this area. The NN interpretation was completed as a first-pass approximation, required some arbitrary simplification of some polygons, and is not sufficiently detailed to identify faults believed to exist within the resource area. Refinement of the model through reconciliation of polygons between orthogonal sections and plans, and through additional information from drill logs such as structure and alteration intensity is recommended.

Table 7-1: Summary of Simplified Lithological Domains

Lithology Code	Rock Type	Lithology Domain	Domain Description
Ovb	0	10	overburden
cPx	1	101	clinopyroxenite
cPx-CS	1	101	clinopyroxenite
cPxDu	1	101	clinopyroxenite
cPxHb	1	101	clinopyroxenite
cPxocPx	1	101	clinopyroxenite
cPxspWh	1	101	clinopyroxenite
cPxWh	1	101	clinopyroxenite
ocPx	2	102	olivine clinopyroxenite
ocPxaltWh	2	102	olivine clinopyroxenite
ocPxcPx	2	102	olivine clinopyroxenite
ocPxDk	2	102	olivine clinopyroxenite
ocPxDu	2	102	olivine clinopyroxenite
ocPxspWh	2	102	olivine clinopyroxenite
ocPxWh	2	102	olivine clinopyroxenite
mtcPx	1.5	103	magnetite clinopyroxenite
mtcPxHb	1.5	103	magnetite clinopyroxenite
Du	4	104	dunite and wehrlite
DucPx	4	104	dunite and wehrlite
DuDk	4	104	dunite and wehrlite
DugDu	4	104	dunite and wehrlite
DuGs	4	104	dunite and wehrlite
DuocPx	4	104	dunite and wehrlite
DuSp	4	104	dunite and wehrlite
DuWh	4	104	dunite and wehrlite
Um	3	104	dunite and wehrlite
Wh	3	104	dunite and wehrlite
WhcPx	3	104	dunite and wehrlite
WhDk	3	104	dunite and wehrlite
WhDu	3	104	dunite and wehrlite
WhgDu	3	104	dunite and wehrlite
WhHb	3	104	dunite and wehrlite
WhocPx	3	104	dunite and wehrlite
gDu	4.5	105	green dunite
gDuWh	4.5	105	green dunite
spgDu	6.5	105	green dunite
Sp	5	106	green dunite
SpcPx	5	106	Serpentinite and serpentinized dunite and wehrlite
spDu	6	106	Serpentinite and serpentinized dunite and wehrlite
spDucPx	6	106	Serpentinite and serpentinized dunite and wehrlite
spDuCS	6	106	Serpentinite and serpentinized dunite and wehrlite
spDuDk	6	106	Serpentinite and serpentinized dunite and wehrlite
spDuDu	6	106	Serpentinite and serpentinized dunite and wehrlite
spDugDu	6	106	Serpentinite and serpentinized dunite and wehrlite

Lithology Code	Rock Type	Lithology Domain	Domain Description
spDuocPx	6	106	Serpentinite and serpentinized dunite and wehrlite
spDuSp	6	106	Serpentinite and serpentinized dunite and wehrlite
spDuWh	6	106	Serpentinite and serpentinized dunite and wehrlite
SpspDu	5	106	Serpentinite and serpentinized dunite and wehrlite
spUm	7	106	Serpentinite and serpentinized dunite and wehrlite
spWh	7	106	Serpentinite and serpentinized dunite and wehrlite
spWhcPx	7	106	Serpentinite and serpentinized dunite and wehrlite
spWhDk	7	106	Serpentinite and serpentinized dunite and wehrlite
spWhDu	7	106	Serpentinite and serpentinized dunite and wehrlite
spWhocPx	7	106	Serpentinite and serpentinized dunite and wehrlite
cpxHb	9.7	107	dikes
Di	8	107	dikes
DicPx	8	107	dikes
Dk	8	107	dikes
DkDu	8	107	dikes
DkSp	8	107	dikes
DkWh	8	107	dikes
fsHb	9	107	dikes
fsHbcPx	9	107	dikes
gDi	8	107	dikes
Hb	9	107	dikes
hbcPx	9.5	107	dikes
HbDi	9	107	dikes
hbmtcPx	9.5	107	dikes
hbmtcPxDk	9.5	107	dikes
hbocPx	9.5	107	dikes
mtHb	9	107	dikes
qDi	8	107	dikes
SpDk	10	107	dikes
CS	12	108	metasediments
CSaltUM	12	108	metasediments
CSDk	12	108	metasediments
GS	11	108	metasediments
Hfs	14	108	metasediments
Inc	11	108	metasediments
MGS	11	108	metasediments
MSD	11	108	metasediments
MSDcPx	11	108	metasediments
Ph	13	108	metasediments
Flt	99	999	undefined
MS	99	999	undefined
Qtz	99	999	undefined

Figure 7-5: Lithology Domains Nearest-Neighbour Interpolation – Plan View

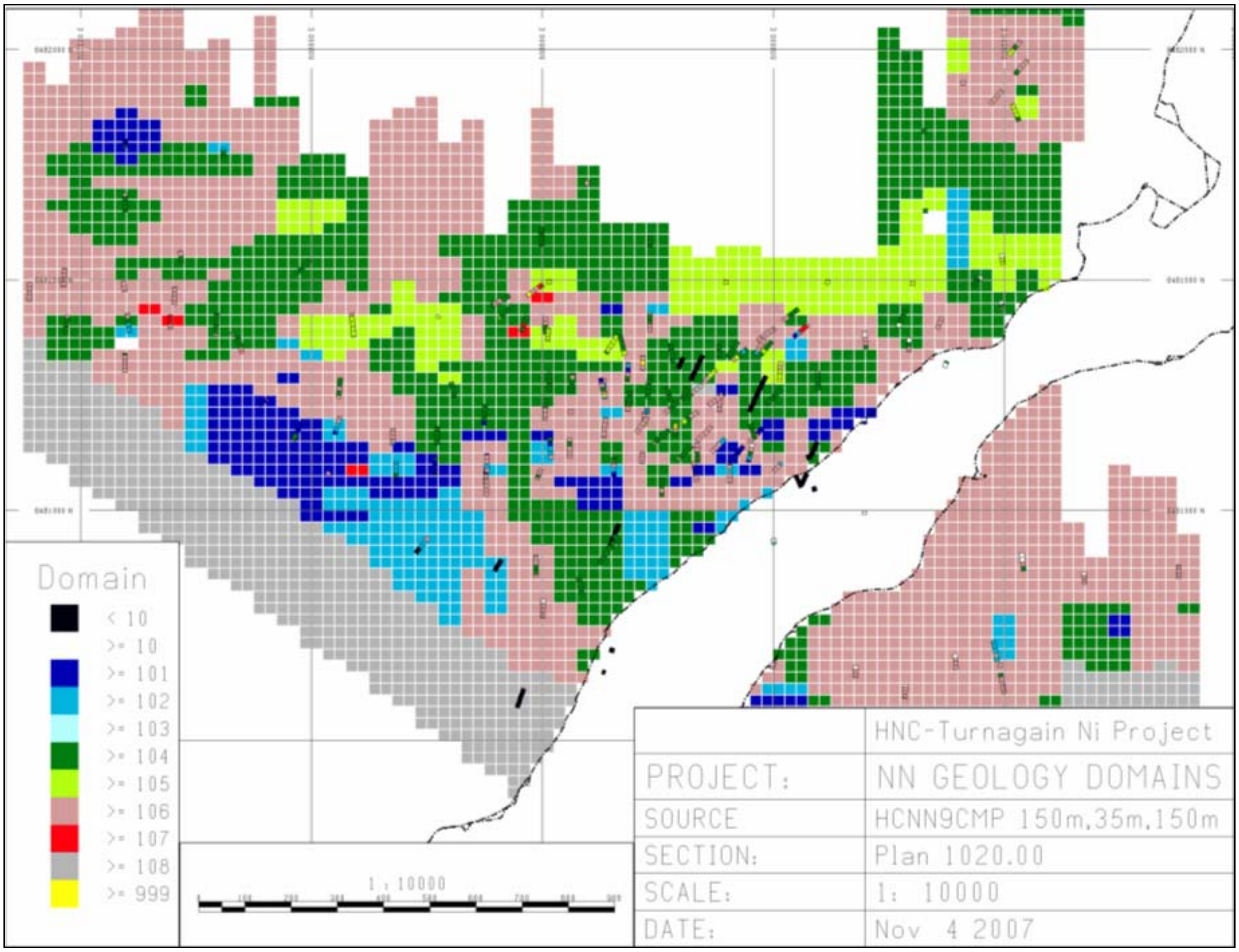
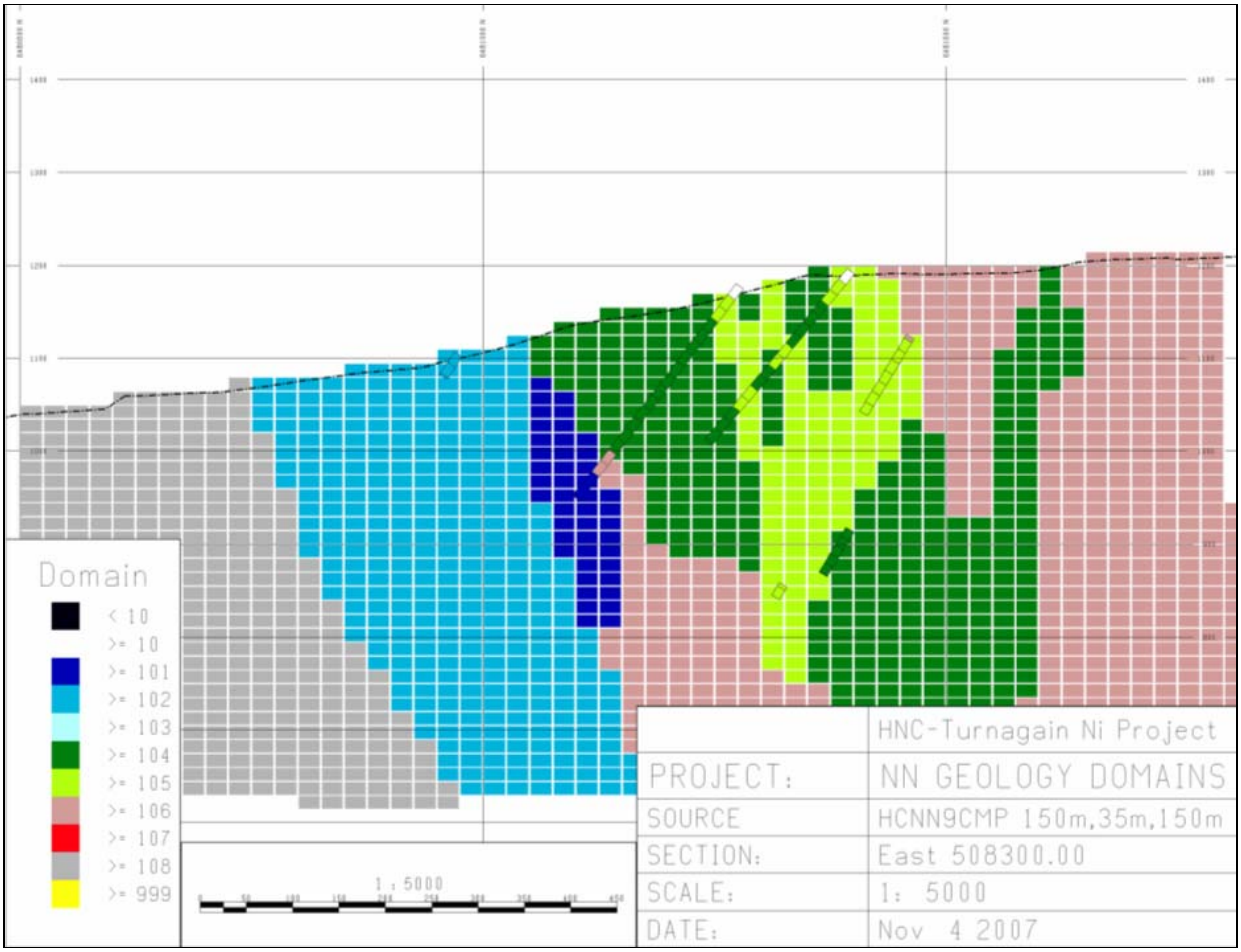


Figure 7-6: Lithology Domains Nearest-Neighbour Interpolation – Cross-Section View



8.0 DEPOSIT TYPES

The geological setting of the sulphide mineralization at the Turnagain deposit is unusual in that it is hosted by an Alaskan-type complex, a magmatic environment that is not generally noted for its sulphide potential. Nixon (1998) concluded that the Fe-Ni-Cu sulphides in the Turnagain complex are of magmatic origin and that wall rock inclusions observed in drill core may have provided a mechanism for sulphur saturation and precipitation of Fe-Ni-Cu sulphides.

Disseminated and rare net-textured mineralization at Turnagain is hosted in dunite, wehrlite, olivine clinopyroxenite and clinopyroxenite and serpentinized equivalents. Sulphides comprise pyrrhotite, pentlandite, chalcopyrite and trace bomite.

8.1 Alaskan-type Ultramafic Complexes

The following description of Alaskan-type ultramafic complexes is summarized from Nixon (1998) and Himmelberg and Loney (1995).

The classification of Alaskan-type complex is based on the silicate mineralogy. Cumulate silicate assemblages that characterize the Alaskan-type complexes comprise dunite, wehrlite, olivine clinopyroxenite and clinopyroxenite and characteristically lack orthopyroxene. The only other common silicate to appear in ultramafic lithologies is amphibole, which forms hornblende clinopyroxenite, clinopyroxene hornblendite and minor hornblendite. Plagioclase may be found in minor amounts in olivine-poor ultramafic lithologies and becomes a dominant constituent in associated mafic rocks such as hornblende or clinopyroxene bearing gabbro and diorite.

Other features typical of Alaskan-type complexes include a crude internal zonation from dunite at the core outwards to wehrlite, olivine clinopyroxenite, and clinopyroxenite, and more rarely to hornblendite and gabbro. Zonation may not be well developed or some zones may be missing. There is a general lack of layering in outcrop and an absence of high temperature tectonite fabrics. Contacts between major rock units range from gradational to sharp, and may be intrusive.

Alaskan-type complexes range in size from sills of the order of tens of metres in thickness to intrusions up to 10 km in maximum exposed widths.

8.2 The Turnagain Ultramafic Complex

The Turnagain complex displays the following key features of an Alaskan-type complex (Nixon, 1998):

- Ultramafic cumulates are restricted to mixtures of olivine and clinopyroxene with minor chromite, rare amphibole and trace phlogopite; orthopyroxene is absent.
- Centimetre-scale layering is comparatively rare.
- Localized chromitite layers (<1 m in thickness) in the dunite have been entirely remobilized to form schlieren and syndepositional folds, features that are characteristic of all Alaskan-type occurrences in BC.
- Clinopyroxene compositions are diopsidic, characteristic of Alaskan-type intrusions.

9.0 MINERALIZATION

The following description is modified from the Mineral Resource Update Turnagain Nickel Project, (Simpson, 2007) and is considered an appropriate description and interpretation of the Turnagain mineralization.

Showings of semi-massive and massive sulphides have been identified by work to date. The locations of these mineralized zones are shown on Figure 7-3. These semi-massive and massive sulphide zones, plus broad zones of disseminated sulphides, are invariably hosted by dunite and wehrlite near the southern and eastern margins of the ultramafic body. The central dunite is essentially devoid of sulphide minerals although it is worthy of note that the highly magnesian olivine is more enriched in nickel (up to 0.20–0.30 weight percent) than the olivines in the peridotites and pyroxenites which have been reported to be depleted in nickel in areas of sulphide mineralization. Nixon (1998) suggests that these features are further evidence of fractional crystallization of the ultramafic magma.

Primary sulphide minerals consist mainly of pyrrhotite with lesser pentlandite and minor chalcopyrite. Some bornite has been reported. Intercumulus and blebby sulphides with grain sizes ranging from 1 to 4 mm are evident in widespread disseminated zones seen in drill cores. Rarely these intercumulus sulphide grains coalesce to form net-textured sulphides. Semi-massive and massive sulphides and rare sulphide matrix breccias are also noted in drill cores over intervals not exceeding a few tens of centimetres.

Narrow fracture-filling sulphide lenses, commonly featuring chalcopyrite and minor pentlandite along with the more prevalent pyrrhotite, appear to be products of remobilization of primary sulphides adjacent to dykes, altered xenoliths and serpentinized areas.

Secondary nickel and copper sulphides, including violarite and valleriite, have been noted in serpentinized zones and both primary and secondary sulphides are associated with graphite. Recent petrographic and microprobe studies of drill core samples from the Horsetrail zone (Kucha, 2005) have identified additional nickel sulphide minerals including mackawinite, heazlewoodite, godlevskite and millerite. Platinum group element minerals identified to date include vysotskite, a palladium-iron-nickel sulphide and sperrylite, a platinum arsenide mineral.

Where serpentinization of the olivine is weak to moderate, the sulphides are in sharp contact with silicate minerals. With increasing serpentinization, the rims of the sulphide grains are replaced by magnetite and locally intergrown with serpentinite flakes.

The principal mineral zones identified to date on the Turnagain property (Figure 7-3) include the following:

- The Horsetrail zone and surrounding area have been the focus of most of the historic and recent diamond drilling. Results to date suggest a northwest to west-northwest trend for these zones which consist of broadly dispersed, disseminated to intercumulus sulphide mineralization in both dunite and wehrlite and serpentinized equivalents. Sulphide grains range in size from 0.5 to 5 mm and commonly rim olivine grains. Drill core samples from the Horsetrail zone have a median of 0.23% total nickel with grades ranging from 0.01% to 4.89% total nickel. AC-Ni-based (ammonium-citrate–hydrogen-peroxide leaching assay method) grades range from 0.01% to 4.06% and have a median grade of 0.14%. Total cobalt grades range from 0.001% to 0.480% with a median of 0.013% Co. There appears to be a spatial relationship between graphitic xenoliths, increasing clinopyroxene content in the ultramafic host rocks and the incidence of sulphide mineralization. Where present, chalcopyrite occurs along the margins of pyrrhotite and in narrow veinlets. Relatively unaltered dunite adjacent to the Horsetrail zone may contain total nickel values of 0.20% to 0.30%, virtually all of which is in the crystal lattices of the silicate mineral olivine and consequently is not of economic importance.
- The Northwest zone is contiguous with, and lies northwest of, the Horsetrail zone. This zone has mineralization styles and grades similar to the Horsetrail zone but is intruded by several mafic and felsic dikes which dilute the overall grade. Drill core samples from the Northwest zone have a median grade of 0.20% total nickel with grades ranging from 0.01% to 2.86%. AC-Ni-based grades range from 0.01% to 0.756% and have and a median of 0.14%. Total cobalt grades range from 0.001% to 0.166% Co. The Horsetrail and the Northwest zones form a zone approximately 2,000 m long in the east-west direction, and 550 m wide from north to south and have been tested by approximately 150 drill holes.
- The Hatzl zone mineralization consists of disseminated and net textured pyrrhotite and pentlandite hosted by dunite and wehrlite. This mineralization is similar to the Horsetrail zone and may be continuous with Horsetrail. The Turnagain River flows between the two zones and the region below the river has not been sufficiently drill-tested to exclude the potential of additional mineralization. The Hatzl zone is 1,150 m long in a northeast direction and 300 m wide in a northwest direction and has been tested by 13 drill holes.
- The Duffy zone mineralization lies 500 m northeast of the Horsetrail zone and consists of disseminated sulphides similar to those within the Horsetrail zone. Grades range from 0.014% to 0.525% total nickel and 0.007% to 0.388% AC-Ni. The Duffy zone is 300 m in diameter, lies 70 m below the surface topography, does not crop out and was discovered by exploration drilling in 2006. The zone has been tested by six drill holes.

Other mineralized zones on the property include the Bench, DJ, and DB prospects, which host platinum group element (PGE) mineralization, the Mandible, Davis, Highland, and Discovery prospects, which host Ni-Co mineralization and the Cliff and Central area prospects which host Ni-Co and PGE mineralization. These zones are exploration targets undergoing initial drilling.

10.0 EXPLORATION

Exploration in the period 1957–1995 is documented in Section 6 of this report. This section covers only work competed by HCNC and its precursor companies since acquisition of the Project in 1996.

10.1 Geological Mapping

Sulphide-bearing outcrops in the Davis, Horsetrail, Discovery and Cliff zones were relocated, then prospected and mapped during 1996.

In 1998, a global positioning survey (GPS) using a Trimble Geoexplore 2 instrument was undertaken by company personnel to locate and map drill holes, claim posts and other geographical positions.

Detailed geological mapping was undertaken by Clark (1976) at various scales from 1 inch:50 ft to 1 inch:1,000 ft as part of his thesis work. Additional mapping was completed by HCNC geologists, and Scheele (2007) at metric scales ranging from 1:1,000 to 1:5,000.

10.2 Geochemical Surveys

The following discussion modified from Carter (2005) is considered thorough and to reasonably represent the surface geochemical soil sampling programs completed on the property.

Of particular interest are the results of a 1971 soil geochemical survey conducted by Union Miniere over mineral claims contiguous with Falconbridge claims, and covering the northeastern margin of the ultramafic complex and the Cliff zone east of Turnagain River. More than 800 samples were collected from B and C soil horizons at 200 ft intervals along grid lines spaced 400 ft (122 m) apart and samples were analyzed for nickel, copper and cobalt. Values of greater than 650 ppm nickel and 300 ppm copper were considered to be distinctly anomalous; cobalt values were erratic. Best results were obtained from a 900 x 450 m area west of the Discovery zone where anomalous nickel values ranged from 800 to 2,000 ppm.

A geochemical sampling program carried out in 2003 consisted of the collection and analyses of 250 soil samples at an 100 m spacing along four topographic contour lines between 1,300 and 1,460 m elevation, northwest and upslope of the principal mineralized zones. An analysis and interpretation of the results obtained from these samples was undertaken by Dr. Colin E. Dunn, P.Geo. on behalf of the company in early 2004 (Carter, 2005).

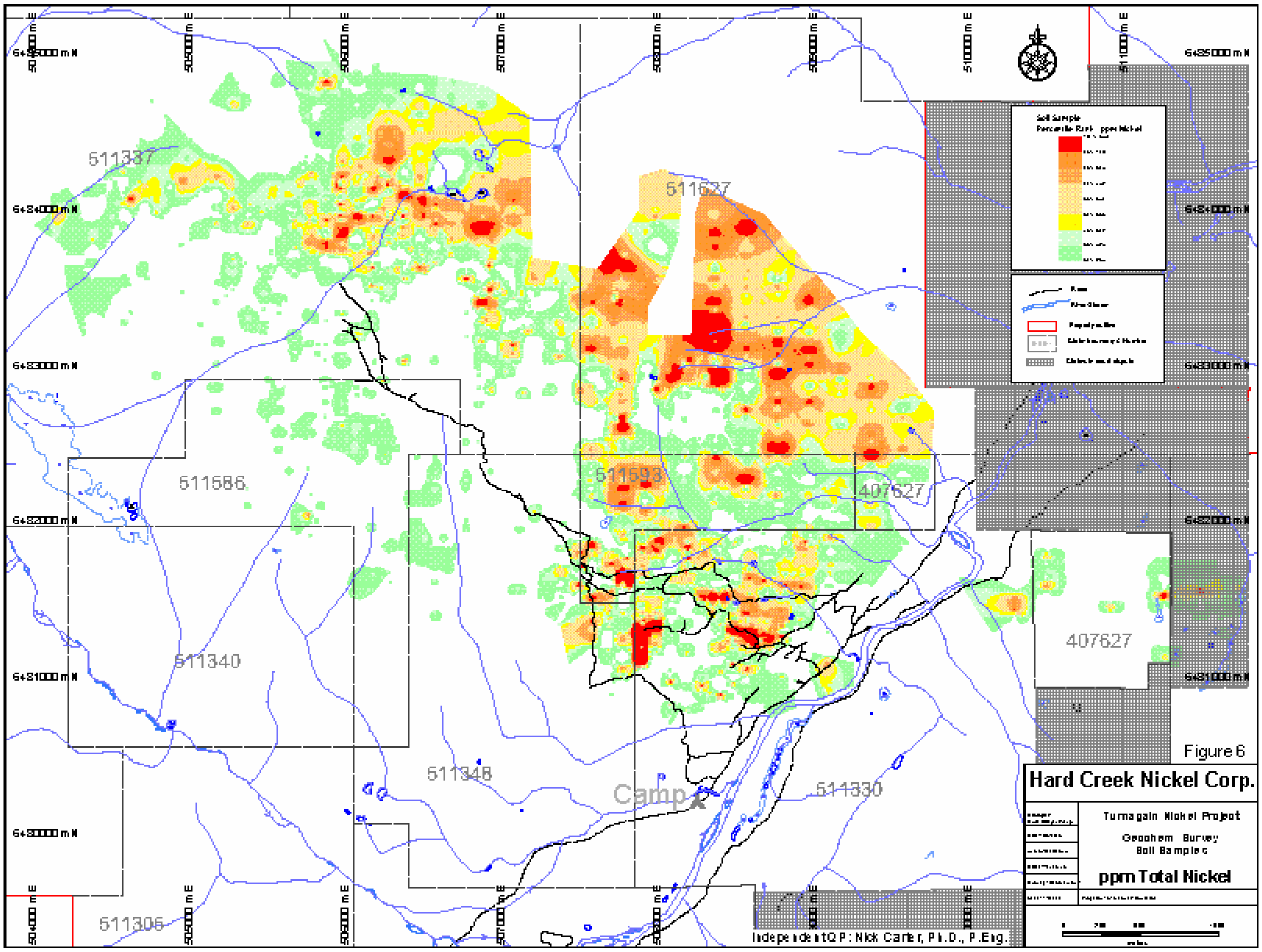
Results for copper, nickel, cobalt and platinum+palladium were kriged and contoured as 90th, 80th, 70th and 50th percentiles. Coincident and high copper, cobalt and platinum+palladium values are concentrated within a poorly-explored area between 3 and 4 km west-northwest of the Horsetrail zone. Elevated nickel values in soils are more widespread and are coincident with the Horsetrail zone as shown in Figure 10-1, and immediately northwest of the copper, cobalt and platinum+palladium anomalies.

A reconnaissance biogeochemical survey, carried out in April, 2004, consisted of the collection of 132 twig and bark samples along four transects over the Turnagain ultramafic intrusion. Analytical results were not as definitive as those obtained from previous soil sampling and a comprehensive geochemical sampling program was initiated in mid-2004 to follow up and expand upon results of the 2003 surveys.

The 2004 program also consisted of the collection of more than 2,000 soil samples collected at 25 m intervals along survey lines spaced 100 m apart within an area of 15 km^2. More detailed sampling was undertaken in areas yielding anomalous base and precious metals results. Results of this survey highlighted two strong copper-in-soil anomalies 2.5 km northwest of the Horsetrail zone with values exceeding 430 ppm copper with peaks to 3,219 ppm copper over areas of 1,500 x 1,100 m and 900 x 600 m. These anomalous areas flank the hornblende diorite-granodiorite intrusion which cuts the older ultramafic rocks in this area. Anomalous platinum-palladium values in soils, in part coincident with the DJ zone, extend from the northern part of the larger copper-in-soils anomaly. Anomalous nickel values in soils are widespread over the northern part of the Turnagain ultramafic intrusion and within and adjacent to the Horsetrail zone. The geochemical interpretation requires that anomalous nickel values in soils are paired with copper so that the highly mobile nickel originating from olivine can be screened. Copper occurs only in sulphide minerals and when present in ultramafic rocks with nickel can be used successfully to indicate nickel anomalies of exploration significance.

The 2004 geochemical program also included the collection and analyses of 330 rock float and 243 bedrock samples from within, and adjacent to, the soil geochemical grid. Results for total nickel and platinum+palladium indicated significant total nickel results (>0.20% to a maximum of 1.9%) in both float and bedrock samples, which are mainly clustered in the area of the Horsetrail zone and in a smaller area north of the DJ zone, known as the Central area.

Figure 10-1: Nickel-in-Soils



Note: Figure from Carter, 2005

10.3 Geophysical Surveys

The following discussion, modified from Carter (2005), is considered thorough and to reasonably represent the geophysical survey programs completed on the property.

Airborne Surveys

Scintrex completed a helicopter EM (HEM) and magnetic survey for Falconbridge in July 1969 (680 line-km), and Questor completed a fixed wing "high resolution" magnetic survey for Bren-Mar in August 1996 (400 line-km). A third airborne geophysical survey was completed over the Turnagain property by AeroQuest Limited in late September of 2004.

The AeroQuest survey utilized a helicopter-borne AeroTEM II time domain electromagnetic system and a high sensitivity caesium vapour magnetometer. Continuous readings on both instruments were obtained from northeast-southwest oriented survey lines at 100 to 200 m spacing; precise locations were established using a GPS. Terrain clearance was 30 m and the survey totalled 1,866 line-km. The AeroQuest magnetic response confirmed the results of earlier surveys, accurately outlining the limits of the Turnagain ultramafic intrusion as shown in Figure 10-2. Magnetic data ranged from lows of 55,000 nanoteslas (nT) to highs of 63,000 nT; average background was 57,800 nT. The AeroQuest survey also highlighted electromagnetic anomalies within the intrusion area of interest.

Ground Magnetic Surveys

Ground magnetic surveys using an Overhauser magnetometer commenced in 1997–1998 to further define two of the airborne anomalies, Davis (Grid A) and Northwest (Grid B).

The Grid A survey used north–south lines, at a 100 m spacing, with stations every 25 m along lines. A total 12.3 line-km was surveyed within an approximate 15 km^2 area. A number of magnetic highs identified from the survey were correlated with pod-like serpentinized and magnetite-banded peridotite intrusions; however, four of the magnetic anomalies were considered to be potentially due to the presence of sulphides.

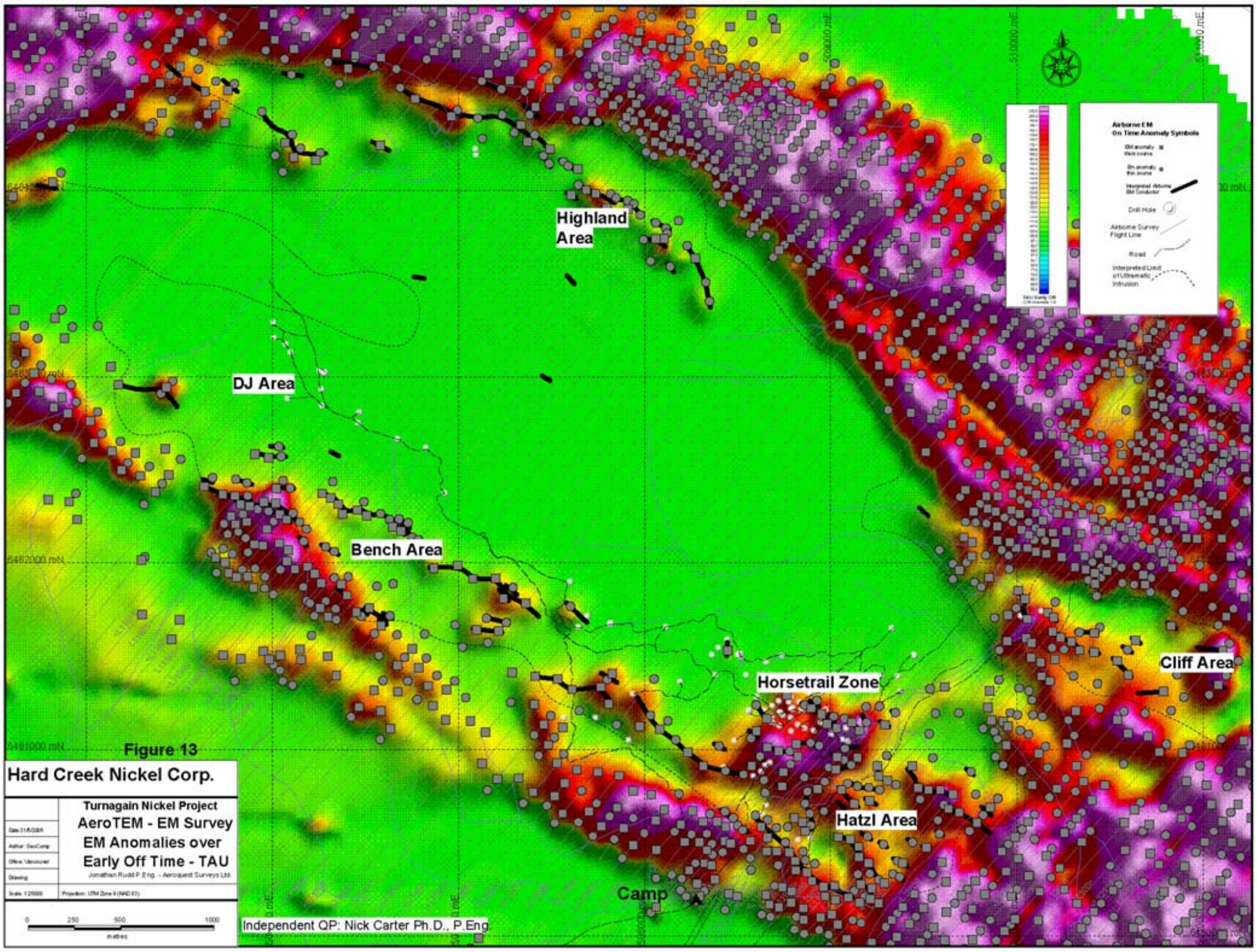
Grid B comprised 100 m spaced east–west lines, with stations at 25 m along lines, for a total survey distance of 5.6 line-km. The survey identified a strong positive magnetic anomaly covering an approximate area of 6 km^2.

Results of the grid-based surveys showed that the areas of high total field magnetic readings do not necessarily coincide with sulphide-rich rocks as there appears to be little correlation between trends and known mineralized showings. Some prospects are on magnetic highs (i.e., Northwest zone), some on magnetic lows (i.e., Discovery zone) and others in areas of mixed magnetic response (i.e., Horsetrail and Fishing Rock zones).

Down-hole Geophysics

Borehole pulse electromagnetic surveys were undertaken of four drill holes (97-9, 98-1, -4 and -5) in 1998. All of these holes were drilled to test the southern part of the Horsetrail zone and major in-hole anomalies were interpreted as being caused by two sheet-like, shallowly south-dipping conductive horizons which in part correlate with zones of sulphide mineralization containing elevated (+0.30%) nickel values and with talc/serpentinite zones.

Figure 10-2: Airborne Magnetic Survey Image



Note: Figure from Carter, 2005. The figure 13 cited above the title block is a reference to the Carter report. Grey squares represent an EM thick source anomaly, grey circles represent an EM thin source anomaly

10.4 Drilling

At the effective date of this report, the Turnagain ultramafic intrusion has been tested by 74,364 m of diamond drilling in 302 holes since 1966 (Table 10-1).

Table 10-1: Summary of Drill Programs

Year	Operator	Holes (#)	(m)
1967	Falconbridge	13	1,304.90
1970	Falconbridge	15	1,458.00
1996	Bren-Mar Resources Ltd.	5	795.30
1997	Bren-Mar Resources Ltd.	9	1,855.30
1998	Bren-Mar Resources Ltd.	5	1,264.10
2002	Canadian Metals Exploration	7	1,686.63
2003	Canadian Metals Exploration	*22	8,672.0
2004	HCNC	49	7,633.42
2005	HCNC	37	7,143.10
2006	HCNC	**68	19,121.80
2007	HCNC	***75	24,600.00
Total		**305**	**75,534.55**

* One 2003 drill hole was extended
** One 2005 drill hole, and one 2006 drill hole were extended
*** Three holes remained to be drilled at the effective date of this report. Nineteen of these holes are used in this resource estimate update

By the end of the 2007 drilling season, HCNC will have drilled an additional 3 holes totalling 1,170 m. Drilling in 2007 will comprise 75 holes for a total of 24,600 m. This includes the collection of approximately 4,275 m of PQ-sized drill core for a variety of crushing, grinding and abrasion testwork. Analytical results for 19 holes drilled in 2007 and returned by 25 September 2007 (the effective date of this report) are reported in Section 11 of this report.

10.5 Other Studies

During 1972–1975, a PhD dissertation was completed on the geology of the Turnagain intrusion (Clark, 1976). The work comprised geological mapping, lithogeochemical sampling and petrographic studies.

In 2007, a master thesis was completed on the age and origin of the Turnagain intrusion and associated mineralization (Scheele, 2007).

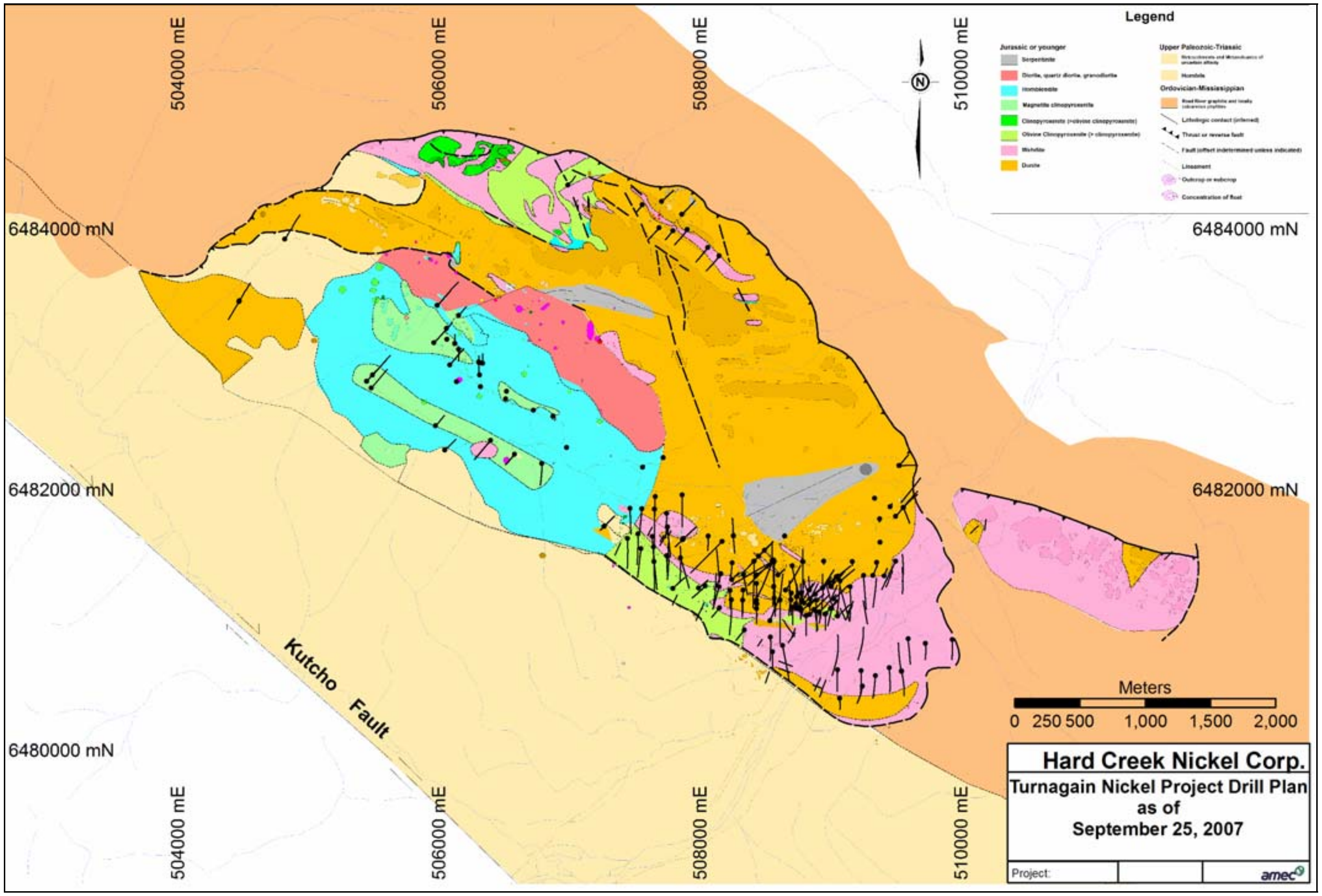
11.0 DRILLING

The Turnagain drill hole database comprises 248 holes, including 19 holes drilled in 2007 for a total of 58,797 m of drilling. Two hundred of these holes totalling 51,861 m were drilled in the resource area. Results for an additional 56 holes (16,737.55 m) drilled on the property in 2007 were not available for review at the writing of this report. These 56 holes include metallurgical test holes in the resource area and drilling in areas other than the resource area. The locations of the 248 holes are shown in Figure 11-1.

Drilling in 2007, as analyzed as at the effective date of this report, continued to intersect similar grade, mineralization style, and interval lengths as previously drilled on the property. The 19 holes in the resource area were drilled to test extensions of the known zones and to upgrade resource classification in parts of the 2006 resource estimate. Significant results for the 2007 holes are shown in Table 11-1. AC-Ni grades greater than 0.1% AC-Ni are intersected from surface to the total depth of drilling in most holes. Irregular and gradational contacts and the disseminated style of mineralization preclude interpretation of true thickness. There is a zone central to the Horsetrail area that is 300 m in the north-south direction and 900 m in the east-west direction, with several intersections returning greater than 0.2% AC-Ni. These grades are not continuous throughout the zone.

Most of the holes drilled to date have been inclined. Initial drilling by Falconbridge recovered QXT (20 mm) and AQ (27 mm) size core. BQ size (36.4 mm) core was recovered until 1996, and since 2004 contactors DJ drilling have recovered NQ size (47.6 mm) core. Part of the 2007 drill program will include some PQ size (85 mm) core collected for metallurgical purposes. Core recoveries are excellent, averaging 95%. Prior to 2006, most drill core sample intervals were 2 m. Since 2006, core sampling has been completed predominantly on 4 m intervals. An initial check on grade versus sample length shows no indication of grade bias. A more detailed check on the impact on grade, due to the change in sample length, is recommended before proceeding to more advanced work.

Figure 11-1: Drill Hole Location Plan (effective date 25 September 2007)



Note: Figure from Hard Creek Nickel Corporation

Table 11-1: Significant Results for 2007 Drill Holes used in this Resource Estimate (19 Holes)

Hole ID		From (m)	To (m)	Length (m)	TD Ni (%)	TD Co (%)	AC-Ni (%)	AC-Co (%)
07-176		6.4	160	153.6	0.27	0.015	0.220	0.011
	incl.	16	88	72	0.35	0.018	0.277	0.012
		200	228	28	0.28	0.019	0.243	0.013
		280	316	36	0.21	0.010	0.179	0.007
07-177		48	104	56	0.25	0.013	0.179	0.007
		136	160	24	0.26	0.016	0.148	0.007
		200	280	80	0.33	0.014	0.273	0.010
07-178		16	32	16	0.3	0.012	0.237	0.009
		76	192	116	0.35	0.018	0.340	0.016
	incl.	76	104	28	0.46	0.020	0.473	0.021
	and incl.	156	188	32	0.4	0.017	0.390	0.014
		200	216	16	0.21	0.014	0.183	0.010
		272	360	88	0.25	0.017	0.170	0.010
		376	424	48	0.27	0.017	0.179	0.009
	incl.	400	420	20	0.37	0.021	0.247	0.011
		448	483.1 EOH	35.1	0.24	0.013	0.233	0.011
07-179		11.3	36	24.7	0.26	0.011	0.229	0.008
		80	104	24	0.24	0.011	0.212	0.008
		120	272	152	0.3	0.016	0.224	0.010
		280	296	16	0.25	0.016	0.207	0.010
		364	424 EOH	60	0.24	0.019	0.197	0.011
07-180		144	160	16	0.20	0.012	0.197	0.010
		176	188	12	0.22	0.009	0.212	0.008
		216	324	108	0.32	0.017	0.275	0.013
	incl.	236	260	24	0.44	0.02	0.425	0.017
	and incl.	316	317.1	1.1	1.33	0.099	1.254	0.091
		332	416	84	0.26	0.014	0.219	0.011
	incl.	356	372	16	0.34	0.014	0.283	0.010
	and incl.	392	404	12	0.36	0.015	0.306	0.013
07-181		72	364	292	0.28	0.015	0.261	0.012
	incl.	72	84	12	0.39	0.014	0.301	0.009
	and incl.	104	120	16	0.35	0.017	0.345	0.015
	and incl.	192	224	32	0.42	0.021	0.429	0.019
07-182		132	152	20	0.24	0.012	0.172	0.009
		188	276	88	0.30	0.017	0.240	0.012
	incl.	212	228	16	0.41	0.018	0.324	0.011
		312	336.5 EOH	24.5	0.23	0.017	0.188	0.014
07-183		48	252	204	0.29	0.018	0.266	0.015
	incl.	92	128	36	0.34	0.020	0.297	0.017
	and incl.	180	208	28	0.39	0.020	0.349	0.015
		284	348	64	0.28	0.018	0.273	0.016
	incl.	324	344	20	0.34	0.016	0.331	0.014
07-184		3.7	28	24.3	0.23	0.017	0.175	0.013
		60	92	32	0.29	0.015	0.221	0.009
		352	368	16	0.27	0.016	0.240	0.007
		380	537.9 EOH	157.9	0.25	0.016	0.207	0.008

Hole ID	From (m)	To (m)	Length (m)	TD Ni (%)	TD Co (%)	AC-Ni (%)	AC-Co (%)
07-186	20	36	16	0.22	0.013	0.133	0.008
	52	268	216	0.27	0.013	0.169	0.009
	300	324	24	0.25	0.014	0.216	0.010
07-190	8	20	12	0.24	0.014	0.165	0.010
	28	44	16	0.22	0.014	0.176	0.010
07-192	3.7	152	148.3	0.25	0.012	0.169	0.010
	164	264	100	0.25	0.014	0.209	0.009
07-195	6	94	88	0.27	0.014	0.223	0.008
	114	126	12	0.26	0.011	0.237	0.008
	174	198	24	0.21	0.013	0.200	0.009
	214	306	92	0.30	0.014	0.249	0.008
	406	422	16	0.22	0.010	0.159	0.008
07-103 (extension of 05-103)	233	281	48	0.24	0.012	0.163	0.010
07-185	64	84	20	0.28	0.014	0.191	0.008
	104	124	20	0.26	0.014	0.182	0.010
	192	208	16	0.23	0.013	0.215	0.013
	252	280	28	0.21	0.018	0.201	0.009
	424	436	12	0.21	0.012	0.207	0.011
07-187	33.3	52	18.7	0.23	0.014	0.249	0.011
	84	108	24	0.23	0.013	0.231	0.009
	192	212	20	0.21	0.013	0.205	0.008
	228	312	84	0.22	0.012	0.210	0.007
	404	428	24	0.35	0.045	0.330	0.033
07-189	32	56	24	0.24	0.014	0.128	0.010
	76	92	16	0.29	0.015	0.148	0.009
	120	132	12	0.31	0.016	0.201	0.012
	156	200	44	0.25	0.014	0.189	0.008
	240	256	16	0.21	0.01	0.178	0.010
	296	360	64	0.24	0.017	0.205	0.007
07-191	12.2	56	43.8	0.24	0.013	0.141	0.010
	72	100	28	0.23	0.013	0.146	0.009
	136	164	28	0.21	0.011	0.177	0.008
	224	244	20	0.24	0.018	0.227	0.006
	300	420.3 EOH	120.3	0.33	0.022	0.333	0.011
07-194	92	132	40	0.25	0.014	0.243	0.008
	144	188	44	0.21	0.011	0.222	0.008
	324	340	16	0.25	0.012	0.238	0.011

Collar Surveys

HCNC planned drill holes in advance and then spotted the collar in the field using a handheld GPS. Because of the high magnetic background, HCNC set the direction of drilling using foresights and backsights which are also spotted by HCNC staff using a GPS.

The hole dip is set with an inclinometer on the drill rod. After completion of the hole, the collar location is resurveyed using a backpack Trimble GPS and the dip is taken from the first Reflex Maxibor down-hole survey measurement. Finally the collar location, azimuth, and dip are surveyed by a professional Land Surveyor. Data in the collar table for holes used in the resource estimate are a mixture of all methods. The survey method is not indicated in the database. A review of the drill collar data is presented in Section 14.2.

Down-hole Surveys

HCNC began using a Reflex Maxibor survey tool for down-hole survey measurements in 2005. HCNC employees perform the survey and collect measurements for all new holes and have collected Maxibor surveys for some pre-2005 holes. Measurements are take approximately every 3 m down-hole and are collected digitally in a hand-held recorder. Readings are stored in raw data files which include depth and change in azimuth and dip from the collar. The data are post-processed with Reflex proprietary software SProcess® to assign a true azimuth and dip using the collar azimuth as determined by the collar surveys. This method makes accurate down-hole surveys dependant on the collar surveys. Prior to 2005, holes were surveyed down-hole for changes in dip by acid-etched tube methods or were not surveyed down-hole. The down-hole survey method is not indicated in the database. The impact of the down-hole surveys on the resource estimate is discussed further in Section 14.2.

12.0 SAMPLING METHOD AND APPROACH

12.1 Geochemical Surveys

There is no information that was made available to AMEC for the various geochemical sampling programs. As the data has been superseded by drill hole sampling in the areas that are currently included in the resource estimate (Section 17), geochemical surveys are not considered further in this report.

12.2 Drilling

AMEC has been provided with no information regarding drilling or logging procedures in place during the 1967–1998 exploration programs, and thus can make no comment on these data.

Geotechnical Data

In 2007 HCNC contracted Piteau and Associates Engineering Ltd. to provide geotechnical core logging guidance. Piteau provided HCNC geologists with instructions for recording core RQD, recovery, joint frequency, joint condition, hardness, and weathering. HCNC have posted the geotechnical protocol in several places in the core shack and are using standardized geotechnical core logs. In addition, HCNC geologists are performing point load tests on core following instructions set out by Piteau. Geotechnical logging between 2002 and 2007 included RQD and recovery only. AMEC reviewed the protocol and observed the geotechnical logging by HCNC geologists, and found the method and application to be satisfactory. AMEC reviewed core for several holes and concluded that the core is generally competent and recovery is very good.

Geological Data

In 2006 HCNC established a core logging and sampling protocol which is posted as a flowsheet in the core shack. AMEC reviewed the protocol and observed the geological logging by HCNC geologists and found the method and application to be satisfactory. Prior to any geological logging, the core is re-aligned, driller block measurements are converted to metres, and sample intervals are marked at 4 m intervals generally ignoring lithology. Drill core was sampled at 2 m intervals or less during the 2004 and 2005 programs and on 4 m intervals since 2006. Sample intervals are marked with a red or yellow marker and sample numbers are assigned from a pre-printed Acme Analytical Labs assay tag book. Core is photographed three boxes at a time on the logging rack in the core shack with a tripod mounted camera from a pre-set height.

Core was halved by use of a hydraulic core splitter and/or a diamond saw. Half of the core is stored in core boxes on site and half sent for analysis.

Density

HCNC collects specific gravity (SG) measurements by water immersion method every 20 samples, using up to 50 cm of un-split core. A protocol for SG measurements is posted in the logging tent. Specific gravity calculated as follows:

SG = weight in air / (weight in air – weight in water)

AMEC reviewed the protocols and observed the procedure and concluded the SG measurements are reasonable. Data is recorded manually on paper and later transferred to a digital file. Data entry errors due to transposition of numbers or poor written records are possible.

AMEC recommend double data entry for any manual entry of data into a database. AMEC also recommends HCNC create a SG standard to use periodically to ensure the scale is working properly.

13.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

13.1 Sample Preparation and Analyses

AMEC has been provided with no information regarding sample preparation, analytical procedures or sample security measures in place during the 1967–1998 exploration programs, and thus can make no comment on these data. Assay results from these years are not used in the estimate reported in Section 17.

Drill core samples from the 2002 to 2007 programs, received by Acme Analytical in Vancouver, were checked against requisition documents prior to being dried, weighed, crushed, split and pulverized before being subjected to a variety of analytical techniques. Acme is a certified ISO:9000 facility.

Prior to 2004, samples were analyzed for nickel, copper, cobalt, and approximately twenty major and minor elements by aqua regia digestion followed by an inductively coupled plasma emission spectroscopy (ICP-ES) finish. Samples collected from the 2004 to 2007 programs were subjected to a four acid (HNO_3-$HClO_4$-HF and HCl) digestion followed by ICP-ES analyses to determine values for total nickel, copper, cobalt, and 22 other elements, including sulphur.

In 2003, to assist in distinguishing nickel in sulphide from nickel in silicate phases, every 10^{th} sample was analyzed for nickel, cobalt, and magnesium with an ammonium citrate-hydrogen peroxide leach. Beginning in 2004, all new samples and most of the older samples have been analyzed with AC-Ni.

In 2004 and 2005 sulphur content was analyzed by the Leco furnace method. In 2006 sulphur content was analyzed by ICP-ES after a four acid digestion. In 2007 sulphur content was analyzed by Leco and ICP-ES.

For some core, particularly outside the resource area, samples were analyzed for platinum, palladium, and gold by lead-collection fire-assay fusion followed by ICP-ES and results reported in parts per billion (ppb).

13.2 Quality Assurance and Quality Control (QA/QC)

Laboratory quality control during 2004 to 2007 was maintained by routinely analyzing internal standards, sample blanks and duplicate samples. HCNC staff also insert reference sample pulps in the field as samples ending in 00, 01, 25, 26, 50, 51, 75, 76, 100, 101 and blank samples are inserted every 30 samples. Acme is instructed to create and analyze duplicate pulps from crushed core every 30^{th} sample. Pulps from every 10^{th} sample are sent to a check lab. In 2007 International Plasma Laboratories Ltd (IPL) in Richmond was used as a check laboratory and analyzed pulps for total nickel, sulphide nickel and sulphur. Prior to 2007 ALS Chemex in Vancouver was used as a check laboratory and pulps were analyzed for total nickel and sulphur. IPL is ISO:9001 certified; the ALS Chemex laboratory is ISO:9002 certified.

The HCNC reference standards used for Ni, Cu and Co were two CANMET reference samples labelled “U.M. 2” and “U.M. 4” (Cameron, 1975). Both were derived from small lenticular masses of peridotite that occur along a major east-west fault zone in the Werner Lake District of northwestern Ontario. CANMET analyzed the material for ascorbic acid-hydrogen peroxide soluble nickel and also by use of a four acid digest for total Ni contents. The CANMET certification was completed in 1974 and is not supported by current industry standards requiring a round robin approach using several labs.

HCNC have two reference materials, HNC-94 and HNC-103, prepared from mineralized drill core from the resource area. Barry Smee certified these through a round robin process for total digestion nickel, iron, copper, and sulphur.

Initial attempts by HCNC to certify the CANMET and Company standards for soluble nickel were unsuccessful. Through diligence and careful work HCNC have resolved inter lab issues with respect to the analytical protocols and have initiated a new multi lab round robin for these standards. Results for this round robin program are expected by the end of 2007. The resource estimate in Section 17 of this report is reported in terms of NiS which in turn is based on AC-Ni. Results of the new round robin should be carefully reviewed.

Three other standards, labelled PGMS-1, WGB-1 and WMG-1, were used for monitoring Pt and Pd.

Field blanks for the 2004 and 2005 programs were considered to be questionable. The field blank material was obtained from Meadows Landscape Supply in Pitt Meadows and described as “⅜ inch crushed fine granite”; however, as 17 blank samples returned up to five times the detection limit for Ni (0.001%), Simpson (2006) recommended that an alternative source of blank material be investigated.

The field blank material used during 2006 was crushed granite gneiss obtained from Squamish.

HCNC inspect QC samples on receipt of Acme certificates. Failures are identified by inspection of values. Standards and blanks are reviewed based on acceptable limits, and duplicates based on straight line graph. When failures are identified, Acme is notified to do re-run analyses on the batch.

13.3 Sample Security

Drill core was transported from the drill site to the exploration camp for logging by the drill contractor. Split core samples were numbered and bagged, and transported from the site by helicopter in sealed numbered bags in 300-350 kg lots, to Dease Lake or by plane to Smithers. The samples were then shipped by commercial transport to the primary laboratory, Acme, in Vancouver. Requisition forms were faxed to HCNC Vancouver office with date and number of samples shipped, and Acme notified HCNC upon receipt of samples.

Drill core from holes drilled between 1996 and 2002 is stored in racks at the Boulder camp on Wheaton Creek, 15 km west of the property. Core recovered from all the 2003 to 2007 programs is stored in sturdy racks at the camp, on the property. Sample security and core storage are considered to conform to industry standards.

14.0 DATA VERIFICATION

14.1 AMEC Site Visit, 2007

AMEC traversed the extents of the known mineralized areas of the Horsetrail and the Northwest zones and observed several drill sites all with well marked collar monuments. For the most recent drilling, casing has been left in the hole and a 2 m wooden fence post, which is tagged with an aluminium sheet which identifies the hole number, azimuth and dip, has been placed beside the collar location. Older holes without casing are commonly identified by a boulder monument at the collar location, and many of these holes have been marked with a 2 m fence post. AMEC used a handheld GPS unit to verify collar locations of the holes listed in Table 14-1. The absolute maximum difference between the handheld GPS and the database coordinates in east or north direction is 6.5 m which is within the tolerances considered practical for a non-differential handheld GPS.

Table 14-1: Drill hole Collar Checks in the Field

BHID	Year	Size	Length	East	North	Elevation	Zone	AMEC East	AMEC North	Δ East	Δ North
02-06	2002	BQ	485.2	508512.3	6481513.4	1,162.7	Horsetrail	508507.0	6481507.0	5.3	6.4
03-21	2003	BQ	333.8	508606.0	6481280.8	1,117.2	Horsetrail	508603.0	6481282.0	3.0	-1.2
03-22	2003	BQ	281.9	508382.5	6481398.5	1,167.9	Horsetrail	508379.0	6481395.0	3.5	3.5
04-24	2004	BQ	370.7	508789.6	6481215.5	1,083.7	Horsetrail	508787.0	6481222.0	2.6	-6.5
04-65	2004	NQ	136.0	508363.2	6481504.4	1,185.2	Horsetrail	508362.0	6481500.0	1.2	4.3
05-92	2005	NQ	205.8	508995.4	6481385.9	1,082.3	Horsetrail	508997.0	6481389.0	-1.6	-3.1
06-127	2006	NQ	367.3	507476.5	6481712.7	1,276.8	Northwest	507474.0	6481709.0	2.5	3.7
06-166	2006	NQ	305.7	507412.8	6481908.2	1,300.7	Northwest	507410.0	6481903.0	2.8	5.2

AMEC reviewed split core for the entire length of holes 06-121, 06-159, 06-170, 07-179 (230–350 m), and 05-107 (every second box):

- The core in these boxes is generally competent, split evenly and the sample intervals are marked adequately.
- Core boxes are well marked and stored in sturdy racks. The geological contacts are gradational and rock type classification somewhat subjective; however, the logged lithology generally matched lithology observed.
- The core is competent. In 2006 HCNC established a geology key which defines codes for major rock type and modifiers, structural modifiers, sulphide minerals, and xenoliths. This key is posted through out the core shack and is valid for the most recent holes only.
- At least one other geology key was used prior to 2006 and the database reflects this, having 277 unique rock types (including modifiers).

AMEC reviewed cross-sections for the entire resource area:

- The sections were in various states of interpretation. Some looked east; some looked west and colour codes were not consistent from section to section.
- Section interpretation was incomplete and not systematic, varying from plotting of drill holes only, grade shell polygons or lines, logged lithology polygons or lines, lithology polygons or lines based on Mg/Ca ratios, or based on Mg only.
- Section interpretation varies by drill campaign and in some cases two sections within 15 m of each other showed the same drill holes.

AMEC made recommendations to rationalize lithology from all holes in the resource area to the new 2006 geology key and to complete a set of comprehensive geology cross sections across the resource area. HCNC agreed and the work was completed. The results are described in Section 7 of this report.

Twelve half-core samples from the Northwest, Horsetrail, and Hatzl zones were collected by AMEC and submitted to Acme Analytical Laboratories, in Vancouver, for preparation and analysis using HCNC procedures. The check samples confirm the grades reported in the drill hole database from previous exploration on the property (Table 14-2).

Table 14-2: AMEC Check Sample Results

Drill Hole	AMEC Sample Number	Sample Depth	Logged Rock Type	HCNC Sample Number	TD-Ni %	AC-Ni%	Reassay TD-Ni %	Reassay AC-Ni%	Reassay AR-Ni%
06-121	gkk1	7.0	gDu	352694	0.226	0.108	0.255	0.091	0.256
	gkk2	9.7	gDu	352694	0.226	0.108	0.186	0.068	0.177
	gkk3	72.1	spDu	352711	0.216	0.175	0.309	0.299	0.290
	gkk4	75.0	spDu	352712	0.201	0.124	0.399	0.388	0.368
06-159	gkk5	54.3	spDu	355814	0.188	0.161	0.170	0.157	0.167
	gkk6	52.3	spDu	355814	0.188	0.161	0.226	0.228	0.220
	gkk7	200.0	spDu	355793	0.328	0.311	0.410	0.433	0.388
	gkk8	200.9	spDu	355793	0.328	0.311	0.203	0.209	0.194
06-170	gkk9	54.1	spDu	357034	0.271	0.206	0.335	0.205	0.331
	gkk10	57.8	spDu	357034	0.271	0.206	0.192	0.161	0.190
	gkk11	278.5	spDu	357093	0.182	0.172	0.190	0.190	0.184
	gkk12	280.7	spDu	357093	0.182	0.172	0.154	0.167	0.153

Note: TD-Total Digestion, AC-Ammonium Citrate, AR- Aqua Regia

14.2 Database Validation

AMEC completed an independent review of the Turnagain project database in 2007. The results of this review are summarized below:

Drill Collars

A review of the collar table data revealed that collars for 26 holes which have been surveyed by professional surveyors had not been updated in the drill hole database. In addition, four holes, 02-05, 02-07, 03-01, and 04-70, have no decimal precision in the collar coordinates and are assumed to be planned coordinates and therefore are considered unsurveyed.

Drill hole collars were checked against the 5 m contoured DTM topography used in the resource estimate. The check used north-south sections in MineSight® software, and a discrepancy of one contour interval (i.e., 5.0 m) was used as a guideline to highlight potential problems in the collar or topography data. Four holes, 02-02, 02-03, 02-04, and 06-154 have discrepancies ≥5.0 m.

The risk or uncertainty implied by the lack of collar survey or surveys not updated with the professional survey data is considered minor. The Turnagain collar database is considered suitable for this resource estimate but best practice is to use the most current and accurate survey measurements and topographic surface in a resource estimate.

HCNC should collect professional survey measurements for all holes, where practical, prior to commencing more advanced work, and should document those holes which are no longer able to be surveyed. Collar survey method should also be documented in the database.

Down-hole Surveys

The survey table contains 29 holes with a maximum down-hole survey depth of zero metre. These holes are assumed to be unsurveyed. There are 24 holes in the survey table with multiple records with the same azimuth which are assumed to be surveyed by acid etched tube methods. The remaining holes all have multiple records with varied down-hole azimuths and are assumed to have been surveyed by Reflex Maxibor methods.

Ninety-one holes with Maxibor down-hole surveys were used to examine down-hole deviation. Surveyed drill hole locations were compared to the projected locations assuming no deviation. Four of the holes showed deviation greater than 25 m at 300 m down-hole. Twelve of the 54 holes which have not been surveyed or lack down-hole azimuth measurements have total lengths greater than 300 m. These un-surveyed holes add risk to the location of samples used in the resource estimate but are spread throughout the resource area and are supported by nearby surveyed holes.

The risk or uncertainty implied by the lack of down-hole collar surveys data is considered minor. The Turnagain down-hole collar database is considered suitable for this resource estimate but best practice is to use the most current and accurate survey measurements.

HCNC should collect Maxibor down-hole surveys for as many holes as practical. Holes which are no longer accessible should be documented. The down-hole survey method should be documented in the drill hole database to allow for assessment of impact of data quality at the pre-feasibility or feasibility stage.

14.3 Sampling Method Review

The data most important to the resource model are the sulphide-nickel assays. The assay method employed by Acme Laboratories consists of a concentrated hydrogen peroxide plus ammonium acetate leaching solution that is believed to be selective at dissolving nickel from sulphide mineral species while leaving nickel in silicates undissolved (AC-Ni assay). This is a critical issue for deposits of this type because of the need to obtain accurate estimates of the nickel that can be recovered by established metallurgical methods (flotation of sulphide minerals). The mining industry does not yet have a widely recognized and well established method of achieving this. HCNC is pioneering the use of the AC-Ni method for resource estimation.

The resource estimate is reported in terms of nickel sulphide, a calculated value whose value is determined according to the following formulae:

- NiS = AC-Ni if Sulphur ≥0.2%
- NiS = 0 if Sulphur <0.2%

In cases where the sulphur result is less than 0.2% S, the AC-Ni result is assigned a value of zero. This is a precautionary step in case the AC-Ni assay result partially leaches nickel from some silicate minerals; this precaution may cause an underestimation of the grade of the nickel resource, particularly if the sulphur assays have a low bias. The underestimation could be large enough to materially impact the estimation of nickel grade, but limits the possibility that an overestimation of the grades of the nickel resource will occur. Until 2006, the sulphur assay method was the Leco furnace method; this is the industry standard method. Additional sulphur assays in the database, since 2004, have been reported from ICP analysis. From 2006, the Leco sulphur assays were discontinued, thus samples with reported sulphur values in 2006 are only from ICP readings of the multi-acid digestion. A discussion of the ICP versus Leco methods is presented in Section 14.4, and includes an impact of the analytical accuracy of the ICP readings.

A total of 17% of the composites used in the resource estimate (446 composites) were set to zero as a result of the sulphur value being less than 0.2% S. The green dunite unit (Unit 105), which is not considered to host sulphide mineralization, contained 141 of these samples. For Units 104 and 106, 11% of the composites were set to zero. AMEC's approach limits the potential overestimation of any metal content; however AMEC recommends that the potential upside should be assessed if it is determined that the AC-Ni analysis does not extract silicate Ni, and that the AC-Ni results can be used without the adjustment due to sulphur.

Other assay results do not materially impact the resource model. There are total digestion nickel results (TD-Ni) for all drill samples. These data currently consist of results from two different assay methods. Some historical assays were performed using an aqua regia digestion; this digestion dissolves most, but not all nickel. As performed, it obtains nickel results that are greater than AC-Ni results but less than multi-acid Ni results; the amount of difference varies depending upon the sample's mineralogy. The majority of TD-Ni results (assays since 2003) are based upon a multi-acid digestion that is believed to dissolve all nickel. The TD-Ni results provide an upper limit check on the AC-Ni results, because the TD-Ni result should equal or exceed the AC-Ni result for each sample.

14.4 QA/QC Review

Certified Reference Materials

HCNC included samples of CANMET certified reference materials UM2 and UM4 in its submission of samples to Acme Laboratories in Vancouver. These are the only reference materials presently available from CANMET (or, to AMEC's knowledge, from any other vendor of certified reference materials) that are certified for sulphide nickel rather than total nickel.

The certified values for sulphide nickel assigned by CANMET are limited to a sulphide nickel result that was obtained by multiple assays from a single laboratory; usually CANMET certified values are based upon the consensus of more than 20 independent assay laboratories. CANMET did not certify UM2 and UM4 for any other determinations, such as sulphur and total nickel. For this reason, HCNC contracted Smee and Associates Consulting in Vancouver to perform additional certifications on UM-2 and UM-4 (see Section 13.2).

The assay protocol for determining sulphide-nickel is different from the one used by Acme Laboratories. The CANMET procedure leached samples using a mixture of concentrated hydrogen peroxide and ascorbic acid. Although the intent (selectively leaching only the nickel-in-sulphide minerals) is the same for both the Acme and CANMET methods, the differences in leaching reagents and methods will likely result in some systematic differences in the results (Cameron, 1975).

The mean AC-Ni values obtained by Acme and the certified values are compared in Table 14-3.

The Acme results are about 10% lower than the CANMET-certified value of the higher-grade UM-2, and show close agreement for the lower-grade UM-4 certified material.

HCNC constructed control charts; these show more than 90% of Acme's results on UM-2 are between 0.24% and 0.28% AC-Ni, and more than 90% of Acme's results on UM-4 are between 0.180% and 0.205% AC-Ni.

Table 14-3: Sulphide Ni Comparison of Acme AC-Ni Means and CANMET Certified Values for UM2 and UM4

Drilling Campaign	UM-2		CANMET	Relative Difference
	Number	Acme Mean %AC-Ni	% sulphide- Ni	(%)
2004 DDH	57	0.26	0.29	-10
2005 DDH	52	0.26	0.29	-10
2006 DDH	72	0.25	0.29	-14
2006 Infill	53	0.27	0.29	-7
Overall	**234**	**0.26**	**0.29**	**-10**
Drilling Campaign	**UM-4**		**CANMET**	**Relative Difference**
	Number	**Acme Mean %AC-Ni**	**% sulphide- Ni**	**(%)**
2004 DDH	130	0.19	0.19	0
2005 DDH	85	0.20	0.19	5
2006 DDH	118	0.19	0.19	0
Overall	**333**	**0.193**	**0.19**	**2**

This is good performance for a selective leach determination. Shifts from one time frame to another can be seen, but the shifts are unlikely to be large enough to significantly impact local estimations of sulphide nickel grades.

Thus on the basis of the CANMET certified value, the Acme AC-Ni assay results slightly underestimate the sulphide-nickel content. For "total" results, most commercial laboratories can obtain mean values on CANMET base metal standards that agree within 10% of the certified value. However, selective leach assaying is much more difficult; based on similar analyses, this is good agreement for a selective leach method.

Given the nature of selective leach assays and the fact that the CANMET certified values are based upon a single laboratory, HCNC is including samples of UM-2 and UM-4 in a round robin presently being performed for new in-house standard reference materials made from rocks collected on the property. HCNC has obtained sulphide nickel assay results on UM-2 and UM-4 from two other laboratories at the effective date of this report. International Plasma Laboratories (IPL) followed Acme's method, after some experimentation with the leaching time. SGS followed a published method (Young, 1974) that was the basis of, but modified by, Acme. The comparison is shown in Table 14-4.

Acme results on both UM-2 and UM-4 are neither the highest of the four laboratories, nor are they the lowest.

Table 14-4: Comparison of Laboratory Results on CANMET Standards UM-2 and UM-4

Laboratory	Sulphide-Ni %	
	UM-2	UM-4
Canmet	0.29	0.19
Acme	0.26	0.19
IPL	0.26	0.20
SGS	0.23	0.17

Under contract to HCNC, Smee and Associates Consulting Ltd (Smee Associates) performed a round robin on CANMET standards UM-2 and UM-4 for total nickel, total sulphur (Leco), and other elements (Smee, undated). The round robin consisted of only five laboratories, including Acme. In the case of sulphur, there were results from only four laboratories because SGS Lakefield reported there was not sufficient sample. This is too few laboratories to provide a reliable estimate of the population mean of all assay laboratories, which is the goal of a round robin.

In the case of sulphur, Smee Associates did not certify a sulphur value for UM-4 because of the lack of adequate agreement between the four laboratories. The values for total nickel and sulphur are shown in Table 14-5. AMEC used the round robin raw assay data and agrees with the assignments made by Smee Associates. Smee Associates does not calculate the uncertainty in the certified value. The uncertainties in the mean values shown in the table were calculated by AMEC; however the calculated uncertainties are not very reliable because the round robin performed by Smee Associates has too few laboratories to obtain a reliable estimate of the uncertainty.

Table 14-5: Acme Performance on UM-2 and UM-4 for Total Nickel

Standard	Certified Total Ni %	Uncertainty of Mean (95% conf.)	Mean Acme Total Ni%		
			2004	2005	2006
UM-2	0.35	±0.02	0.36	0.36	0.35
UM-4	0.24	±0.02	0.25	0.24	0.23

HCNC compiled a portion of the sulphur results on UM-2 and UM-4 that were performed by Acme (the primary laboratory) and by ALS Chemex (the check laboratory). Unlike the preceding years, Acme did not perform Leco S determinations in 2006. Hence the 2006 ICP sulphur data are used, and are shown separately (Table 14-6). The Acme results indicate an 8% to 10% drop in average S grade for the 2006 Acme ICP data compared to the average of Acme's Leco S data.

Table 14-6: Sulphur Results on CANMET Standards UM-2 and UM-4

			ALS-CHEMEX LECO S%			ACME LECO (pre-2006) or ICP (2006) S%		
Name	**Best Value S%**	**Year(s)**	**N**	**Mean**	**95% CONF.**	**N**	**Mean**	**95% CONF.**
UM-2	1.03	pre-2006	15	1.02	0.02	15	1.06	0.08
		2006	11	0.93	0.06	11	0.96	0.08
UM-4	0.50	pre-2006	16	0.48	0.01	16	0.51	0.01
		2006	13	0.47	0.03	13	0.47	0.02

Check Assays

ALS Chemex performed check assays on about 5% of the Turnagain Project sample pulps. Total nickel was determined using a four-acid digestion and ICP reading. Numerous other elements were determined by ICP from the same acid digest. Sulphur was determined using Leco furnace. Sulphide-nickel assays were not performed because ALS Chemex does not perform such an analysis.

A check assay program has been established using IPL. Some check assay results for samples assayed by Acme in 2007 have been completed and the plot is shown in Figure 14-1. There are no obvious outliers, the correlation is good (R^2 = 0.979) and there is no measurable bias (slope equals 1.00).

IPL performed assays of 2005 metallurgical samples collected from Turnagain Project sample rejects to determine measured head grades of samples undergoing metallurgical testwork. The method they employed at that time was slightly different; it used the same reagents but had a shorter agitation time (a few minutes) combined with a long settling time (overnight). These measured head grades can be compared to length-weighted average grades of the Acme AC-Ni results. The comparison (Figure 14-2) shows reasonable agreement. The slope of the linear fit is about 1.08, offset by a negative y-intercept of about -0.02% AC-Ni. This is good agreement.

The results of the Leco sulphur check assays show time periods of large disagreement. Assay certificates were placed in order by Acme certificate number, reflecting the time sequence that samples were assayed by Acme. Acme results having Chemex check results were then grouped so that each group would contain at least five results, but results in the same Acme certificate were always kept together, regardless of the number of results. Grouping the data reduces the noise. The Acme mean was then divided by the Chemex mean to obtain a ratio that provides an estimate of the bias. Results are shown in Figure 14-3.

Figure 14-1: IPL Check Assays for Sulphide-Nickel

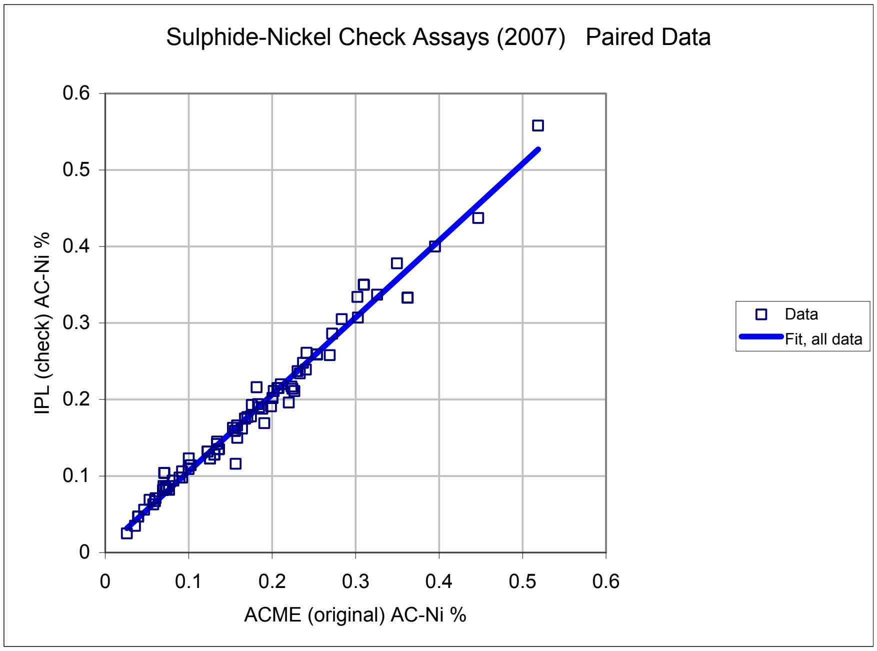
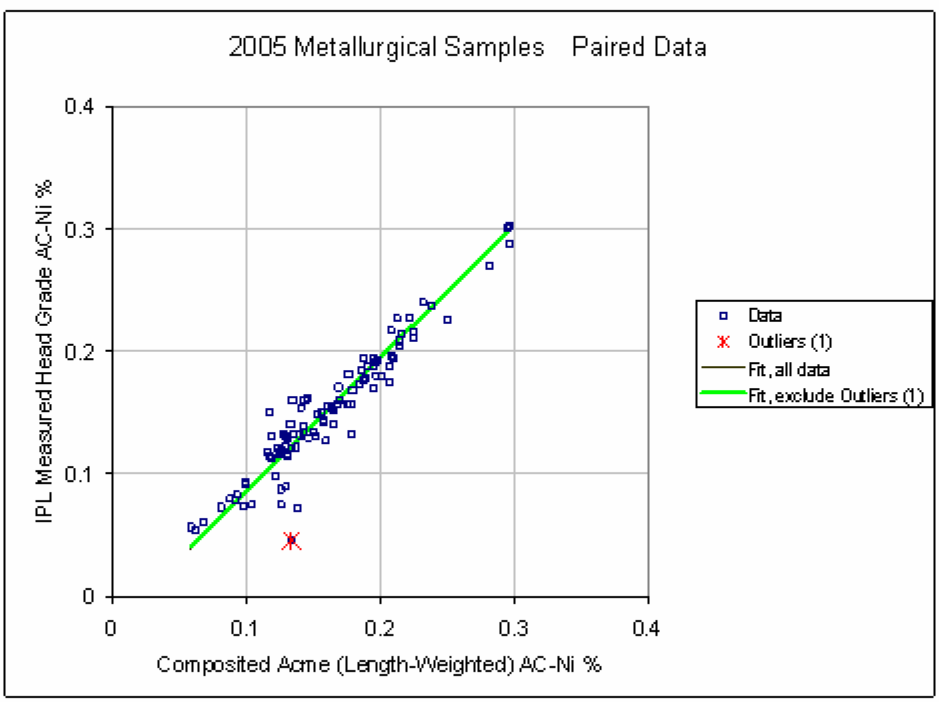
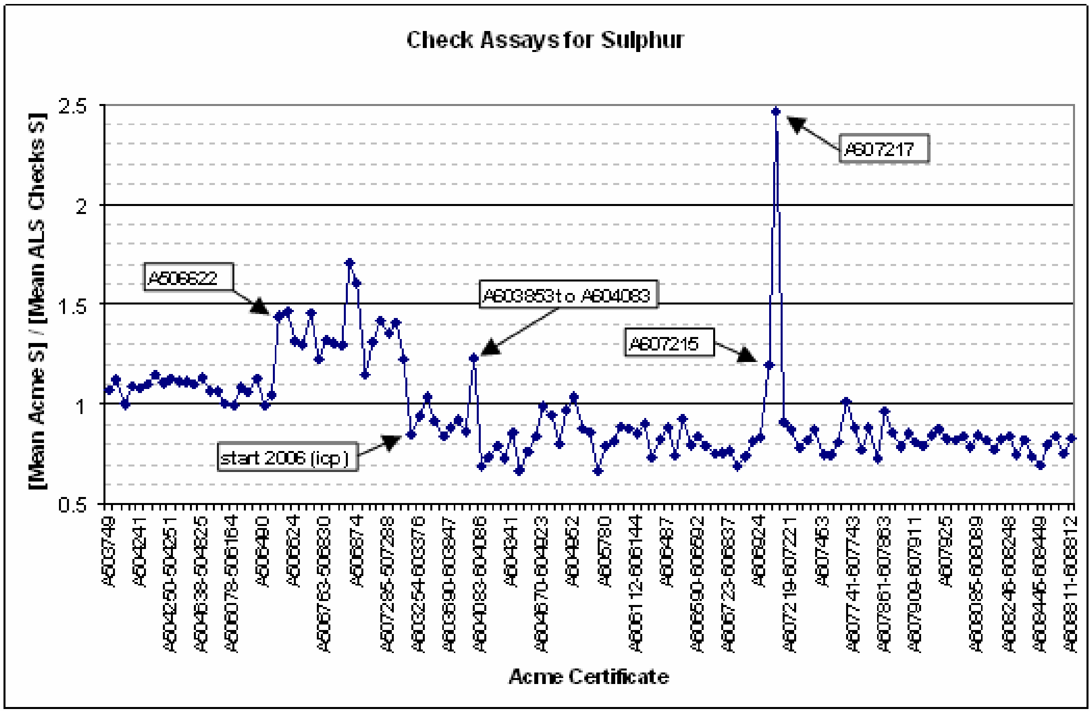


Figure 14-2: Comparison of Acme and IPL Metallurgical Sample AC-Ni Head Grades



Note: Line of fit for all data is present, but over-plotted by the line of fit that excludes outliers.

Figure 14-3: Chemex Leco Sulphur Check Assays over Time



Validation of AC-Ni Using Metallurgical Test Recovery

The intent of using a selective leach method for calculating the nickel resource is to more accurately determine the recoverable nickel. Different approaches of predicting recoverable nickel were undertaken by AMEC (Long, 2007a).

Using the AC-Ni data in combination with total nickel and rock type to determine a “background” level of silicate Ni for each rock type was not useful for predicting metallurgical recovery.

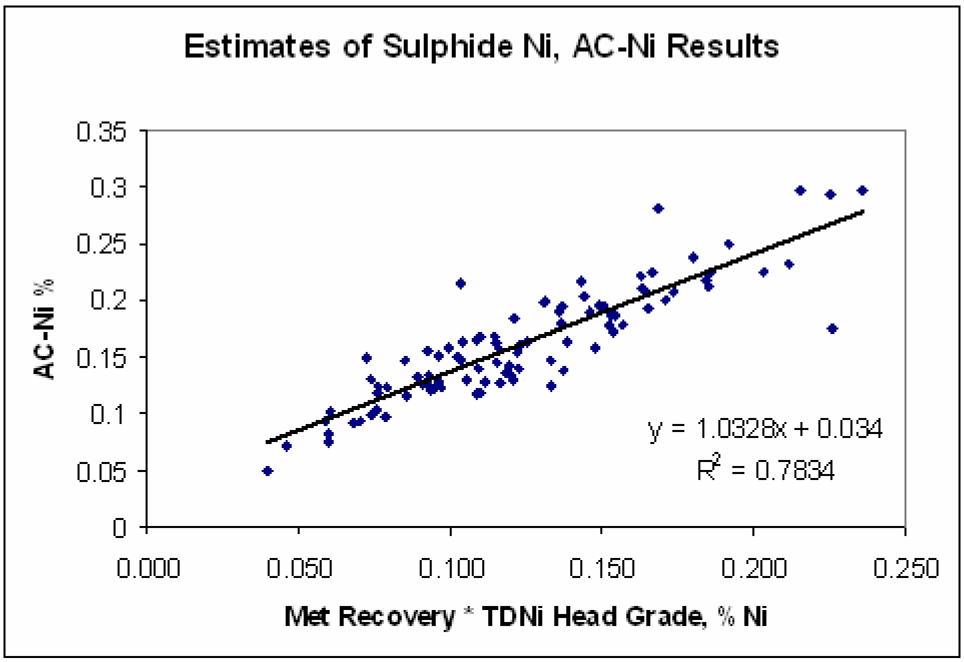
Francois-Bongarcon (2007) derived a method of determining a recovery factor by taking the logarithm of measured sulphur head grades and calculating a linear regression against metallurgical recovery of total nickel. The method was evaluated by calculating the logarithm of Acme sulphur results for each sample interval, using Francois-Bongarcon’s regression equation to obtain a recovery factor, and applying the recovery factor to Acme total nickel result to obtain the recoverable (sulphide)-nickel grade.

The length-weighted average grade was then calculated for each metallurgical sample and compared to the grade obtained by multiplying the metallurgical test recovery factor by the measured total nickel head grade. This approach yielded reasonably good correlation (R2 = 0.69). However the accuracy issue of the Acme sulphur analyses, particularly for

2006 data which was based only on ICP measurements, and not on the industry-standard LECO method, prevent this approach from being employed (Long, 2007a).

The best correlation is obtained using the AC-Ni assay results (Figure 14-4). Applying a metallurgical recovery factor to the AC-Ni results better predicts metallurgical recovery as obtained in the metallurgical test results than can be obtained from the total Ni or other assay results, even used in combination with rock and alteration type (Long, 2007b).

Figure 14-4: Comparison of AC-Ni to Sulphide-Ni Calculated from Metallurgical Test Data



Bulk Density

Acme checked the density of coarse rejects that had sample intervals encompassing a sample that HCNC measured for density. The sample interval of the coarse reject is typically much longer than the sample HCNC measured (Table 14-7). The Acme density results agree within 5% for all rock types except for the one sample of hornblendite, where the HCNC result is about 10% higher than the Acme result. Hornblendite is not common.

The checks by Acme validate the field measurements by HCNC. A review of the bulk density data indicates that this data has been collected in a manner that will allow the measurements to be used in resource estimation.

Table 14-7: Bulk Density Checks, by Rock Type

Rock Type	No	Mean Density gm/cm^3 HCNC (site)	Mean Density gm/cm^3 Acme	HCNC / Acme
Hornblende-Diorite	1	3.01	3.06	0.98
Altered Dunite	1	2.72	2.74	0.99
Green Dunite	2	3.08	3.08	1.00
Olivine Clinopyroxenite	1	3.24	3.21	1.01
Serpentinized Dunite	8	2.86	2.82	1.01
Wehrlite	2	2.93	2.89	1.02
Magnetic Clinopyroxenite	1	3.26	3.21	1.02
Dunite	8	3.11	3.01	1.03
Hornblendite	1	3.24	2.95	1.10

Note: HCNC = Hard Creek Nickel

14.5 Conclusions

AMEC considers that the verification conducted on the deposit model and geological and mineralization data has shown that the geological understanding of the deposits is sufficient to support resource estimation.

Following review of the sampling and analytical methods, database and an independent sampling program, AMEC considers that the data collection and data base are sufficiently precise and accurate to support resource estimation.

15.0 ADJACENT PROPERTIES

There are no immediately adjacent properties that host a similar mineralization style to the Turnagain Property.

16.0 MINERAL PROCESSING AND METALLURGICAL TESTING

16.1 Introduction

The plan is to process the Turnagain mineralization through an on-site concentrator and hydrometallurgical process facility that will produce nickel, cobalt and copper precipitation products. The planned nominal milling rate is 50,000 t/d. ROM open pit mineralization will be crushed in a gyratory crusher, followed by fine crushing, ball milling and concentration by flotation. The nickel-cobalt-copper concentrate be finely reground, pressure oxidized in an autoclave, and leaching of the copper, cobalt and nickel will follow with precipitation of copper, cobalt and nickel occurring sequentially.

Three products are planned to be produced; copper sulphide, nickel hydroxide, and cobalt hydroxide. These products will be trucked to the Port of Stewart for shipment to end users. Saleable product will be paid for on the basis of 85% for nickel contained in nickel hydroxide and 80% for cobalt in cobalt hydroxide; 80% of the copper is considered to be payable. The copper sulphide will be sent to a smelter for processing.

The tailings from the process will be impounded in a tailings pond; water will be reclaimed from the tailings pond and re-used in the process.

It should be noted that there are a variety of issues related to the production of saleable products by concentration and subsequent hydrometallurgical processing that represent a technical and financial risk to the project and will have to be addressed in future work.

A summary of the metallurgical testing up to and including 2004 is reported in Carter (2005). Further laboratory bench scale testing was performed by Process Research Associates Ltd., of Richmond, BC from 2005 to 2007. The majority of testwork considered is from 2005 and 2006 and took place on drill core produced in 2005. Drill core produced in 2006 was primarily on areas outside the proposed pit area and although examined, testwork on this material does not contribute to the design. Testwork was also performed at SGS-Lakefield in 2007 on material to produce concentrate for column cleaner work and for further hydrometallurgical testing. In both the previous and latest testwork, the supervision of the work has been done by F. Wright, P. Eng.

Progress has been made in relating geology to the metallurgical results. With the definition of the material into domains as defined by the new geological model, more insight has been gained into the relationship of the mineralization to test variables.

16.2 Metallurgical Testwork (2004 – 2006)

16.2.1 General

Mineral processing testwork conducted by Process Research Associates (PRA) under the supervision of F. Wright, P. Eng., in 2004 to 2006, indicates that the Turnagain deposit responds relatively well to processing by conventional grinding and flotation to produce a nickel-cobalt concentrate. It should be noted that the basis of the PA design is primarily the 2005 and 2006 work program, PRA0506908.

The PRA program of flowsheet development studies and metallurgical response was based on the following series of objectives:

- Study the mineral composition and hardness characteristics of mineralization from the Horsetrail zone.
- Devise a set of common treatment parameters for processing the Horsetrail mineralization zone including flotation feed sizing, and developing a reagent regime and flowsheet configuration.
- Assess the concentrate quality with regard to mineral composition and minor element concentrations.

The results of this testwork were used for the design of the plant in this PA. The results indicate nickel sulphide flotation at a relatively coarse primary grind (nominal 150 µm), with cleaner flotation to produce a concentrate. This concentrate is generally predicted to average 8% to 12% nickel at a nickel sulphide recovery between 60.0% and 80.0%, with cobalt and copper recovery at approximately 70% and 50% respectively. A laboratory analysis of the test concentrate products indicates that level of MgO content is of concern and needs to be addressed, either by lowering the level of MgO to produce a saleable concentrate or alternatively to produce saleable products by a hydrometallurgical route.

The use of flotation columns was attempted in 2007 at SGS-Lakefield. This work indicated that there was potential to upgrade the material but that substantial work would be necessary to validate this approach.

The hydrometallurgical route in processing the concentrate has been subject of two test programs, one at Cominco Engineering Services Ltd and the other at SGS-Lakefield. Preliminary hydrometallurgical testwork was conducted at Cominco Engineering Services Ltd (CESL) in 1999 under the supervision of F. Wright, P. Eng. The hydrometallurgical flowsheet used in this assessment is based on that testwork and has been devised by Dr. D. Dreisinger, P. Eng. The hydrometallurgical testwork indicated that high extractions of the payable elements could be produced by leaching of the flotation concentrate. In this flowsheet, the leaching is followed by precipitation of valuable elements as hydroxides (for the nickel and cobalt) and as sulphide (for the copper). It should be noted that the recovery portion of the flowsheet was not tested at that time.

The hydrometallurgical flowsheet would have the mill concentrate undergo ultra-fine regrinding followed by sulphur oxidation in a high pressure autoclave. The material is then discharged from the autoclave and further leached in a reactor. Discharge from the reactor undergoes filtration with the solids being discharged to a tailings impoundment area and the solution undergoing a series of precipitation steps to produce a nickel hydroxide product, a cobalt hydroxide product and a copper sulphide product. A further solids discharge product containing magnesium leached in the process is discharged at the end of the process together with the neutralized solution.

16.2.2 Description of Mineralization

The geology of the Turnagain ultramafic nickel deposit within the proposed pit walls (currently the Horsetrail Zone is 100% of the resource used in this study) is expected to comprise primarily of the mineralized intrusive that hosts the economic mineralization. The economic minerals are nickel, copper and cobalt sulphides. From a mineralogical viewpoint, the samples examined in the metallurgical program are typical of mineralization found in ultramafic nickel deposits with recoverable nickel concentrated in the sulphides and background nickel present as silicates. Metallurgical performance has been evaluated on the basis of nickel sulphide recovery and grade. It is the nickel sulphides that contain economically recoverable nickel in the Turnagain deposit. To a large degree, the metallurgical response of the mineralization is influenced by the talc content in the mineralization which influences critical reagent consumptions and concentrate quality.

With the definition of the resource into different mineralization types as defined by domain, the opportunity was provided to further determine the response of the mineralization in producing a concentrate, responding to grind, etc. The principal domain groups looked at comprised:

- 101 – clinopyroxenes, approximately 2% of the resource
- 102 – olivine clinopyroxenes, approximately 5% of the resource
- 104 – dunites and wehrlites, approximately 35% of the resource
- 106 – serpentinized dunites and wehrlites, approximately 56% of the resource

For the purposes of the resource definition, the following recoveries have been assigned to the various mineralization types (Table 16-1).

Resources with low nickel sulphide grades (less than 0.1% Ni as nickel sulphide) have been allotted a recovery of 0%. Despite indications that recovery would be possible from very low grade mineralization, insufficient metallurgical testwork or industry practice exists to support assigning a recovery to the very low-grade material. Further testwork is necessary if this assumption is to change.

Table 16-1: Nickel Recovery and Grade by Domain

	Grade (in terms of % NiS)		
Mineralization Type – Domain	**0.10 to 0.15**	**0.15 to 0.20**	**>0.20**
101	80	80	80
102	75	75	75
104	70	80	80
106	60	70	80

Note: Insufficient information exists to define the response of domains 103 and 105. The extent of these domains however is not significant, providing only 2% to 3% of the total resource.

It should also be noted that the testwork indicated that the pyroxenite groups, contained in domains 101 and 102, are difficult to concentrate into a higher grade concentrate. This would limit the ability to produce a concentrate of sufficient nickel grade that would be attractive to a smelter. Instead a lower-grade concentrate (<6% Ni) would be produced which could be fed to an onsite hydrometallurgical facility. In contrast, the indications are that a flotation concentrate could be produced from Domains 104 and 106 which would be suitable either for a smelter or for a hydrometallurgical facility.

The metallurgy and preliminary assessment mill design is primarily based on Domains 104 and 106 which represent 91% of the resource.

16.2.3 Density

Specific gravity (SG) measurements were carried out on 503 core samples from the current resource area. The average value of 3.09 was used for the tonnage estimations in this study.

Specific gravity of core samples was measured in the field by the immersion method. A piece of whole core up to 50 cm in length was weighed in air and in water and the specific gravity calculated as follows:

SG = weight in air/(weight in air – weight in water)

As part of the metallurgical test program, PRA measured SG using the pycnometric method with -10 Tyler mesh assay rejects. Their results were within 5% of SG determinations measured by Acme Laboratory using a similar method.

16.2.4 Metallurgical Samples

Sample Origin

A metallurgical test program was conducted at PRA (Process Research Associates) based on metallurgical samples defined by HCNC. The majority of the material used for the testing programs performed in 2005 and 2006 comes from the 2004 and 2005 exploration programs. There was additional flotation testwork performed on drill core produced from 2006. However as these samples were almost exclusively outside the proposed pit limits, these samples were not incorporated into the analysis. Relevant tests were considered to be those based on samples from within the forecast pit area.

The relevant tests performed were divided by domain type into the following groups:

- 101 – clinopyroxenes
- 102 – olivine clinopyroxenes
- 104 – dunites and wehrlites
- 106 – serpentinized dunites and wehrlites

Sample Preparation

Samples were received at PRA as -6 mesh assay rejects from Acme. The sample weights in individual bags were recorded on the sample-receiving log sheet, which also included the sample identification numbers and characteristics. Samples were composited as per HCNC's instructions. The individual samples for each composite were blended three times before riffling out into 2 kg test charges.

Splits from each composite were taken for individual head assays.

Analytical Procedures (Metallurgical Samples)

A four acid digestion was performed on each sample prior to the quantitative determination of total nickel. Samples were leached in a mixture of hydrogen peroxide and ammonium citrate for sulphide nickel determination.

Multiple elements were assayed by Inductively Couple Plasma (ICP) scans with acid digestion. The International Plasma Laboratories (IPL, ISO9002 certified) conducted all assays with appropriate quality control procedures including use of duplicate assays.

16.2.5 Mineralogy

Pentlandite, pyrrhotite, and chalcopyrite, and to a lesser extent, bornite are the principal sulphide minerals of the Turnagain deposit. A variety of secondary nickel mineralization also occurs to a minor extent. Little information is available on the mode of occurrence of the cobalt.

The non-sulphide gangue minerals include a mixture of olivine, chromite, pyroxene, serpentine, and talc.

16.2.6 Grinding

Testwork

Bond Work Index tests have been performed on the material. These produced an average Bond Ball Work Index of approximately 20 kWh/t. This value was used as the basis of design for the comminution circuit. Grinding work performed in the preparation of material for flotation indicates a fair amount of variation in the grindability of the samples. It is recommended to further examine this in upcoming testwork.

Primary Grind and Mineral Liberation

A primary grind of 150 µm K_{80} was determined as appropriate for the liberation size used for this PA. Testwork indicated reasonable recoveries at this particle size with the liberation of the majority of sulphides and non-sulphide gangue minerals. AMEC analysis of the data by domain type indicated that metal recoveries for Domains 101 and 102 are moderately sensitive to grind. Metal recoveries for Domains 104 and 106, which form the majority of the resource, are more sensitive to grind.

It is recommended that in future testwork, more emphasis be given to flotation liberation size as defined by the domain types. As grinding represents a major portion of the processing capital and operating costs, the optimized particle grind size will impact the potential development of the deposit.

16.2.7 Flotation Testwork

Bench Scale Flotation (2004 – 2006)

This subsection presents a summary of testwork performed on drill core samples produced in the years 2004 and 2005. Emphasis was placed upon the examination of testwork performed in 2005 within the confines of the potential pit area. The 2006 testwork was examined but not used in the analysis as it did not correspond to the potential pit area.

A total of 110 bench tests and two locked cycle tests from the 2006 program were provided by HCNC as indicative of the metallurgy found in the Turnagain deposit. A variety of factors were examined in these tests including pulp density, reagent dosage, flotation residence time and grind size effect.

The testwork indicated that it was possible to produce a rougher concentrate that could be further upgraded to produce a sulphide concentrate with grades ranging from 5% to over 15% nickel. The testwork on the Turnagain mineralization indicates a fairly slow floating material (55 minutes) with a high mass pull of 15% to 20% required. The PA design has utilized an overall residence time of 50 minutes in rougher flotation together with a mass pull of 15% for preliminary design purposes. Insufficient work exists on the flotation kinetics to provide a definitive residence time and more work in this area is necessary.

Variation in the concentrate grade was attributed to be more dependent on other factors than simply head grade. F. Wright, P. Eng., consulting metallurgist on testwork, indicates that concentrate grade could be related to the sulphur content, and degree of sulphur dissemination in the gangue, as well as the ratio of pyrrhotite to pentlandite. This assessment is preliminary and more work is required in this area.

Principal flotation reagents consisted of potassium amyl xanthate as a collector, copper sulphate as an activator, MIBC as a frother, and CMC as a dispersant. Significant reductions in reagent consumption have been achieved in ongoing testwork.

Evaluation of nickel recovery in 2006 indicated that for material ranging from 0.1% to 0.5% nickel sulphide, recovery would vary between 75% to 82%. Using the 2007 AMEC geological model of 2007, it was possible to partition the samples into domain groups allowing analysis of metallurgical performance.

Recoveries were estimated by plotting rougher recovery values against the final concentrate grades produced in the same batch cleaner tests for each domain. This provides a preliminary indication of the possible recovery-grade relationship. It does not account for losses occurring in the cleaning of the concentrate material for upgrading it. It is essential in future flotation testwork to quantify the level of losses incurred in upgrading the concentrate. Typically for deposits of this nature, initial testwork producing higher grade concentrates at high recoveries from low grade feeds is problematic. The locked cycle work done in 2006 indicates that this problem can be solved. However until there is more metallurgical testwork, this area represents a significant potential risk to the project.

Several stages of cleaning will be necessary to produce a concentrate adequate either as a saleable material or as a material to be treated in the hydrometallurgical process. The initial cleaning in the first and second cleaners will be to reject non-sulphide gangue. Subsequent cleaning will be to reject both non-sulphide and sulphide gangue. Although regrinding has not been utilized in the testwork, an allowance has been provided in the design to add capacity after the second cleaning stage prior to the cleaner columns. An allowance, based on a design weight of 5% of the original plant feed, has been included. Continued testwork is necessary to develop a robust grade-recovery curve and to optimize cleaning, and regrind particle size selection.

The rougher recovery was also plotted against the feed grade in the 4 principal domains. This indicated that percent recovery in Domains 101 and 102, which are the clinopyroxenes, either stays constant with feed grade or declines. These two domains, however, comprise less than 5% of the resource. In Domains 104 and 106 (approximately 90% or more of the resource), a more typical relationship is seen where percent recovery increases with increasing feed grade.

Plotting the concentrate grade against feed grade indicates that in general, the concentrate grade will increase as the feed grade increases. Two points should be noted: firstly that the scatter of the data in this analysis for Domains 104 and 106 indicates that this response is conditional to flotation operating practice, and secondly that a possible maximum concentrate feed grade of approximately 8% to 9% Ni is possible in Domains 101 and 102.

Recovery plotted against the grind size indicates that the P_{80}=150 µm target in the design should be suitable for Domains 101, 102 and 106. Domain 104 shows more sensitivity to grind size indicating that it may be necessary to achieve a size distribution closer to P_{80}=100 µm. Further testwork must continue in this area to delineate the sensitivity of the metal recovery to particle size grind.

The analysis conducted by AMEC is based on a summary of data put together by HCNC and produced from testwork by PRA of Vancouver. The flotation test procedure has been witnessed by I.A. Lipiec, P.Eng. (AMEC), and follows standard testing protocols. Random data sheets have been examined and confirm the summary data as provided.

Locked Cycle Confirmation Testing (2006)

Locked cycle testwork was performed on two composites, MG 06-1 and MG 06-2, with the two samples being essentially the same. The results indicate that at this level of process development, recovery is a function of concentrate grade. An analysis of the concentrate was performed on material from both tests and the results indicate high levels of MgO when attempting to produce a high grade concentrate (Table 16-2, FLC 2). This suggests that a hydrometallurgical approach treating a lower-grade concentrate may be required to maximize the economic value of the deposit. It should be noted, however, that process development is still at a fairly early stage for this deposit and more work is recommended to improve grade recovery performance to produce a saleable concentrate for smelting as an alternative.

Table 16-2: Nickel Concentrate Analysis (2006)

Element/Compound	Symbol/ Formula	Unit	Locked Cycle Test #FLC 1	Locked Cycle Test #FLC 2
Aluminium	Al	ppm	884	464
Antimony	Sb	ppm	<5	<5
Arsenic	As	ppm	58	67
Barium	Ba	ppm	6	<2
Bismuth	Bi	ppm	<2	<2
Cadmium	Cd	ppm	<0.2	<0.2
Calcium	Ca	ppm	2,249	1,332
Chromium	Cr	ppm	2,006	981
Cobalt	Co	ppm	4,417	7,572
Copper	Cu	ppm	6,195	6,965
Iron	Fe	ppm	390,382	412,463
Lead	Pb	ppm	<2	<2
Magnesium	Mg	ppm	102,267	57,583
Manganese	Mn	ppm	438	262
Mercury	Hg	ppm	<3	<3
Molybdenum	Mo	ppm	53	749
Nickel	Ni	ppm	90,414	155,805
Phosphorous	P	ppm	<100	<100
Potassium	K	ppm	<100	<100
Silver	Ag	ppm	7	14
Sodium	Na	ppm	<100	<100
Selenium	Sr	ppm	7	4
Titanium	Ti	ppm	<100	<100
Tungsten	W	ppm	7	<5
Vanadium	V	ppm	35	29
Zinc	Zn	ppm	116	147
Zirconium	Zr	ppm	7	7

16.3 Pilot Plant (SGS-Lakefield, 2007)

In 2007, pilot plant flotation work was performed at SGS-Lakefield (Peterborough, Canada) with two principal goals. The first goal was to produce enough material for subsequent hydrometallurgical testing and concentrate characterization. The second purpose was to provide some material to conduct cleaner testwork using a flotation column. The recovery numbers seen in this work were lower than what has been utilized for design (Table 16-3). This was expected, however since the sulphide minerals in the material selected for the work may have oxidized while being stored at room temperature conditions for over two years. Such oxidation would reduce the material's ability to float as compared to freshly mined material.

Table 16-3: Nickel Concentrate Analysis (2007)

Element/Compound	Symbol/Formula	Unit	Pilot Plant
Aluminium	Al	ppm	2,300
Antimony	Sb	ppm	<1
Arsenic	As	ppm	33
Barium	Ba	ppm	32
Bismuth	Bi	ppm	20
Cadmium	Cd	ppm	2
Calcium	Ca	ppm	6,100
Chromium	Cr	ppm	1,200
Cobalt	Co	ppm	1,900
Copper	Cu	ppm	1,800
Iron	Fe	ppm	Not available
Lead	Pb	ppm	410
Magnesium	Mg	ppm	190,000
Manganese	Mn	ppm	750
Mercury	Hg	ppm	Not available
Molybdenum	Mo	ppm	18
Nickel	Ni	ppm	45,000
Phosphorous	P	ppm	41
Potassium	K	ppm	220
Silver	Ag	ppm	5
Sodium	Na	ppm	220
Selenium	Sr	ppm	30
Titanium	Ti	ppm	30
Tungsten	W	ppm	Not available
Vanadium	V	ppm	24
Zinc	Zn	ppm	550
Zirconium	Zr	ppm	Not available

In the column cleaning testwork performed, potential for reducing the MgO content in the concentrate was shown to be feasible. The test indicated that the concentrate grade was impacted by the level of residence time in the unit, wash water applied, and froth depth. Further work is deemed necessary to optimize results for recovery and grade performance. Indicated recoveries in the cleaner flotation program produced lower recoveries than utilized for design. This lower recovery is suspected to be caused by a combination of material oxidation and the test parameters used.

16.3.1 Concentrate Dewatering

Settling and filtration tests were not performed on concentrates. The basis of the PA design work was based on information from similar projects.

16.3.2 Tailings Characterization

Neutralization potential tests were carried out on material from the 2005 drilling program and test results indicate that the vast majority of samples would appear to be acid consuming rather than acid generating. This in combination with the relatively coarse grind of the flotation feed would suggest that the tailings material would be suitable for the construction of tailings dams as is currently practiced in the industry. This assumption requires further testwork and validation.

16.4 Hydrometallurgical Testwork

Only limited work has been performed on the hydrometallurgical processing on concentrate from the Turnagain property.

Testwork was conducted in 1999 and was performed on material (detailed information on the source of this sample has not been provided to AMEC) provided by Bren-Mar Resources (under the supervision of F. Wright, P. Eng.) to Cominco Engineering Services Ltd in Vancouver, B.C. Pressure oxidation leaching of this material indicated high extractions for nickel (approximately 97%) and cobalt (approximately 93%) as shown in Table 16-4. The condition under which this work was performed is similar but slightly different from that utilized for the process design in the AMEC report. The design approach followed by AMEC was based on recommendations by HCNC's hydrometallurgical consultant, Dr. D. Dreisinger, P. Eng.

Further work was performed in 2007 and confirms, on a preliminary basis, the extractions seen in the 1999 work. AMEC has considered this to be a positive development. It should be noted that AMEC is relying on the hydrometallurgical approach as recommended by D. Dreisinger.

Table 16-4: Turnagain Pressure Oxidation Tests Summary (CESL, 1999)

Test #	Nickel Extraction	Cobalt Extraction
1405	91.9	93.0
1406	94.3	92.8
1409	96.7	NA
1426	98.2	NA
1427	97.2	92.9
1430	95.6	NA
1431	95.5	NA
1433	96.8	NA

In 2006, AMEC modeled this recommended approach utilizing METSIM to provide reagent consumptions and recovery estimates. The level of work done in the area hydrometallurgy is adequate for a preliminary assessment but would have to be greatly expanded prior to pre-feasibility or feasibility work. In particular, reagent consumption, metal recovery by precipitation, and solid-liquid separation would all have to be examined closely.

In 2007, further work was performed on the pilot plant concentrate produced the same year. Two leaching methodologies were examined in this program: Total Pressure Oxidation (Total POX) and the low temperature (110°C) pressure leach process. The objective was to test the suitability of the leach processes for the flotation concentrate.

The concentrate head sample used for the test assayed: 4.5% Ni, 0.23% Al, 0.19% Co, 0.18% Cu, 20% Fe, 19% Mg, and 9.48% sulphur.

Both leaching processes worked well for the concentrate. A nickel extraction of 97.6% was achieved with Total POX and 98.4% with the low temperature pressure leach process. Cobalt extraction was 97% and 98% respectively while for copper extraction it was 96% and 95%. The lower temperature process leached less of the non-payable components within the concentrate.

One area which was flagged for further study was the difficulty in filtration which occurred with the residues.

Substantial work is required to optimize the leach process, subsequent solid liquid separations and downstream metal recovery.

16.5 Metallurgy and Process Design

This section describes the planned on-site processing of Turnagain mineralization using a concentrator and hydrometallurgical facility to produce separate nickel and cobalt hydroxide products that are intended to be trucked and shipped out of the province through the Port of Stewart, for sale to end users. In addition, a separate copper sulphide product can be sold to a copper smelter. Processing is planned to be based on a conventional nickel sulphide flotation flowsheet to produce a sulphide concentrate followed by a hydrometallurgical process to convert nickel and cobalt from its sulphide forms to a hydroxide form.

The mineralization determining the design characteristics of the concentrator is the dunite and wehrlite material which comprises approximately 90% of the overall resource within the preliminary pit. Material from this zone can produced by open pit mining methods, mixed and processed at a rate of 50,000 t/d.

Design Basis – Mineralization Types

The proposed mining operation is a conventional shovel and truck open pit mine feeding a 50,000 t/d processing plant using standard mineral flotation technology. Mining and processing of the deposit is proposed to begin with a starter pit. The pit is designed to expand in phased pushbacks until the ultimate pit limits are reached. Mineralized material is hauled to a primary crusher located near the southwest pit rim of the Main Pit.

The mine plan has been scheduled to maximize the production of high-grade material, during the first five years, to minimize the capital payback period.

The metallurgical response of the deposit is highly influenced by the talc content in the mineralization.

A detailed mass balance covering all streams within the process plant has been prepared based on daily throughput requirements, plant availability, and the design feed grades for nickel. The nominal mass balance representing the expected flows of solids, water, and slurry when processing mineralization at the average Year 1 to 5 feed grades of 0.195% Ni is provided in the PA.

Mill production, mineralization characteristics, grades, and recovery estimates by milling period are summarized in Table 16-5.

Table 16-5: Mill Production, and Mineralization Characteristics

Description/Phase (year)	Unit	Year 1 – 5	Year 6 – 10	Year 11 – 15	Year 16 – 29	LOM
Production						
Operating Days/Year	(d)	360	360	360	360	360
Tonnes Milled/Calendar Day	(t/d)	50,000	50,000	50,000	50,000	50,000
Tonnes Milled/Year	(kt/a)	18,000	18,000	18,000	18,000	18,000
Mineralization Characteristics						
Mineralization Specific Gravity	(SG)	3.09	3.09	3.09	3.09	3.09
Mineralization Bulk Density	(t/m^3)	1.8	1.8	1.8	1.8	1.8
Mineralization Moisture Content	(%)	2.0	2.0	2.0	2.0	2.0
Ball Mill (average)	(kWh/t)	20.0	20.0	20.0	20.0	20.0

16.5.1 Projected Metallurgy and Metallurgical Balance

A summary of the projected metallurgical recoveries and metal balance is provided in Table 16-6. The recoveries of nickel to the sulphide concentrate from any particular domain and feed grade ranged from 60% to 80% while cobalt and copper recoveries, based on limited data, are assumed to be 65% and 50% respectively in the concentrator.

It should be noted that all concentrator and hydrometallurgical recoveries are based on recovered nickel sulphide rather than total nickel. The portion of the nickel not present as nickel sulphide is generally present as nickel silicates and virtually none of this material is recoverable to any significant extent to a saleable product.

Table 16-6: Overall Metallurgical Recoveries

Description/Phase (year)	Units	Year 1 – 5	Year 6 – 10	Year 11 – 15	Year 16 – 29	LOM
Mill Feed Grade and Recovery						
Mill Feed Grade						
Nickel	(%)	0.198	0.175	0.151	0.154	0.166
Cobalt [1]	(%)	0.011	0.011	0.011	0.011	0.011
Copper [1]	(%)	0.027	0.027	0.027	0.027	0.027
Recoveries (Concentrator)						
Nickel	(%)	69.8	70.1	71.6	70.2	70.3
Cobalt [2]	(%)	70.0	70.0	70.0	70.0	70.0
Copper [2]	(%)	50.0	50.0	50.0	50.0	50.0
Recoveries (Hydromet)						
Nickel	(%)	96.0	96.0	96.0	96.0	96.0
Cobalt [2]	(%)	96.0	96.0	96.0	96.0	96.0
Copper [2]	(%)	96.0	96.0	96.0	96.0	96.0
Cumulative Production Recoveries						
Description/Phase (year)	**Units**	**Year 1 – 5**	**Year 6 – 10**	**Year 11 – 15**	**Year 16 – 29**	**LOM**
Ni Product Production	(%)	67.0	67.3	68.7	67.4	67.5
Co Product Production	(%)	67.2	67.2	67.2	67.2	67.2
Cu Product Production	(%)	48.0	48.0	48.0	48.0	48.0

Note:
1. Because of erratic grades of cobalt, the average assay of cobalt has been used in the mill feed grade rather compiling it from the block model. This assumption was subsequently checked and found to be in reasonable agreement with the block model.
2. Recovery data on cobalt is limited so a 70% recovery has been identified as appropriate for this level of study. Copper recovery was found to be lower and has been set at 50% in the concentrator. Both cobalt and copper are assumed to have the same recovery as nickel in the hydrometallurgical facility.

The recoveries in the hydrometallurgical plant were estimated to be 95% on the basis of preliminary hydrometallurgical work performed on a similar but different flowsheet to that incorporated in the PA.

The impact of the payable factor for the nickel is assessed in the financial analysis section.

During the first few years of mine operation, the effect on recovery of storing low-grade mineralization for later processing towards the end of the mine life should be investigated.

16.5.2 Process Description

Process Facilities

The Turnagain material is planned to be processed through an on-site concentrator and hydrometallurgical process facility to produce nickel, cobalt and copper precipitation products. The nominal milling rate is 50,000 t/d. The planned layout is shown in Figure 16-1.

ROM open pit material will be crushed in a gyratory crusher. The crushed material will be processed by means of a fine crushing circuit in combination with ball mill grinding, followed by rougher flotation, conventional cleaning, regrind, column cleaner flotation, and dewatering, to produce a nickel-cobalt-copper concentrate for storage in holding tanks. The concentrate will be finally reground and then pressure oxidized in an autoclave. Leaching of the copper, cobalt and nickel will then take place. Precipitation of copper, cobalt and nickel would happen sequentially.

Product will be trucked by the existing road network to the Port of Stewart for shipment to smelters. The truck haulage of concentrate will be contracted out. Tailings from the process will be impounded in a tailings pond; water will be reclaimed from the tailings pond and re-used in the process.

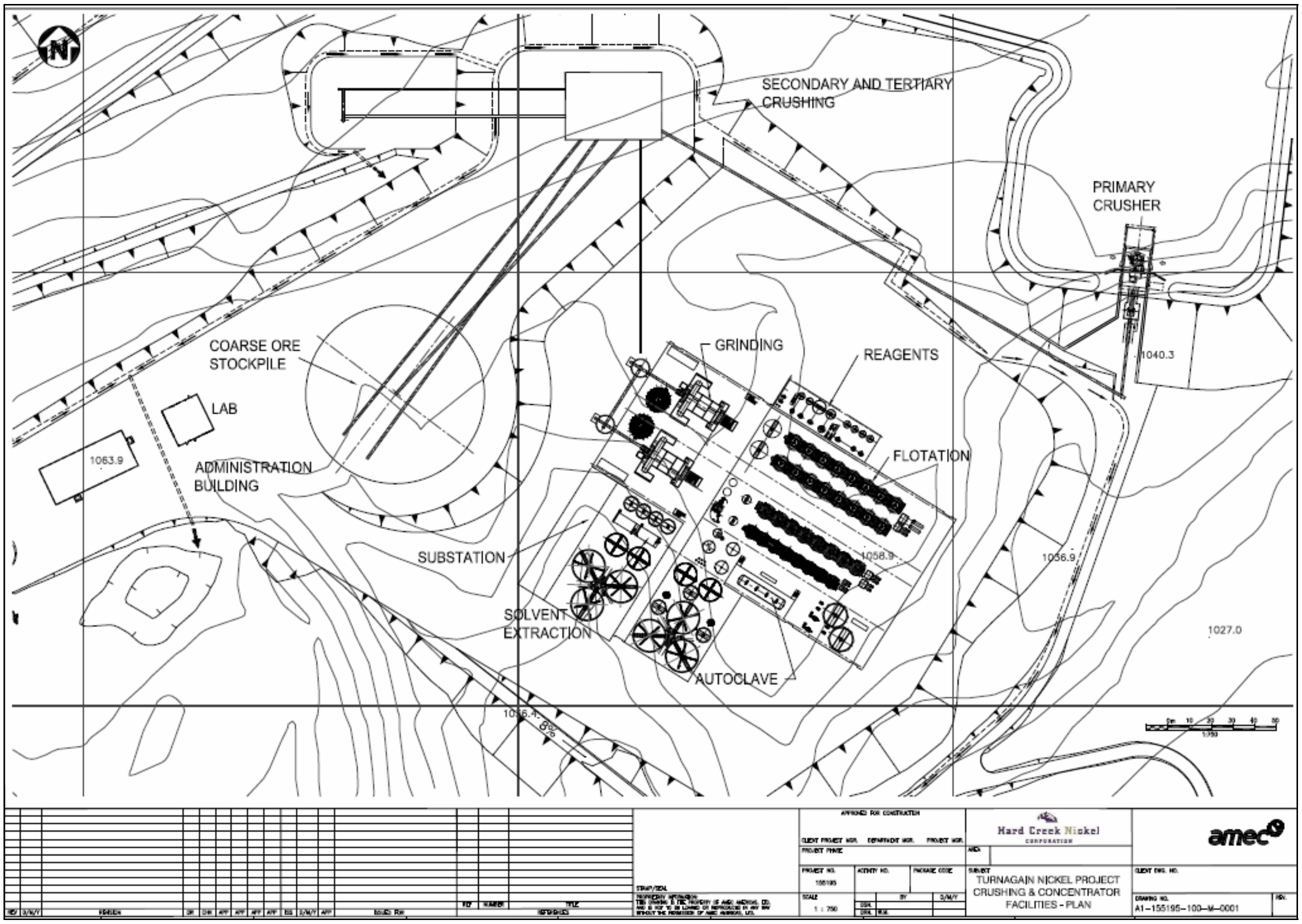
The mill building will be a pre-engineered, steel-framed and steel-clad insulated building founded on spread footings.

Concentrator

Primary crushing is proposed to be undertaken using a 1,524 mm x 2,261 mm (60" x 89") gyratory crusher. Discharge from the primary crusher is sent to two primary screens within the Fine Crushing Building. From these screens, the undersize is sent to the Fine Ore Bins while the coarse oversize is delivered to the coarse mineralization stockpile. The coarse mineralization stockpile will have a nominal live capacity of 25,000 t. Material reclaimed from the coarse mineralization stockpile is fed through 2 secondary crushers adjacent to the feed conveyors. Material is conveyed a surge bin feeding 4 secondary screens. The undersize from each screen passes to the discharge conveyor while the oversize enters a tertiary cone crusher (4 tertiary crushers in total). Finished product from the secondary screens is transferred by the discharge conveyors to the Fine Ore Bin System which is sized to provide 12 hours of capacity for the mills.

The grinding circuit was designed to achieve the required annualized throughput with single stage ball milling. An average 20 kWh/t ball mill work index was selected as the design basis. The circuit has been sized to grind 2,126 t/h to a product size of 80% passing 150 µm at an availability of 98%. A small regrind circuit helps to reduce the size of the minerals to assist in producing a higher concentrate grade.

Figure 16-1: Planned Metallurgical Plant Layout



AMEC selected a rougher, three stages of conventional cleaning with a cleaner scavenger circuit, and one further stage of column cleaning for the flotation circuit design. Cleaning will be critical in the flowsheet, and as a result, higher than normal levels of cleaning equipment have been incorporated into the flowsheet and capital cost estimate. The metallurgical testwork showed that a fairly high mass recovery to the rougher concentrate is required to obtain an acceptable nickel recovery. The expected mass pull into the rougher concentrate is about 15% of the flotation feed weight. The reagents that gave the best results included copper sulphate as a sulphide mineral promoter, PAX as the nickel collector, and methyl isobutyl carbinol (MIBC) as the frother. CMC is used as a talc depressant and slimes dispersant.

The concentrate thickener is of conventional design. An underflow target density of 60% has been adopted for scale-up. No testwork has been performed. It has been assumed that the concentrate settling characteristics are relatively typical and free settling.

Hydrometallurgical Processing

Preliminary sizing was conducted for a hydrometallurgical facility suitable for the treatment of the nickel sulphide concentrate. This facility consists of concentrate storage tanks, intensive regrinding, a pressure oxidation autoclave, and multiple precipitation (all tankage for precipitation reactions has been sized for a one hour retention time), counter-current decantation (CCD) washing and filtration circuits together with a small solvent extraction circuit. Three products are produced in this circuit; copper sulphide, nickel hydroxide and cobalt hydroxide. Insufficient data exists to provide design information for the solvent extraction and filtration circuit so only an allowance has been provided for the capital in this area.

Product Shipping

Moisture limitations for the precipitation products are set to fairly low levels, typically 3%, to provide ease of handling while avoiding dusting. All products will be shipped in one-tonne concentrate shipping bags. The products will be hauled on trucks carrying approximately 40 t at a time. Assuming the storage area is maintained at 50% capacity, in the event of road closure for weather or maintenance, there will be room for one week of storage in the hydrometallurgical process building.

Tailings Handling

The tailings impoundment will provide storage for all process tailings generated by the project and are designed to manage the changing requirements of the tailings impoundment development over the life of the mine. This includes provision for development of required tailings beaches and for construction of the downstream shells of Dam No.1 and No. 2 using cycloned sand.

During periods of tailings dam construction, rougher tailings will be sent to a cyclone sands plant, located at each of the two tailings dams, where coarse tailings sand solids will be separated from the fines.

Reagents

Reagents used within the mill will include:

- potassium amyl xanthate (PAX) – a conventional collector
- methyl isobutyl carbinol (MIBC) – a standard frother
- a polymer flocculant – to enhance concentrate and precipitate settling,
- copper sulphate – a promoter for sulphide flotation
- carboxyl methyl cellulose (CMC) – a reagent used for talc depression and slimes dispersion.

Other reagents will be used in the hydrometallurgical facility including:

- quicklime (calcium oxide, CaO) for pH modification and precipitation
- sulphuric acid for leaching
- sodium hydrosulphide for copper sulphide precipitation
- magnesium oxide for nickel and cobalt hydroxide precipitation
- a solvent extractant suitable for cobalt and zinc
- a diluent suitable for use in the SX circuit.

Plant Services

The process plant will use two kinds of water: fresh water and process water. Freshwater will be used for reagent mixing, process pump gland water, and cooling water for lubricant systems. Fresh make up water will be obtained from groundwater pump wells located in the floodplain of the Turnagain River.

Process water will be reclaimed from the tailings impoundment and pumped back to the concentrator and hydrometallurgical facilities. Process water will be used primarily in the grinding, flotation, and regrind circuits. Pumping to transfer reclaim water from the tailings pond to the plant site will be from the reclaim barge which is equipped with vertical turbine pumps.

Both low-pressure flotation air and compressed air will be supplied to various areas of the plant.

Instrumentation, in the form of a particle size analyzer for the grinding circuit and separate on-stream x-ray analyzers for the flotation and hydrometallurgical circuits, will be provided to measure the metal content of various process solutions and slurries. Slurry samplers will be used to provide samples for metallurgical accounting purposes.

16.5.3 Process Control

Process control for the Turnagain plant site will be by a programmable logic controller (PLC) or distributed control system (DCS), depending on which is most cost-effective for the application.

The control system will handle all process plant digital controls including motor control, interlocks, switches and indicator lights as well as all analogue process control loops, process indicators, and analogue control devices.

16.5.4 Process Operating Cost

Process operating costs for the Turnagain metallurgical complex are shown in Table 16-7.

Table 16-7: Estimated Plant Operating Costs (Typical)

Plant Operating Costs	Cost (CDN$/t milled)
Manpower	0.59
Reagents	3.09
Grinding Steel	0.86
Comminution Wear Parts	0.34
Spare Parts	0.16
Consumables Freight	1.13
Power Consumption	1.37
Miscellaneous	0.06
Total Mill Operations	**7.60**

Reagents form a major portion of the cost. In particular, the use of talc depressants can be critical for the ultramafic nickel deposits. Consumption can be difficult to predict and this poses a risk for operating cost.

Because the material is very hard, there is a significant cost associated with grinding steel, comminution wear parts, and power consumption. Further testwork could have a significant effect in reducing these costs further.

Freight costs of consumables, in particular those for the hydrometallurgical plant are significant. Costs are based on information from vendors and transportation companies.

The mill manpower includes staffing for the crushers, concentrator, hydrometallurgical and packaging facility. Staffing assumes the requirement for a moderate amount of technical support. During start-up of the plant, additional staffing may be required and this is covered within the contingency.

The metallurgical facilities are run as an integrated facility with common supervision and maintenance. A total of 135 staff is planned for the plant complex.

17.0 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

An update of the mineral resource estimates for the Turnagain deposit was performed by Mr. Guillermo Pareja of AMEC, under the guidance of Mr. Greg Kulla, P.Geo. of AMEC. The resource estimates were made from three-dimensional (3D) block models utilizing commercial mine planning software (MineSight®). Project limits are in UTM coordinates. Project limits are 507000–510000 E, 6480250–6482525 N, and 600 m to +1,350 m elevation. Cell size for the project was 25 m east x 25 m north x 15 m high.

There are 248 holes in the Turnagain database. Two hundred of these holes are in the resource area, all of which were used in developing the resource lithology model. One hundred ninety-seven holes in the resource area have assays; 158 holes, representing 42,128 m of sampling in 16,421 intervals drilled since 2002, were used for interpolating grade in the resource estimate. Assays for 40 holes drilled prior to 2002 are not supported by adequate control samples and lack complete analysis for NiS by AC-Ni. A scoping study (AMEC, 2006) showed removal of the pre-2002 holes results in a very slight increase in tonnage and no significant change in grade.

17.1 Geological Models

A significant difference between this estimate and previous estimates is the use of geological domains in the model. Previous models, which used grade shells and did not have the benefit of a lithological model, showed conditional bias and local excessive smoothing.

The geological model used in this estimate is based on the nearest-neighbour interpolation of geology as described in Section 7.3 of this report. This geological model is a first pass attempt at a comprehensive interpretation of the lithology through the resource area. Improvements are expected with more work, in particular the nature and direction of contacts between the key lithological units is expected to be refined.

17.2 Mineralized Zones

In the previous Technical Report, the Turnagain deposit was subdivided into domains (grade shells) based on a 0.1% NiS cut-off (Simpson, 2007). Because of the use of geological domains in the current model, it was decided that grade shells were not necessary. Thus, the 6 previously used domains were grouped into 3 spatially continuous zones (see Figure 17-1) for this estimate:

1. Main (encompassing the Northwest, Horsetrail North and Horsetrail Main domains)
2. Hatzl (encompassing the Hatzl North and Hatzl South Domains)
3. Duffy

Figure 17-1: Map View of Zone and Domain Locations

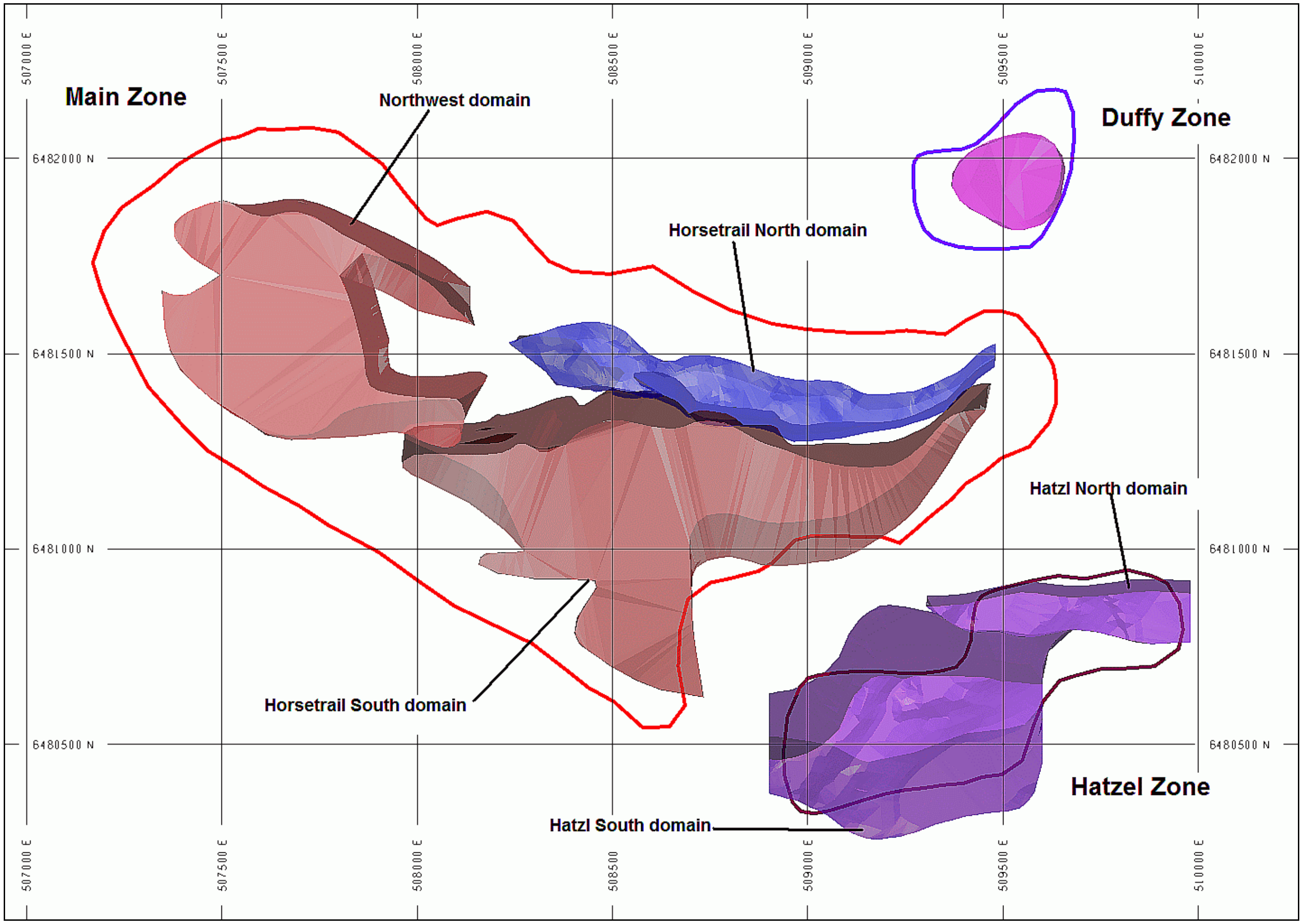


Figure 17-2 shows that the spatial distribution of AC-Ni grades above a 0.10% AC-Ni cut-off is fairly continuous within each of the zones, both along the drill holes and in between drill holes. NiS grades (Figure 17-3) are less continuous, particularly at the north-central area of the Main zone. NiS is a calculated field assigned a background value of zero if AC-Ni was not analysed or if sulphur was less than 0.2%. The north-central area has several intervals which were deemed non-mineralized by visual inspection and were not assayed for AC-Ni.

Figure 17-2: Map View of Composites with Grades ≥0.10% AC-Ni

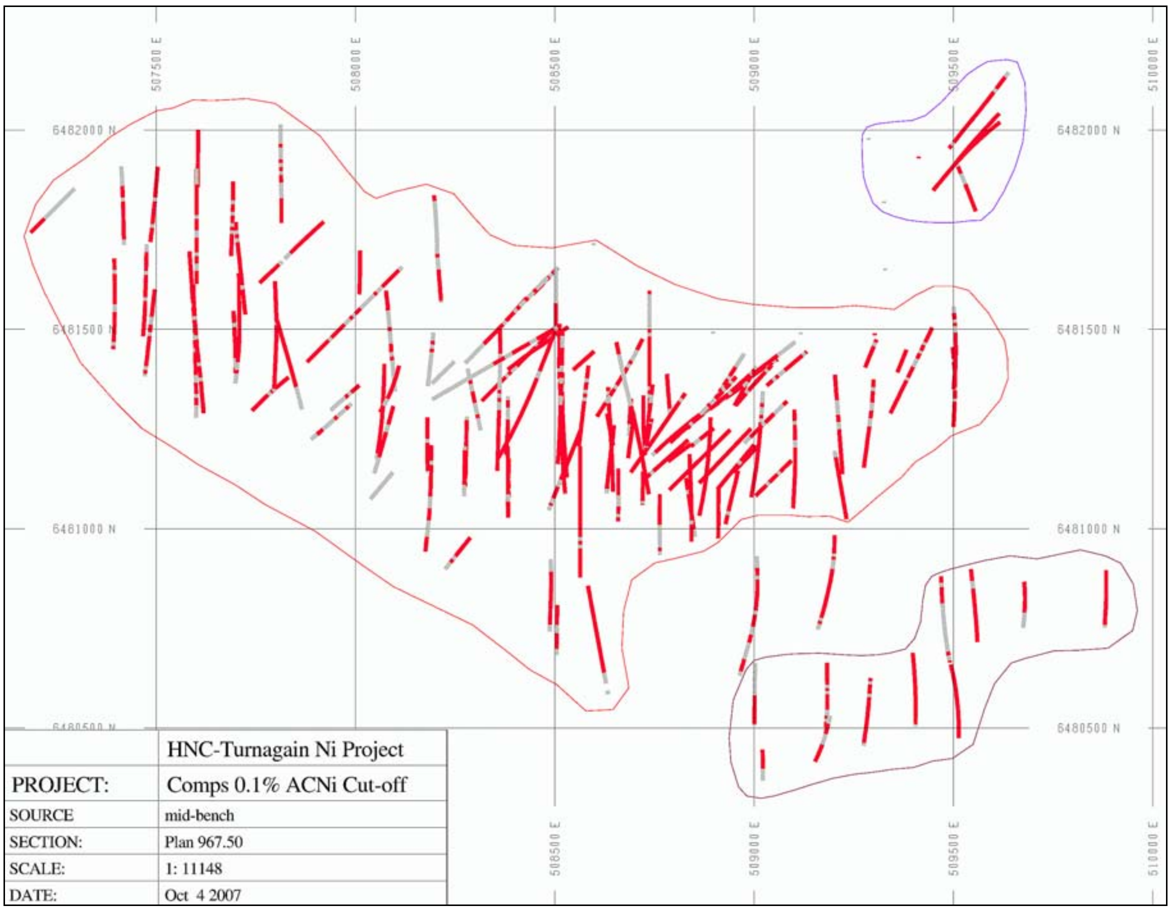
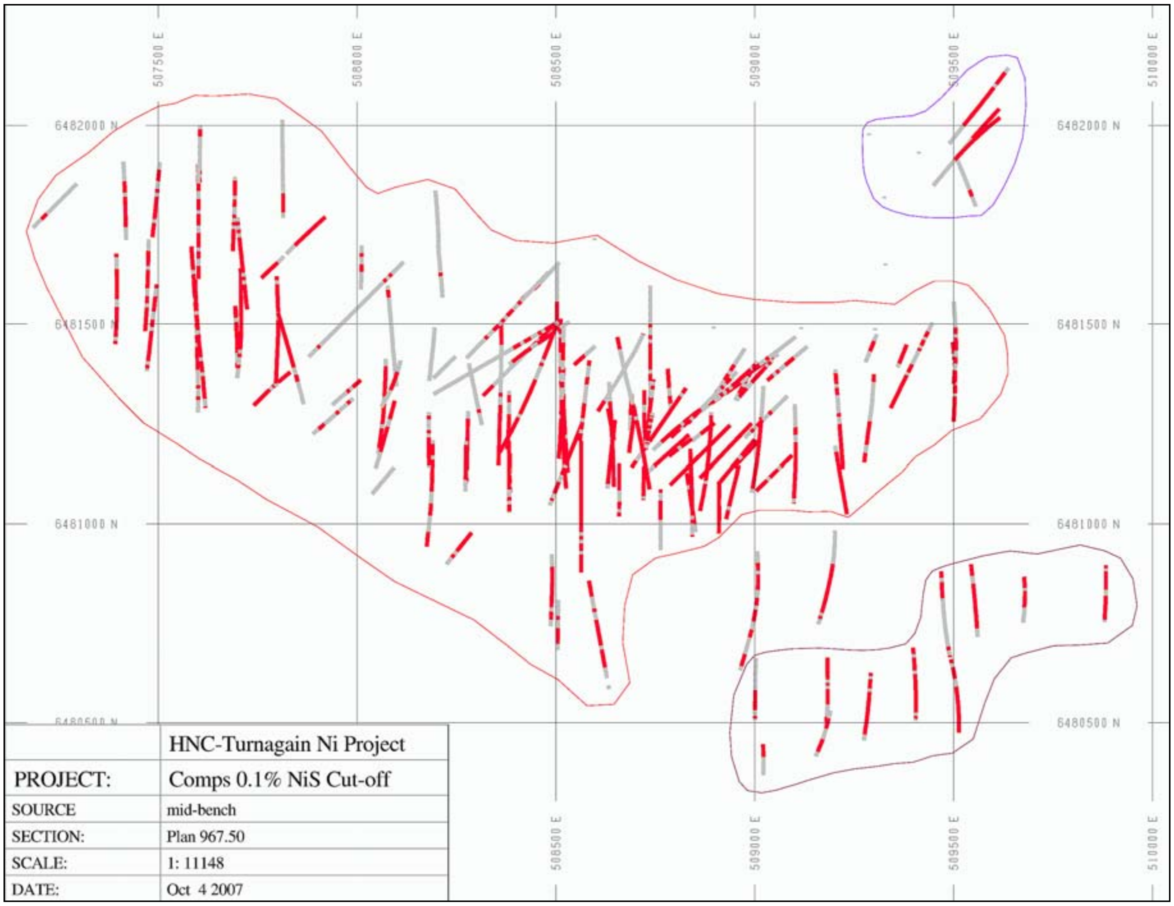


Figure 17-3: Map View of Composites with Grades ≥0.10% NiS



17.3 Composites

The drill hole assays were composited into fixed-length 15 m down-hole composites, a size which was considered to best support the selective mining unit (SMU). Intervals that were less than 15 m in length represent individual residual composites from end-of-hole intervals. Composites that were less than 7.5 m long were removed from the dataset that was used in interpolation.

Composites which honour the geology in the drill holes and shorter composite lengths (e.g., 7.5 m) should be examined in future models as this may provide better resolution along domain boundaries and further reduce unintentional smoothing.

17.4 Data Analysis

The lithological and mineralized domains were reviewed to determine the appropriate estimation or grade interpolation parameters. Several different procedures were applied to the data to discover whether statistically distinct domains could be defined using the available geological objects. For each zone, key lithological categories were investigated.

Six "elements" were interpolated: nickel by the AC method (AC-Ni), Total nickel (TD-Ni), nickel sulphide (NiS), cobalt by the AC method (AC-Co), S, and Mg.

Descriptive statistics and contact grade profile plots were completed for all elements (AC-Ni, TD-Ni, NiS, Co, S, and Mg) by zone and/or lithological domain. Results obtained were used to guide the construction of the block model and the development of estimation plans. Data analyses were conducted on composited assay data (15 m down-hole composites).

Units 107 (dikes) and 108 (meta-sediments) have non-recoverable nickel contents and are considered waste. Unit 103 (mtcPx) has a very small number of composites (6) and no blocks tagged with that unit. Thus, those three units are not included in the following discussions.

The statistical properties of all the elements are summarized by lithological domain in Tables 17-1 and 17-2, for the 15 m composites at Turnagain. In these tables CV = coefficient-of-variation or standard deviation/mean, a measure of relative dispersion.

17.5 Evaluation of Extreme Grades

Most elements, except for sulphur, have highly symmetrical, low skewed distributions, with moderate to low CVs; because of this, it is considered that extreme grades are not a cause of concern for interpolation, and it was decided not to perform a high-grade capping (or top-cutting) on any of the elements for all rock types. Outlier restriction, or "dynamic capping" was performed, see Section 17-10.

Table 17-1: Statistics for 15 m Composites – AC-Ni%, NiS%, and Total Ni% Data

Lithological Domain Code	Lithological Domain Rocktype	AC-Ni Number	AC-Ni Mean	AC-Ni CV	NiS Number	NiS Mean	NiS CV	Total Ni Number	Total Ni Mean	Total Ni CV
101	cPx	136	0.104	0.75	136	0.101	0.79	136	0.122	0.67
102	ocPx	188	0.143	0.54	188	0.139	0.57	188	0.170	0.53
103	mtcPx	6	0.022	0.40	6	0.020	0.55	6	0.030	0.31
104	DuWh	1,022	0.148	0.52	1,022	0.118	0.81	1,017	0.226	0.32
105	gDu	207	0.094	0.64	207	0.026	2.37	207	0.256	0.17
106	Sp	1,198	0.180	0.45	1,198	0.156	0.64	1,193	0.229	0.34
107	Dk,Di,Hb	49	0.078	0.76	49	0.069	0.83	49	0.099	0.65
108	MSD	34	0.031	1.01	34	0.027	1.17	34	0.043	0.89
All		**2,840**	**0.153**	**0.54**	**2,840**	**0.126**	**0.79**	**2,830**	**0.217**	**0.38**

Table 17-2: Statistics for 15 m Composites –S%, Co% and Mg% Data

Lithological Domain Code	Lithological Domain Rocktype	S Number	S Mean	S CV	Co Number	Co Mean	Co CV	Mg Number	Mg Mean	Mg CV
101	cPx	136	1.54	0.55	132	0.010	0.38	135	15.3	0.33
102	ocPx	188	1.39	0.64	186	0.010	0.36	187	18.3	0.21
103	mtcPx	6	0.68	0.58	6	0.005	0.45	6	12.8	0.11
104	DuWh	1,022	0.61	1.08	993	0.009	0.51	980	22.1	0.24
105	gDu	207	0.10	1.44	193	0.005	0.38	207	25.7	0.11
106	Sp	1,198	0.72	0.93	1,190	0.011	0.44	1,179	22.8	0.18
107	Dk,Di,Hb	49	0.66	0.85	45	0.006	0.59	49	9.5	0.48
108	MSD	34	1.29	0.77	26	0.004	0.71	32	7.1	0.68
All		**2,840**	**0.72**	**1.02**	**2,771**	**0.009**	**0.49**	**2,775**	**21.7**	**0.25**

17.6 Contact Grade Profile Analyses

Contact profiles for AC-Ni, TD-Ni, S and Mg show sharp differences in grade across most domain boundaries. Cobalt shows generally soft to gradational contacts.

17.7 Estimation Domains

The data analysis indicates that lithological control is the main control on the spatial distribution of grades. Thus, it was decided that for grade interpolation the data should be subdivided by lithological domain and zone; the use of grade shells was discontinued in this model. Additionally, it was decided to define all inter-lithological domain contacts as hard boundaries. The use of grade shells in combination with lithological domains should be explored in future models, particularly to control grade-smoothing within lithological domains.

17.8 Variography

Variograms were only calculated for the primary payable metal assay AC-Ni. Data for the DuWh and Sp lithological domains (within the Main zone only) were combined in order to have enough samples to define a good variogram. The modelled variogram was subsequently used for all lithological domains, for all zones, and for all metals.

Correlograms (a type of variogram) were computed on 15 m down-hole composites. Directional variograms were calculated and modeled in 37 directions. The AC-Ni variogram model consisted of a nugget effect, two-nested structure variance contributions, ranges for the variance contributions and the model type (spherical). The nugget effect was modeled using down-hole variograms. Two structures were fitted, setting the main axis orientation parallel to the main orientation of the Main zone. MineSight® rotation conventions were specified.

The nugget effect, or random variation components of spatial variation, is moderate at 25% of the total variation. The first structure is isotropic with a relatively short range (80 m); the second structure is strongly anisotropic (isotropic ratio >2.6), with moderate to long ranges.

17.9 Model Setup

The block model was set up using MineSight® software. The block model size selected was 25 m x 25 m x 15 m. This allowed consistency with previous modelling in the Turnagain deposit (see Simpson, 2007). The extents of the model are shown in Table 17-3. Various coding were done on the blocks in preparation for grade interpolation. The block model was coded according to zone and lithological domain based on block centroids. Dykes and meta-sediments were assumed to represent zero grade waste.

Table 17-3: Block Model Extents

	Min	Max	Extent	Size	Number
Easting	507000	510000	3000	25	120
Northing	6480250	6482525	2275	25	91
Elevation	600	1350	750	15	50

17.10 Mineral Resource Estimation Plan

The Turnagain estimation plan, or sets of parameters used for estimating blocks, was designed using a philosophy of restricting the number of samples for local estimation. AMEC has found this to be an effective method of reducing smoothing and producing estimates that match the actual recovered grade-tonnage distributions. While local predictions based on a small number of samples are uncertain, and can be conditionally biased, this method can produce reliable estimates of the recovered tonnage and grade over the entire deposit, i.e., the global grade–tonnage curves from the estimations are accurate predictors of the actual grade-tonnage curves.

Interpolation was limited to the mineralized lithological units (cPx, ocPx, DuWh, Sp, and gDu). Only blocks within those units were interpolated, and only composites belonging to those units were used. Metal values within blocks belonging to all other units (dykes and metasediments) were set to zero.

Modelling consisted of grade interpolation by Ordinary Kriging (OK). Both restricted and unrestricted grades were interpolated to allow calculation of the metal removed by the outlier restriction. Unrestricted nearest-neighbour grades were also interpolated for validation purposes. Blocks and composites were matched on lithological domain and zone.

The search ellipsoids were oriented parallel to the general orientation of the mineralization in the main zone. The search strategy employed concentric expanding search ellipsoids. The first pass used a short search ellipse relative to the long axis of the variogram ellipsoid. For the second pass, the search ellipse was increased by 50% (up to the full range of the variogram) to allow interpolation of grade into those blocks not estimated by the first pass. A last third pass was performed using a larger search ellipsoid (with axis ranges equivalent to the full range of the variogram for the corresponding axis).

The number of composites used in estimating grade into a model block followed a strategy that matched composite values and model blocks sharing the same zone and domain. The minimum and maximum number of composites was adjusted to incorporate an appropriate amount of grade smoothing (Table 17-4). To ensure that at least two boreholes were used in the estimate, the number of composites from a single drill hole that could be used was set to one less than the minimum number of composites.

Table 17-4: Search Ellipsoids for Turnagain

		First Pass	Second Pass	Third Pass
Rotation Angles	**Z**	105	105	105
	X	0	0	0
	Y	-20	-20	-20
Ranges (m)	**Y**	80	200	400
	X	80	100	150
	Z	80	150	280
Number of Composite	**Min**	3	3	3
	Max	12	8	8
	Max per DDH	2	2	2

Note: MIN = minimum number of composites; MAX = maximum number of composites, MAX per DDH = maximum number of composites derived from a single borehole. Axis rotations are left-hand, right-hand, left-hand for the Z, X and Y axis, respectively.

The final parameters used were based on the geological interpretation, data analyses, and variogram analyses, and were optimized for the interpolation of AC-Ni. Those same parameters were subsequently used in the interpolation of all the other metals.

For all metals, an outlier restriction was used to control the effects of high-grade samples within the domains, particularly in the low-grade domains, where unrestricted high-grade composites tended to result in 'blow outs' from extreme grade composites. In outlier restricted kriging, outliers (i.e., values above a specified cut-off) are cut down to the specified threshold value if their distance to the interpolated block is greater than the specified distance. If the distance to the interpolated block is less than the specified distance, then outliers are used at their full value. The outlier thresholds applied were defined at the 98 percentile of their respective population. The thresholds are shown in Tables 17-5 and 17-6.

Table 17-5: Outlier Thresholds Applied to Lithology Domains by Element (First Pass)

Element	cPx	ocPx	DuWh	gDu	Sp
AC-Ni	0.25	0.28	0.35	0.22	0.40
NiS	0.25	0.28	0.35	0.22	0.40
Total Ni	0.3	0.4	0.4	0.3	0.4
S	3.5	3	3	0.8	3
Co	0.02	0.2	0.022	0.01	0.03
Mg	25	28	-	-	-

Table 17-6: Outlier Thresholds Applied to Lithology Domains by Element (Third and Second Passes)

Element	cPx	ocPx	DuWh	gDu	Sp
AC-Ni	0.15	0.20	0.20	0.15	0.20
NiS	0.15	0.20	0.20	0.15	0.20
Total Ni	0.18	0.22	0.28	0.28	0.25
S	3.5	3	3	0.8	3
Co	0.02	0.2	0.022	0.01	0.03
Mg	25	28	-	-	-

The block model was assigned mean bulk density values based on rock type. The mean bulk density of each lithology is shown in Table 17-7.

No additional manipulation of grade or recovery was applied to simulate the effects of mining. The resources for the Turnagain deposit were tabulated and reported using these undiluted grade values.

Table 17-7: Average Bulk Density

Lithology	Bulk Density
cPx	3.14
ocPx	3.06
DuWh	3.06
gDu	3.06
Sp	2.99
Dk	3.13
MSD	2.97

17.11 Validation

Validation was focused on AC-Ni and NiS results. Validation tools included visual inspection, model checks for global bias (histograms, boxplots), model checks for local bias (swath plots), metal reduction, contact plots and Herco validation.

The validation results shown in this section indicate that the methods used and results obtained in the estimation of the resource model for the Turnagain deposit are adequate.

Visual Inspection

AMEC completed a detailed visual validation of the Turnagain resource model. The model was checked for proper coding of drill hole intervals and block model cells, in both section and plan and found to be properly done. Grade interpolation was examined relative to drill hole composite values by inspecting sections and plans. The checks showed good agreement between drill hole composite values and model cell values (Figures 17-4 and 17-5). The hard boundaries between lithological domains appear to have constrained grades to their respective estimation domains. The addition of the outlier restriction values succeeded in minimizing grade smearing in regions of sparse data.

Figure 17-4: North-South Longitudinal Section 508500 E Showing AC-Ni Grades in Drill Holes and in the Block Model

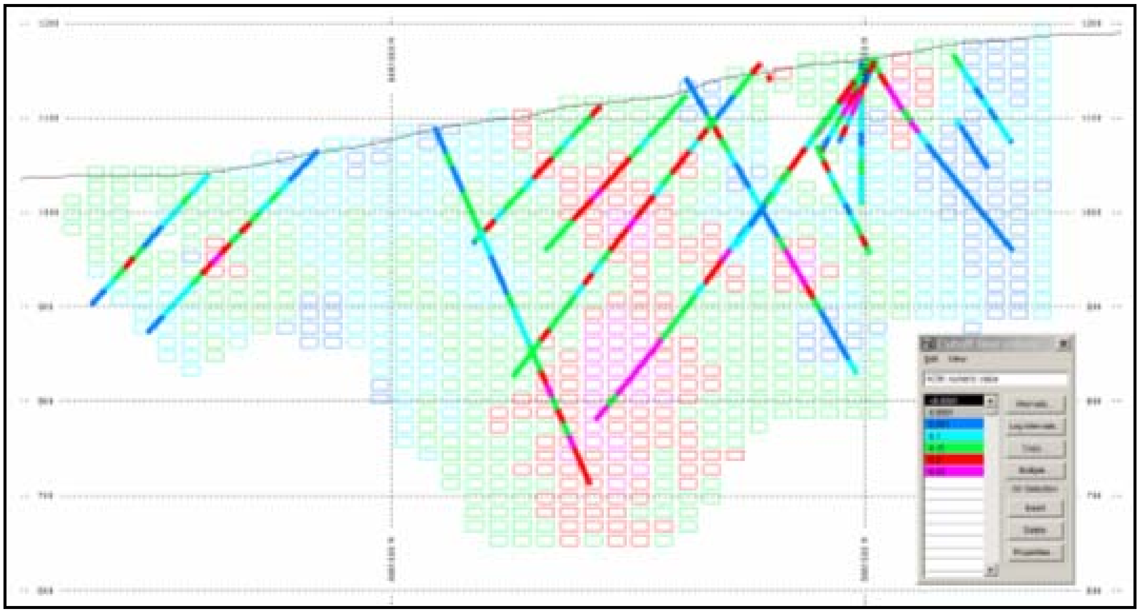
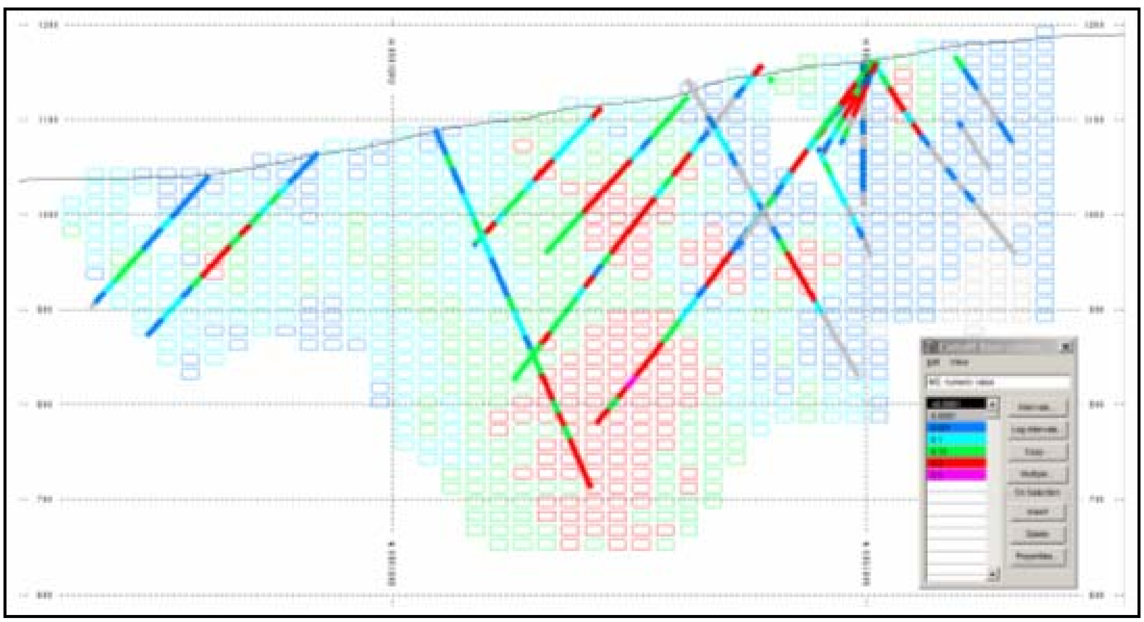


Figure 17-5: North-South Longitudinal section 508500 E Showing NiS Grades in Drill Holes and in the Block Model



Model Checks for Global Bias

AMEC checked the block model estimates for global bias by comparing the average metal grades from the unrestricted OK model with the average metal grades from unrestricted nearest-neighbour estimates. Results showed no problems with global bias in the estimates.

Model Checks for Local Bias

AMEC checked for local biases in the grade estimates using grade slice or swath checks. The two trends behaved as predicted and show no significant differences for any of the metals in the estimates.

Metal Reduction

The effective amount of metal removed by outlier restriction can be evaluated by comparing the block models kriged with and without outlier restriction. The quantity of metal removed is mostly less than 5%, which is considered reasonable for the type of deposit.

Contact Plots

Comparison of the composite contact plots against their corresponding OK block model contact plots show that overall the OK block model correctly reproduced the boundary relationships among all the rock types.

Herco

Results of Herco validation indicate that the level of smoothing in the current model is appropriate.

17.12 Cut-off Determination

The use of a 0.10% NiS cut-off has been determined as a result of parameters selected for pit optimization based on work by AMEC (2006). Table 17-8 shows mining and economic assumptions used for pit optimization in 2007 and the resulting external cut-off of 0.072%. The cut-off used in this estimate is 0.10% NiS considering NiS is a calculated field set to zero for samples without AC-Ni analysis or with S less than 0.2%.

Table 17-8: External Cut-off Determination

Items	
CDN$/tonne milled	5.08
CDN$/tonne mined	1.4
CDN$/US$1	1.05
% recovery	70
% metal price paid for transportation, treatment and refining costs	39.15
lb/tonne	2204.62
nickel price (US$/lb))	8.25
Ni cut-off %	0.080

Note: % recovery is an average for domain 101,102,104, and 106.
Percent metal price paid is sell cost/nickel price (3.23/8.25)
Ni cut-off is estimated by: ((5.08+1.40)/1.05)/((8.25-(8.25*39.15/100)*(75/100))/2204.62=0.080%

17.13 Mineral Resource Classification

The mineral resources of the Turnagain deposit were classified in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves as referenced in NI 43-101. The mineralization of the Project satisfies sufficient criteria to be classified into Measured, Indicated and Inferred mineral resource categories.

Measured Mineral Resources

The Measured Mineral Resource category is supported at Turnagain where the drill spacing is on approximately 75 x 50 m centres. Geological and grade continuity is demonstrated by inspection of the model and drill hole data in plans and sections over the various zones, combined with spatial statistical work and investigation of confidence limits in predicting planned quarterly production. Considering these factors, AMEC concludes that blocks covered by this data spacing may be classified as Measured Mineral Resource.

Indicated Mineral Resources

The Indicated Mineral Resource category is supported at Turnagain where the drill spacing is on approximately 100 x 100 m centres. Geological and grade continuity is reasonably assumed by inspection of the model and drill hole data in plans and sections over the various zones, combined with spatial statistical work and investigation of confidence limits in predicting planned annual production. Considering these factors, AMEC concludes that blocks covered by this data spacing may be classified as Indicated Mineral Resource.

Inferred Mineral Resources

All interpolated blocks that did not meet the criteria for Measured or Indicated Mineral Resource were classified as Inferred Mineral Resource if they fell within 200 m of two drill holes. A further limitation was added that the block should be within 50 m (in the vertical direction) of the nearest composites, in order to control extrapolation of grades, particularly in the vertical direction.

17.14 Mineral Resource Summary

In accordance with CIM best practice of reporting resources which have a reasonable expectation of economic extraction, the classified resources of this model were constrained in a Lerchs-Grossman optimized pit using Whittle software. The parameters considered when constraining the resources include mining method, mining cost, metallurgical recovery, process method, process cost, general and administrative (G&A) costs, long term metal prices, treatment and refining costs (TC/RC), and capital costs. Parameter assumptions are listed in Table 17-9.

Table 17-9: L-G Parameter Assumptions

Items	Assumptions
Mining Method	Best case open-pit
Mining Cost	CDN$1.40/tonne
Recoveries	Variable (average 75%)
Process Method	On-site concentrate; off-site smelting and refining
Process Cost and G&A	CDN$5.08/tonne processed
Metal price	US$8.25/lb Ni produced
TC/RC	US$3.12/lb Ni produced

Resources stated here are based on concentrate production on site with off-site smelting and refining. Hydrometallurgical processing of the concentrate on-site shows more favourable economics and is considered in more detail in Section 19 of this report.

Using the assumptions in Table 17-9, the Turnagain property contains an estimated 489 Mt of Measured and Indicated Resources at 0.163% NiS at a 0.10% NiS cut-off and an additional 560 Mt of Inferred Resources at 0.152% NiS at a 0.10% NiS cut-off. A resource summary for a range of cut-offs is reported in Table 17-10. Resources by zone for a range of cut-offs are shown in Table 17-11.

The tonnage above cut-off is extremely sensitive to the cut-off grade. If the cut-off is increased to 0.16% NiS, the Measured and Indicated tonnage decreases by approximately half and the contained metal decreases by approximately 25%.

Table 17-10: Mineral Resource Estimate (All Zones); Effective Date 25 September 2007, G. Kulla QP

	Measured					Indicated					Measured & Indicated				
NiS Cut-off %	Tonnage (kt)	NIS %	AC_NI %	NI_TOT %	AC_CO %	Tonnage (kt)	NIS %	AC_NI %	NI_TOT %	AC_CO %	Tonnage (kt)	NIS %	AC_NI %	NI_TOT %	AC_CO %
>2	20,709	0.252	0.258	0.300	0.014	68,714	0.229	0.237	0.274	0.014	89,423	0.234	0.242	0.280	0.014
>0.18	27,056	0.237	0.244	0.287	0.013	127,918	0.211	0.220	0.256	0.013	154,974	0.216	0.224	0.261	0.013
>0.16	34,858	0.222	0.230	0.274	0.013	196,928	0.195	0.206	0.244	0.013	231,786	0.199	0.209	0.248	0.013
>0.14	43,417	0.208	0.218	0.263	0.012	277,330	0.181	0.194	0.233	0.012	320,747	0.184	0.197	0.237	0.012
>0.12	52,099	0.195	0.207	0.255	0.012	361,855	0.169	0.183	0.224	0.012	413,954	0.172	0.186	0.228	0.012
>0.10	59,464	0.184	0.199	0.250	0.011	429,688	0.160	0.175	0.218	0.012	489,152	0.163	0.178	0.222	0.012

	Inferred				
NiS Cut-off %	Tonnage (kt)	NIS %	AC_NI %	NI_TOT %	AC_CO %
>2	36,351	0.223	0.225	0.253	0.014
>0.18	116,654	0.200	0.203	0.238	0.013
>0.16	220,231	0.186	0.190	0.228	0.012
>0.14	329,470	0.174	0.181	0.219	0.012
>0.12	448,321	0.162	0.172	0.210	0.011
>0.10	560,052	0.152	0.164	0.204	0.011

Table 17-11: Mineral Resource Estimate (by Zone)

Measured Main

NiS Cut-off %	Tonnage (thousands)	NIS %	AC_NI %	NI_TOT %	AC_CO %
>2	20,709	0.252	0.2577	0.300	0.0140
>.18	27,056	0.237	0.2441	0.287	0.0134
>.16	34,858	0.222	0.2302	0.274	0.0129
>.14	43,417	0.208	0.2181	0.263	0.0123
>.12	52,099	0.195	0.2074	0.255	0.0118
>.10	59,464	0.184	0.1994	0.250	0.0114

Indicated Main

NiS Cut-off %	Tonnage (thousands)	NIS %	AC_NI %	NI_TOT %	AC_CO %
>.2	62,840	0.231	0.2371	0.275	0.0126
>.18	117,307	0.212	0.2194	0.257	0.0121
>.16	178,449	0.197	0.2059	0.245	0.0115
>.14	248,572	0.184	0.1937	0.235	0.0110
>.12	324,438	0.171	0.1829	0.226	0.0105
>.10	385,926	0.162	0.1752	0.220	0.0101

Indicated Hatzl

NiS Cut-off %	Tonnage (thousands)	NIS %	AC_NI %	NI_TOT %	AC_CO %
>.2	5,874	0.227	0.2378	0.261	0.0153
>.18	10,611	0.210	0.2212	0.246	0.0149
>.16	18,479	0.193	0.2052	0.229	0.0139
>.14	28,758	0.177	0.1939	0.218	0.0134
>.12	37,417	0.166	0.1837	0.207	0.0134
>.10	43,762	0.159	0.1750	0.198	0.0132

Inferred Main

NiS Cut-off %	Tonnage (thousands)	NIS %	AC_NI %	NI_TOT %	AC_CO %
>.2	25,151	0.224	0.2253	0.257	0.0142
>.18	90,319	0.199	0.2013	0.240	0.0131
>.16	168,369	0.186	0.1894	0.231	0.0122
>.14	249,340	0.174	0.1798	0.222	0.0114
>.12	347,950	0.161	0.1702	0.212	0.0108
>.10	443,469	0.150	0.1622	0.205	0.0103

Inferred Hatzl

NiS Cut-off %	Tonnage (thousands)	NIS %	AC_NI %	NI_TOT %	AC_CO %
>.2	5,246	0.225	0.2276	0.255	0.0161
>.18	13,534	0.202	0.2047	0.235	0.0146
>.16	33,977	0.182	0.1888	0.218	0.0133
>.14	57,793	0.169	0.1786	0.208	0.0130
>.12	76,193	0.160	0.1707	0.200	0.0130
>.10	90,815	0.152	0.1637	0.193	0.0128

Inferred Duffy

NiS Cut-off %	Tonnage (thousands)	NIS %	AC_NI %	NI_TOT %	AC_CO %
>.2	5,953	0.219	0.2242	0.234	0.0118
>.18	12,801	0.204	0.2098	0.222	0.0112
>.16	17,885	0.194	0.2012	0.217	0.0109
>.14	22,337	0.186	0.1950	0.213	0.0106
>.12	24,177	0.182	0.1936	0.213	0.0105
>.10	25,768	0.177	0.1923	0.213	0.0105
Total	**25,768**	**0.177**	**0.1923**	**0.213**	**0.0105**

17.15 Risks Associated with Current Model

Risks in the current model are:

- The geological model: because of the strong lithological control on grades, the influence of the geological model on the final block model grades is very high; thus, it is necessary that the geological model be of the highest quality possible; AMEC recommends that future models should include an update of the geological model.
- The use of AC-Ni and NiS: the use of the calculated NiS values for the final model is conservative; this creates the risk of underestimating the value of the deposit.
- Smoothing of grades within lithological domains is considered minor but the use of grade shells in combination with lithological domain controls should be explored.

18.0 ADDITIONAL REQUIREMENTS FOR TECHNICAL REPORT ON DEVELOPMENT PROPERTIES AND PRODUCTION PROPERTIES

The Turnagain project is not a development property as defined under NI 43-101. Information relating to the PA is included in Section 19 of this report.

19.0 OTHER RELEVANT DATA AND INFORMATION

This section includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary assessment based on these resources will be realized. The results of the economic analyses discussed in this section represent forward-looking information as defined under Canadian securities law. The results depend on inputs that are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here.

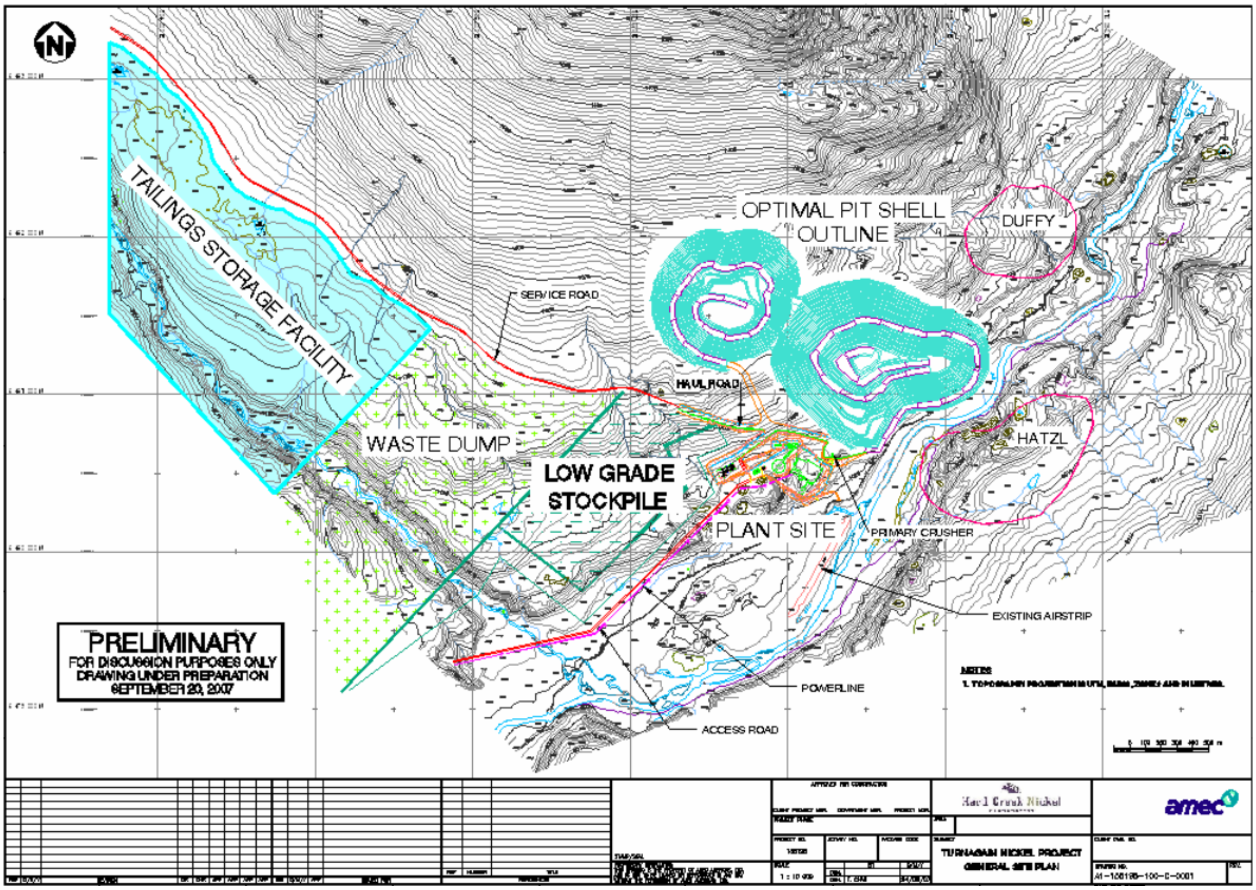
19.1 Mining

AMEC has addressed initial pit resources and design, reviewed tailings and waste considerations, reviewed ancillary and infrastructure requirements, and proposed a project execution plan. The information from these activities is presented in this subsection. A different nickel price was used for the mine planning (US$7.50/lb) to that used to declare resources (US$8.25/lb). The objective of the work completed as part of the PA was to derive a preliminary production schedule for two processing methods, evaluate the practicality of the process methods, and select a method for further study. In addition, several sensitivity studies were performed at various metal prices, selling costs, process costs and slope angles. Eight schedules based on different nickel prices were prepared for the financial analysis described in Section 19.7. A proposed site plan is included as Figure 19-1.

19.1.1 Assumptions

- low-grade stockpiles are used in the schedules
- only nickel sulphide (NiS) grade is used in the optimization process
- the pushbacks are selected based on the five year production phases
- only the Main zone mineralization is used in the production schedule
- the pit is developed using a setback of 100 m from the Turnagain River
- only blocks classified as Measured, Indicated or Inferred Resources were used; no unclassified material was considered. About 30% of the tonnes contained in the ultimate pit shell are in the Inferred Resource category. The presence such low-confidence material in the schedules carries both technical and financial risks. AMEC recommends that HCNC proceed with further investigations in order to move the Inferred resources to a higher-confidence mineral resource category.
- mining and transport costs are based on AMEC's knowledge of similar projects in Canada.

Figure 19-1: Proposed Site Plan



19.1.2 Data Available

- grade block model for total nickel, nickel sulphide, cobalt, sulphur, magnesium for the bench plans with the nickel sulphide and total nickel (indicated as NiS and Ni_T)
- lithological block model consisting of six lithological units (see Section 7)
- the block model containing the confidence classification
- three triangulations representing individual mineralized zones: Main, Duffy and Hatzl zones
- the extended surface topography, which covers a sufficient area to accommodate the proposed pit shells
- the location of a single claim with 4% royalty rights belonging to a third party
- mill throughput of 50,000 t/d

- rock density for the mineralised units and waste (Table 19-1):

Table 19-1: Specific Gravity for Major Lithological Units

Major Lithological Units	Unit	
101 clinopyroxenite	g/cm^3	3.14
102 olivine clinopyroxenite	g/cm^3	3.06
104 dunite-wehrlite	g/cm^3	3.06
105 green dunite	g/cm^3	3.06
106 serpentinite	g/cm^3	2.99
Waste	g/cm^3	3.05

19.1.3 Process Methods

Two process methods were considered for the recovery of nickel: one method is to produce a saleable concentrate and the other is a hydrometallurgical process. Each method has a separate set of parameters for the optimization process. In both cases, the production rate considered was 50,000 t/d. There was no throughput analysis performed.

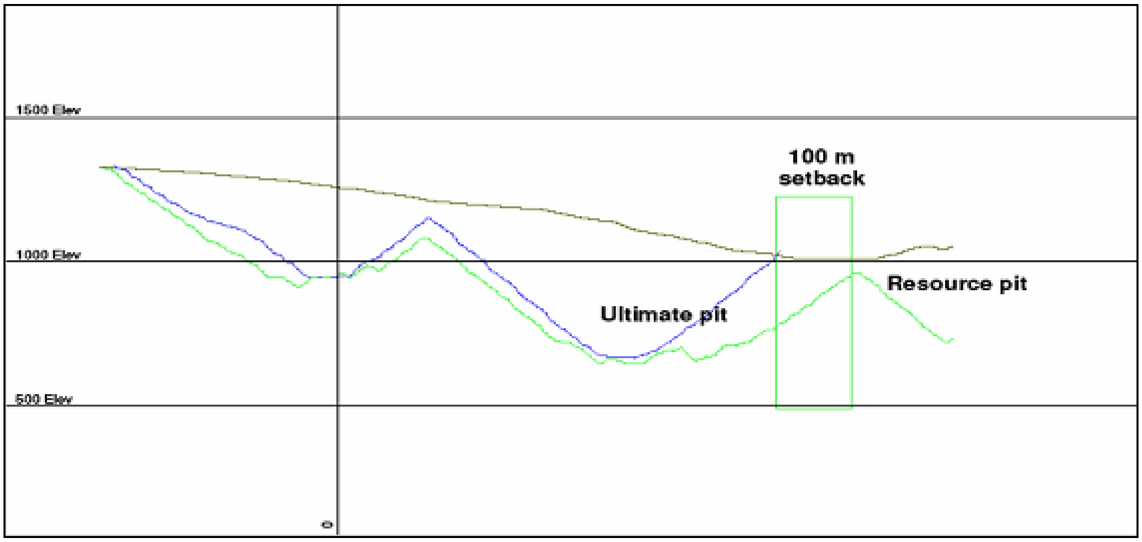
River Proximity and Resources

The river proximity was considered while optimizing the pits for both processing methods. The Turnagain River passes through the HCNC property and divides the mineralized zones. A 100 m setback from Turnagain River was used in this PA; this distance needs to be confirmed for future studies.

The cross section in Figure 19-2 demonstrates the setback from the Turnagain River; an overall pit slope of 45° was used in the optimization process.

Table 19-2 shows how much of the ‘in pit resources are removed from the inventory due to a setback of 100 m from the Turnagain River with a price of US$8.25/lb for nickel used to constrain the mineral resources. The Revenue Factor 1 pit shell was used to delineate the portion that met the “reasonable prospect for economic extraction” test.

Figure 19-2: Setback from Turnagain River



Note: The above section shows the pit shell resulting from the optimization, considering the 100 m setback from the Turnagain River. The larger shell is the Revenue Factor 1 shell used to constrain the resources. The vertical rectangular outline is representing 100 m setback from the river on both sides.

Table 19-2: Impact on In-pit Shell Resources

Items	Mill Feed (Mt)	NiS Grade (%)	Waste (Mt)	Life Of Mine (years)
Hydrometallurgical option all	1,025	0.158	1,140	61
Hydrometallurgical option minus river setback	823	0.156	742	47
Saleable Concentrate option all	1,003	0.158	1,048	59
Saleable Concentrate option minus river setback	849	0.155	847	50

Note: includes Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that these data will be realized.

19.1.4 Saleable Concentrate Option

Optimization Parameters

Table 19-3 shows the inputs used in the optimization for the Saleable Concentrate option, without considering differences in haul distance for mineralization and waste or increases in haul distance during the mine life. Sustainable capital cost for mining equipment was also not considered. As the royalty payment is only applicable for a small part of the mill feed, it was not considered. Selling costs included ocean freight, smelting and refining costs.

Table 19-3: Inputs, Saleable Concentrate Option (costs in Q3 2007 Canadian Dollars)

Items	Unit		Assumptions
Mining Cost	CDN$/t mined	1.40	
Process + G&A Costs	CDN$/t milled	5.08	
Ni Price	CDN$/lb	7.89	based on US$ 7.50/lb
Selling Cost	CDN$/lb	3.40	based on US$ 3.23/lb
Exchange Rate	US$/CDN$	0.95	
Nickel Recovery	Rock Type	101	RA (NIS.G,0 ,0.1 ,0.80)
Nickel Recovery	Rock Type	102	RA (NIS.G,0 ,0.1 ,0.75)
Nickel Recovery	Rock Type	104	RA (NIS.G,0 ,0.1 ,0.7,0.15,0.8)
Nickel Recovery	Rock Type	106	RA (NIS.G,0 ,0.1 ,0.6,0.15,0.7,0.2,0.8)

Note: nickel recovery is expressed as a range for the different rock types. For example, for rock type 106, if the NiS grade is below 0.1 %, the recovery is 0%; if the NiS grade is between 0.1% and 0.15%, the recovery is 60%; if the NiS grade is between 0.15% and 0.2%, the recovery is 70%; and if the grade is higher than 0.2%, the recovery is 80%.

Sensitivity Analysis

AMEC carried out a set of sensitivity analyses for the saleable concentrate option to demonstrate how the NPV changes with variations of the nickel price, selling cost and process operating cost. The Ni price was varied from US$7.50 +60% to -20%. The base selling cost of US$3.23 was varied from +20% to -20%; this includes: refining/smelting, metal transport, insurance, marketing, and shipping. The process cost of CDN$5.08, including general and administrative (G&A) costs, was varied from +20% to -20%.

The ultimate pit was selected base on the maximum NPV. There were no pushbacks selected in the analyses. From the previous preliminary assessment, it was found that schedules based on one pushback per five years of production results in NPV values closest to the Best Case NPV value. The sensitivity analysis shows that the deposit is very sensitive to the nickel price i.e., a 10% increase in metal price from the base case results in an increase of the Best Case NPV to nearly CDN$380 million.

A discount rate of 10% was used in the NPV analysis. All NPV calculations in this analysis excluded capital costs. At the time of this analysis, the capital cost was estimated to be about CDN$1 billion. Consequently, it was decided between HCNC and AMEC to continue the study on the basis of using the hydrometallurgical process where NPV values were more economically viable than the saleable concentrate option.

19.1.5 Hydrometallurgical Process Option

Lerchs-Grossmann Pit Shells

The inputs to the LG program are shown in Table 19-4.

Table 19-4: Inputs to L-G Shells

Items	Unit		Assumptions
Mining Cost	CDN$/t mined	1.40	
Process + G&A Costs	CDN$/t milled	7.94	
Ni Price	CDN$/lb	7.89	based on US$ 7.50/lb
Selling Cost	CDN$/lb	0.00	all transport costs included in G&A
Exchange Rate	US$/CDN$	0.95	
Nickel Recovery	Rock Type	101	RA(NIS.G,0 ,0.1 ,0.65)
Nickel Recovery	Rock Type	102	RA(NIS.G,0 ,0.1 ,0.61)
Nickel Recovery	Rock Type	104	RA(NIS.G,0 ,0.1 ,0.57,0.15,0.65)
Nickel Recovery	Rock Type	106	RA(NIS.G,0 ,0.1, 0.48,0.15,0.57,0.2,0.65)

The mining cost estimate was based on a comparison with similar mines in Canada. The process cost was estimated by AMEC, based on a comparison with operations in similar settings. The recoveries were based on the metallurgical testwork completed to date by HCNC. AMEC reviewed the work and further refined the values. The process recovery varies with the nickel sulphide grade. The recovery factor takes into account recovery in the concentrator, hydrometallurgical facility and the amount of payable nickel by the final purchaser. Variable specific gravity values were used depending on the lithological unit. The nickel price used was US$7.50 is lower than that used in the resource estimations (US$8.25 for resources).

19.1.6 Revenue per Tonne Basis

The summary of the ‘in pit shell resource’ was prepared based on the break-even cut-off of 0.1% nickel sulphide. The cut-off is dictated by the process recoveries assuming that no nickel is recovered for the mineralization grading less then 0.1% NiS. This constraint is due to the limited number of samples with grade less then 0.1% NiS.

To understand the magnitude of the further potential of processing “low-grade mineralization” (i.e., less than 0.1% NiS), the break-even cut-off was calculated using the average recovery for two major lithological units. The quantity of Dunite-Wehrlite together with Serpentinite accounts for more then 80% of the deposit. The calculated average recovery for Dunite-Wehrlite and Serpentinite is 70%. This number was used in the calculations to establish the break–even cut-off, if the recovery for lower then 0.1% NiS was known (Table 19-5).

The break-even cut-off calculation, providing nickel can be recovered below 0.1% NiS grade, is as follows:

$$((7.94+1.4)/0.95)/((7.5)*(70/100))/22.0462=0.085\%\ \text{NiS}$$

Table 19-5: Break-even Cut-off Calculation Inputs

Item	
CDN$/tonne Milled	7.94
CDN$/tonne Mined	1.40
CDN$/tonne Rehandled	0.50
US$/CDN$	0.95
% Recovery (0 to 100)	70
% of metal price paid for product transportation and insurance	-
lb/tonne	22.0462
Price (CDN$/lb Ni)	7.89

From the calculation it appears that mineralization with a grade over 0.085% NiS is economically interesting. It should be noted that no mining dilution was considered in this study as most mineralization/waste contacts are contacts between material slightly above cut-off and slightly below cut-off. Also, no recovery reduction of stockpiled material was considered.

For the remaining part of this study, the cut-off was 0.1% NiS.

19.1.7 Geotechnical Assessment

The geotechnical database is not complete, but in general the rock is expected to be competent. The main lithological types are peridotite and pyroxenite. At the southwest side of the planned pit, some phyllite may occur in the pit wall.

At this stage of the study, a preliminary overall pit slope angle, including ramps, of 45° was used as the rock is expected to be generally competent. AMEC performed sensitivity analyses on two slope angles: 41° and 43° against the base case of 45°. A 2° reduction in slope angle results in a 3% loss in mill feed, while a 4° reduction results in 5.5% loss in mill feed.

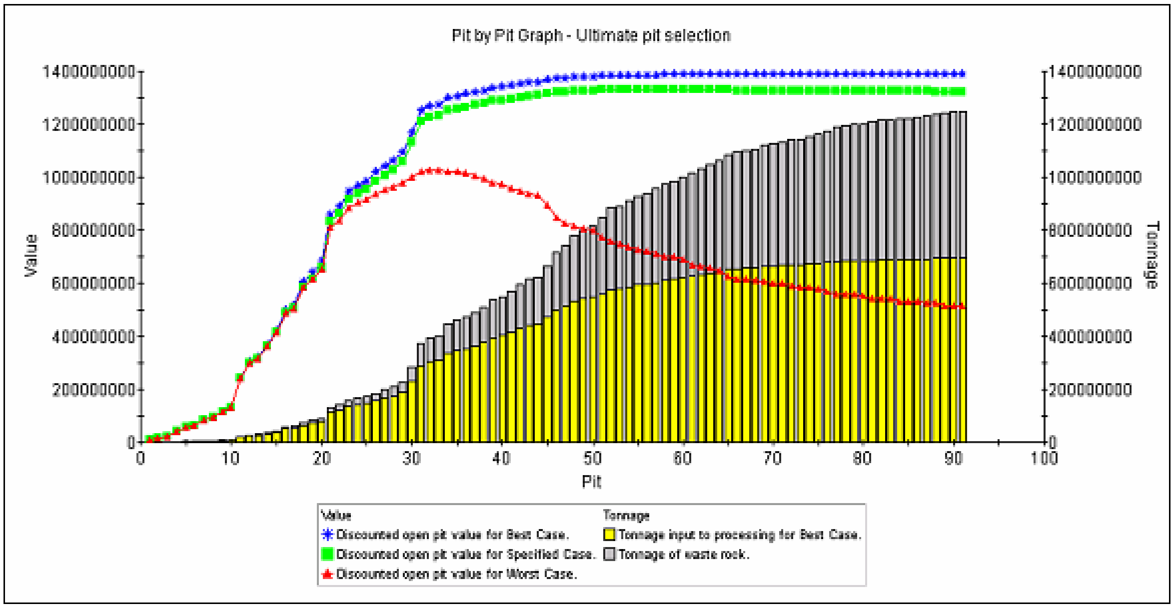
The slope angles within the potential pits and dumps should be further optimized as the project proceeds towards the pre-feasibility stage. The regional structure, major faults and discontinuities, if any, should be identified for further refining of the slope angles.

19.1.8 Ultimate Pit Selection

AMEC optimized the Main deposit using NiS and cobalt in order to quantify the additional potential when considering revenue from cobalt. When compared to using NiS only, the additional tonnage was approximately 20 Mt. As this is a relatively small tonnage, and only preliminary recovery information for cobalt was available, it was decided to limit all further optimizations to NiS only.

The ultimate pit shell was selected based on the ‘Pit by Pit’ graph output (see Figure 19-3).

Figure 19-3: Pit by Pit Analysis



There are 91 pits resulting from optimization. The revenue factors used are multiples of 0.01. The starting factor equals to 0.30 while the last one equals to 1.50.

The two curves represent NPV for the Best Case Mining, an ideal mining scenario using a large number of pushbacks being mined shell by shell without considering capital cost or practical mining widths. The ‘Best Case Mining’ curve is the top curve in blue color. The ‘Worse Case Mining’ red curve represents the NPV per pit in the case that the pit is mined bench by bench without any pushbacks. From recent studies of the Turnagain deposit, it was found that with several pushbacks, the NPV for the specified case (i.e., a pushback every five years) approaches close to the Best Case Mining NPV.

A discount rate of 10% was used in the NPV analysis. Capital investment was not considered in the Pit by Pit analysis.

Based on the Pit by Pit graph, pit 47 with a revenue factor 0.76 was selected as an ultimate pit shell for the mining schedule and financial analysis. As shown on the graph the NPV doesn’t increase significantly beyond pit shell 47. More investigation of the schedule could prove that a smaller ultimate pit shell or alternatively five pushbacks instead of six could result in an improved NPV value.

At this stage of the project the reporting of grade and tonnes was based on the undiluted resource model. More detailed studies of the dilution factor and dilution grade should be performed to account for the equipment sizing and the nature of the deposit. For the next level of study any small irregularities should be removed from the pit shells before the final schedule is derived.

Table 19-6 shows the contained tonnes and grade in the L-G ultimate shell. This includes a 100 m setback from the Turnagain River, and a break-even cut-off grade of 0.10% NiS.

Nearly 70% of the resources contained in the ultimate pit shell are in the Measured or Indicated category. In order to assess the impact of using Inferred material in this PA, AMEC undertook an additional optimization process using the same set of parameters, allowing only material classified as Measured or Indicated to be considered as mill feed.

From the Pit by Pit graphs obtained for both cases, MII (i.e., Measured, Indicated and Inferred categories) and MI (i.e., Measured and Indicated categories), it appears that the ultimate pit shell lies in the same range of revenue factors. An RF of 0.76 was used in both cases. The pushbacks were chosen on a five year production basis. In the MII case, six pushbacks were used, while the MI case four pushbacks were selected.

The mill feed tonnage is about 40% less when not considering the Inferred material. The difference in NPV when comparing these two schedules is about CDN$170 million. AMEC is aware that HCNC performed additional drilling in the zones currently classified as Inferred. This additional drilling may help to reduce the risk (or impact) of Inferred material by converting it to a higher confidence classification. For a pre-feasibility level study Inferred resources cannot be considered as potential mill feed and must be treated as waste. Table 19-7 compares the tonnage of mill feed, waste and average feed grade for NiS resulting from these two sets of optimizations. MII is used in one case, and MI is used exclusively in the other case.

Table 19-6: Tonnes and Grades Contained in Ultimate Pit Shell

NiS >0.10% , undiluted 'in pit resources in ultimate pit shell	Mill Feed (kt)	Waste (kt)	NI_TOT %	AC_NI %	S %	MG %	NIS %	AC_CO %
Measured >0.10% NiS	56,611	-	0.252	0.203	1.052	22.644	0.187	0.012
Indicated >0.10% NiS	290,871	-	0.222	0.176	0.860	21.600	0.164	0.010
Measured + Indicated >0.10% NiS	347,482	-	0.227	0.181	0.891	21.770	0.167	0.010
Inferred >0.10% NiS	169,941	-	0.199	0.155	0.875	20.554	0.145	0.010
Waste	-	229,541	-	-	-	-	-	-

Note: It could be expected that about 5% of the resource will be lost in the internal ramp. Includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that these data will be realized.

Table 19-7: Comparison of Contained Tonnes and Grades

	NPV specified Case 10% DR ($000's)	Mill Feed (kt)	Waste (kt)	Avg. NiS %
MI	1,221,495	305,064	192,140	0.170
MII	1,389,868	516,619	225,737	0.160

Note:
1. Mineral resources are not mineral reserves and do not have demonstrated economic viability
2. The preliminary assessment is preliminary in nature, it includes Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PA will be realized.
3. MII (i.e., Measured, Indicated and Inferred categories); MI (i.e., Measured and Indicated categories)

In the base case, the Measured and Indicated mineral resources in mill feed add up to 347 Mt. By removing the Inferred mineral resources, the tonnes contained in Measured and Indicated resources are 305 Mt, which corresponds to a reduction of 42 Mt or 12% of the mill feed.

19.1.9 Production Schedule

The Main zone extraction method consists of two pits which join towards the surface and have several common benches. The production schedule is common for both pit bottoms. The schedule is based on a milling rate of 50,000 t/d. No trade-off study was performed on throughput.

The schedule is preliminary in nature and more work is required to smooth the feed to the plant. The bench advance is reasonable and varies from 4 to 12 per year to be mined by two loading units and a spare wheel loader. The mine plan considers a minimum mining width of 50 m.

19.1.10 Pushback Selection

The annual schedule is based on a starter pit shell and five pushbacks. The pushbacks are selected based on five years of production. The revenue factors for the selected pushbacks are 0.48 (for the starter pit shell, producing for four years), 0.56, 0.59, 0.63, 0.71 and the ultimate pit shell 0.76 respectively. Table 19-8 shows the data related to the six pushbacks selected for the final schedule. The data is obtained through Pit by Pit analysis.

The selected pushbacks were processed with the Whittle module "Minimum Mining Width" which is used to reformat the selected pit shells into a new set of pit shells representing the defined number of pushbacks and the 50 m minimum mining width to ensure sufficient working space for the planned shovel and truck operation.

Table 19-8: Pit by Pit Analysis

Pit Shell Number	Revenue Factor	NPV$ Best Case ($ disc)	NPV$ Specified Case ($ disc)	NPV$ Worst Case ($ disc)	Processed Tonnage (kt)	Waste Tonnage (kt)	NiS grade %	Mine Life (years)
19	0.48	642,770	617,401	617,401	71,577	12,372	0.183	4.0
27	0.56	1,042,336	1,004,812	953,625	166,713	31,026	0.170	9.3
30	0.59	1,170,578	1,130,713	997,964	229,244	55,467	0.160	12.7
34	0.63	1,301,147	1,255,122	1,019,913	338,075	109,509	0.150	18.8
42	0.71	1,352,703	1,301,643	947,303	429,640	163,290	0.145	23.9
47	0.76	1,375,880	1,322,946	827,976	516,619	225,737	0.141	29.0

Note:

1. Mineral resources are not mineral reserves and do not have demonstrated economic viability
2. The preliminary assessment is preliminary in nature, it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary assessment will be realized.

Mining Schedule

While analyzing the preliminary schedules, the NPV values were used to establish the favourable mining scenarios. Previous studies showed that by limiting the process by a single, selected elevated cut-off resulted in lower NPV then the value obtained with the Whittle Opti-Cut cut-off optimizer. In order to maximize the NPV, AMEC used the ‘Opti-Cut’, cut-off optimizer based on the Milawa algorithm. Several scenarios with a range of various equipment capacities were run. The mining limits used were 35 Mt/a, 40 Mt/a, and 50 Mt/a. The final schedule used in the financial analysis (Section 19.7) used a mining limit set to 35 Mt/a.

One low-grade stockpile was incorporated into the schedule allowing for processing of higher-grades earlier in the schedule while lower grade mill feed is being stockpiled. The material from low-grade stockpile is used during the years where there is a shortage of mill feed for processing. The remaining portion of the low-grade material is processed in last years of Life-of-Mine (LOM) Years 26 through 29. The cost related to rehandling from the stockpile is CDN$0.50/t moved. More detailed cost analysis will be required after refining of the stockpile location and layout.

The preliminary schedule is shown in Figure 19-4. The tonnes and grades produced per year are shown in Table 19-9. Production is planned for 29 years. In the last four years the material from the stockpile will be processed. In Years 10, 11, 14 and 15 small amounts of mill feed are taken from the stockpile.

AMEC created the preliminary pit design shown in Figure 19-5 with 32 m wide haul roads. The design is preliminary but using a double benching approach, it fits the ultimate pit shell closely.

Figure 19-4: Preliminary Schedule

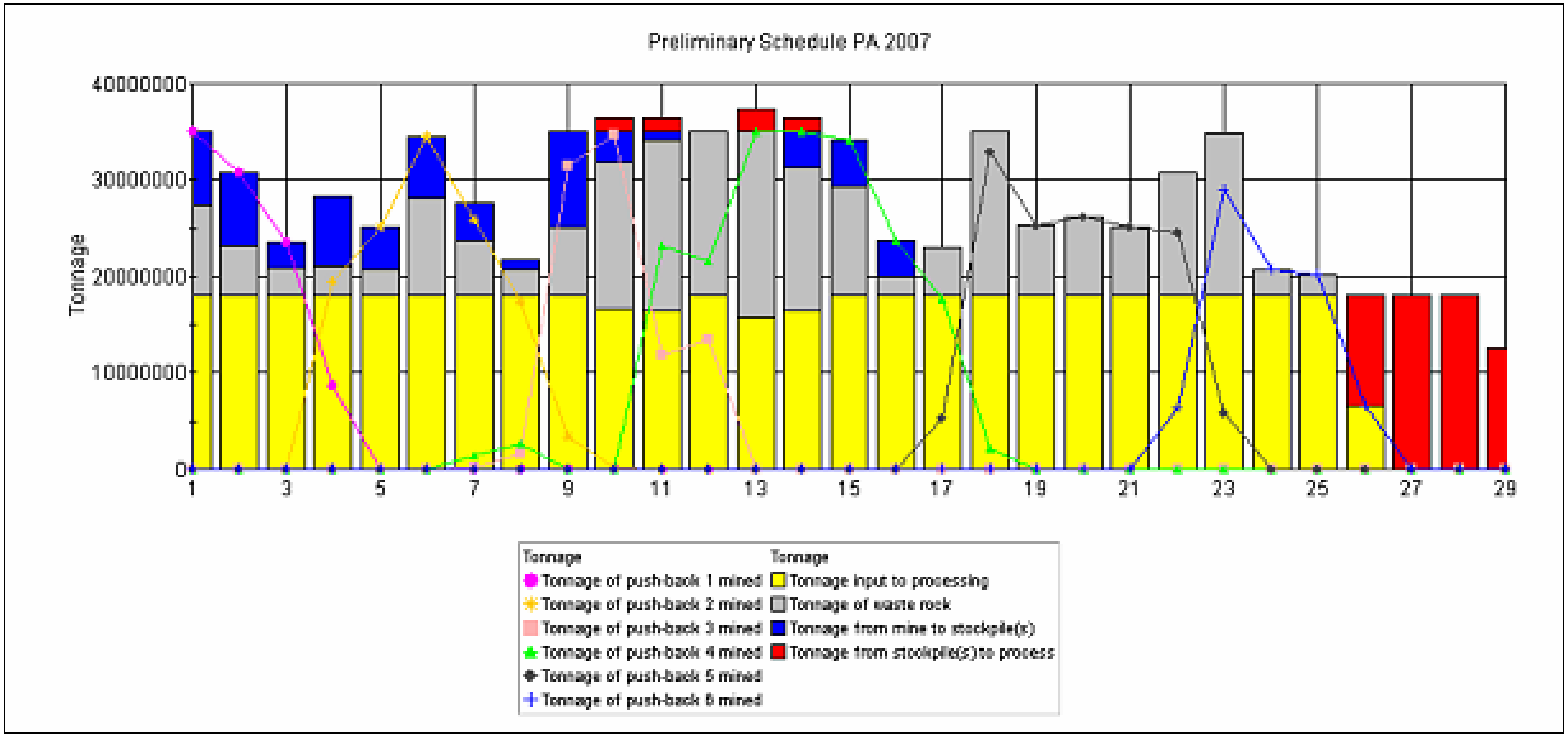
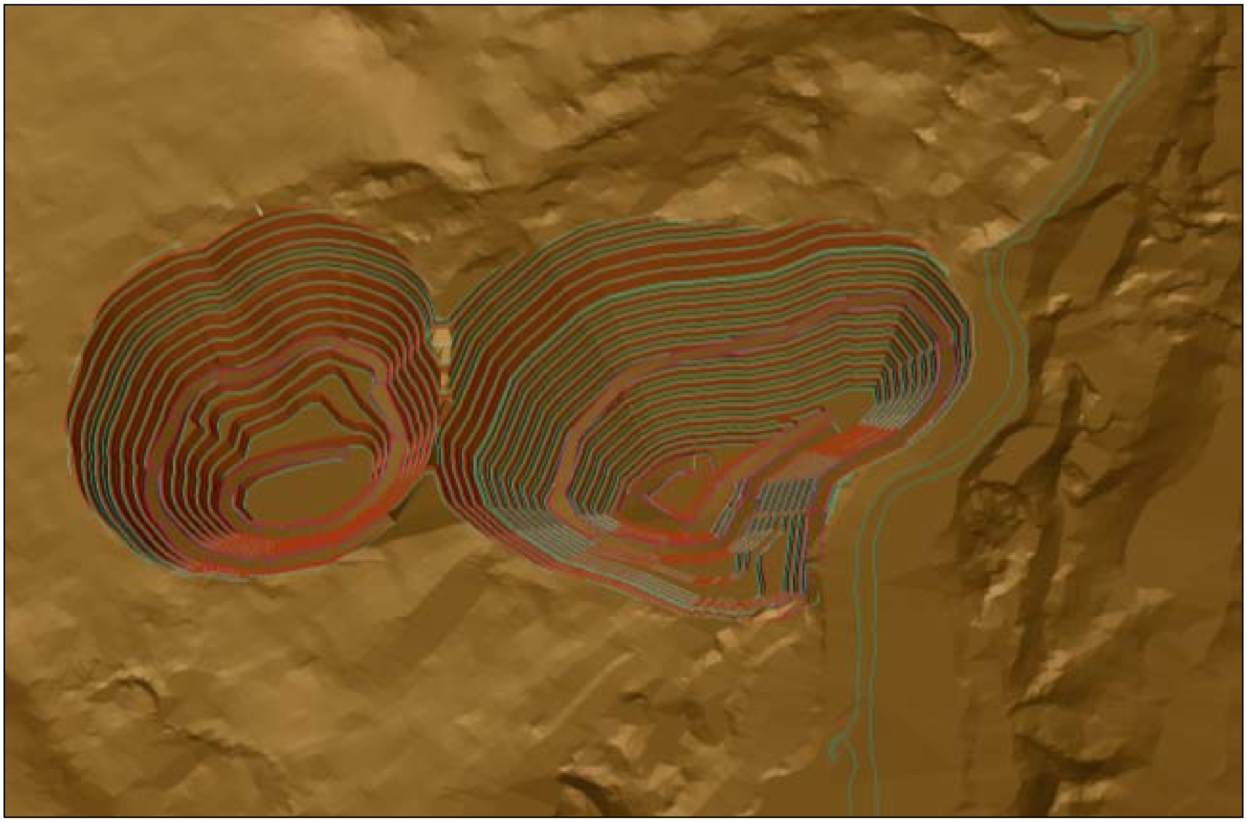


Table 19-9: Preliminary Schedule

Period Year	Mill Feed Produced (t)	Waste (t)	Strip Ratio	Avg. NiS %	To Stockpile (t)	From Stockpile (t)	Avg. NiS% Mined+ Stockpile	Push Back1	Push Back2	Push Back3	Push Back4	Push Back5	Push Back6
1	17,980,159	9,541,376	0.95	0.185	7,478,465	-	0.185	35,000,000	-	-	-	-	-
2	18,000,000	5,201,359	0.71	0.201	7,587,115	-	0.201	30,788,474	-	-	-	-	-
3	18,000,000	2,861,055	0.31	0.224	2,684,014	-	0.224	23,545,069	-	-	-	-	-
4	18,000,000	2,990,790	0.57	0.196	7,312,843	-	0.196	8,759,226	19,544,407	-	-	-	-
5	18,000,000	2,685,910	0.40	0.182	4,481,978	-	0.182	-	25,167,888	-	-	-	-
6	18,000,000	10,169,510	0.92	0.170	6,363,707	-	0.170	-	34,473,842	59,376	-	-	-
7	18,000,000	5,759,028	0.53	0.181	3,802,629	-	0.181	-	25,923,017	139,819	1,498,822	-	-
8	18,000,000	2,696,417	0.21	0.188	1,030,684	-	0.188	-	17,365,073	1,651,334	2,710,695	-	-
9	17,999,977	7,159,136	0.94	0.167	9,840,888	23	0.167	-	3,427,844	31,572,155	-	-	-
10	16,593,771	15,236,041	1.11	0.168	3,170,189	1,406,229	0.164	-	263,919	34,736,081	-	-	-
11	16,532,295	17,597,193	1.12	0.150	870,512	1,467,705	0.148	-	-	11,775,953	23,224,047	-	-
12	17,991,942	17,008,058	0.95	0.172	-	8,058	0.172	-	-	13,493,863	21,506,137	-	-
13	15,672,770	19,327,230	1.23	0.135	-	2,327,230	0.133	-	-	-	35,000,000	-	-
14	16,540,707	14,762,211	1.12	0.140	3,697,082	1,459,293	0.138	-	-	-	35,000,000	-	-
15	18,000,000	11,385,207	0.90	0.152	4,765,664	-	0.152	-	-	-	34,150,871	-	-
16	18,000,000	1,998,440	0.31	0.179	3,634,738	-	0.179	-	-	-	23,633,179	-	-
17	18,000,000	5,050,292	0.28	0.183	-	-	0.183	-	-	-	17,658,875	5,391,417	-
18	18,000,000	16,984,142	0.94	0.143	-	-	0.143	-	-	-	2,082,380	32,901,762	-
19	18,000,000	7,373,567	0.41	0.143	-	-	0.143	-	-	-	-	25,373,567	-
20	18,000,000	8,262,058	0.46	0.147	-	-	0.147	-	-	-	-	26,262,058	-
21	18,000,000	7,082,827	0.39	0.147	-	-	0.147	-	-	-	-	25,082,827	-
22	18,000,000	12,851,640	0.71	0.150	-	-	0.150	-	-	-	-	24,551,909	6,299,732
23	18,000,000	16,910,610	0.94	0.150	-	-	0.150	-	-	-	-	5,911,777	28,998,833
24	18,000,000	2,629,284	0.15	0.146	-	-	0.146	-	-	-	-	-	20,629,284
25	18,000,000	2,213,507	0.12	0.146	-	-	0.146	-	-	-	-	-	20,213,507
26	6,587,404	481	-	0.159	-	11,412,596	0.132	-	-	-	-	-	6,587,885
27	-	-	-	-	-	18,000,000	0.117	-	-	-	-	-	-
28	-	-	-	-	-	18,000,000	0.123	-	-	-	-	-	-
29	-	-	-	-	-	12,639,375	0.123	-	-	-	-	-	-
Total	**449,899,025**	**225,737,369**			**66,720,508**	**66,720,509**		**98,092,769**	**126,165,990**	**93,428,581**	**196,465,006**	**145,475,317**	**82,729,241**

Figure 19-5: Preliminary Pit Design with Haul Roads



Due to the preliminary nature of the design, no tonnage and grade comparisons were undertaken for the ultimate pit shell and detailed design. In the future a trade-off study for varying ramp location options will need to be considered. The location of the waste rock pile and plant are of a preliminary nature and are also should be subject to a trade-off study that will also consider the tailings pumping costs. The layout of the main haul roads and ramps might change with a better understanding of any geotechnical limitations.

19.1.11 Capital Cost Estimate (Mining)

Capital Cost Estimate

The capital cost estimate for mining is based on the preliminary equipment fleet selected for the Turnagain Nickel project. The fleet size is based on supplying 18 Mt/a of feed to the mill. The haulage fleet will consist of 220 ton or 240 ton trucks, to be loaded by electric cable shovels and a wheel loader. Waste and low-grade stockpile material will be moved at an average ratio of 0.65 per tonne milled. At the end of mine life, the low-grade stockpile will be recovered to the mill. A proposed equipment list is included as Table 19-10.

The configuration may need to change when more information becomes available. Considering the recent increases in capital costs, energy prices and delivery times, a future trade-off study between electric cable shovels and electric hydraulic shovels should be considered.

Table 19-10: Mine Equipment List

Year	1	2	3	4	10	15	20	25	29
Drills	2	2	2	2	2	2	2	2	0
Shovels	2	2	2	2	2	2	2	2	1
Loaders	1	1	1	1	1	1	1	1	1
Trucks	8	8	7	7	12	14	9	9	4
Track Dozers	4	4	4	4	4	4	4	4	2
Graders	2	2	2	2	2	2	2	2	1
Wheel Dozer	1	1	1	1	1	1	1	1	1
Water/ Sand Truck	1	1	1	1	1	1	1	1	1
Backhoe w/ rockbreaker	1	1	1	1	1	1	1	1	1
Backhoe for ditches	1	1	1	1	1	1	1	1	1
Dewatering pump+pipes	1	1	1	1	1	1	1	1	1
Lowbed truck+trailer	1	1	1	1	1	1	1	1	1
Fuel/lubrication truck	1	1	1	1	1	1	1	1	1
Tire manipulator	1	1	1	1	1	1	1	1	1
Forklift 30 t	1	1	1	1	1	1	1	1	1
Forklift 10 t	1	1	1	1	1	1	1	1	1
Water truck for drills	1	1	1	1	1	1	1	1	1
Maintenance truck	1	1	1	1	1	1	1	1	1
Multi tool truck	1	1	1	1	1	1	1	1	1
Blasting ancillary truck	1	1	1	1	1	1	1	1	1
Blasting loader	1	1	1	1	1	1	1	1	1
Mobile crane, 100 t	1	1	1	1	1	1	1	1	1
Lighting plant	3	3	3	3	3	3	3	3	3
Ambulance	1	1	1	1	1	1	1	1	1
Crew bus	1	1	1	1	1	1	1	1	1
Pickup trucks	10	10	10	10	10	10	10	10	10
Dispatch System	1	1	1	1	1	1	1	1	1
Engineering Software	1	1	1	1	1	1	1	1	1
Engineering Equipment	1	1	1	1	1	1	1	1	1

Equipment replacement is based on the life cycle of the equipment, which was assumed to be 60,000 operating hours or 10 years of production for the trucks and drills, 42,000 operating hours or 7 years for support equipment and loaders and no replacement for the cable shovels. This schedule led to the mine equipment capital cost shown in Table 19-11. The initial capital required for mine production equipment, mine operations support equipment, and mine maintenance mobile equipment, as well as mine engineering software licenses and equipment are included in this table.

No pre-stripping was considered, although some in-pit material might be suitable for the construction of the tailings dam and therefore could benefit both.

Table 19-11: Mine Equipment Capital Cost (CDN$M)

Year	-1	1	2	3	4	10	15	20	25	29
Drills	7.8	-	-	-	-	7.8	-	7.8	-	-
Shovels	38	-	-	-	-	-	-	-	-	-
Loaders	4.4	-	-	-	-	4.4	-	4.4	-	-
Trucks	29.6	-	-	-	-	44.4	-	33.3	-	-
Track Dozers	6.8	-	-	-	6.8	6.8	6.8	6.8	3.4	-
Graders	3.2	-	-	-	-	3.2	-	3.2	-	-
Wheel Dozer	1.6	-	-	-	-	1.6	-	1.6	-	-
Water/ Sand Truck	2	-	-	-	-	2	-	2	-	-
Backhoe w/ rockbreaker	0.55	-	-	-	-	0.55	-	0.55	-	-
Backhoe for ditches	0.88	-	-	-	-	0.88	-	0.88	-	-
Dewatering pump+pipes	0.5	-	-	-	0.5	0.5	0.5	0.5	0.5	-
Lowbed truck+trailer	2.7	-	-	-	-	2.7	-	2.7	-	-
Fuel/lubrication truck	0.4	-	-	-	-	0.4	-	0.4	-	-
Tire manipulator	0.8	-	-	-	-	0.8	-	0.8	-	-
Forklift 30 t	0.4	-	-	-	-	0.4	-	0.4	-	-
Forklift 10 t	0.2	-	-	-	-	0.2	-	0.2	-	-
Water truck for drills	0.3	-	-	-	-	0.3	-	0.3	-	-
Maintenance truck	0.25	-	-	-	-	0.25	-	0.25	-	-
Multi tool truck	0.3	-	-	-	-	0.3	-	0.3	-	-
Blasting ancillary truck	0.3	-	-	-	-	0.3	-	0.3	-	-
Blasting loader	0.27	-	-	-	-	0.27	-	0.27	-	-
Mobile crane, 100 t	2	-	-	-	-	-	2	-	-	-
Lighting plant	0.09	-	-	-	0.09	0.09	0.09	0.09	0.09	-
Ambulance	0.07	-	-	-	-	-	0.07	-	-	-
Crew bus	0.04	-	-	-	0.04	0.04	0.04	0.04	0.04	-
Pickup trucks	0.4	-	-	-	0.4	0.4	0.4	0.4	0.4	-
Dispatch System	1.5	-	-	-	-	1.5	-	1.5	-	-
Engineering Software	0.25	-	-	-	0.25	0.25	0.25	0.25	0.25	-
Engineering Equipment	0.2	-	-	-	0.2	0.2	0.2	0.2	0.2	-
Total	**105.8**				**8.28**	**80.53**	**10.35**	**69.43**	**4.88**	

Note: The numbers are all in third quarter 2007 Canadian dollars, using an exchange rate of CDN$1 = US$0.95 and have an accuracy of ±35%.

Besides these investment costs, the following capital should be included: $5 million for mine electrical infrastructure, $2.5 million for a permanent screening plant for road sanding and stemming, $4 million for spare parts for mining equipment and $2.4 million for tires.

19.1.12 Haulage Considerations – Mill Feed, Waste Dump and Low-grade Stockpile

AMEC reviewed mill feed haulages and proposed distances. Approximate distances include 1 km from the rim of the Northwest pit to the crusher, located immediately southwest of the Main pit; and 400 m from the Main pit rim to the plant. All roads are assumed to have ramps at a maximum of 10%. The 100 m wide area between the river and the pit rim will also have a 40 m wide haul road, preferably on the higher cliff.

The area between the crusher and plant location and the proposed tailings storage facility is indicated for the waste dump and low-grade stockpile locations, whereby the low-grade stockpile will be closest to the crusher and the waste dump could be integrated with the tailings dam.

The entrance of the low-grade stockpile is considered to be 750 m from the Northwest pit entrance and 1,500 m from the Main pit entrance. The haul distance on the low-grade stockpile is considered to increase by 150/a, 5% uphill.

The waste rock dumps can be developed as end-dumped, angle-of-repose structures within the indicated areas on the general site plan. The waste rock is expected to stay at an angle of repose of approximately 35° and end-dumping, even at higher dump heights, should be possible if the base of the dump is also in good condition. Reclamation objectives for the waste dumps, which will be confirmed during the environmental permitting process in progress, may dictate alternative geometries. After re-sloping and reclamation, the waste dump will require a larger footprint. The waste dump entrance is considered to be 1,700 m from the Northwest pit entrance and 2,450 m from the Main pit entrance. The haul distance on the waste dump is considered to increase by 100 m/a, 5% uphill. No waste dump is considered at the northeast side of the Main deposit as it cannot be ruled out that a potential resource may be present in this region as already proven with the Duffy deposit.

Consideration should be given to the possibility to use material from within the deposit for the initial dam construction. This rock, if suitable, might also be used in the tailing storage dam, when this facility needs to have its height increased. More detailed geotechnical investigations are also required to indicate a safe distance between the plant site and low-grade stockpile.

Limited data were provided concerning potential acid rock drainage and no special facilities have been considered. This however needs to be confirmed with comprehensive sampling and detailed environmental work.

Exploration drilling should be conducted to confirm that the potential waste dumps, mill site and tailings facility will not be built on top of potential future resources.

19.1.13 Operating Cost Estimate (Mining)

Mining Cost Estimate

The operating cost estimate for mining is based on recent information and has been benchmarked against other similar operations in North America. The estimated average cost is CDN$1.40/t moved including drilling, blasting, loading, hauling, support and general mine expense cost, which consists of operational, maintenance and engineering overhead as shown in Table 19-12.

Table 19-12: Estimated Mine Operating Costs

Mine – Unit Operating Costs	Cost (CDN$/t)
General Mine Expense	0.16
Drilling	0.07
Blasting	0.18
Loading	0.20
Hauling	0.60
Support	0.20
Total Mine Operations	**1.40**

The fleet is expected to be capable of mining sufficient material to supply 18 Mt/a of feed to the mill as well as the associated waste, which might be moved forward a bit in the later years to smooth mining capacity requirements. The actual costs will vary with the strip ratio, haul distances, pit depth, required mill feed and other parameters. This cost estimate is based on a diesel fuel price of CDN$0.85/L and third quarter, 2007 Canadian dollars at an exchange rate of CDN$1.00 = US$0.95 without allowance for inflation or escalation.

In Years 10 to 14, small tonnages from the Low-Grade Stockpile are fed to the mill from Year 26 to 29 the mill feed only consists of material from the Low-Grade Stockpile. The cost considered for rehandling this material is estimated at CDN$0.50/t.

Estimated fuel consumption for the mining equipment is 14 ML/a initially, peaking at 20 ML/a later in the mine life. ANFO requirements varies between 6 Mkg/a and 9 Mkg/a, assuming (based on industry norm) an average powder factor of 0.25 kg/t.

Work Force

Mining work force estimates are based on the mine fleet and a maintenance staff to operating staff ratio of 0.6. The staffing schedule includes positions related to mine operations, engineering, geology, and maintenance. Mine work force numbers range from a minimum of 116 persons during Year 29, to a maximum of 219 persons in Year 15.

19.2 Tailings and Waste Management

Knight Piésold Ltd. (KPL) conducted an evaluation of potential tailings management facility options in the vicinity of the Turnagain Property in April 2006 (Knight Piésold, 2006). The evaluation included six potential tailings impoundment sites in four different valleys. The potential tailings impoundment sites were sized for an ultimate capacity of approximately 300 Mm^3 of waste, anticipated to be generated from the proposed operation. The evaluation indicated that an integrated waste and water management facility along Hard Creek represents the most economic and environmentally preferable option for development. Other factors that may have a major influence on the location and/or arrangement of the preferred waste and water management facilities include land tenure and/or fisheries and aquatic resources. These were not factored into the 2006 evaluation but were recommended for future study programs. It was also recommended that the optimum configuration for the Hard Creek impoundment be updated once mineral reserves have been estimated, and when rough quantities and characteristics of the waste materials are determined.

KPL completed a preliminary mine development alternatives assessment for the Turnagain Project in July 2007 (Knight Piésold, 2007a). This assessment included a re-evaluation of the potential Hard Creek tailings management facility options. Five potential tailings management facilities were considered in the Hard Creek Valley, sized to store approximately 900 Mm^3 of a combination of tailings, waste rock, and water. An evaluation of the five sites indicated two of the potential tailings management facilities were preferred. Two preliminary mine development alternatives were developed based on the two preferred tailings management facilities, including preliminary conceptual layouts for other water and waste management components (i.e., waste dump locations, tailings and water reclaim pipelines, etc.). Mine Development Alternative 1 (MDA 1) was determined to be the preferred alternative based on a number of criteria. It was noted that more detailed engineering design; environmental impact assessment and permitting work; and economic analyses will be required for future studies.

KPL updated the MDA 1 water and waste management layout in November 2007, to provide a basis from which future optimizations and alternatives may evolve as the project design basis is further defined (Knight Piésold, 2007b). The updated water and waste management layout includes a preliminary design of the preferred waste management facility, to store tailings and potentially acid generating waste rock. The updated layout also includes preliminary designs of a non-reactive waste dump; a water management system including surface water diversions; a tailings delivery system; and a water reclaim system.

A geotechnical site investigation program was carried out in August and September 2007. The fieldwork included geotechnical drilling, installation of standpipe piezometers, falling head permeability tests, test pit excavation and surficial mapping. The drill hole and test pit locations were selected by KPL based on the proposed arrangement of tailings storage facility embankments, dumps, and stockpiles to help develop an understanding of regional geotechnical conditions.

19.3 Infrastructure

19.3.1 Plant Site Location

Information from HCNC is that bedrock exposure is confined to the higher-relief drainages and along mountainous ridges. A thin layer of glacial till (0 to 6 m in depth) covers the majority of the property outside of the floodplain of the Turnagain River.

The process facilities and ancillary facilities will be located on a cut/fill benched platform located immediately to the west of the open pit, north of the Turnagain River and to the east of Hard Creek. The planned site layout was included as Figure 19-1.

19.3.2 Process Facilities

Process facilities include:

- primary crushing
- coarse mineralization stockpile and fine crushing
- mill complex and structures, including hydrometallurgical facilities and product load-out.

19.3.3 Ancillary Facilities

Ancillary facilities include:

- maintenance shops and warehouse
- assay laboratory
- administration, security, and first aid offices.

19.3.4 Electrical Distribution

Electricity to the primary crushing plant is expected to be distributed via a 25 kV overhead line from the main substation; the line would continue on to feed sands plants and tailings dam. The secondary and tertiary crushing plant would be supplied by an underground 25 kV cable. The grinding mills would be fed off the main substation 25 kV bus, with converters to step down the voltage to items such as the motor starters, substations, sewage and water treatment plants, water and fire pumps. An allocation for synchronous condensers allows for power factor correction for the rest of the plant, bolsters the fault level of the 25 kV bus and provides voltage stability.

Emergency power will be provided by 600 V diesel generators. UPS units will be used where necessary.

19.4 Project Execution Plan

An engineering, procurement, and construction management (EPCM) approach is proposed for the project, with the EPCM contractor providing staff in the field to manage sub contractors who will be constructing the plant. Pre-engineered buildings and modular construction will be used to the greatest extent possible, to accelerate construction and reduce on-site labour costs.

Detail engineering will follow, where possible, the concepts and mine plan developed during a future feasibility study. A basic engineering phase and scope will then be identified and executed, to review, firm-up and freeze key design documentation.

There are three principal areas of detail design to be undertaken:

- processing systems
- on-site infrastructure systems and ancillary buildings
- external infrastructure systems: access road and power supply.

Mine systems will be designed to BC and North American standards and include a maximum of pre-assembly and modularization of components. The Project will be designed using Metric (SI) units.

A typical schedule for a project of this size, in its setting, predicts an overall elapsed time from completion of project financing through to project completion, of 24 to 30 months, most of which will be required for EPCM activities.

19.5 Capital Costs

The total estimated capital cost to design and build the Turnagain Project described in this report is CDN$1,381 million. A capital cost estimate summary is included in Tables 19-13 and 19-14.

Table 19-13: Capital Cost Estimates – Directs

Area (Directs)	($M)
Mining	116.3
Site Development	66.2
Process Facilities	496.9
Ancillary Buildings & Facilities	66.7
Tailings Facility	44.0
Utilities	2.2
Total Directs	**792.3**

Table 19-14: Capital Cost Estimates – Indirect Costs

Area (Indirect Costs)	($M)
Owner's Costs	79.2
Construction Indirects	64.7
Engineering Procurement Construction Mgmt	95.1
Construction Camp & Catering	32.4
Capital Spares	22.0
Freight	38.5
Start-up & Commissioning Allowance	6.5
Total Indirect Costs	**338.4**
Contingency	249.9
Total Capital Cost Estimate	**1,380.6**

All capital costs are expressed in Q3 2007 Canadian dollars incorporating an exchange rate of CDN$1.00 = US$0.91 where appropriate. There are no allowances for escalation, interest during construction, taxes, or duties. All due allowances have been considered for items such as:

- using actual productivity factors and unit prices from similar ongoing and previous projects
- understanding of the open shop contracting environment in BC at this time
- using capital equipment costs provided by AMEC
- using estimated budget costs for the power line and main access road
- using actual current costs of bulk materials such as cement and steel
- using experienced construction management supervision during construction

- using traditional methods of applying percentages for the costs associated with piping, electrical and instrumentation work
- using mine fleet sizes estimated by AMEC
- assuming that the backfill for the crusher structure is provided by the mine operations from the pit
- estimating building costs from layouts combined with database information
- using estimated costs for camp
- assuming that the explosives supplier provides the above ground facilities
- using the configuration of the reclaim water and surplus water system derived
- understanding of remoteness of project
- batching of concrete on site and locally available suitable aggregate.

The estimate covers the direct field costs of executing this project, plus the indirect costs associated with design, procurement, and construction efforts.

Labour rates have been derived as a composite for 70 h/wk, 7 d/wk schedule including travel. The approximately CDN$112.32/h average wage rate is a combined rate for an estimated crew mix (supervision, skilled/unskilled labour) using the rates from the AMEC database for a general contractor in the mining industry in BC.

Owner’s costs have been developed in conjunction with the Owner and include items such as:

- all operating costs for the three-month preproduction period
- insurances
- head office staff assigned to the project
- Owner’s allowances for field operations offices and supplies
- Owner’s travel costs during construction.

An allowance of 25% of the direct and indirect costs has been added to the estimate. Earthwork is considered to have the greatest risk in this project due to limited information.

19.6 Operating Costs

19.6.1 Summary

Data for this estimate was based on actual quotations, current regional salary rates, and information from similar operations, and supplemented with additional information using the Mining Journal Reference Book.

The operating cost summary is shown in Table 19-15.

Table 19-15: Operating Cost Estimates

Area	$/t
General & Admin	
G&A Labour	0.14
Direct	0.29
Subtotal	0.43
Mining	1.40
Processing	
Process Labour	0.59
Consumables	5.59
Power	1.37
Miscellaneous	0.06
Subtotal	7.60
Total (CDN$/t milled)	**9.43**

19.6.2 Workforce

Table 19-16 lists workforce required over the life of the mine.

Table 19-16: Workforce Totals

Area	Year 1-2	Year 3-9	Year 10-14	Year 15-19	Year 20-28	Year 29
Administration	29	29	29	29	29	29
Processing	135	135	135	135	135	135
Mining	168	161	193	219	174	116
Total	**332**	**325**	**357**	**383**	**338**	**280**

19.6.3 Realization Costs

The realization costs used in the cash flow model include the following:

- trucking to port
- port storage
- ship loading
- draft survey
- umpire sampling
- ocean freight
- payable levels for hydroxide products
- losses
- ocean freight
- insurance.

19.6.4 Sustaining Capital

Sustaining capital for the project over 29 years is CDN$173.5 million.

19.7 Financial Analysis

AMEC carried out a financial analysis of the Project after capital and operating cost estimates were developed.

The results of the following economic analysis represent forward-looking information as defined under Canadian securities law. Forward-looking information in this analysis includes, but is not limited to, statements regarding future mining and mineral processing plans, rates and amounts of metal production, capital and operating costs, tax and royalty terms, smelter and refinery terms, the ability to finance the project, and metal price forecasts.

The analysis depends on inputs that are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here. This assessment is preliminary in nature, includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary assessment will be realized.

Some of the key technical risks include:

- changes in government and changes in regulations affecting the ability to permit and operate a mining operation
- discussions with stakeholders that may be impacted by any proposed mining operation are at a very early stage
- the fact that mineral resources are estimates based on limited sampling data, interpretation of geology and assumptions applied that may change with increased exploration, development and mining
- proximity to the Turnagain River, which may require a deeper setback than 100 m, negatively impacting resources
- actual mining and metallurgical recoveries that may be achieved
- there are a variety of issues related to the production of saleable products by concentration and subsequent hydrometallurgical processing that represent a risk to the project and will have to be addressed in future work
- variations in operating and capital costs
- provision of power

- future metal prices may change from those used in the economic model; the project is very sensitive to metal prices.

19.7.1 Valuation Methodology

The Turnagain Project has been valued using a discounted cash flow approach. The analysis includes sensitivities to variation in nickel prices, operating costs, capital costs, and exchange rates.

The economic analysis has been run with no inflation (constant dollar basis). Capital and operating costs are expressed in third quarter 2007 Canadian dollars, unless otherwise noted.

Financial model parameters incorporated in the analysis, include:

- resource estimates and mine life
- metallurgical balance
- mill and hydrometallurgical plant performance
- nickel product sales terms
- product transportation costs
- metal prices
- exchange rate
- operating costs: mine, process and G&A
- capital costs over LOM.

Estimated annual net cash flows were discounted to the beginning of Project Year -2 at real discount rates using cumulative net cash flow, 5%, 8%, 10%, 12% and 15%.

It is important to note that taxation is not considered in this analysis. Typically taxation is not a factor at this level of assessment.

It should be noted that all financial figures are in Canadian dollars except for the commodity prices.

19.7.2 Sensitivity Analysis

The results of the Base Case are summarised in Tables 19-17 to 19-20, whereas the sensitivity analysis is summarized in Figures 19-6 to 19-9. All financial numbers are in Canadian dollars except for the commodity prices, which are in US dollars.

Table 19-17: Average Operating Margin ($/t) – Base Case

Item	LOM
Net Smelter Return (NSR)	18.52
Operating Cost	9.43
Margin	5.47

Table 19-18: Pre-tax Net Cash Flows – Base Case

Item	($M)
Years 1-5	1,117
Years 6-15	1,602
Years 16-29	1,489

Table 19-19: Pre-tax Rate of Return and Payback (100% equity basis) – Base Case

Item	
IRR (%)	12.2
Payback (years)	6.4

Table 19-20: Pre-tax Net Present Value – Base Case

Item	($M)
Cumulative net cash flow	2,828
5% discount	954
8% discount	422
10% discount	187
12% discount	13
15% discount	(172)

Notes to accompany Tables 19-17 to 19-20: All commodity prices are in $US/lb. All other values including the resulting NPVs are in million Canadian dollars.

Figure 19-6: Sensitivity of Cumulative Net Cash Flow

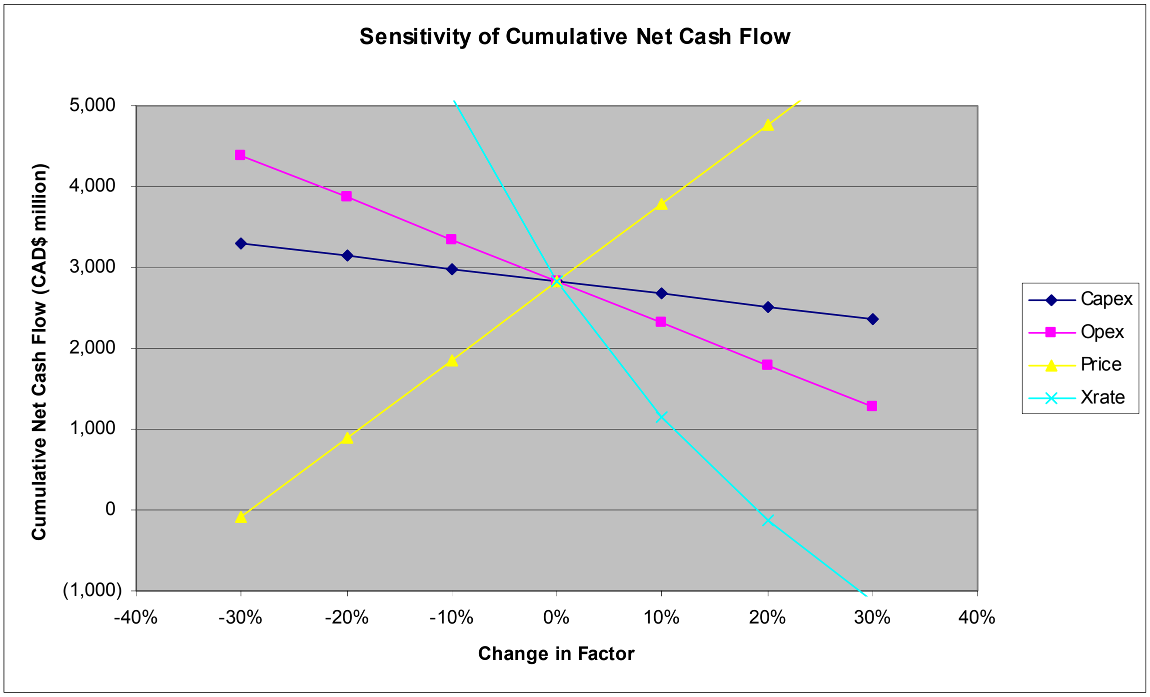


Figure 19-7: Sensitivity of NPV at 5%

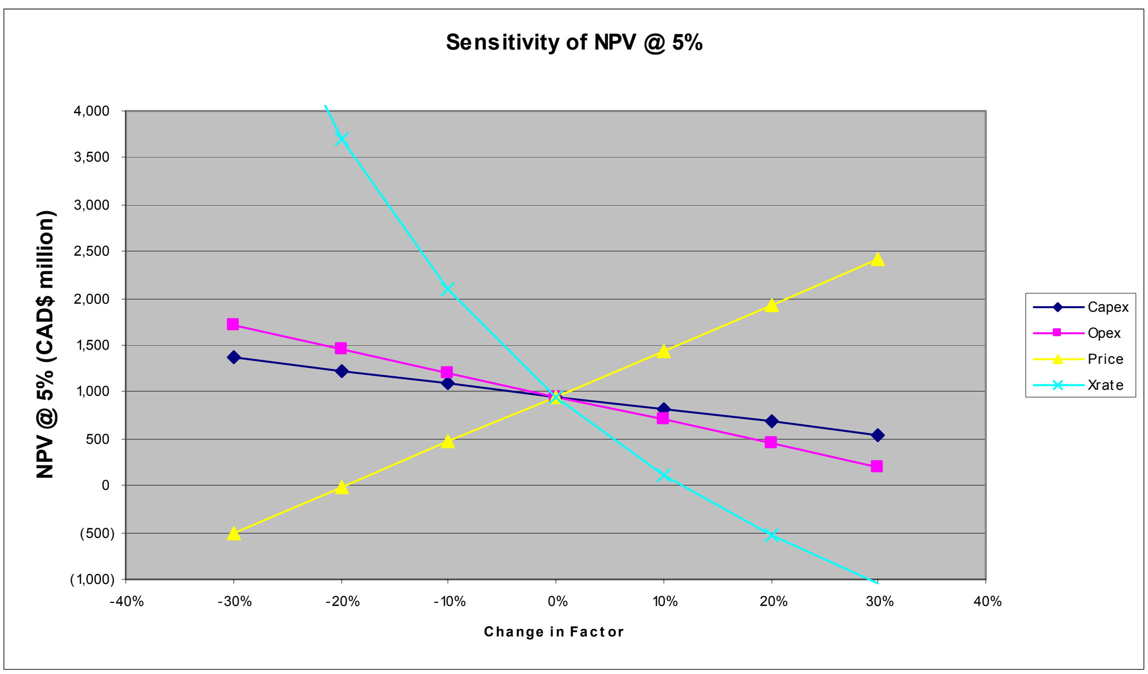


Figure 19-8: Sensitivity of NPV at 10%

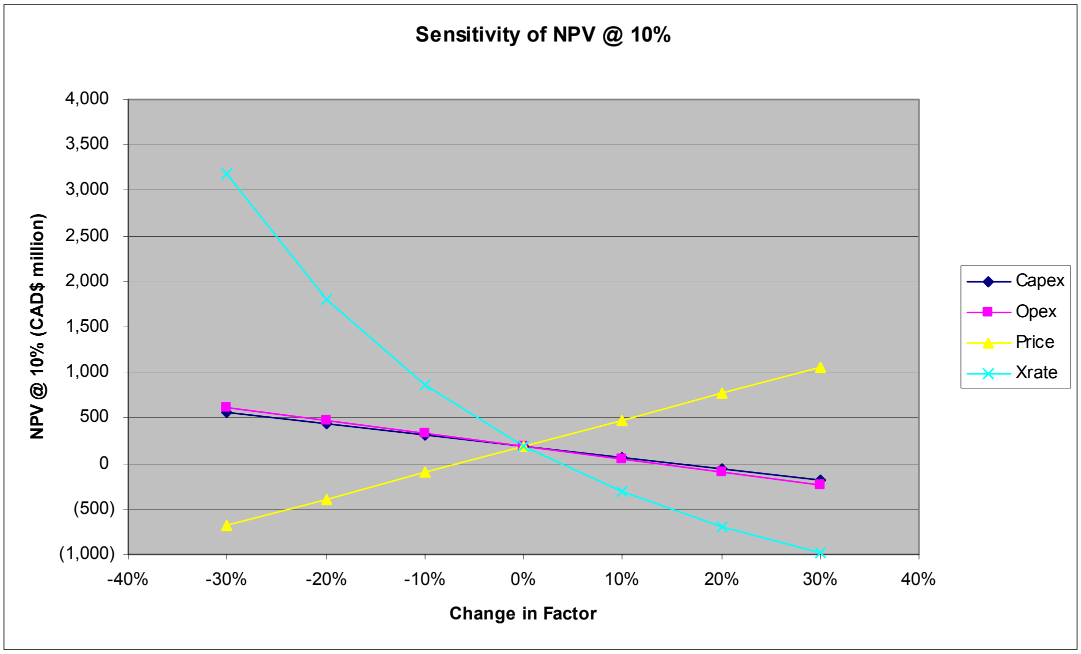


Figure 19-9: Sensitivity of NPV at 12%

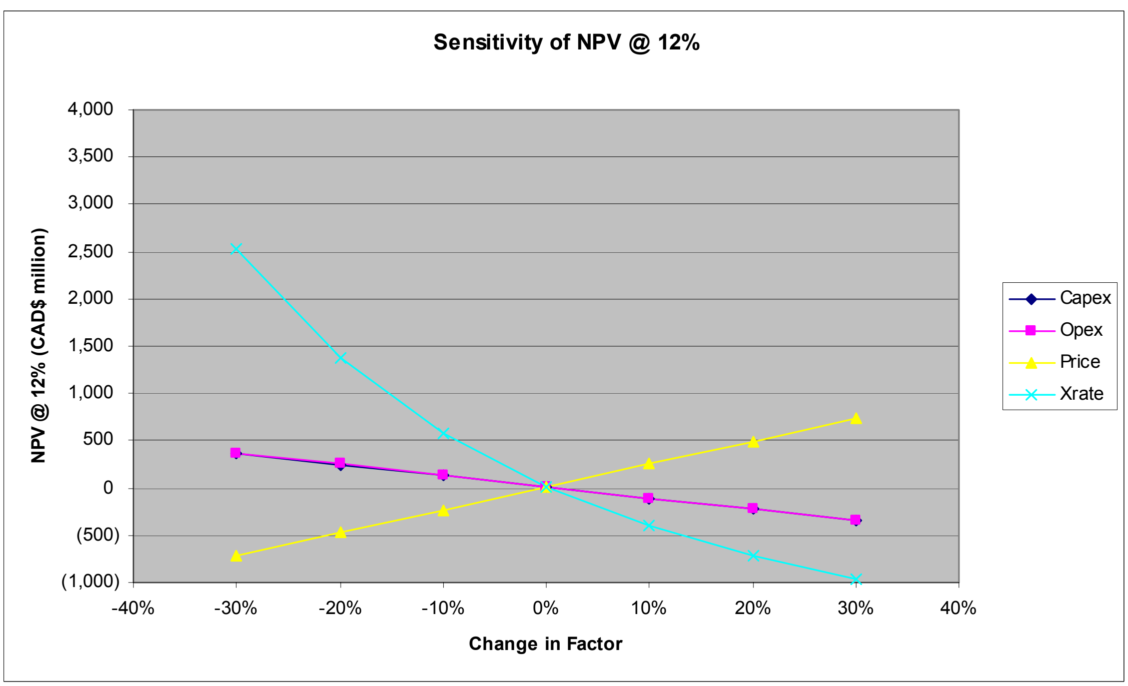
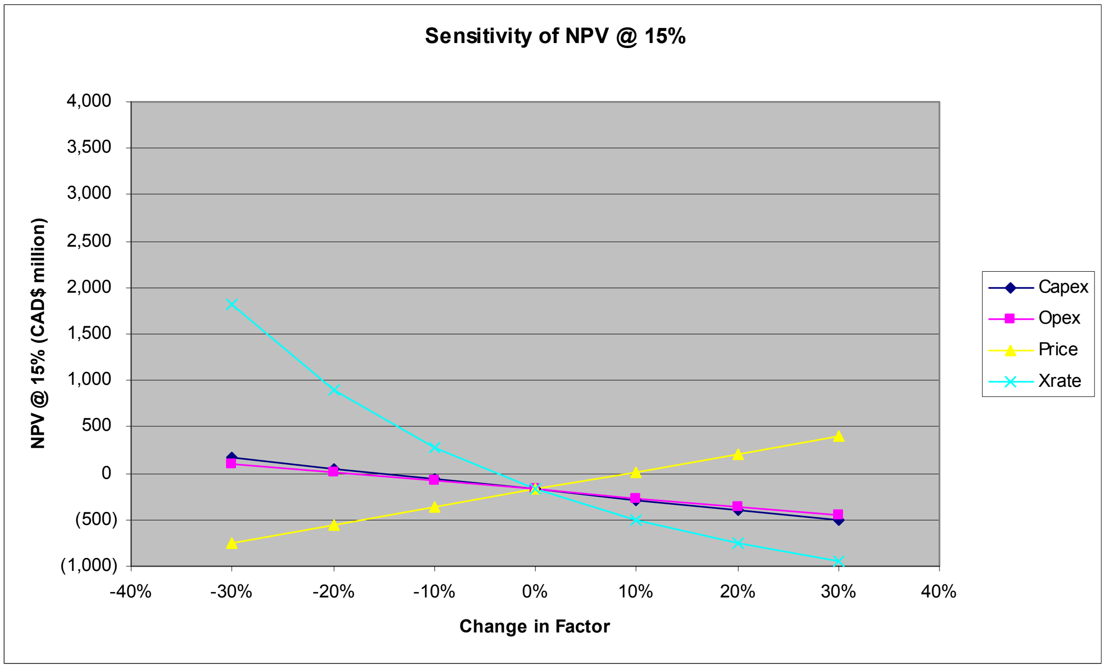


Figure 19-10: Sensitivity of NPV at 15%



Base Case Assumptions

- Ni price: US$7.50/lb
- Co price: US$11.00/lb
- Cu price: US$1.40/lb
- Foreign exchange: US$0.95 = CDN$1.00.

It should be noted that AMEC is not in the business of forecasting metal prices. Other nickel prices, with an accompanying NPV value, are provided in Table 19-21. Case 8 in this table represents approximate metal prices as at 30 November 2007.

Table 19-21: Pre-tax Internal Rate of Return & Net Present Value – Various Cases

Case	Base	2	3	4	5	6	7	8	9
Commodity Prices									
Nickel Prices[1]	7.50	5.25	6.00	6.75	8.25	9.00	9.75	12.00	14.00
Cobalt Prices[1]	11.00	7.70	8.80	9.90	12.10	13.20	14.30	34.00	14.00
Copper Prices[1]	1.40	0.98	1.12	1.26	1.54	1.68	1.82	3.15	3.00
NPV									
%IRR	12.2	N.A.	4.8	8.8	15.3	18.3	21.1	31.8	34.9
Cumulative net cash flow ($M)	2,828	(80)	889	1,859	3,797	4,767	5,736	9,905	10,819
5% discount ($M)	954	(511)	(22)	466	1,443	1,931	2,419	4,484	4,991
8% discount ($M)	422	(629)	(279)	72	773	1,123	1,474	2,945	3,324
10% discount ($M)	187	(679)	(390)	(102)	476	764	1,053	2,258	2,578
12% discount ($M)	13	(713)	(471)	(229)	255	497	738	1,745	2,018
15% discount ($M)	(172)	(744)	(553)	(362)	19	210	401	1,191	1,412

1. All commodity prices are in $US/lb.

19.8 Risks and Opportunities

- This PA was based on power being supplied from an extension to the North American grid that currently terminates at Meziadin Junction to the south. The project would be supplied from this line by an approximately 70 km long connection from the vicinity of Dease Lake, following the existing road to the mine site. The project viability hinges on this power being available.
- Low-grade material below 0.1% NiS needs to be tested for metallurgical recoverability. Additional material could increase the resource base and possibly be recovered at no extra cost.
- The Duffy and Hatzl deposits have the potential of being included in the resources after further geological and metallurgical investigations.
- A throughput optimization study might indicate a more optimal throughput than the 50,000 t/d basis of this PA.
- Most higher-grade mineralization is located close to the Turnagain River so if a setback larger than the currently assumed 100 m is required, resources would be negatively impacted.
- Overall pit slope angles were based on 45°, specific geotechnical information was not available at the time of this study. Shallower pit slopes would result in a loss of mineralization and increase of stripping ratio. Hydrogeology could also impact the pit slope angle.
- Neutralization potential tests indicated that the waste rock is non-acid generating (NAG). No special measures were considered in this PA for ARD (acid rock drainage). More testwork should be carried out to confirm this item.
- Removal of low margin material in the last laybacks could reduce the contained nickel from the current estimates.
- The Project is very sensitive to metal prices.
- There is a lack of developed information on grinding, settling and filtration. Available information is suitable for a PA, not for pre-feasibility or feasibility work. Grinding, in particular, is a significant cost in this project and major effort is warranted in determining its impact.
- Flotation testwork is still at a preliminary stage and should be considered adequate for a PA but not for feasibility level work. Further work is recommended. In particular, testwork is required to verify the production of concentrate at a grade of 8% to 9% Ni while maintaining the recoveries roughly at the levels seen at the roughing stage.
- Leaching and precipitation testwork to produce metal hydroxide products is at a preliminary stage and will require significant testing to verify. In addition, the testwork was performed on concentrate produced prior to the testwork used for the concentrator

design in this PA. This process has not been proven on the industrial scale and represents a potential execution risk to the project.

- There is potential for the sale of a high-grade concentrate but the effect on recovery of this option needs to be determined. Also, preliminary concentrate saleability and the cost of treating have to be determined. The control of MgO will be a very important issue.
- Variability testing will be necessary to determine the response of the mineralization throughout the orebody to a final flowsheet.
- In the current financial model, the purity of the precipitation products is assumed to be high with no significant impurities. Testing is necessary to determine if this is possible.

20.0 INTERPRETATION AND CONCLUSIONS

20.1 Geology, Mineralization, Database and Resources

Using a 0.10% NiS cut-off, the Turnagain Nickel deposit has an estimated open-pit resource of 489 Mt of Measured and Indicated Resources at 0.163% NiS and an additional 560 Mt of Inferred Resources at 0.152% NiS. The tonnage above cut-off is extremely sensitive to the cut-off grade. If the cut-off is increased to 0.16% NiS, the tonnage decreases by half and the contained metal decreases by 25%.

Assigning NiS to 0.001% for low sulphur analyses in case the AC-Ni analysis partially leaches nickel from some silicate minerals infers an incomplete understanding of the AC-Ni analysis. The uncertainties of the partial digestion for estimating sulphide nickel are less than those obtained using total nickel and therefore the partial digestion approach provides a more reliable estimate of "recoverable" nickel for resource modeling purposes. The ongoing round robin program is intended to provide better support for the use of AC-Ni in the resource estimate. Additional mineralogical work to determine the habit and distribution of nickel sulphides is also warranted.

Alternatively, if the AC-Ni analysis is successfully leaching only sulphide nickel then the assignment of NiS to a background value for low sulphur intervals and for interval missing AC-Ni analyses may lead to underestimation of the mineral resource.

The use of lithological domains in this model has decreased both conditional bias and smoothing relative to previous models. The nearest-neighbour interpolation of lithology used to create the lithological domains is preliminary and should be refined by reconciling polygons on orthogonal sections and plans and should consider other logged information such as structure and alteration. Smoothing of grades within lithological domains is considered minor but the use of grade shells in combination with lithological domain controls should be explored.

20.2 Metallurgy and Process Plant

Progress has been made in relating geology to the metallurgical results. With the definition of the material into domains as defined by the new geological model, more insights have been gained into the relationship of the mineralization to test variables.

Indications are that a flotation concentrate, could be produced from Domains 104 and 106 of the Main zone, which would be suitable either for a smelter or for a hydrometallurgical facility. The metallurgy and PA mill design is primarily based on Domains 104 and 106 which represent 91% of the resource.

Testwork undertaken between 2004 and 2007 resulted in the following:

- Grinding work performed in the preparation of material for flotation indicates a fair amount of variation in the grindability of the samples. A primary grind of 150 µm K_{80} was determined as appropriate for the liberation size used for this PA, giving reasonable recoveries at this particle size with the liberation of the majority of sulphides and non-sulphide gangue minerals.
- Bench testwork indicated that it was possible to produce a rougher concentrate that could be further upgraded to produce a sulphide concentrate with grades ranging from 5% to over 15% nickel. The PA design has utilized an overall residence time of 50 minutes in rougher flotation together with a mass pull of 15% for preliminary design purposes.
- Variation in the concentrate grade was attributed to the sulphur content, and degree of sulphur dissemination in the gangue, as well as the ratio of pyrrhotite to pentlandite. This assessment is preliminary and more work is required in this area.
- Principal flotation reagents consisted of potassium amyl xanthate as a collector, copper sulphate as an activator, MIBC as a frother, and CMC as a dispersant. Significant reductions in reagent consumption have been achieved in ongoing testwork.
- Evaluation of nickel recovery in 2006 indicated that for material ranging from 0.1% to 0.5% NiS, recovery would vary between 75% to 82%. It is essential in future flotation testwork to quantify the level of losses incurred in upgrading the concentrate. Typically for deposits of this nature, initial testwork producing higher grade concentrates at high recoveries from low grade feeds is problematic. The locked cycle work done in 2006 indicates that this problem can be solved. However until there is more metallurgical testwork, this area represents a significant potential risk to the project.
- Several stages of cleaning will be necessary to produce a concentrate adequate either as a saleable material or as a material to be treated in the hydrometallurgical process.
- Locked cycle testwork suggests that a hydrometallurgical approach treating a lower-grade concentrate may be required to maximize the economic value of the deposit. It should be noted, however, that process development is still at a fairly early stage for this deposit and more work is recommended to improve grade recovery performance to produce a saleable concentrate for smelting.
- From 2007 pilot plant testwork, column cleaning testwork showed that potential for reducing the MgO content in the concentrate was feasible. Concentrate grade was impacted by the level of residence time in the unit, wash water applied, and froth depth. Further work is deemed necessary to optimize results for recovery and grade performance. Indicated recoveries in the cleaner flotation program produced lower recoveries than utilized for design.
- Settling and filtration tests were not performed on concentrates. The basis of the PA design work was based on information from similar projects.

The Turnagain mineralization is planned to be processed through an on-site concentrator and hydrometallurgical process facility that will produce nickel, cobalt and copper precipitation products. The planned nominal milling rate is 50,000 t/a. ROM open pit material will be crushed in a gyratory crusher. The crushed mineralization will be processed by means of a fine crushing circuit in combination with ball mill grinding, followed by rougher flotation, conventional cleaning, regrind, column cleaner flotation, and dewatering, to produce a nickel-cobalt-copper concentrate to be stored in holding tanks. The concentrate will be finely reground and then pressure oxidized in an autoclave. Leaching of the copper, cobalt and nickel will follow with precipitation of copper, cobalt and nickel occurring sequentially.

Three products are planned to be produced: copper sulphide, nickel hydroxide, and cobalt hydroxide. These products will be trucked to Port Stewart for shipment to smelters. The truck haulage of concentrate will be contracted out. Tailings from the process will be impounded in a tailings pond; water will be reclaimed from the tailings pond and re-used in the process.

Saleable product will be paid for on the basis of 85% for nickel contained in nickel hydroxide and 80% for cobalt in cobalt hydroxide and copper in copper sulphide; 80% of the copper is considered to be payable.

It should be noted that there are a variety of issues related to the production of saleable products by concentration and subsequent hydrometallurgical processing that represent a technical and financial risk to the project and will have to be addressed in future work.

20.3 Mine Planning

AMEC completed initial pit resource and design plans, reviewed tailings and waste considerations, reviewed ancillary and infrastructure requirements, and proposed a project execution plan. Several sensitivity studies were performed at various prices, selling costs, process costs and slope angles. Eight schedules based on different nickel prices were prepared for financial analysis.

Mineralization grading over 0.085% NiS was considered economically interesting. No mining dilution was considered, as most mineralization/waste contacts are contacts between material slightly above cut-off and slightly below cut-off. No recovery reduction of stockpiled material was considered. The cut-off used in the study was 0.1% NiS as no recovery data was available below this grade.

The proposed mining operation is a conventional shovel and truck open pit mine feeding 50,000 t/d to a processing plant that will use hydrometallurgical technology. Mining and processing of material will commence in a starter pit. The pit will expand in phased pushbacks until the ultimate pit limits are reached. Mineralized material will be hauled to a primary crusher located near the southwest pit rim of the Main Pit.

The mine plan has been scheduled to maximize the production of high-grade material, during the first 5 years, to minimize the capital payback period. Mine life is estimated on the plan at 29 years. The planned workforce ranges from a peak of 383 persons in Year 15 of the proposed mine plan, to 280 persons in Year 29.

20.4 Financial Analysis

The results of the economic analysis that follows represent forward-looking information as defined under Canadian securities law. The results depend on inputs that are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here. The PA is preliminary in nature, includes Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PA will be realized.

The total estimated capital cost to design and build the Turnagain Project described in this report is CDN$1,381 million. Total operating costs are estimated at CDN$9.43 per tonne milled. Sustaining capital for the project over 29 years is CDN$173.5 million.

The financial base case assumed metal prices of US$7.50/lb for Ni, US$11.00/lb for Co, US$1.40/lb for Cu, and an exchange rate of US$0.95 = CDN$1.00. The base case assumed a net smelter return of CDN$18.52/t, operating costs of CDN$10.04/t, for a margin of CDN $5.47/t. Using a 100% equity basis, the internal rate of return was estimated at 12.2%, and payback in 6.4 years. Cumulative cash flow was estimated at CDN$2,828 million. The base case NPV ranged from CDN$954 M at NPV 5% discount to CDN$-172 million at NPV 15% discount.

NPV was also estimated for a range of cases at different metal prices.

21.0 RECOMMENDATIONS

This PA indicates that the Turnagain deposit is potentially mineable and that further work is justified. It is recommended that work in all major areas be undertaken to advance this project to the pre-feasibility level to better determine the economic viability of this project. This work could include some drilling to better define the resources and upgrade these into reserves. Major metallurgical work is necessary to ensure that saleable metal products can be produced. More geotechnical testing and environment work is necessary to allow the project to move forward. Confirmation and availability of the required power from the North American grid extension is critical to this project.

A three-discipline approach is recommended for future work on the Turnagain project. The programs outlined below total CDN$1.08 million and are estimated to take about 6 to 9 months to complete. Once the data from these programs are available, have been evaluated, and indicate that additional work is warranted, HCNC should consider completing a pre-feasibility level study on the project.

21.1 Geology

Before proceeding to a more advanced level of study, HCNC should:

- complete a review of the new round robin work on standard reference material
- complete detailed mineralogical analysis on the host rocks to determine the habit and distribution of sulphides and the possibility of a silicate nickel component in AC-Ni analysis.
- the entire drill hole database should be converted to a relational database. Consideration should be given to developing the database in a manner suitable for importing at a later date into more sophisticated database application. AMEC recommend double data entry for any manual entry of data into a database.
- analytical laboratory and procedure used and certificate number should be documented for each sample in the database. Analysis for nickel and sulphur by different procedures should be separated into unique database fields.
- collar and down-hole survey methods should be documented in the database to allow the impact of data quality to be assessed when classifying mineral resources and reserves.
- refine the lithological model through reconciliation of polygons between orthogonal sections and plans and through additional information from drill logs such as structure and alteration intensity.

This phase of work, excluding the mineralogical analysis, is expected to take four weeks and cost approximately CDN$30,000. It should be undertaken concurrently with the mining and metallurgical programs outlined in Sections 21.2 and 21.3, but is not contingent on them.

During the next resource estimation, HCNC should:

- examine the use of grade shells in combination with lithological domains to control grade-smoothing within lithological domains
- examine the use of composites which honour the geology in the drill holes and the use of shorter composite lengths (e.g., 7.5 m) as a means of providing better resolution along domain boundaries and to further reduce unintentional smoothing
- examine the impact on grade due to the change in sample length from 2 to 4 m.

In addition, to improve the data quality HCNC should:

- collect professional survey measurements for all drill holes
- collect Maxibor down-hole surveys for as many holes as practical
- HCNC should create a SG standard to use periodically to ensure the weight scale used in SG determinations is working properly.

21.2 Mining

Before proceeding to a more advanced study, HCNC should address the following points:

- The PA study is based on an overall pit slope angle of 45°. Geotechnical recommendations are required to define slope angles at all sides of the pit to support a pre-feasibility study.
- In this PA, a setback from the Turnagain River of 100 m was used as there are no clear regulations. As significant amounts of high-grade mineralization are located close to the river, the setback should be defined as soon as geotechnical recommendations are available.
- A trade-off study should be conducted to define the optimal plant, waste dump and low-grade stockpile location.

This phase of work is expected to take about 6 to 9 months from inception; note that some items, such as the geotechnical drilling, would need to be conducted during the northern summer. AMEC estimates the costs associated with the program at approximately CDN$500,000. The work should be conducted concurrently with the geological program outlined in Section 21.1, and the metallurgical program defined in Section 21.3, but is not contingent on them.

Once results of this work are to hand, and have been assessed, HCNC should review if a pre-feasibility study on the project is warranted from the mining perspective. If development factors, in particular metal prices, capital cost estimates, and exchange rates, are favourable, HCNC should proceed with a pre-feasibility study.

21.3 Process

Substantial metallurgical work is required to determine the best approach to producing a saleable product (smeltable concentrate or hydroxide products). This future work includes:

- samples for process development work
- samples for variability work
- samples to prepare a bulk concentrate for hydrometallurgical work
- sample aging studies
- crushing Index Work
- Bond Work for all mineralization types in the deposits
- SAG Mill Power Index testwork (alternatively drop weight and SMC tests allowing JKSimmet analysis) for grinding
- process development work (further define ability to produce concentrate grade and recovery)
- process development work (optimizing reagent quantities, etc.)
- variability testwork to test response of mineralization to the flowsheet (mineralization characterization)
- settling testwork (concentrates and tailings)
- filtration testwork (mill)
- smelter testing of the copper sulphide concentrate
- ultra-fine grinding definition
- pressure oxidation work
- production of precipitation products (nickel and cobalt)
- residual leaching
- precipitation studies
- solvent extraction testing for cobalt and zinc.

This phase of work is expected to take six to nine months, and cost approximately CDN$550,000. It should be conducted concurrently with the geological and mining programs outlined in Section 21.1 and 21.2, but is not contingent on results from these programs. The additional metallurgical information is material, and will drive the cost assumptions used to delineate any reserves that may be declared in future work on the project, and is important in informing any decision to proceed with a pre-feasibility level study.

21.4 Other Areas of Investigation

AMEC notes that, to date, discussions with stakeholders that may be impacted by any proposed mining operation are at a very early stage. Such discussions will be necessary as the Project is advanced, in order to assess any socio-economic impacts.

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22.2 Units of Measure

a	Annum (year)
%	Percent
°	Degrees
°C	Degrees Celsius
cm	Centimetres
dwt	Deadweight tonnes
g	Grams
g/cm^3	Grams per cubic centimetre
g/m^3	Grams per cubic metre
h	Hour(s)
ha	Hectares (10,000 square metres)
HP	Horsepower
kg	Kilograms
klm^3	Thousands of loose cubic metres
km	Kilometres
km^2	Square kilometres
KN	Kilo newtons
kW	Kilowatts
M	Millions
m	Metres
m^3	Cubic metres
masl	Metres above sea level
mm	Millimetres
Mm^3	Million cubic metres
MN/m^2	Million Newtons per square metre
Mt/a	Million dry tonnes per year
MW	Megawatts
ppm	Parts per million
st	Short tons
t	Tonnes (metric)
t/h	Tonnes per hour
t/a	Tonnes per year
CDN$ M	Million Canadian Dollars
US$ M	Million US Dollars
$/t	Canadian dollars per tonne
US$/t	US dollars per tonne
US$/T	US dollars per short ton
wt %	Weight percent

23.0 DATE AND SIGNATURE PAGE

The effective date of this Technical report, entitled 'Turnagain Nickel Project, British Columbia NI43-101 Technical Report on Preliminary Assessment is 25 September, 2007.

"Signed"	Dated
AMEC Americas Limited	11 January 2008

Per:

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