Updated NI 43-101 Technical Report for the Nickel King, Main Zone Deposit

Northwest Territories, Canada

02 June 2010

Prepared by

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Abbreviations, Symbols, and Acronyms

Aber Resources	Aber
Absolute Difference	ABS
Acid-Base Accounting	ABA
ACME Analytical Laboratories Ltd.	ACME
Annual Information Form	AIF
Canada Mining Regulations	CMR
Canadian Institute of Mining	CIM
Canadian Nickel Company Ltd (subsequently Vale Inco)	CANICO
Cobalt	Co
Copper	Cu
Crone Geophysics Ltd.	Crone
Department of Indian and Northern Affairs Canada	DIAND
Diamond Drill Hole	DDH
Electromagnetic	EM
Elevation	El.
Falconbridge Ltd.	Falconbridge
Fugro Airborne Surveys	Fugro
Geological Survey of Canada	GSC
GEOTECH Ltd.	GEOTECH
Gigaannum	Ga
Gold	Au
Gross Overriding Royalty	GOR
Highwood Resources Ltd.	Highwood
Hydrochloric Acid	HCl
Inductively Coupled Plasma-Mass Spectrometry	ICP-MS
Lower Sills	LS
Mackenzie Valley Land and Water Board	MVLWB
Mackenzie Valley Resource Management Act	MVRMA
Microsoft Excel spreadsheets	XLS
National Instrument 43-101	NI 43-101
National Topographic System	NTS
Navigator Exploration Corp.	Navigator
Net Smelter Royalty	NSR
Neutralizing Potential/Acidifying Potential	NP/AP
Nickel	Ni
Nitric Acid	HNO_3
Northwest Territories	NWT

Palladium	Pd
PEG Mining Consultants Inc.	PEG
Platinum Group Elements	PGE
Platinum	Pt
Potential Mineral Deposit	PMD
Proficiency testing program for mineral analysis	PTP MAL
Quality Assurance/Quality Control	QA/QC
Silver	Ag
Smelter Returns Royalty	NSR Royalty
Snowbird Tectonic Zone	STZ
Spatially Integrated Information for DIAND	SID
Strongbow Exploration Inc.	Strongbow
Three Dimensional	3D
Time Domain Electromagnetic	TDEM
Tundra Airborne Surveys Ltd.	Tundra
Upper Sills	UP
Very Low Frequency Electromagnetic	VLF-EM
W. Kizan	Kizan

Units of Measure

10,000 parts per million = 1%	ppm vs %
Centimetres	cm
Degrees Celsius	°C
Degrees	°
Dollars per metre	$/m
Grams	g
Hectares	Ha
Inches	"
Kilograms	kg
Metre	m
Metres above sea level	masl
Million pounds	Mlb
Million tonnes	Mt
Parts per billion	ppb
Parts per million	ppm
Pounds	lb
Tonne	t

1.0 SUMMARY

Strongbow Exploration Inc. (Strongbow) commissioned PEG Mining Consultants Inc. (PEG) to update the independent Mineral Resource Estimate and Technical Report for the Nickel King Main Zone deposit. The Mineral Resource Estimate has not changed from the news release of the original technical report on 25 February 2009 (and subsequent technical report on 9 April 2009); however, the technical report has been updated to properly disclose the metallurgy program managed by PEG and conducted at SGS Mineral Services in 2009, as well as highlight the exploration potential of the Nickel King deposit.

The property, located at approximately 60.26° latitude north and 104.527° longitude west, in the Northwest Territories, is only accessible by fixed wing aircraft or helicopter. The town of Stony Rapids, Saskatchewan, Canada, located approximately 130 km south-southwest of the property, is the closest access point. Stony Rapids is accessible via Provincial Highway 905 from La Ronge, Saskatchewan, approximately 400 km to the south. The section of highway 905 between Points North and Stony Rapids was originally built as a seasonal road and is occasionally inaccessible during spring flooding or periods of heavy rainfall. Trans-West Air and Pronto-Air operate regularly scheduled passenger flights from Saskatoon to Stony Rapids.

1.1 GEOLOGY

The property lies within the southern part of the Snowbird Tectonic Zone that forms the boundary between the Archean Rae and Hearne geological provinces.

A number of norite intrusions have been mapped within the property. The most important mineralized zone is the Main Zone consisting of Ni-Cu sulphide mineralization hosted within two arcuate, stacked south dipping norite sills. The sills range from a minimum of about 10 m to 100 m or greater in thickness and are currently interpreted as two limbs of a westerly plunging synform. Typical thickness ranges between 40 m to 60 m. The sills are informally named the Upper Sill and Lower Sill.

Strongbow described the mineralization as sulphide minerals, consisting of pyrrhotite with lesser amounts of chalcopyrite and pentlandite, typically comprising less than 5% in the Upper Sill but locally exceeding 20%. In the better-mineralized portions of the Lower Sill, the sulphide content typically ranges from 5% to 15%, occasionally reaching 30% over short intervals (typically less than 1 m). Initial petrographic work carried out by Strongbow indicates that pentlandite is the primary Ni-bearing sulphide mineral, occurring typically as discrete grains and veinlets and only rarely as exsolution lamellae or flames in pyrrhotite.

At the 0.2% Ni cut-off base case, the geological resource model yielded 11.1 Mt grading at 0.40% Ni, 0.10% Cu and 0.018% Co containing 97.7 Mlb of Ni, 23.5 Mlb of Cu and 4.4 Mlb of Co in the indicated category. The total Inferred Resource is 33.1 Mt grading at 0.36% Ni, 0.09% Cu and 0.017% Co containing 262.4 Mlb of Ni, 63.9 Mlb of Cu and 12.3 Mlb of Co. Seventy five percent of these resource values are in the inferred classification due to the lack of drilling over the extensions of the deposit. For this base case model, CANICO drill holes were removed from the database as the quality of the data was suspect and these holes were only assayed over their high grade composites.

Strongbow has also identified Potential Mineral Deposits (PMD) within the extent of the Nickel King Main Zone deposit. The PMD occupy gaps within the resource estimate where there is insufficient drilling to classify an inferred resource. Total PMD have been estimated as representing between 10 and 27 Mt with an interpolated grade exceeding a 0.2% Ni cut-off. The PMD is conceptual in nature since there has been insufficient exploration to define it as a mineral resource and that it is uncertain if further exploration will result in the target being delineated as a mineral resource.

1.2 METALLURGY

Split core samples from the Nickel King deposit were received at SGS Minerals Services in April 2009 for use in a metallurgy testing program. The samples were crushed and blended to produce an overall composite for a scoping level metallurgical test program that included chemical and mineralogical analysis, as well as grindability, flotation, and heavy liquids testing.

The testwork indicated overall composite nickel and copper average headgrades of 0.65% and 0.14%, respectively. Modal mineralogical analysis by QEMScan revealed that the nickel was associated primarily with pentlandite (~89%), and to a lesser extent with pyrrhotite (~9%) and silicates (~2%). At the target grind size P_{80} of ~110 µm pentlandite liberation was found to be high at 86%.

Grindability testing of the composite indicated a Bond Rod Work Index of 13.2 kWh/t and a Bond Ball Work Index of 15.0 kWh/t.

Heavy liquids testing on the composite sample indicated that ~15% of the mass could be rejected at a ½” crush size while at the same time retaining 95% of the metal values.

Six batch rougher flotation tests were carried out on the overall composite to determine the effect of grind size and reagent addition on flotation kinetics. In addition, four batch cleaner flotation tests were used to develop a flowsheet suitable for the production of saleable concentrates. A locked cycle test based on the optimum conditions identified in the batch program was carried out to evaluate the effect of recycle streams in the flowsheet and to generate a representative concentrate product for minor element analysis. Table 1-1 presents the metallurgical projection based on the locked cycle test results.

Table 1-1: Locked Cycle Test Metallurgical Projection – Nickel King Composite

Product	Weight %	Assays				Distribution			
		Cu %	Ni %	S %	Co %	Cu %	Ni %	S %	Co %
Final Concentrate	3.0	4.21	16.5	28.3	0.74	89.1	78.4	18.5	63.5
Cleaner Scav. Tail	6.1	0.05	0.69	11.3	0.030	2.1	6.7	15.0	6.0
Pyrrhotite Tail	12.1	0.03	0.59	23.3	0.024	2.9	10.3	56.2	8.3
Low Sulphur Tail	78.9	0.01	0.04	0.65	0.010	5.9	4.6	10.3	22.2
Head (calc.)	100.0	0.14	0.63	4.62	0.035	100.0	100.0	100.0	100.0

Acid-Base Accounting (ABA) testing of the low sulphur tailings indicated a low possibility of being acid generating. Minor element analysis conducted on the final concentrate described above did not indicate elevated levels of any deleterious elements.

Metallurgical performance is judged to be good with production of a bulk concentrate grading copper and nickel grades of 4.21% Cu and 16.5% Ni, respectively. The concentrate is a saleable product that we expect will be attractive to many of the current domestic and foreign consumers. Further separation of the copper and nickel minerals to provide individual copper and nickel concentrates may add further value and will be examined as part of future testwork programs.

2.0 INTRODUCTION AND TERMS OF REFERENCE

This report describes the results of the first metallurgical testwork and mineral resource estimation of the Nickel King Deposit, which is owned by Strongbow Exploration Inc. based in Vancouver, Canada. The report is written to comply with standards set out in National Instrument 43-101 (NI 43-101) for the Canadian Securities Administration. The report is written in support of Strongbow's Annual Information Form (AIF) for the year ended 31 January 2010 and was prepared at the request of Mr. Ken Armstrong, President and Chief Executive Officer of Strongbow Exploration Inc. under the direct supervision of:

- **Pierre Desautels, P.Geo**. – Principal Resource Geologist with PEG Mining Consultants Inc. Mr. Desautels directed the review of the 2008 digital data as well as the estimation of the resource for the Nickel King Deposit and is responsible for the overall technical report. Mr. Desautels also visited the project site from 31 March 2009 to 1 April 2009 to review drill core logging and sampling procedures, collect representative character samples, and verify drill hole collar locations.

- **Andy Holloway, P.Eng., C.Eng.** – Principal Process Engineer with PEG Mining Consultants Inc. Mr. Holloway designed and directed the 2009 metallurgical program.

The following individual provided the regional and local geology and the historical information on the Nickel King Deposit:

- **David Gale, P.Geo. P.Geol.** – Vice President Exploration is a registered Professional Geoscientist in the Province of British Columbia, the Northwest Territories, and in Nunavut. He completed a Masters degree from Queen's University in Kingston, Ontario in 1997, following the completion of his undergraduate geology degree at Memorial University in St. John's, Newfoundland. Mr. Gale has conducted gold and base metal exploration with a number of companies, including Westmin Resources and BHP Minerals Ltd. Mr. Gale joined Homestake Canada Inc in 1998, and following the takeover by Barrick Gold in 2001, stayed with the company for seven years. Work during that period mainly focused on exploration around the Eskay Creek mine, a precious metal enriched VMS deposit in northern British Columbia. Mr. Gale joined Strongbow Exploration in January 2005 and is managing all gold and base metal exploration for the company.

Information, conclusions, and recommendations contained herein are based on a field examination, including a study of relevant available technical data and discussions with Strongbow's site geologist, Mr. David Gale, Vice President Exploration.

All units used in this report are metric unless otherwise stated; grid references are based on the UTM NAD 83 coordinate system.

The sections on Mining Operations, Process Metal Recoveries, Markets, Contracts, Environmental Considerations, Other Relevant Data and Information, Taxes, Capital and Operating Cost Estimates, Economic Analysis, Payback, and Mine Life, are not applicable to this report. All Illustrations are embedded within the body of the report.

3.0 RELIANCE ON OTHER EXPERTS

PEG has followed standard professional procedures in preparing the content of this resource estimation report. Data used in this report has been verified where possible and this report is based upon information believed to be accurate at the time of completion.

PEG has not verified the legal status or legal title to any claims and the legality of any underlying agreements for the subject properties regarding mineral rights, surface rights, permitting, and environmental issues in sections of this technical report; PEG has relied on the information gathered during the site visit supplied by Strongbow's representative.

The author has also relied on several sources of information on the property, including digital geological and assay data, and geological interpretations by Strongbow. Therefore, in writing this report, the qualified person relies on the truth and accuracy as presented in various sources listed in the references section of this report.

4.0 PROPERTY DESCRIPTION AND LOCATION

The Nickel King project is located in the south-eastern Northwest Territories, approximately 550 km southeast of Yellowknife and 130 km northeast of Stony Rapids, Saskatchewan (Figure 4-1). The area lies within portions of National Topographic System (NTS) 1:50,000-scale map sheets 075A/01, 075A/02, 075A/07 and 075A/08. Nickel King project land holdings consist of eleven contiguous mineral claims and mining leases covering a total area of 7,595.18 ha (Figure 4-2 and Table 4-1). The Main Zone deposit is situated on mining lease 4954 and is located at UTM 526200E and 6680160N (North American Datum 83, Zone 13). Strongbow is the registered owner of a 100% interest in all of the mineral claims and mining leases. The property boundaries shown in this report reflect those displayed on the Department of Indian and Northern Affairs Canada (DIAND) NWT website, 'SID Viewer' (Spatially Integrated Information for DIAND), located on the World Wide Web at http://nwt-tno.inac-ainc.gc.ca/ism-sid/index_e.asp. PEG verified the status of these mineral claims and mineral leases from the above-mention website and found them to be in good standing.

Tenure to mineral claims in the Northwest Territories is governed by the Canada Mining Regulations (CMR) and administered by Indian and Northern Affairs Canada's Mining Recorders Office in Yellowknife. Mineral claims are acquired by traditional ground staking methods. An individual claim may cover a maximum area of 1,045.1 ha (2,582.5 acres). Annual exploration expenditures of $2.00 per acre are required to keep a mineral claim in good standing. Provided sufficient exploration expenditures are filed, a mineral claim can remain valid for a period of ten years (i.e., cumulative expenditures of $20/acre over ten years). Upon reaching its tenth anniversary a mineral claim must be legally surveyed and an application filed to convert the mineral claim to a mining lease. If a mineral claim is not taken to lease at its tenth anniversary then it must be relinquished. A mining lease is established by:

- filing an application with the Mining Recorder within 30 days of the tenth anniversary of the recording date of a mineral claim
- conducting a legal survey of the boundaries of the claim
- making annual rental payments of $1.00 per acre (equivalent to $1.00 per 2.471 ha). A mining lease is typically valid for a period of twenty-one years and renewable for an additional twenty-one year period.

Figure 4-1: Location Map

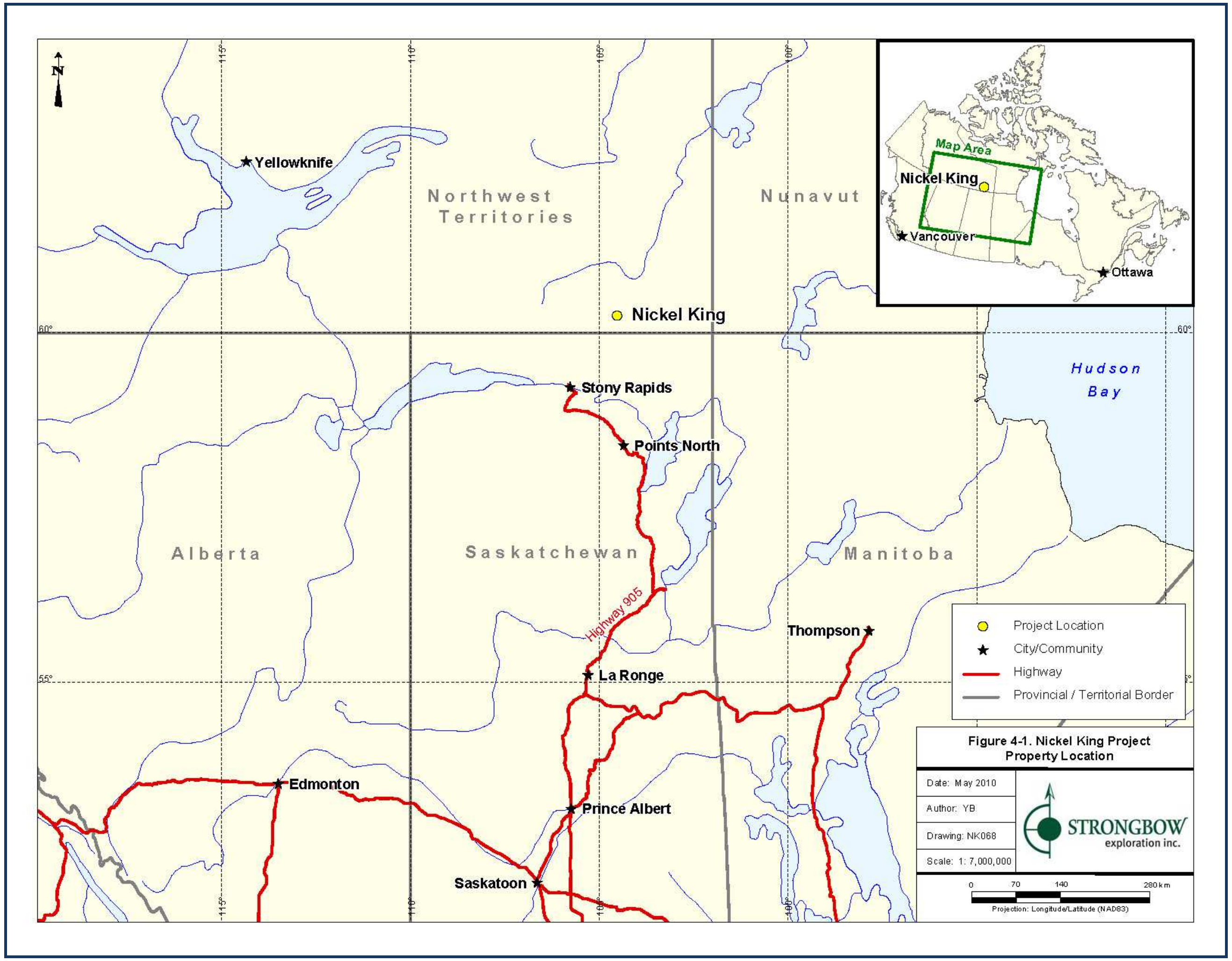


Figure 4-2: Land Holdings

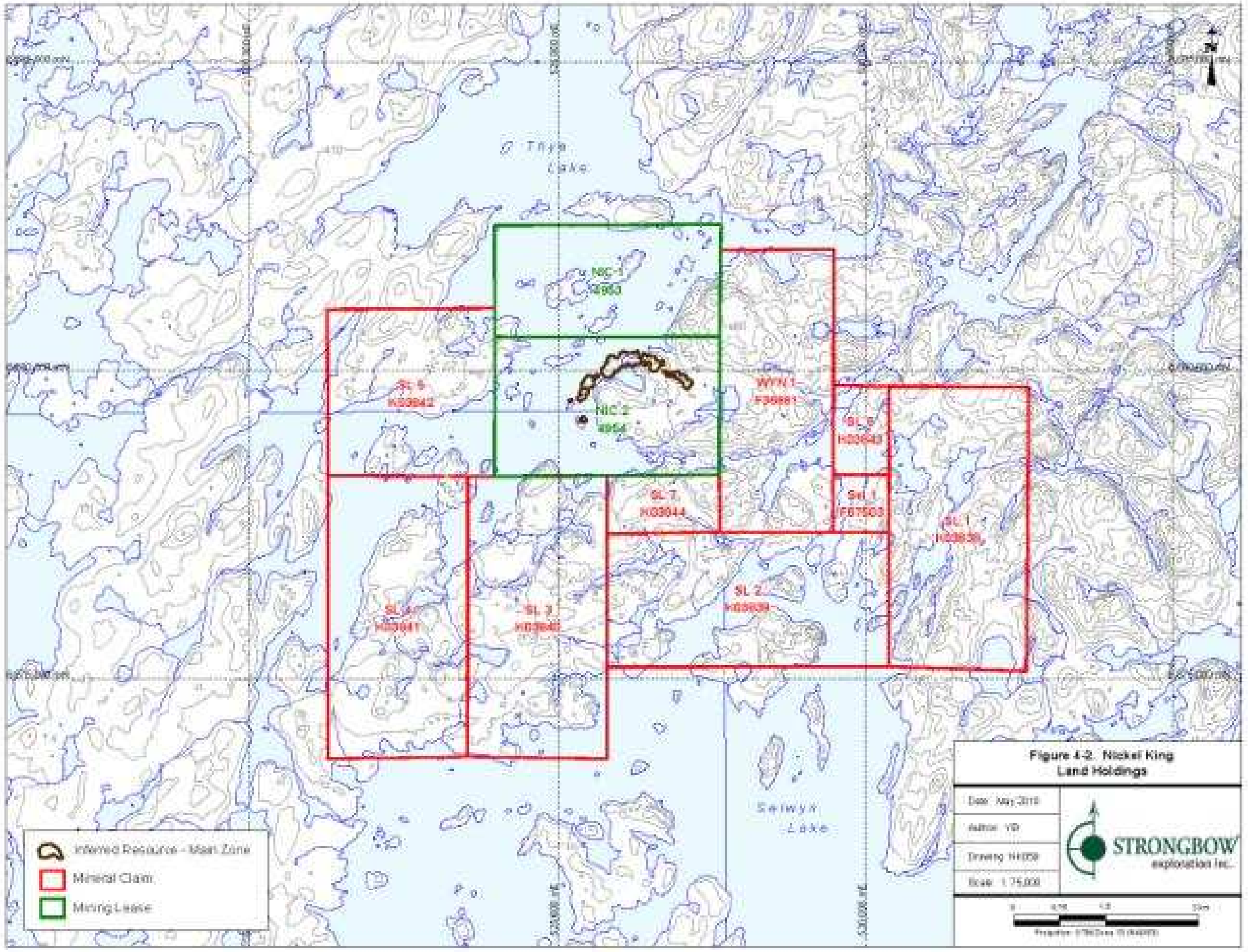


Table 4-1: Summary of Mineral Leases and Claims

Lease or Claim No.	Lease or Claim Name	Mining Division	Type	Acres	Hectares	Recording or Issue Date (d/m/yr)	Anniversary or Expiry Date (d/m/yr)	Comments
4953	NIC 1	NWT	Lease	1,700.0	687.82	20-Nov-97	15-Nov-28	Converted to mineral lease Nov. 2007
4954	NIC 2	NWT	Lease	2,044.0	827.00	20-Nov-97	15-Nov-28	Converted to mineral lease Nov. 2007
F67503	SEL-1	NWT	Claim	203.4	82.30	10-Sep-01	10-Sept-11	Legal land survey completed in 2007 in anticipation of conversion to mineral lease in 2011
K03838	SL 1	NWT	Claim	2,582.5	1,045.10	01-Dec-06	01-Dec-16	-
K03839	SL 2	NWT	Claim	2,582.5	1,045.10	01-Dec-06	01-Dec-16	-
K03840	SL 3	NWT	Claim	2,582.5	1,045.10	01-Dec-06	01-Dec-16	-
K03841	SL 4	NWT	Claim	2,582.5	1,045.10	01-Dec-06	01-Dec-16	-
K03842	SL 5	NWT	Claim	1,859.4	752.47	01-Dec-06	01-Dec-16	-
K03843	SL 6	NWT	Claim	309.9	125.41	01-Dec-06	01-Dec-16	-
K03844	SL 7	NWT	Claim	387.4	156.78	01-Dec-06	01-Dec-16	-
F36681	WYN-1	NWT	Claim	1,936.0	783.00	04-Jul-00	04-Jul-10	Legal land survey completed in 2007 in anticipation of conversion to mineral lease in 2010
Total	-	-	-	**18,770.1**	**7,595.18**	-	-	-

The Nickel King project was assembled by several generations of staking. The NIC 1 and NIC 2 mineral claims were legally surveyed during August 2007, reached their ten year anniversary in November 2007 and subsequently converted into mining leases 4953 and 4954, respectively (Table 4-1). The annual rent associated with these leases is $1,700 for 4953 and $2,044 for 4954. Sufficient exploration expenditures have been approved for the WYN 1 and SEL 1 mineral claims to remain in good standing through to their respective tenth anniversaries. As such, the WYN 1 and SEL 1 claims were also legally surveyed in August 2007, in advance of conversion to mining leases on or around 4 July 2010 and 11 September 2011, respectively.

Mineral claims SL-1 through SL-7, inclusive (SL claims) were staked in 2006 and have not been legally surveyed. Sufficient exploration expenditures have been filed with and approved by the Mining Recorder to extend each of the SL claims to their tenth anniversary, 1 December 2016, at which time a legal survey will be required to convert these claims to mining leases. The current annual cost to Strongbow of maintaining tenure to the Nickel King project is $3,744 covering the rental fees to mining leases 4953 and 4954. The rental fees will increase to approximately $5,883 once the WYN 1 and SEL 1 mineral claims are converted to lease in 2010 and 2011, respectively.

The Nickel King project is subject to two separate underlying royalty agreements. Mining leases 4953 and 4954 are subject to a 3% net Smelter Returns Royalty (NSR Royalty) on base and precious metals production. The NSR Royalty also applies to a two-kilometre area of interest extending from the boundaries of the two leases. This NSR Royalty is payable to Lloyd E. Anderson and William W. Kizan, (original agreement dated 9 August 1995; and as amended on 14 July 2000). Strongbow maintains the right to purchase the entire NSR Royalty for C$1.5 million at any time. Mining leases 4953 and 4954 are subject to an additional 2% gross overriding royalty (GOR Royalty) in favour of Aber Diamonds Inc., subject to an agreement dated 7 February 2000. Strongbow retains the right to purchase one-half of the GOR Royalty (1%) for C$2.5 million at any time.

The Mackenzie Valley Resource Management Act (MVRMA) as administered by the Mackenzie Valley Land and Water Board (MVLWB) governs land use and mineral exploration activities in the Nickel King areas. A Type A land use permit (No. MV2006C0036) was granted to Strongbow by the MVLWB in February 2007 and subsequently extended for a two year period in February 2010. This permit governs Strongbow's activities at Nickel King, including camp construction, mapping, prospecting, ground geophysics, diamond drilling, and water use.

On an annual basis prior to commencement of exploration work in the Northwest Territories, a work and safety plan must be submitted to the Workers Compensation Board (Yellowknife, NT). Commencement of work is contingent upon approval of the health and safety plan from the Chief Inspector of Mines and is generally issued and valid for the calendar year barring any significant changes to the scope of the proposed program. To date Strongbow has not experienced any delays, deficiencies, or impediments to the timely approval of the submitted work and safety plans.

No previously identified environmental liabilities are associated with the Nickel King project.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

The project area is subject to a sub-polar climate with normal summer temperatures that may range up to 30°C in July but typically average around 17°C. Normal winter temperatures average around -20°C, but may fall into the -40°C range. Freeze-up and break-up of ice on the lakes occurs in mid October and late April or May, respectively. Annual precipitation is approximately 50 cm/d, of which 35 cm falls as rain, and the remainder falls as snow. The terrain is generally gently rolling glacial topography characterized by glacial troughs, scoured and lake-filled rock basins, streamlined till forms, and till plains. Elevations on the Nickel King Claims range from 395 m to 460 m above standard mean sea level (masl). Native vegetation in the area consists of black spruce, white spruce, jack pine, tamarack, birch, and willows. A forest fire burned much of the project area in 1998.

The project area is remote with no direct road access. Access is primarily by ski-equipped (winter) or float-equipped (summer) fixed wing aircraft from the hamlet of Stony Rapids or Points North Saskatchewan, located 130 km southwest and 190 km southeast of the project area, respectively. The nearest airstrip (1,830 m) is situated 77 km to the east at Obre Lake Lodge, NT, which is serviced by regular charters from Winnipeg, Manitoba during the summer. Access to the property may also be accomplished by helicopter from Stony Rapids. Ground exploration (prospecting, mapping, etc.) is generally limited to the summer season, between break up and freeze up (May to September). Drilling may be accomplished year round once sufficient supplies, air support, and accommodation are in place.

Road access to Stony Rapids and Points North is possible via Provincial Highway 905, connecting Stony Rapids to La Ronge, Saskatchewan, approximately 400 km to the south. La Ronge is the nearest larger community and supply centre for Stony Rapids. Highway 905 has a paved surface as far north as La Ronge and a gravel surface from about 30 km north of La Ronge to Stony Rapids. The stretch of highway 905 between Points North and Stony Rapids was originally built as a seasonal road and is occasionally inaccessible during spring flooding or periods of heavy rainfall. Currently the province of Saskatchewan plans to upgrade this stretch to an all season surface. Trans-West Air and Pronto-Air operate regularly scheduled passenger flights from Saskatoon to Stony Rapids (six days a week in offseason and seven days a week in peak season).

6.0 HISTORY

6.1 CANICO 1952-53

In 1952, the Canadian Nickel Company Ltd. (CANICO; subsequently Vale Inco) discovered nickel sulphide mineralization in the Thye Lake area. Mapping and geophysical surveys led to diamond drill programs in 1952 and 1953. A total of eighteen EX size diamond drill holes, totalling 3,528.8 m, were drilled. Other than the diamond drill logs, no first-hand information regarding the results of CANICO's exploration program is available.

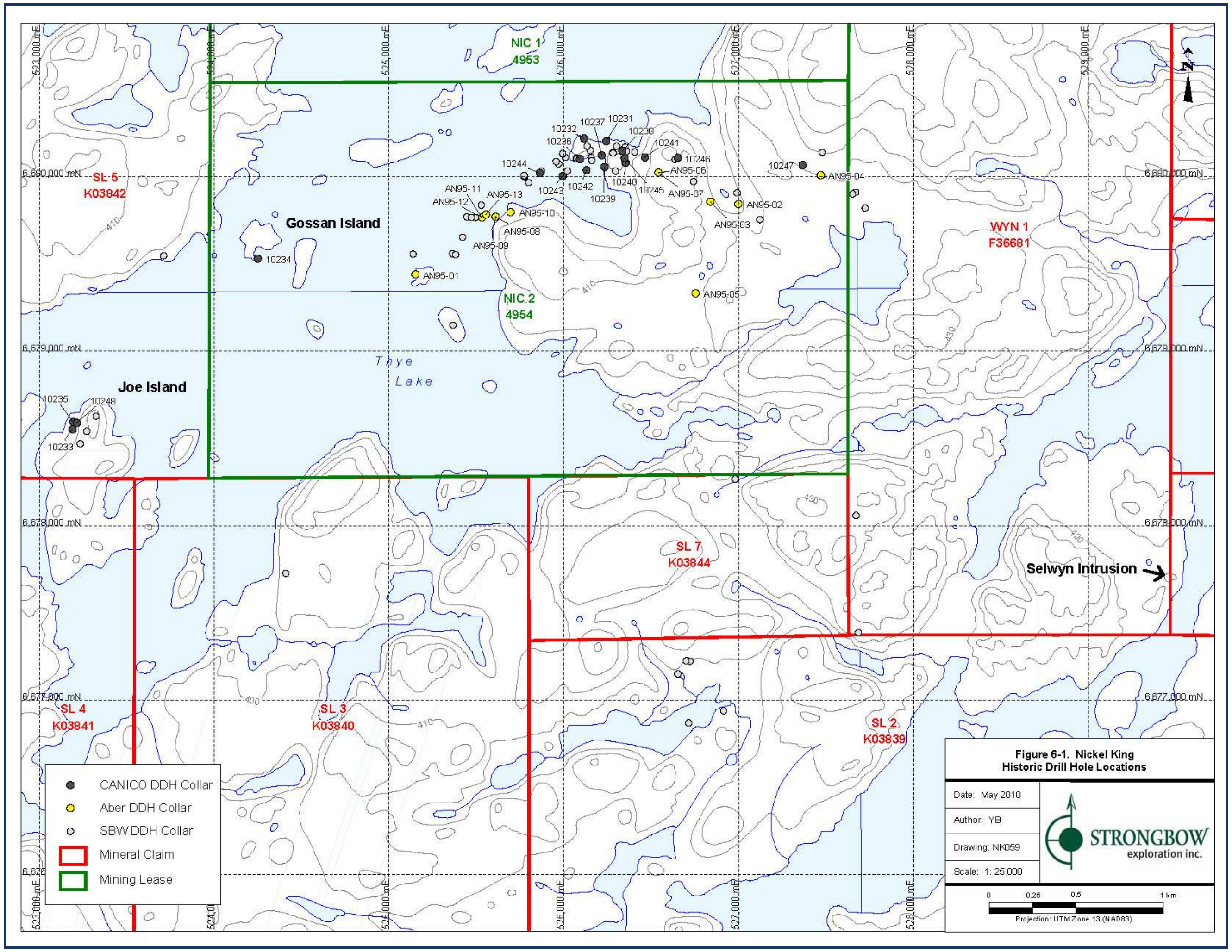
Thirteen of the CANICO drill holes (2,801.51 m) tested mineralization in the Main Zone over an 800 m strike length centred on the peninsula within mining lease 4954 (Figure 6-1). Mineralization was described as disseminated to semi-massive nickel-copper-iron sulphides within two, stacked mafic to ultramafic-gabbro/pyroxenite to norite composition sills. Both sills were mineralized, with the best assay results returned from hole 10240 which intersected 1.21% Ni and 0.36% Cu over 13.72 m within the Lower sill.

The remaining five drill holes tested three target areas outside of the Main Zone, including Joe Island (three holes), Gossan Island (one hole) and the Central Zone (one hole - 10247) (Figure 6-1). Each of these holes intersected mafic intrusive rocks and insignificant to weak sulphide mineralization. Subsequent to the 1953 exploration program, CANICO reported no further work and allowed the mineral claims to lapse.

6.2 GEOLOGICAL SURVEY OF CANADA 1954-1970

In 1954, the Geological Survey of Canada (GSC) flew an aeromagnetic survey over NTS sheet 75A which was published as Map 1126A. Regional scale geological mapping by the GSC was conducted in the area in 1963 and 1969, and the resulting map was published the following year (Taylor, 1963 and Taylor et al., 1970).

Figure 6-1: Historic Drill Hole Locations



6.3 HIGHWOOD 1975

In 1975, Highwood Resources Ltd. (Highwood) acquired the Nickel King prospect from Nemco Exploration Ltd., who had staked claims in the area through a grub staking agreement with G. Thomas. Highwood conducted very limited prospecting of the area, compiled an internal report and study of the CANICO drilling, and concluded that claims hosted excellent exploration potential for nickel-copper and possibly cobalt-silver mineralization (Thomas, 1976). Thomas (1976) recognized that the sulphide mineralization occurred in two distinct stacked norite sills. Recommendations were made for the application of modern ground geophysical survey techniques, additional drilling, mapping, and prospecting; however, follow-up on these recommendations was not undertaken and the claims were allowed to lapse.

6.4 KIZAN 1987-89

In 1987, the Main Zone was re-staked as the "Anki Claims" by W. Kizan (Kizan). Kizan conducted prospecting and blasting (trenching) of mineralization exposed at surface. In 1988 the Anki 1, Anki 2, and Anki 3 claims were optioned to Hartz Equities Ltd. who conducted grid refurbishing, mapping, and trenching over the property, and commissioned a resource evaluation for the Main Zone by an independent consultant, Eckhardt Buhlmann. Buhlmann (1989) prepared a geological resource estimate for the Main Zone deposit of 15.04 Mt grading 0.45% Ni and 0.12% Cu at a cut-off of 0.1% Ni. This resource included 4.9 Mt grading 0.72% Ni and 0.19% Cu at a cut-off of 0.5% Ni. Buhlmann did not identify the resource category of the historic estimate, and as such, no comparison of the estimate can be made to the accepted categories. This historic estimate is not current and does not meet NI 43-101 or CIM definition standards. A qualified person has not evaluated this resource estimate on behalf of Strongbow. This historical resource is reported here for information purposes only and should not be relied upon.

6.5 ABER 1995

In 1995, Aber Resources (Aber), subsequently Harry Winston Diamond Corp., optioned the Anki claims from Kizan, erected a new grid over the Main Zone, and completed ground geophysical magnetometer and EM (horizontal loop) surveys, mapping, and diamond drilling (Bryan and Naeher, 1995a, 1995b). Thirteen diamond drill holes (2,248 m) (Figure 6-1) tested a variety of ground EM targets. All 13 drill holes intersected mafic to ultramafic intrusive rocks with eight holes returning anomalous nickel-copper mineralization. The highest-grade intersection was reported as 1.08% Ni and 0.26% Cu over 12 m in hole AN95-11 (Bryan and Naeher, 1995b). A forest fire destroyed all Aber drill core in 1998.

Drill holes AN95-06 and AN95-07 tested an area within the Main Zone. Both holes, drilled from the same location, intersected the gabbro/norite intrusion including zones of disseminated sulphides. Drill hole AN95-06 intersected 0.88% Ni and 0.23% Cu over 7 m, while hole AN95-07 intersected 0.39% Ni and negligible Cu over 5.89 m.

The remaining eleven holes tested targets outside of the Main Zone as defined by CANICO. Six drill holes (AN95-08 to -13) targeted ground geophysical (EM and magnetic) anomalies defined beneath Thye Lake, approximately 200 m to 400 m southwest of the Main Zone. All six holes intersected a mafic (norite) intrusion. Four of the six holes intersected significant mineralization ranging from 0.35% Ni and 0.11% Cu over 2.50 m in hole AN95-12 to 0.90% Ni and 0.22% Cu over 17.5 m in hole AN95-11. Drill logs for the remaining two holes describe zones of disseminated mineralization; however, these zones were not assayed.

Drill hole AN95-01 tested a ground EM anomaly approximately 900 m southwest of the Main Zone and 500 m southwest of the AN95-08 to -13 drilling. AN95-01 intersected disseminated and stringer sulphide mineralization in a mafic (norite) intrusion, including 0.31% Ni and 0.10% Cu over 11.50 m.

Drill holes AN95-02 and AN95-03 tested ground geophysical anomalies approximately 300 m to 500 m southeast of the Main Zone. Both holes intersected mineralized mafic (norite) intrusions with a best reported grade of 0.23% Ni over 27 m in hole AN95-02.

Drill hole AN95-04 tested a target approximately 800 m east of the Main Zone near CANICO drill hole 10247, intersecting a weakly mineralized mafic intrusion (0.15% over 3 m).

Drill hole AN95-05 tested an EM target located approximately 850 m south of the Main Zone. Although several zones of mafic intrusive were encountered, no significant nickel or copper mineralization was intersected.

Falck (1995) conducted a mapping program over the Nickel King showings on behalf of Aber. Falck interpreted the gabbro (mafic to ultramafic) intrusions as younger than the major folding events and that brittle faulting affected the gabbro, and may be responsible for complications observed in the CANICO drill logs. The nickel-copper mineralization was considered as a predominantly primary feature with only minor evidence for redistribution or remobilization.

The Anki mineral claims were allowed to lapse and in 1997, Kizan and Aber re-staked the NIC 1 and NIC 2 claims to cover the Main Zone and associated historic drilling. Aber's interest in the property was transferred to Navigator Exploration Corp. (Navigator) through an agreement dated 7 February 2000.

6.6 FALCONBRIDGE 2000-2001

In 2000, Falconbridge Ltd. (Falconbridge) optioned the NIC 1 and NIC 2 claims from Navigator and staked several more claims (WYN 1 to WYN 4, and SEL 1) adjacent to the NIC claims. The new claims were incorporated into the Navigator-Falconbridge option agreement. In 2000 and 2001, Falconbridge completed grid re-furbishing, mapping (1:2,500 scale), and prospecting and regional scale litho-geochemical surveys. Falconbridge's mapping results indicated that the mafic to ultramafic sills associated with the Main Zone can be traced to the north and south of the mapped area and were affected by at least two stages of folding and displayed a complex geology (Blair and Pattison, 2000). Falconbridge further noted that nickel mineralization was associated with orthopyroxene-rich, variably amphibolitized mafic to ultramafic sills. Litho-geochemistry indicated that the sulphide minerals have a high sulphide metal content (nickel tenor) of approximately 5% to 8% Ni in 100% sulphide, however, nickel grades were observed to be generally low (i.e., <0.5%; Blair and Pattison, 2000). A ground based EM (UTEM) geophysical survey was recommended to be completed on a new grid cut orthogonal to the historic grid as a test of the complex sill geometry; however, this work was not undertaken.

A new mineralized intrusion was discovered on the northwest shore of Selwyn Lake (the Selwyn Intrusion) in 2001 (Figure 6-1). The Selwyn Intrusion displays similar litho-geochemical characteristics and sulphide metal contents to the intrusions that host the Main Zone. Falconbridge suggested that litho-geochemical surveys might prove a useful tool for identifying the Ni-Cu potential of other intrusions in the area (Pattison, 2002).

In 2003, Falconbridge terminated the Navigator-Falconbridge option agreement and Navigator retained ownership of the mineral claims staked by Falconbridge. The WYN 2, WYN 3, and WYN 4 mineral claims were subsequently allowed to lapse.

In May 2004, Navigator amalgamated with Strongbow Resources Inc. to form Strongbow Exploration Inc. Because of the amalgamation, Strongbow assumed the rights and obligations of Navigator, including the Nickel King project.

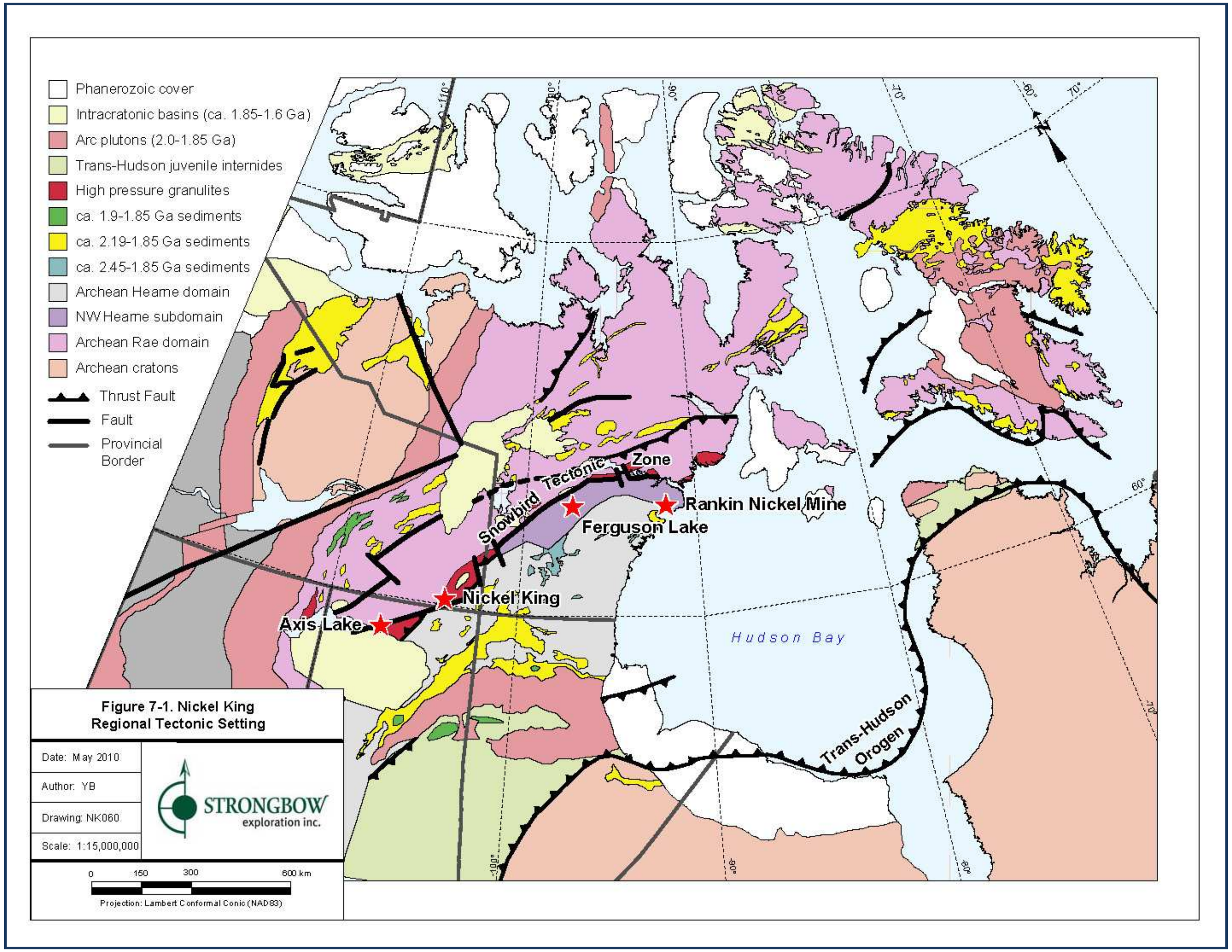
7.0 GEOLOGICAL SETTING

7.1 REGIONAL GEOLOGY

The Nickel King project lies within the Snowbird Tectonic Zone (STZ) in the northwestern part of the Canadian Shield (Figure 7-1). The STZ is a >2,800 km long crustal scale structural break that coincides with the boundary between the Archean Hearne (to the southeast) and Rae (to the northwest) domains of the Churchill Province (Hoffman, 1988). This structural break has variably been interpreted as a suture of the 1.9 Ga collision between the Rae and Hearne domains (Hoffman, 1988); an intracontinental shear zone at 2.6 Ga with limited Paleoproterozoic reworking (Wallis, 1970; Schau and Tellas, 1993; Hanmer et al., 1995) or an incipient intracontinental rift within an older, ca. 2.55 Ga, orogenic zone (Flowers et al., 2006). The tectonic interpretation and chronology of the STZ is complex and remains controversial.

The western Churchill Province is dominated by neoarchean, amphibolite to granulite granitoid gneisses and greenstone belts that are overlain by 2.45-1.75 Ga volcano-sedimentary sequences, and intruded by 1.83-1.75 Ga granite suites (Berman et al., 2007). The Nickel King project lies within the Rae domain, proximal to the northwest side of the STZ. The Rae domain consists of predominantly granulite grade orthogneiss, upper amphibolite to granulite grade paragneiss (Hurwitz Group), lesser mafic intrusive and volcanic rocks, and minor banded iron formation (Martel, 2005). Protoliths of the paragneiss are interpreted as pelitic to psammitic rocks derived from turbidite basins (Stubley, 2007). Martel (2005) has reinterpreted a thin zone of magnetic supracrustal rocks in NTS map Sheet 65D (Hearne domain) as Archean in age but this is inconsistent with the known Proterozoic age of other supracrustal units (Hurwitz group affinity) in the Churchill Province. The area underwent high pressure (granulite facies) peak metamorphism in Archean (2.6-2.5 Ga) and Proterozoic (ca. 1.9 Ga) time (Martel, 2005). Proterozoic age (1.9 Ga.) Chipman mafic dykes are particularly abundant within the Chipman panel (Flowers et al., 2006; Martel, 2005). The Nickel King claims and leases cover the west margin of a magnetically subdued basin-shaped domain within the Rae domain and are situated approximately 20 km northwest of the interpreted STZ.

Figure 7-1: Regional Tectonic Setting



7.2 LOCAL AND PROPERTY GEOLOGY

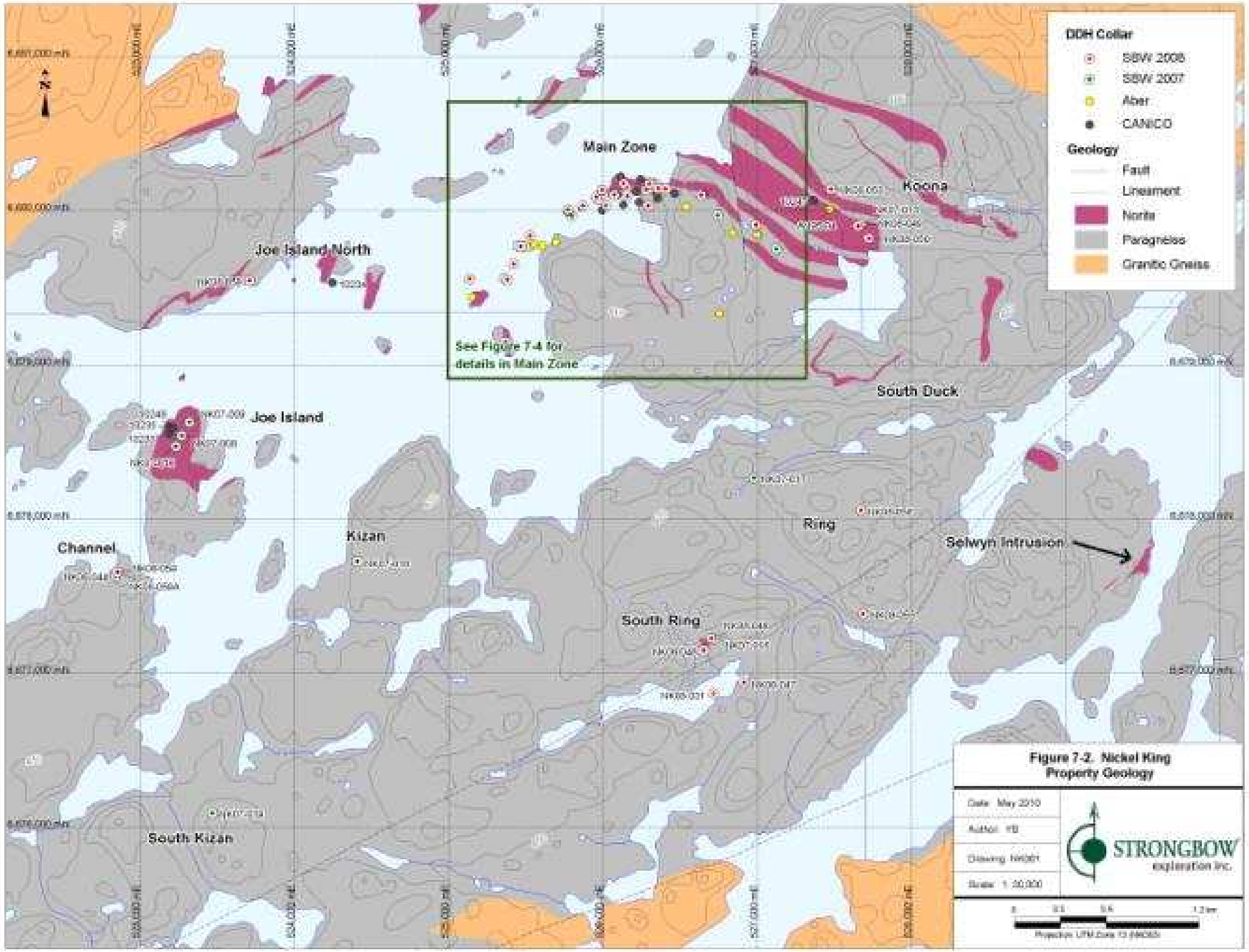
7.2.1 LITHOLOGY

Late Archean or early Proterozoic paragneisses dominate the local geology of the Nickel King area (Figure 7-2). Locally, mafic with minor ultramafic intrusive rocks (norite, gabbro, pyroxenite, and peridotites) and pegmatite dykes intrude the paragneiss. The paragneisses are considered pelitic to psammitic in composition, consistent with a turbiditic protolith. These rocks typically contain variable amounts of biotite, garnet, sillimanite, quartz, and feldspar. Local increased concentrations of hornblende may indicate a volcanic protolith and some garnet rich lithologies are interpreted as possible remnants of metamorphosed iron formation. Rare, relic bedding is observed and defined by grading of clasts in a meta-conglomerate. Sulphides (pyrite and lesser pyrrhotite) and minor graphite occur locally within the paragneiss, although no graphite has been identified in paragneiss in the immediate vicinity of the Main Zone.

Past workers have variably described the mafic/ultramafic intrusions in the Nickel King area as norites, gabbros, olivine gabbros, gabbronorites, pyroxenites, amphibolites, and peridotites. Strongbow, however, has identified olivine only at one location (the South Ring showing) and therefore, based on mineralogy, refers to all mafic intrusions as norite and, locally, gabbro or pyroxenite. Norite intrusions host most of the important mineralization in the area, including the Nickel King Main Zone deposit, and occur as medium to coarse-grained rocks comprised of orthopyroxene, clinopyroxene, hornblende, plagioclase, and phlogopite. The rocks appear to be fresh with very little alteration but in thin section display a variety of metamorphic textures. The main mineral constituent in the norite is enstatite, which occurs as equant subhedral grains with a little interstitial anhedral plagioclase. In most unaltered samples, the average grain size is 0.5 mm to 1 mm, with some grains up to 2 mm. Thin sections from the norite contain two textural populations of orthopyroxene. The first population is comprised of coarse grained orthopyroxene and the second population exhibits a significantly finer grain size and granoblastic texture. There also appears to be a significant grain size reduction at the margins of the coarse grained domains transitional into the more finely grained granoblastic host (Barnett and Renaud, 2010). Alteration of orthopyroxene to fibrous actinolite is locally developed, as well as minor sericitization of plagioclase.

At least two generations of pegmatite are noted to crosscut the norite intrusions and paragneiss (Falck, 1995). These pegmatite dykes are dominated by potassium feldspar, quartz, and biotite, with coarse (1 cm to 2 cm) deformed graphic textures in feldspars and biotite crystals locally exceeding 10 cm. The pegmatites are locally tourmaline rich with black glassy tourmaline ranging from fine radial agglomerates to one centimetre wide blades.

Figure 7-2: Property Geology



7.2.2 STRUCTURE

The paragneiss at Nickel King has a pronounced gneissosity. This dominant fabric has locally been mapped as parallel to, and partly defined by, relict bedding (Stubley, 2007). This predominant foliation, defined by aligned biotite, is everywhere parallel to gneissosity. Complex fabric relationships and folding of the gneissosity are preserved within the paragneiss. Tight to isoclinal folds with shallow-dipping (0° to 50°) axial planes are the dominant feature with variations in dip of axial planes largely attributed to a younger event that produced upright open folds (Stubley, 2007).

Norite sills, plugs, and dykes intrude the paragneiss at many localities within the project area (Figure 7-2). With few exceptions, the norite intrusions appear devoid of all tectonic fabrics except locally along marginal contacts. In thin section, evidence for strain has been interpreted by examining the larger crystals of orthopyroxene that preserve an extensive amount of wandering extinction. In addition there is abundant textural evidence supporting the interpretation that individual grains within these coarse domains have sutured grain margins and exhibit a consistent subgrain development in marginal areas that are transitional into the more granoblastic host (Barnett and Renaud, 2010). These relationships suggest some level of metamorphism and deformation, similar to the surrounding host rocks, but the exact timing and extent is uncertain. Where contacts are observed in outcrop, the norite intrusions are parallel to the gneissosity in the enclosing paragneiss, and this lack of transection of gneissosity suggests the norite has the same “corrugated isoclinal fold structure” as the paragneiss (Stubley, 2007). An outcrop observed east of the Main Zone peninsula preserves a near-profile view of an isoclinal fold of a norite-paragneiss contact, and demonstrates the folding style common in both units. Figure 7-3 shows a near-profile view of recumbent tight to isoclinal fold of norite-paragneiss contact (outlined by red dashed line in the image); the pencil lies along sub-horizontal axial plane (UTM Grid: 527078E 6680386N; taken from Stubley, 2007).

Two stacked norite sills host mineralization at the Main Zone. The sills are individually referred to as the Upper Sill and the Lower Sill and collectively as the Main Zone Norite. The Upper and Lower Sills are interpreted to form a recumbent, south facing arcuate shaped and south dipping (approximately 30° to 40°) synform that plunges gently to the west and southwest (approximately 14°). Drilling data suggests the true widths of the sills vary from 40 m to 110 m. The Upper Sill outcrops on a peninsula extending westward into Thye Lake whereas the Lower Sill outcrops immediately northeast of the peninsula (Figure 7-4).

Figure 7-3: Norite-Paragneiss Contact (Stubley, 2007)

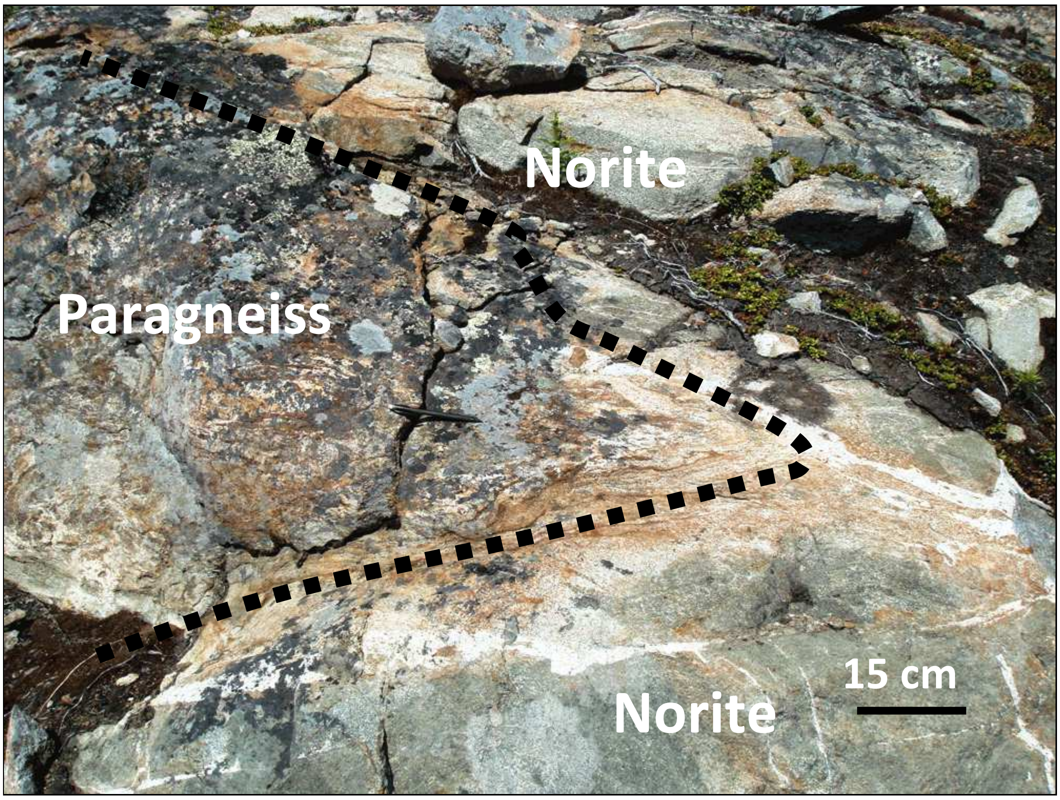
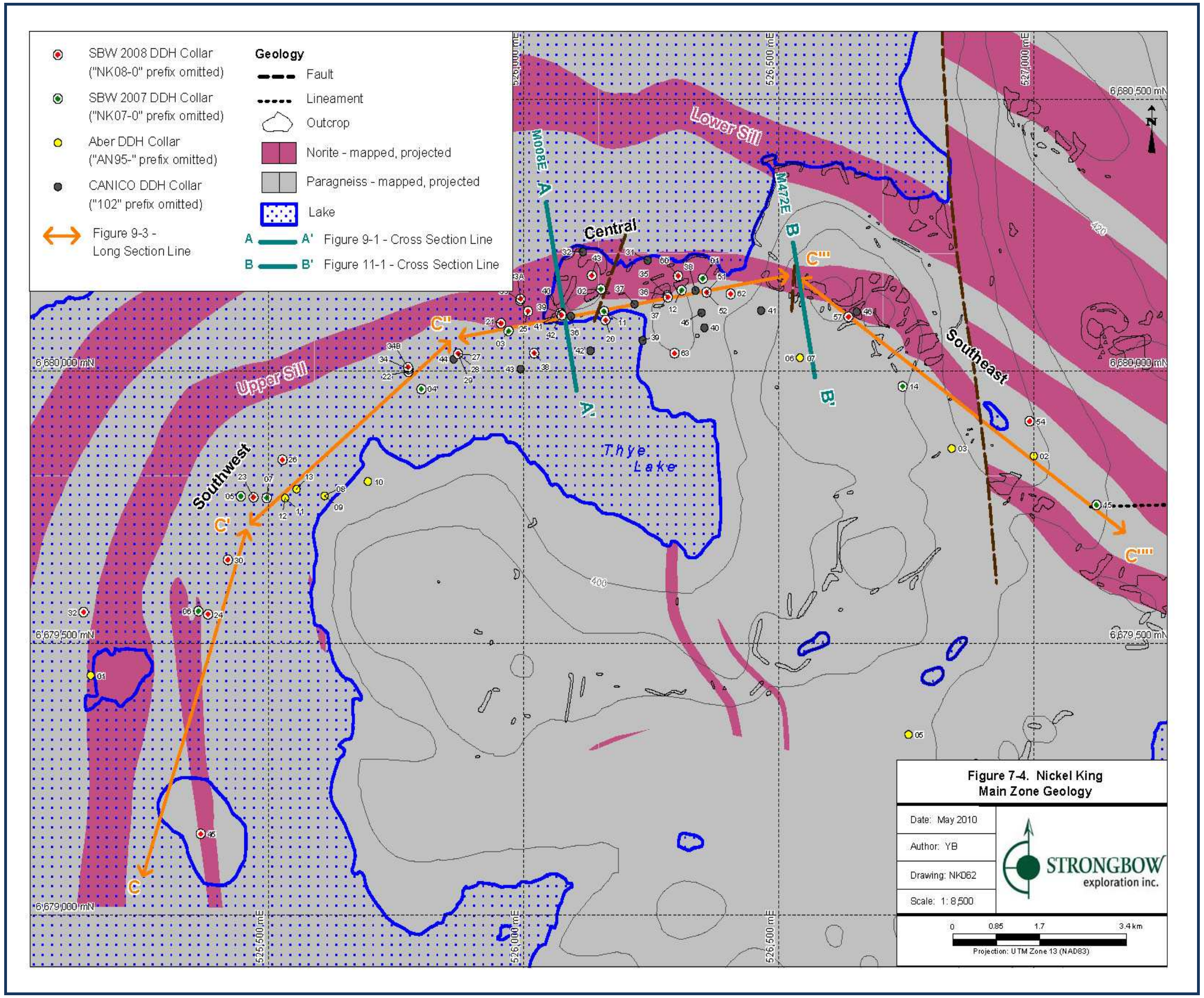


Figure 7-4: Main Zone Geology



Bedrock mapping has identified several north- to north-northeast-trending faults that cross-cut the Main Zone norite (Stubley, 2007). A north-northeast trending fault transects the Main Zone norite between drill holes NK-07-002 and NK07-011 (Figure 7-3). Limited drill data suggests that this fault has resulted in an approximately 40 m dextral displacement of the norite, but additional drilling is necessary to add confidence to this interpretation.

A second fault, located east of the peninsula (around 526880E, 6680250N) appears to truncate the Upper Sill of the Main zone. Drill results suggest that east of the fault the Upper Sill was probably uplifted and subsequently eroded.

8.0 DEPOSIT TYPES

The project area contains nickel-copper-cobalt sulphide mineralization hosted in mafic (norite) intrusions, sharing many characteristics with a diverse class of mineral deposits commonly referred to as “magmatic sulphide deposits.” Examples of large magmatic sulphide deposits (nickel-copper ± cobalt and platinum group metals) in Canada include the Sudbury nickel belt in Ontario, the Thompson nickel belt in Manitoba, the Raglan Nickel Belt in northern Quebec and the Voisey’s Bay nickel deposit in Labrador. Examples of large magmatic sulphide deposits elsewhere in the world include the Bushveld igneous complex in South Africa, Norilsk in Russia and the Stillwater complex in Montana, USA.

As a group, magmatic nickel sulphide deposits share many characteristics with respect to geological setting and style of mineralization; however, individual deposits often maintain unique characteristics. In general, these deposits can be subdivided into extrusive, ultramafic (komatiite) hosted deposits and intrusive mafic to ultramafic types. Magmatic sulphide deposits typically require a geological setting that includes deep, mantle-tapping structures, a rising ultramafic magma subjected to some combination of differentiation, magma mixing, and country rock contamination, resulting in the formation of an immiscible sulphide melt. The sulphide melt is then concentrated either by way of a physical trap or gravitational separation followed by continued interaction with sufficient volumes of silicate melt to upgrade the nickel and copper contents of the sulphide phase. The majority of the world’s sulphide nickel deposits occur within extrusive and intrusive ultramafic rocks with a comparably small subset hosted with mafic, gabbro-related rocks.

Nickel King’s mineralization is somewhat unusual in that to date true ultramafic rocks have not been identified within the Main Zone deposit and sulphide minerals are hosted in rocks of noritic composition. One area where norite host rocks are common is the Sudbury nickel belt. However, these deposits were formed as a result of an astrobleme, and as such are not directly analogous to Nickel King.

The Ni-Cu Platinum Group Elements (PGE) sulphide deposits of eastern Botswana, however, do exhibit similar characteristics to Nickel King. A number of mineralized mafic intrusions having gabbroic to noritic compositions (Maier et. al., 2007) have intruded the Tati and the Selebi-Phikwe metamorphic belts. These intrusions occur within the high-grade Limpopo metamorphic belt that occurs between two Archean cratons. At the Selkirk deposit in the Tati belt, high-grade mineralization forms a 20 m thick, 250 m long lens of massive sulphides within a 2 km x 3 km wedge-shaped intrusion (Maier et. al., 2007). The massive sulphides are mantled by a zone of disseminated sulphides. Additional significant resources have been defined within this disseminated zone (165 Mt at 0.28% Ni using a 0.15% cut-off grade; Clegg and Bennett, 2007). Another seven known deposits within the Tati and Selebi-Phikwe belts range from 0.6 Mt to 31 Mt with nickel grades ranging from 0.5% to 2.05% (Maier et. al., 2007). The tectonic relationships, host rock type, and the occurrence of a high-grade massive sulphide lens within a mantle of low-grade disseminated mineralization provide a model comparable to the Nickel King deposit.

Also comparable with the Nickel King deposit is the Las Aguilas Ni-Cu deposits situated in San Juan Province, Argentina. At this deposit mafic-ultramafic rocks intrude an early Paleozoic medium- and high-grade metamorphic complex comprising mostly paragneiss. Mineralization is characterized by pyrrhotite-pentlandite-chalcopyrite. Historical reserves (non-compliant NI 43-101) at the Las Aguilas deposit were 2.2 Mt at 0.52% Ni, 0.50% Cu, 0.037% Co, and 0.4 ppm PGE, but this does not reflect more recent drilling results.

Mineralization at Las Aguilas West deposit is hosted by a sub-vertically dipping orthopyroxenite to melanorite dyke. Similar to Nickel King, the mineralized zone forms a tabular body hosted within the dyke. Drilling to 2008 indicates that the deposit extends at least 250 m down-dip and 500 m along strike, and is open in both directions along strike as well as down-dip. The mineralized zone ranges from 2 m to 21.5 m in true thickness, and in the northern part of the deposit two parallel zones occur. Mineralization is concentrated in the melanorite or more ultramafic portions. Typical intersections include LA08-070 with 18.95 m at 0.40% Ni, 0.48% Cu, 0.03% Co, 0.23 ppm Pt, and 0.31 ppm Pd. The nearby Las Aquilas East deposit is hosted within a 100 m thick mafic-ultramafic intrusion dominantly comprising orthopyroxenite and melanorite, with lesser dunite and harzburgite. Typical intersections include LA08-078 with 19.67 m at 0.58% Ni, 0.42% Cu, 0.03% Co, 0.44 ppm Pt, 0.50 ppm Pd, and 0.22 ppm Au. The host intrusion is funnel-shaped at depth.

This deposit is analogous to Nickel King because of the orthopyroxenite to melanorite host rock, the tabular nature of the mineralization that occurs within a thicker, dyke-like intrusion and the broad zones of moderate nickel and copper grades. It suggests that similar magmatic processes could have controlled the emplacement of these two sulphide nickel deposits.

Exploration for sulphide nickel deposits is guided by regional targeting, mapping and prospecting for mafic to ultramafic intrusions, and locating associated gossans or zones of massive to semi-massive sulphides. The country rock type to favourable mafic intrusives may also be important as a possible sulphur source. Therefore, sedimentary host rocks to mafic intrusives may be important for exploration if it can be demonstrated that they could provide a potential source of sulphur (e.g., iron formation, pyritic horizons, anhydrite horizons, etc.).

Remote exploration methods often begin with airborne magnetic surveys to distinguish favourable geological units and, in some instances, magnetic highs associated with high concentrations of magnetite or sulphide minerals like pyrrhotite that are often associated with pentlandite, the main sulphide nickel ore mineral. In conjunction with magnetic surveys, airborne electromagnetic (EM) surveys are useful for identifying high concentrations of conductive sulphide minerals. In general, the higher the concentration of sulphides, particularly the copper sulphide mineral chalcopyrite, the stronger the EM anomaly (conductive response) will be. EM anomalies associated with magnetic high anomalies are generally expected to be associated with significant pyrrhotite-chalcopyrite-pentlandite mineralization. If ground geophysical surveys (magnetics and EM), prospecting, and lithogeochemical surveys were successful in identifying near surface showings or prospective terrain, then diamond drilling would be required to confirm and define the extent of potential economic mineralization.

9.0 MINERALIZATION

Five areas of mineralization discovered to date within the Nickel King project are the Main Zone, Koona, and South Ring prospect, the Joe Island Trend, and the Selwyn Intrusion (Figure 7-2). In each case nickel-copper sulphide mineralization is hosted within a norite intrusion, ranging from a discrete, plug-shaped intrusive (South Ring) to the extensive, > 2,600 m long arcuate Main Zone norite. Variation occurs in the rocks at South Ring where norites, gabbros, and peridotites have been observed in three drill holes.

The largest mineralized zone at Nickel King is the Main Zone (Figure 7-4). The Main Zone has been defined over a strike length of 2,600 m. Drilling and mapping surveys have led to the interpretation that mineralization is hosted within the limbs of a recumbent, tightly folded norite sill (Figure 9-1), which in turn was folded by a younger folding event that resulted in a south facing, arcuate, open fold geometry.

The Upper Sill of the Main Zone is exposed on the east shore of Thye Lake, trends approximately east-west, typically dips moderately (30° to 50°) to the south, and plunges gently (approximately 10° to 15°) to the southwest beneath Thye Lake. The Lower Sill is located beneath the Upper Sill, dips to the south at a somewhat shallower angle (10° to 40°), and has a similar gentle southwest plunge. Mineralization consists of disseminated to semi-massive iron-nickel-copper sulphides (pyrrhotite-pentlandite-chalcopyrite), concentrated close to the upper margins of both sills. The best mineralized portions of each sill are typically enveloped by a broad zone of disseminated sulphides that ranges from 20 m to 60 m in true thickness. The higher grade, net-textured (Figure 9-2A) to semi-massive sulphide zones (Figure 9-2B) range from 2 m to 14 m thick, with more significant and continuous zones (e.g., greater than 6 m of 0.75% Ni) typically occurring within the Lower Sill.

Disseminated mineralized zones have drill-defined widths (up-dip and down-dip extent) of up to 120 m, but more typically range from 60 m to 80 m (e.g., within the Lower Sill), although mineralization is generally unconstrained to the north (up-dip). Drilling results suggest that the true thickness of mineralization thins in the up-dip direction (Figure 9-1). Less information is available in the down-dip direction, but in at least some areas mineralization appears to be abruptly cut off. Modelled geophysical conductors provide some evidence for potential mineralization further down-dip to the south of at least one of the drill holes (DDH NK08-20) that appears to cut off mineralization.

Figure 9-1: Vertical Section M008E, A – A'

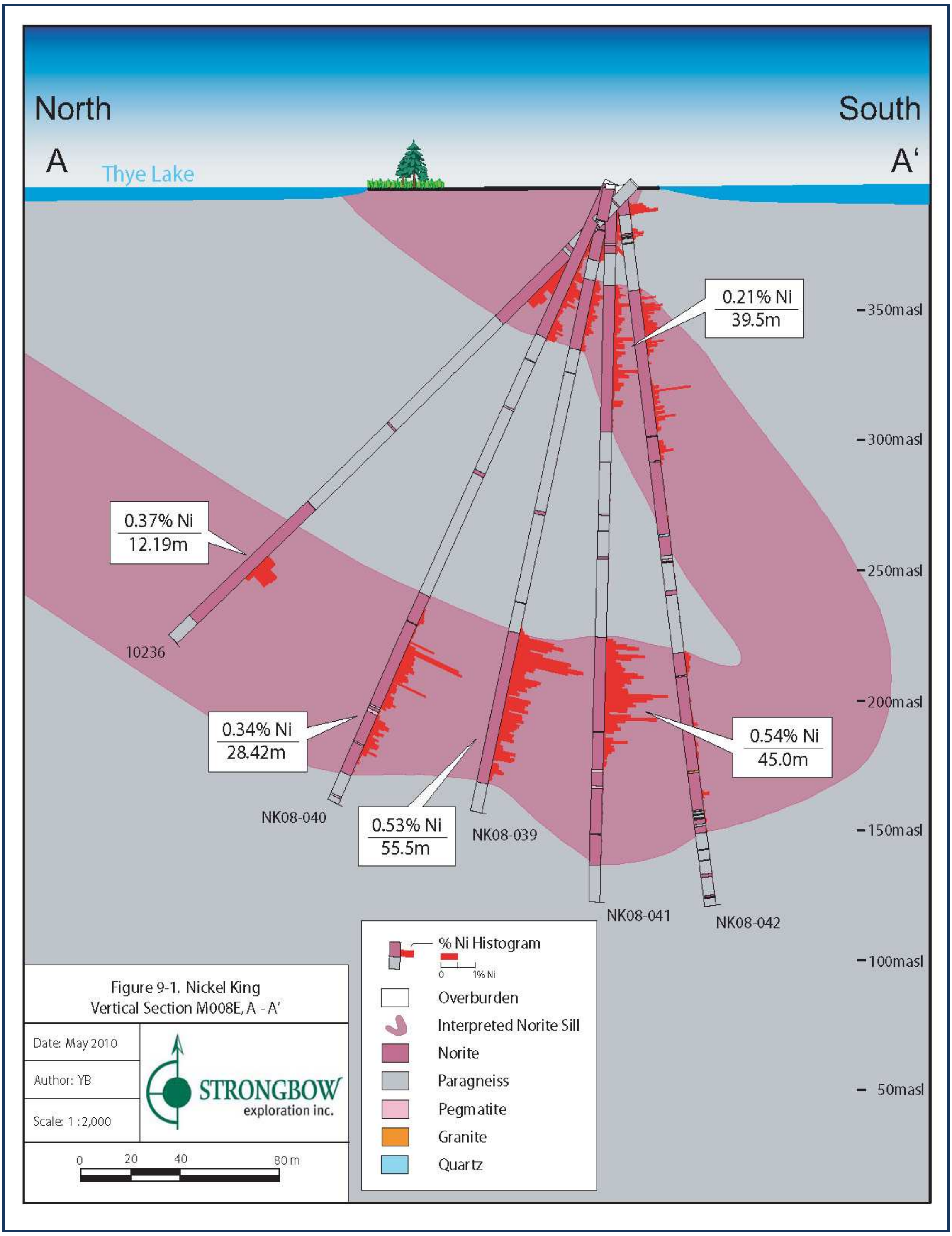


Figure 9-2: (A) Net Textured Sulphides (B) Semi-Massive Sulphides

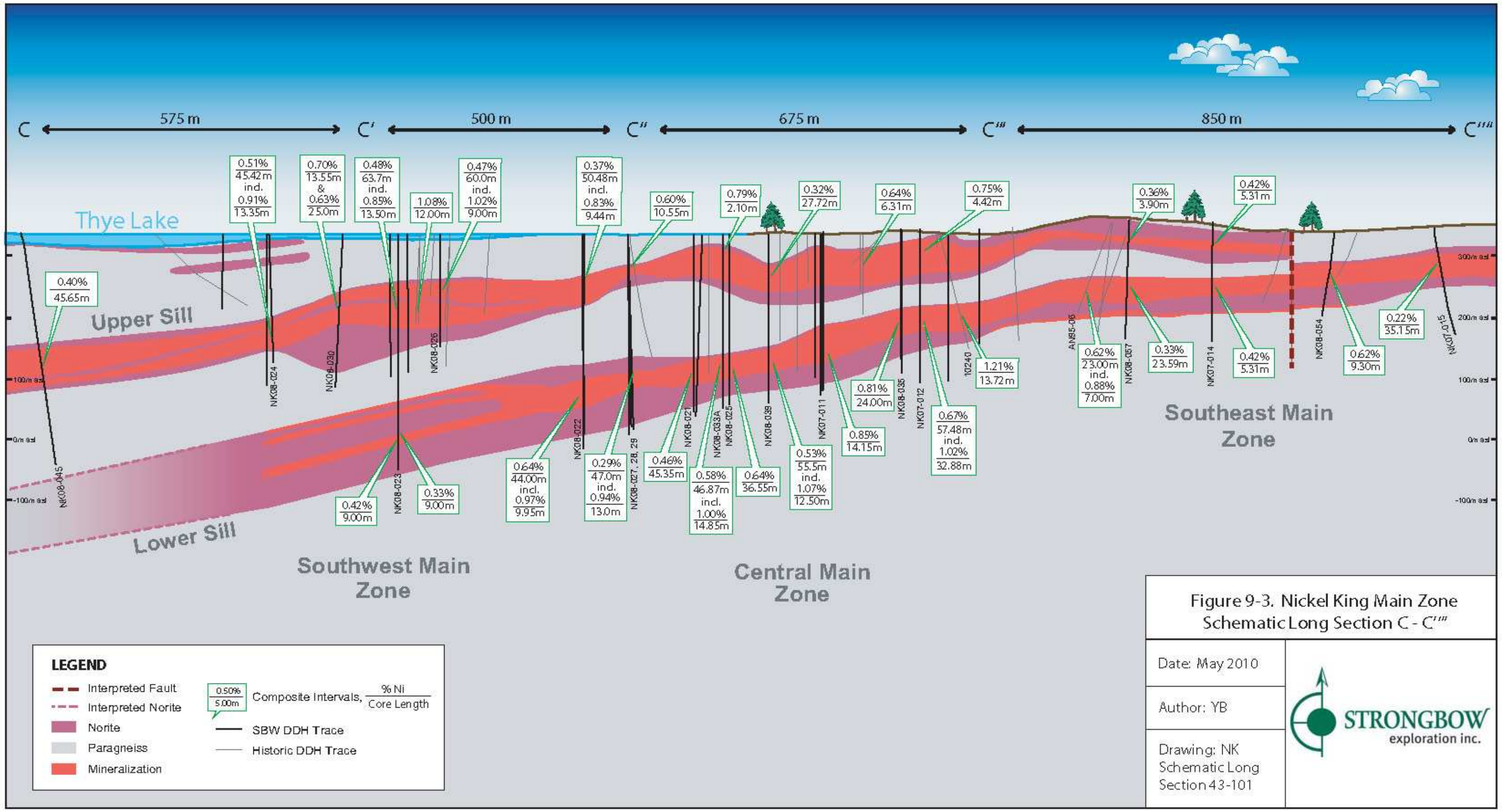
(A) Net textured sulphides @ 140 m in hole NK07-012
Interval returned 1.36% Ni over 1.23 m.

(B) Semi-massive sulphides @ 171 m in hole NK07-11
Interval returned 3.87% Ni over 0.4 m.

The observed sulphide assemblage and textures in the Main Zone are typical of magmatic sulphide deposits in mafic-ultramafic rocks (Le Couteur, 2007). The sulphide disseminations and clots are typically highly irregular in shape because they conform to the spherical triangle shapes of interstices and engulf some silicates. Sulphide minerals vary from less than 0.5 mm up to 7 mm across, and consist mainly of pyrrhotite with variable proportions of inclusions of pentlandite and chalcopyrite. Pentlandite inclusions are generally rather shapeless and blocky, and average 0.5 to 1 mm across, although rare examples reach 2 mm across. Chalcopyrite is also shapeless and generally smaller than associated pentlandite, usually less than 0.5 mm. Pentlandite and chalcopyrite also occur as veinlets in pyrrhotite, mostly less than about 0.1 mm wide. Pentlandite may also occur as small blebs, and a very small amount (<1% total pentlandite observed in thin section) occurs as “flames,” which often show a preferred orientation.

The gradual westerly plunge of the mineralization is remarkably consistent and curves into a southwest-striking geometry that continues under Thye Lake (Figure 7-4 and Figure 9-3). Mineralization also continues to the east and southeast but levels off and appears to demonstrate a more horizontal or very shallow (e.g., ~5°) plunge to the west and north. Along the entire strike length of the Main Zone, net-textured mineralization (most typically hosted within the Lower Sill) is interpreted to occur as ribbons or tubes, porpoising through a disseminated sulphide envelope. Drilling in the central portion of the Southwest Main Zone includes only one drill hole (DDH NK08-23) that was extended deep enough to intersect the Lower Sill. As a result, the majority of mineralization defined to date in this area is hosted in the Upper Sill (Figure 9-3). Mineralization remains open along strike to the southwest and significant potential remains in this direction to extend and expand the deposit in both the Upper and Lower Sills.

Figure 9-3: Schematic Long Section C – C″″



Within the Upper Sill, a distinctive zone of mineralization is interpreted to form a northward-tapering wedge of semi-massive sulphides that exhibit a pseudo-cumulate texture defined by coarse (0.5 cm to 1 cm), sub-angular to sub-rounded pyroxene crystals surrounded by fine to medium grained pyrrhotite, pentlandite, and chalcopyrite (Figure 9-4). This zone ranges from approximately 2 m in thickness at its northern end (NK08-33A) to at least 10 m thick in the south (NK08-24). This cumulate zone is located within or immediately above a broader zone of more typical disseminated sulphide mineralization. The cumulate texture is only rarely observed in drill core to the east of drill hole NK08-33A. This style of mineralization, however, is observed in outcrop within the Upper Sill on the peninsula extending into Thye Lake, although the true thickness of the outcropping cumulate mineralization is difficult to estimate due to strong weathering and alteration. This style of mineralization has not been identified in the Lower Sill.

Figure 9-4: Cumulate-like Texture in DDH NK08-24 at 178 m Downhole

The eastern extent of the Upper Sill is cut off or displaced by a north-south trending fault located to the east of the peninsula (Section 7). Only mineralization within the Lower Sill is interpreted to extend to the east of this fault (Figure 9-3). The thickness of disseminated mineralization in this area typically ranges from 10 m to 30 m, with the higher-grade net textured zones forming 2 m to 5 m thick intervals. The limited density of drilling within the eastern extent of the Main Zone does not allow for an accurate summary on the extent of mineralization in the down-dip, up-dip dimension. The Main Zone norite maintains a south to southwest dip along the entire strike length of the mineralized zone. Depth to the top of the mineralization ranges from 0 (surface) to 140 m; however, the majority of the Main Zone mineralization begins at depths of less than 90 m (eastward from hole NK08-030).

The sulphide metal content (i.e., nickel in 100% sulphides or nickel tenor) of the Main Zone varies between the Lower and Upper Sills. Generally, the Lower Sill returns higher nickel grades and has, on average, a higher sulphide nickel content (4% to 7% Ni in 100% sulphides) than the Upper Sill (3% to 7% Ni in 100% sulphides). The sulphide nickel content of the Upper Sill has been observed to change sharply (3% Ni compared to 5% Ni in 100% sulphides). The Ni:Cu ratios are consistently in the 4:1 or 5:1 range in both sills, whereas Ni:Co ratios range from 25:1 in the Lower Sill to 15:1 in the Upper Sill.

Significant concentrations of PGE are rare in the Nickel King project. Twenty-six samples from the Main Zone drill core returned greater than 100 ppb platinum (Pt). The average Pt value in these twenty-six samples is 231 ppb and the maximum value is 1,011 ppb (drill hole NK08-023). Five Main Zone drill core samples returned Palladium (Pd) values greater than 100 ppb (maximum value of 187 ppb). A majority of the samples (20 of 26) with greater than 100 ppb Pt occur in samples that returned greater that 0.5% Ni, and all five samples with greater than 100 ppb Pd occur in samples with greater than 1.6% Ni. The anomalous PGE values do not appear to be concentrated in any particular location within the Main Zone and are found in both sills. However, there is a tendency for the anomalous PGE values to occur at or near the base of some of the higher-grade nickel mineralized zones.

The following prospects are located in proximity to the Nickel King Main Zone, but do not constitute part of the resource estimation in Section 17 of this report.

The Koona Zone is situated 500 m northeast of the Main Zone and has been tested by six drill holes (Figure 7-2). Mineralization is hosted within a single norite sill, up to 80 m thick and tentatively interpreted to dip to the southwest. Mineralization occurs within the top portion of the sill and consists of net-textured to semi-massive, chaotically banded, pyrrhotite, chalcopyrite, and pentlandite. Broad zones of disseminated sulphides as seen in the Main Zone are not apparently present at Koona and only thin, 0.5 m to 4.0 m zones of mineralization are observed. Drilling has traced mineralization over a 400 m strike length; however, the limited drilling density precludes an accurate interpretation of the geometry of both the mineralization and the host norite sill. All mineralization intersected to date is shallow, occurring within 45 m from surface. The sulphide metal content of the Koona prospect is very similar to the Lower Sill of the Main Zone, ranging from 5% to 7% Ni in 100% sulphides.

The South Ring Zone is situated 3.5 km south of the Main Zone (Figure 7-2). Mineralization is hosted within an outcropping intrusive that is variably noritic to gabbroic to peridotitic in composition, and forms a small, plug-shaped body. Three drill holes have tested the prospect, intersecting a number of stacked, disseminated sulphide zones that range in thickness from 3 m to 21 m. The mafic to ultramafic intrusive contains varying amounts of orthopyroxene and clinopyroxene that in some drill holes have been completely altered to a coarse feathery intergrowth of tremolite and anthophyllite. In some instances (DDH NK08-48) the rock is 50% olivine, locally replaced by green serpentine and intergrown with coarse domains of pargasite, orothopyroxene, phlogopite, and 5% coarse-grained green aluminum-spinel intergrown with sulphides (pentlandite, pyrrhotite, and chalcopyrite). The intrusion hosting mineralization at South Ring is distinct in that it is the only intrusive within the project area that has disseminated magnetite associated with the norite and sulphide mineralization, and contains the only known occurrence of olivine on the Nickel King property. The sulphide metal content of the South Ring Zone is favourable (5.5% Ni) and has a lower Ni:Cu ratio (2:1) than the Main Zone.

The Joe Island Trend is situated approximately 4 km west of the Main Zone and consists of at least three mineralized zones: Channel, Joe Island, and Joe Island North (Figure 7-2). The Joe Island Zone has been tested by six drill holes and is one of the strongest conductors on the property. The mineralization ranges from 5 m to 22 m in thickness, and consists of disseminated to semi-massive zones of pyrrhotite and chalcopyrite with rare pentlandite. Due to the limited number of drill holes, the exact geometry of the host norite sill and related sulphide zones are not known. The Channel target is located 900 m south of Joe Island and was tested by three closely spaced drill holes in 2008. Mineralization begins at or near the bedrock surface and ranges from 2.5 m to 8.5 m in thickness. The geometry of the host norite is unknown. At the very north end of the Joe Island trend, a single drill hole (NK08-58) tested a deep target (Joe Island North), intersecting weakly disseminated pyrrhotite ± chalcopyrite within a 30 m interval of norite. Mineralization in the Joe Island area is distinguished by generally lower sulphide metal contents (<1% Ni and ~2% Ni in 100% sulphide for Joe Island and Channel, respectively).

The Selwyn Intrusion is situated 4.0 km to the southeast of the Main Zone (Figure 7-2). Mineralization consists of discontinuous pods of trace to 2% pyrrhotite, ± trace chalcopyrite within an isoclinally folded norite sill that plunges to the southwest and dips to the southeast. Grab samples have returned from background values up to 0.30% Ni. No significant geophysical anomaly has been identified in this area to date and therefore, it has not warranted drill testing.

10.0 EXPLORATION

Strongbow has actively explored the Nickel King project since June 2004. Three airborne geophysical surveys have been completed on behalf of Strongbow: a fixed wing magnetic survey in 2004 by Tundra Airborne Surveys Ltd. (Tundra), a fixed wing MEGATEM© survey in 2006 by Fugro Airborne Surveys (Fugro) and a helicopter-borne VTEM survey in 2008 by GEOTECH Ltd. (GEOTECH) (Figure 10-1). Crone Geophysics Ltd. (Crone) also conducted Time Domain Electromagnetic (TDEM) ground and borehole geophysical surveys in 2007 and 2008. Strongbow has also conducted bedrock mapping, ground magnetic, and VLF surveys within the project area. Three diamond drill campaigns were completed between 2007 (spring and fall) and 2008.

In June 2004, Tundra completed a 1,089-line kilometres fixed wing horizontal gradient magnetic survey over the project area (Figure 10-1). The survey was flown with a north-south flight line direction, a line spacing of 75 m, and nominal sensor height of 40 m above ground level. The survey defined an arcuate magnetic feature coincident with the Main Zone.

In September 2006, Fugro conducted a MEGATEM© electromagnetic and magnetic survey of the project area. A total of 804-line kilometres of data were collected at a line spacing of 200 m, a nominal magnetometer sensor height of 73 m, and a nominal electromagnetic receiver height of 60 m. The survey was flown in two overlapping orthogonal survey blocks to ensure maximum coupling over the arcuate Main Zone. Survey block 1 was flown in a northwest-southeast orientation while block 2 was flown in a northeast-southwest orientation (Figure 10-1). This survey identified magnetic and electromagnetic anomalies coincident with the known Main Zone mineralized trend. The Fugro survey also identified five additional electromagnetic targets outside of the Main Zone but within the Project area, including the Joe Island trend, Koona trend, and Ring South structure. No significant electromagnetic anomalies were identified over the Selwyn intrusion.

In March 2008, Strongbow contracted GEOTECH to conduct a helicopter borne VTEM survey of the Nickel King project area. Two survey blocks were flown that resulted in one seamless dataset over the central section of the Nickel King project (Figures 10-1 and 10-2). The colours displayed on Figure 10-2 reflect the relative intensity of the Earth's magnetic field; red areas reflect more magnetic rocks relative to blue areas. A total of 485-line kilometres of data was collected using a nominal line spacing of 75 m, a nominal magnetometer sensor height above ground of 65 m, and a nominal electromagnetic sensor height above ground of 30 m. The VTEM survey provided a more detailed dataset, accurate georeferencing, and better depth penetration, which contributed an improved understanding of the extensive magnetic and EM anomalies associated with the Main Zone. EM anomalies were identified along the entire length of the Main Zone and are generally coincident with the magnetic anomalies. These EM anomalies can be seen on Figure 10-3, which shows a Tau image of the Nickel King property. The Tau, also known as the decay constant, is a measure of the rate of decay and therefore the strength of the conductivity of an anomaly. A slow rate of decay, reflecting a high conductivity, will be represented by a high decay constant (red and orange colours on Figure 10-3). This figure highlights the drill-defined 2,600 m extent of mineralization at the Main Zone, as well as the undrilled, along strike potential to the southwest and southeast. The Koona, South Duck, and Joe Island targets are three other areas that have extensive Tau anomalies with only limited drilling.

Figure 10-1: Airborne Geophysical Survey Areas

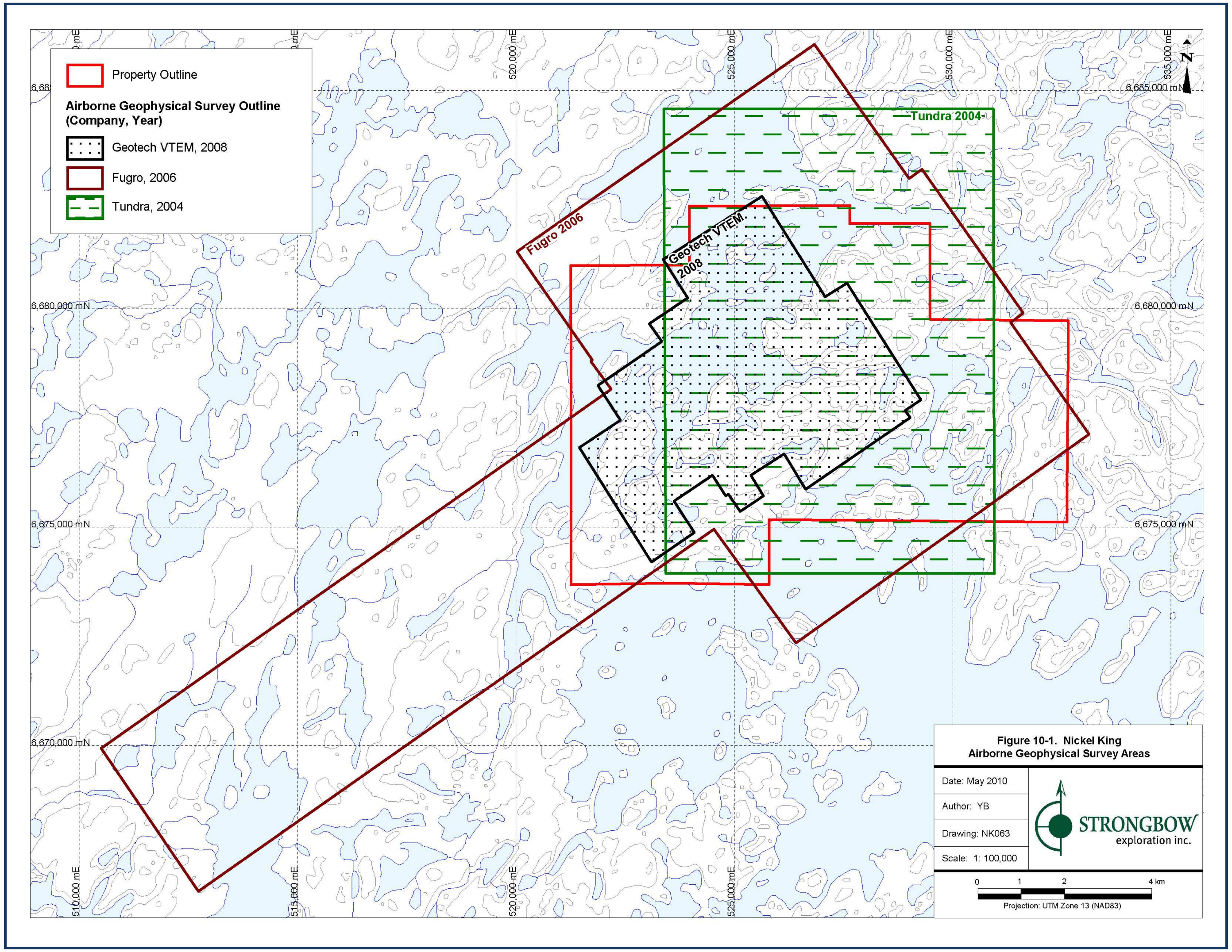


Figure 10-2: Airborne Total Field Magnetics

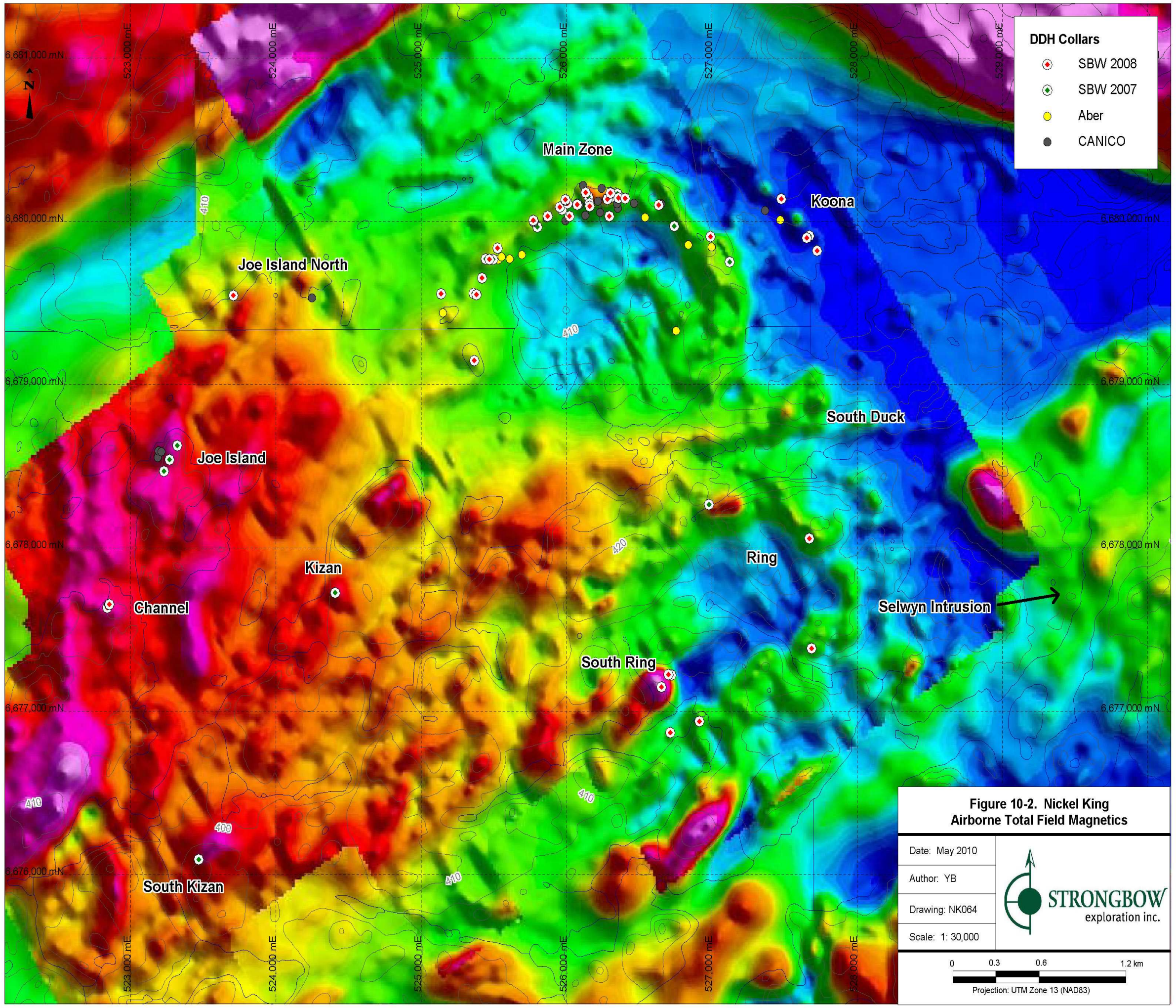
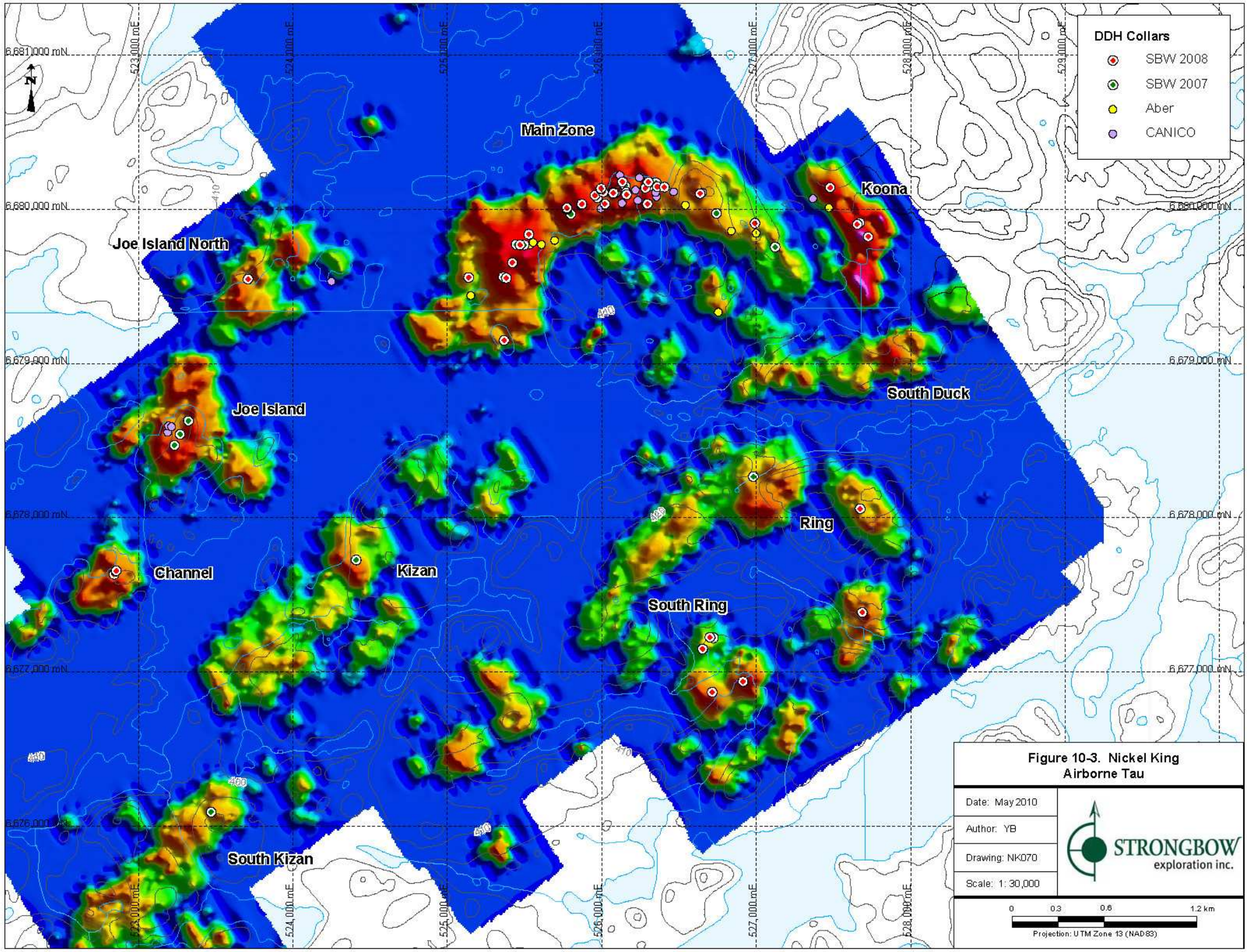


Figure 10-3: Airborne Tau



In 2007 and 2008, ground grids were established on the Main Zone, Koona, Ring, Kizan, South Kizan, and Joe Island areas. Each grid was established utilizing 100 m line spacing and 25 m station spacing. A total of 180-line kilometres of ground grids were established in 2007 and 2008. Selected portions of these grids were refurbished in 2008. The Main Zone was covered by the Main, Main Southwest, and Koona grids, which were all tied into a common local grid (0, 0) coordinate. This (0, 0) coordinate was marked by a steel pin in bedrock on the Thye Lake peninsula and tied into the legal survey of claims for conversion to mineral lease in 2007. The UTM co-ordinate of this pin is 6680096.82N, 526071.75E and 398.08 masl (NAD 83, Zone 13).

Crone conducted TDEM surveys over selected portions of the ground grids and surveyed selected boreholes. The TDEM surveys were typically completed using 100 m line spacing and 25 m station spacing. A total of 49-line kilometres of TDEM ground survey data was collected in 2007 and 35-line kilometres in 2008. Borehole geophysical surveys were completed on a total of 15 drill holes in 2007 and 17 drill holes in 2008. Strongbow completed ground magnetic and Very Low Frequency Electromagnetic (VLF-EM) surveys over selected portions of the Main Southwest, Joe Island, Koona, Kizan, and Ring ground grids in 2007.

The Ground TDEM and magnetic surveys effectively mapped the trend of mineralization within the Main Zone; however, due to the vertically stacked geometry of mineralization within the Upper and Lower Sills, it was generally not possible to distinguish whether the anomalous responses were related to Upper Sill mineralization, Lower Sill mineralization, or a combination of the two. Borehole TDEM surveys proved to be useful for distinguishing between mineralization associated with the Upper and Lower Sills, as well as detecting 'off-hole' anomalies.

Significant conductive responses, generally with coincident magnetic highs, were also established on the Joe Island, Koona, and Ring Grids. Generally, weaker but still significant conductive responses were also established on the Kizan and Kizan South grids.

The VLF-EM surveys were quite limited in extent (confined to portions of the Koona and Ring grids), but support and generally parallel the conductive responses noted in the ground TDEM data.

Stubley (2007) completed 1:10,000 scale-mapping of the Main Zone and produced a geological map that summarized the distribution of the norite and paragneiss units. This work concluded that the Main Zone Norite has been subjected to the same deformation events as the bounding paragneiss, based on the fact that tight to isoclinal folds with shallow-dipping (0° to 50°) axial planes were identified in both the norite and paragneiss. A variation in dip of axial planes is largely attributed to overprinting upright open folds. Three solutions have been proposed to explain the outcrop geometry and distribution of norite in the area of the Main Zone (Stubley, 2007):

1. two or more separate sills with only minimal fold interpretation
2. a recumbent synformal fold of a single sill
3. a sheath fold; this fold style could explain most of the observed features on the property; however, the evidence is not conclusive.

Brozdowski (2009) completed a desktop study and concluded that Stubley's third solution, a sheath fold model, represents a plausible solution to geometrically account for the present configuration of the Nickel King deposit. This model helps explain the parallel and stacked upper and lower mineralized zones, and proposes that the deposit is deformed by a nearly easterly-vergent, shallowly westerly-plunging sheath fold. This interpretation was supported by 3D modelling of the Nickel King geology. If the model is correct, the source or feeder to mineralization is most likely to be found at the western end of the deposit.

11.0 DRILLING

In 1951 and 1952, CANICO completed 18 drill holes (3,528.7 m) in the Nickel King area. Thirteen of these drill holes were completed within the Main Zone. The only available information relating to this drilling are the drill logs. The collar locations were determined with reference to a temporary local grid and it is not known how the drill hole locations were surveyed nor is the method of downhole survey data known (it may be reasonable to assume acid tests were used).

Aber attempted to re-establish the CANICO drill collar locations from data and or files available at that time (Bryan and Naeher, 1995b). In general, Strongbow has utilized the collar locations as determined by Bryan and Naeher (1995b) and confirmed by field evidence for past drilling at each location.

In 1995, Aber completed 13 holes (2,248.7 m) in the Nickel King area. Eleven of these holes were completed in the Main Zone. Aber located the drill collars on a local grid with a baseline azimuth of 099°, line azimuth of 009° and line spacing of 80 m. The grid geometry was similar to CANICO's grid but the exact centre coordinates of the two grids were slightly different. It is unclear whether Aber was able to locate the historic CANICO grid on the ground or simply by geo-reference to historic maps. The drill hole orientation was controlled by acid tests that were generally taken at or near the collar and at the end of the drill hole. Using Aber's documentation, Strongbow converted the drill hole grid locations into UTM coordinates and considers these locations to be accurate within 25 m.

During 2007 and 2008, Strongbow completed three separate drilling programs at Nickel King. A total of 66 drill holes (13,480.9 m) were completed on six general target areas (Table 11-1). The bulk of the drilling has been focused on exploration of the Main Zone with the balance of the drilling testing five other target areas. Table 11-2 shows a complete summary of all the drill holes completed by Strongbow.

Table 11-1: Summary of Strongbow Drilling

Area	Number of DDHs	Total Metres Drilled	Year Tested
Main Zone	45	10,632.2	2007/08
Koona	4	414.7	2007/08
Ring	8	1,295.7	2007/08
Kizan	1	136.9	2007
South Kizan	1	152.0	2007
Joe Island Trend	7	849.5	2007/08
Total	**66**	**13,480.9**	

Table 11-2: Details of all Strongbow Exploration Diamond Drill Holes

Hole_ID	Hole Length	UTM Easting	UTM Northing	Elevation (ASL)	Dip	Azimuth	Year_Drilled	Date_Started	Date_Completed	Area Drilled
NK07-001	242	526350.3	6680170.3	399.1	-85	350	2007	22/03/2007 0:00	25/03/2007 0:00	Main Zone Central
NK07-002	260	526151.2	6680150.7	401	-85	350	2007	26/03/2007 0:00	30/03/2007 0:00	Main Zone Central
NK07-003	296	525970	6680074	396	-85	326	2007	30/03/2007 0:00	03/04/2007 0:00	Main Zone Southwest
NK07-004	350	525799	6679967	394	-85	326	2007	04/04/2007 0:00	08/04/2007 0:00	Main Zone Southwest
NK07-005	232.47	525445	6679770	397	-85	326	2007	09/04/2007 0:00	12/04/2007 0:00	Main Zone Southwest
NK07-006	246.75	525362	6679559	394	-85	288	2007	12/04/2007 0:00	18/04/2007 0:00	Main Zone Southwest
NK07-007	223.42	525496	6679767	397	-85	326	2007	19/04/2007 0:00	21/04/2007 0:00	Main Zone Southwest
NK07-008	146	523269	6678541	406	-85	117	2007	22/04/2007 0:00	25/04/2007 0:00	Joe Island
NK07-009	113	523322	6678628	416	-85	117	2007	26/04/2007 0:00	27/04/2007 0:00	Joe Island
NK07-010	143	523232	6678470	402	-85	117	2007	28/04/2007 0:00	30/04/2007 0:00	Joe Island
NK07-011	257	526157	6680110	397	-85	350	2007	19/08/2007 0:00	24/08/2007 0:00	Main Zone Central
NK07-012	232.81	526309	6680149	402	-85	350	2007	24/08/2007 0:00	27/08/2007 0:00	Main Zone Central
NK07-013	195.62	527670	6679912	418	-85	234	2007	28/08/2007 0:00	31/08/2007 0:00	Koona
NK07-014	193.95	526742	6679972	415	-84	54	2007	01/09/2007 0:00	03/09/2007 0:00	Main Zone Southeast
NK07-015	182	527122	6679754	405	-85	54	2007	04/09/2007 0:00	08/09/2007 0:00	Main Zone Southeast
NK07-016	108.31	526722	6677223	400	-78	312	2007	08/09/2007 0:00	09/09/2007 0:00	South Ring
NK07-017	295.4	526980	6678267	405	-80	148	2007	10/09/2007 0:00	15/09/2007 0:00	Ring Grid
NK07-018	136.9	524409	6677727	406	-72	328	2007	16/09/2007 0:00	18/09/2007 0:00	Kizan
NK07-019	152	523469	6676094	419	-85	133	2007	20/09/2007 0:00	22/09/2007 0:00	South Kizan
NK08-020	261.88	526160	6680094	396	-85	350	2008	28/02/2008 0:00	04/03/2008 0:00	Main Zone Central
NK08-021	289.26	525955	6680088	396	-85	350	2008	05/03/2008 0:00	16/03/2008 0:00	Main Zone Southwest
NK08-022	326.05	525774	6679999	396	-85	324	2008	17/03/2008 0:00	26/03/2008 0:00	Main Zone Southwest
NK08-023	363.7	525470	6679768	396	-85	324	2008	27/03/2008 0:00	08/04/2008 0:00	Main Zone Southwest
NK08-024	209.7	525381	6679553	396	-85	288	2008	08/04/2008 0:00	16/04/2008 0:00	Main Zone Southwest
NK08-025	260.54	526008	6680110	396	-85	350	2008	13/04/2008 0:00	16/04/2008 0:00	Main Zone Central
NK08-026	185.32	525527	6679837	396	-85	324	2008	17/04/2008 0:00	23/04/2008 0:00	Main Zone Southwest
NK08-027	326	525871	6680032	396	-85	324	2008	18/04/2008 0:00	20/04/2008 0:00	Main Zone Southwest
NK08-028	313	525871	6680032	396	-79	324	2008	21/04/2008 0:00	24/04/2008 0:00	Main Zone Southwest
NK08-029	312.27	525871	6680032	396	-70	324	2008	24/04/2008 0:00	27/04/2008 0:00	Main Zone Southwest
NK08-030	244.45	525420	6679653	396	-85	288	2008	24/04/2008 0:00	28/04/2008 0:00	Main Zone Southwest
NK08-031	250	526714	6676870	396	-85	326	2008	28/04/2008 0:00	01/05/2008 0:00	Ring Grid
NK08-032	121.5	525137	6679557	396	-85	270	2008	27/04/2008 0:00	30/04/2008 0:00	Main Zone Southwest
NK08-033	57.69	525993	6680130	396	-85	350	2008	30/04/2008 0:00	01/05/2008 0:00	Main Zone Central
NK08-033A	289	525993.58	6680132.94	396	-86	350	2008	01/05/2008 0:00	03/05/2008 0:00	Main Zone Central
NK08-034	25.3	525774	6680004	396	-85	324	2008	01/05/2008 0:00	03/05/2008 0:00	Main Zone Southwest
NK08-034B	149.81	525773	6680007	396	-84	324	2008	04/05/2008 0:00	06/05/2008 0:00	Main Zone Southwest
NK08-035	212.15	526281	6680139	403	-81	350	2008	03/05/2008 0:00	05/05/2008 0:00	Main Zone Central
NK08-036	200.2	526281	6680139	403	-65	350	2008	05/05/2008 0:00	07/05/2008 0:00	Main Zone Central
NK08-037	230.38	526282	6680136	402	-83	170	2008	07/05/2008 0:00	09/05/2008 0:00	Main Zone Central
NK08-038	279	526020	6680033	396	-85	350	2008	09/05/2008 0:00	11/05/2008 0:00	Main Zone Central
NK08-039	247	526072	6680107	398	-77	350	2008	12/05/2008 0:00	14/05/2008 0:00	Main Zone Central
NK08-040	263	526072	6680107	398	-65	350	2008	14/05/2008 0:00	16/05/2008 0:00	Main Zone Central
NK08-041	274.88	526071	6680106	398	-88	350	2008	16/05/2008 0:00	18/05/2008 0:00	Main Zone Central
NK08-042	278.2	526074	6680103	398	-81	170	2008	18/05/2008 0:00	20/05/2008 0:00	Main Zone Central
NK08-043	237.02	526133	6680176	398	-81	350	2008	21/05/2008 0:00	23/05/2008 0:00	Main Zone Central
NK08-044	141.2	522841	6677634	397	-85	155	2008	23/05/2008 0:00	24/05/2008 0:00	Channel
NK08-045	390	525367	6679149	398	-76	50	2008	24/05/2008 0:00	28/05/2008 0:00	Main Zone Southwest
NK08-046	138	526700	6677226	403	-85	312	2008	28/05/2008 0:00	29/05/2008 0:00	South Ring
NK08-047	120	526914	6676939	396	-85	326	2008	30/05/2008 0:00	01/06/2008 0:00	Ring Grid
NK08-048	111	526652	6677150	396	-85	326	2008	31/05/2008 0:00	01/06/2008 0:00	South Ring
NK08-049	99.02	527653	6679901	417	-85	234	2008	01/06/2008 0:00	02/06/2008 0:00	Koona
NK08-050	75	527724	6679821	416	-85	234	2008	02/06/2008 0:00	03/06/2008 0:00	Koona
NK08-051	201.37	526358	6680145	401	-85	350	2008	03/06/2008 0:00	04/06/2008 0:00	Main Zone Central
NK08-052	197	526358	6680145	401	-60	350	2008	04/06/2008 0:00	06/06/2008 0:00	Main Zone Central
NK08-053	45.04	527478	6680139	428	-85	234	2008	06/06/2008 0:00	06/06/2008 0:00	Koona
NK08-054	150	526991	6679908	399	-64	234	2008	06/06/2008 0:00	08/06/2008 0:00	Main Zone Southeast
NK08-055	172	527686	6677387	417	-85	326	2008	08/06/2008 0:00	09/06/2008 0:00	Ring Grid
NK08-056	101	527672	6678059	429	-85	144	2008	09/06/2008 0:00	10/06/2008 0:00	Ring Grid
NK08-057	195.05	526636	6680100	421	-85	10	2008	10/06/2008 0:00	11/06/2008 0:00	Main Zone Southeast
NK08-058	238.86	523709	6679547	398	-85	125	2008	12/06/2008 0:00	14/06/2008 0:00	Joe Island North
NK08-059	11.72	522853	6677655	398	-85	155	2008	14/06/2008 0:00	14/06/2008 0:00	Channel
NK08-059A	55.68	522855	6677656	398	-85	155	2008	14/06/2008 0:00	15/06/2008 0:00	Channel
NK08-060	249.05	526302	6680175	403	-85	350	2008	15/06/2008 0:00	17/06/2008 0:00	Main Zone Central
NK08-061	174	526405	6680142	401	-85	350	2008	17/06/2008 0:00	18/06/2008 0:00	Main Zone Central
NK08-062	195	526405	6680142	401	-72	350	2008	19/06/2008 0:00	20/06/2008 0:00	Main Zone Central
NK08-063	207	526295	6680033	402	-85	350	2008	20/06/2008 0:00	22/06/2008 0:00	Main Zone Central
Total	13,480.92									

Drill collar locations were located using local grid coordinates. A company geologist would position a collar picket, along with a combination of front and/or back pickets to define the planned azimuth of the drill hole. All collar locations were re-measured using a non-differential hand held GPS unit as soon as the drill was removed. Typical accuracy of the GPS units is approximately ±3 m to 5 m. Drill holes were surveyed using, a Reflex downhole survey instrument to measure azimuth (magnetic north) and dip readings at regular intervals down the hole (typically every 75 m to 100 m) as well as at the top of the drill hole, six to nine metres below the end of the drill casing. Upon completion of all lake based drill holes, the top 50 m to 100 m of each hole was filled with cement (grout).

Almost 80% of the Strongbow drilling has been focused on the Main Zone. All drill holes were oriented with generally north-trending azimuths. As a result, the respective intercepts of mineralization in most of the holes are very close to true thickness (75% to 90% of true thickness). Although mineralization in the Main Zone is continuous along strike, due to the arcuate nature of the Main Zone Norite for descriptive purposes the Main Zone has been broken into three segments based on the general attitude of the sills: the Southeast Main Zone, Central Main Zone, and Southwest Main Zone (Figure 7-4). For the Central Main Zone, grid north (the most common azimuth for the drill holes) is 350° while in the Southwest Main Zone, the typical drill hole azimuth is 326°. In Southeast Main Zone, the typical drill hole azimuth used is 234° or 54°, depending on accessibility issues.

The balance of Strongbow's drilling occurred on five other target areas: Koona, Ring, Kizan, South Kizan, and Joe Island.

11.1 NICKEL KING MAIN ZONE

The Nickel King Main Zone deposit has been drill-defined over a strike length of approximately 2,600 m. Generally, the mineralization is at or close to surface in the Central and Southeast Main Zone and deepens gradually to the west along the arcuate trend of the Main Zone Norite. Typical mineralization consists of broad zones of disseminated sulphides ranging from 20 m to 60 m in thickness, with a higher-grade core, 2 m to 14 m thick consisting of net-textured to rare, semi-massive sulphides. In both sills, the disseminated sulphides form sheet-like zones that are interpreted to be sub-parallel to the bounding intrusive contacts (Figure 9-3). Higher grade, net textured zones are more continuous and consistent in the Lower Sill compared to the Upper Sill where net-textured sulphides form irregularly developed, discontinuous 1 m to 2 m thick pockets within the broader zones of disseminated mineralization. The majority of the drill holes have tested the highest conductivity targets defined by ground electromagnetic surveys and have not been designed to test the up-dip (northern) and down-dip (southern) extent of mineralization. Four weakly mineralized drill holes (NK08-37, NK08-20, NK08-42 and NK08-38, east to west) tested the down dip (southern) extent of mineralization in the Central Main Zone and appear to truncate the mineralization. Two historic drill holes (DDH-10238 and DDH-10236, east to west) test the up-dip (northern) extent of the Main Zone and indicate that mineralization in the Lower Sill does continue to the north, albeit with a reduced thickness (Figure 9-1). Mineralization is generally hosted within the top portion of the Lower Sill, although in detail higher-grade mineralization varies in depth within the sill and is observed to change from section to section. A summary of results from a select number of drill holes along the entire length of the Main Zone (presented from east to west) is provided in Table 11-3.

Drilling has helped refine the model that interprets the Upper and Lower Sills of the Main Zone Norite as south-dipping limbs of an overturned recumbent fold. The geometry of the recumbent fold is interpreted to extend along the entire strike extent of the deposit; however, the full down-dip extent of the mineralization in each sill has not been extensively tested and the fold model is only inferred. Drill holes AN95-06 and AN95-07 provide an example of evidence supporting the fold model (Figure 11-1). These holes were drilled from the same collar site and AN95-06, the shallower dipping hole, intersected the Upper and Lower Sills whereas AN95-07 encountered only a single sill, interpreted to be in the fold hinge where the Upper and Lower Sill converge (Figure 11-1). Conclusive evidence for a fold closure in the Southwest Main Zone and Southeast Main Zone has not been obtained by drilling.

The north-south trending fault mapped in the Southeast Main Zone is interpreted to be a steeply dipping structure (Figure 7-4). A component of dip-slip motion has been inferred based on the absence of Upper Sill in drill holes located on the east side of this fault. The sills immediately west of this fault are typically 10 m to 80 m thick in drill core and are interpreted to dip shallowly to moderately to the southwest.

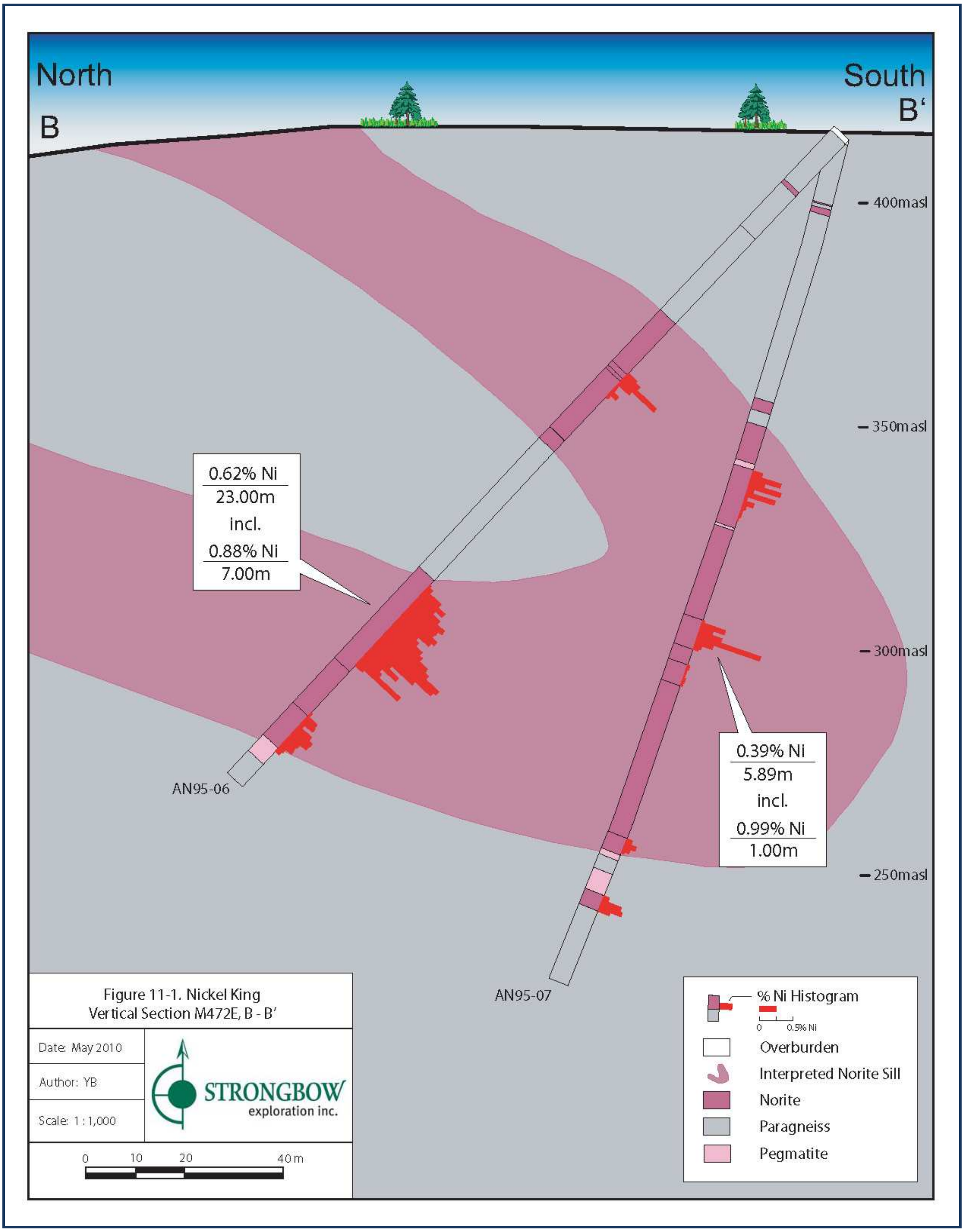
Table 11-3: Assay Results from the Main Zone, Displayed East to West

DDH Name		From	To	Length (m)	0.1% Ni Cut-off			0.5% Ni Cut-off			1.0% Ni Cut-off			Sill
					Ni (%)	Cu (%)	Co (%)	Ni (%)	Cu (%)	Co (%)	Ni (%)	Cu (%)	Co (%)	
Southeast Main Zone														
NK07-15		46.00	81.15	35.15	**0.22**	**0.04**	**0.01**	-	-	-	-	-	-	Lower
	including	54.50	57.15	2.65	**0.63**	**0.14**	**0.03**	-	-	-	-	-	-	Lower
NK08-54		65.05	74.35	9.30	**0.62**	**0.14**	**0.03**	-	-	-	-	-	-	Lower
	including	69.55	71.95	2.40	-	-	-	-	-	-	**1.02**	**0.23**	**0.04**	Lower
		85.20	92.55	7.35	**0.28**	**0.06**	**0.01**	-	-	-	-	-	-	Lower
NK07-14		55.25	59.70	4.45	**0.26**	**0.06**	**0.02**	-	-	-	-	-	-	Upper
		62.01	67.32	5.31	**0.42**	**0.12**	**0.03**	-	-	-	-	-	-	Upper
	including	66.81	67.32	0.51	-	-	-	**0.77**	**0.25**	**0.05**	-	-	-	Upper
NK08-57		33.60	37.50	3.90	**0.36**	**0.07**	**0.01**	-	-	-	-	-	-	Upper
		107.65	120.34	12.69	**0.51**	**0.12**	**0.02**	-	-	-	-	-	-	Lower
	including	117.05	120.34	3.29	**0.85**	**0.16**	**0.04**	-	-	-	-	-	-	Lower
	including	119.54	120.34	0.80	-	-	-	-	-	-	**2.20**	**0.32**	**0.09**	Lower
AN95-06		73.50	77.00	3.50	**0.36**	**0.08**		-	-	-	-	-	-	Upper
		141.48	164.48	23.00	**0.62**	**0.15**		-	-	-	-	-	-	Lower
	including	145.48	159.48	14.00	-	-	-	**0.72**	**0.18**	-	-	-	-	Lower
	including	156.48	158.48	2.00	-	-	-	-	-	-	**1.16**	**0.29**		Lower
Central Main Zone														
NK08-60		3.14	6.00	2.86	**0.22**	**0.05**	**0.01**	-	-	-	-	-	-	Upper
		25.50	29.50	4.00	**0.17**	**0.04**	**0.01**	-	-	-	-	-	-	Upper
		106.89	151.00	44.11	**0.66**	**0.15**	**0.03**	-	-	-	-	-	-	Lower
	including	129.00	147.00	18.00	-	-	-	**0.90**	**0.21**	**0.04**	-	-	-	Lower
	including	133.00	139.00	6.00	-	-	-	-	-	-	**1.31**	**0.26**	**0.06**	Lower
NK08-39		1.78	29.50	27.72	**0.32**	**0.07**	**0.02**	-	-	-	-	-	-	Upper
		38.00	59.00	21.00	**0.16**	**0.04**	**0.01**	-	-	-	-	-	-	Upper
		175.50	231.00	55.50	**0.53**	**0.13**	**0.02**	-	-	-	-	-	-	Lower
	including	177.50	190.00	12.50	-	-	-	**1.07**	**0.26**	**0.04**	-	-	-	Lower
	including	183.50	188.00	4.50	-	-	-	-	-	-	**1.59**	**0.35**	**0.06**	Lower
NK07-21		40.50	54.90	14.40	**0.19**	**0.04**	**0.01**	-	-	-	-	-	-	Upper
		203.90	249.25	45.35	**0.46**	**0.11**	**0.02**	-	-	-	-	-	-	Lower
	including	205.70	209.20	3.50	-	-	-	**0.78**	**0.18**	**0.03**	-	-	-	Lower
	including	206.30	207.20	0.90	-	-	-	-	-	-	**1.22**	**0.18**	**0.04**	Lower
	and including	212.75	214.00	1.25	-	-	-	-	-	-	**1.05**	**0.28**	**0.03**	Lower

… table continues

DDH Name		From	To	Length (m)	0.1% Ni Cut-off			0.5% Ni Cut-off			1.0% Ni Cut-off			Sill
					Ni (%)	Cu (%)	Co (%)	Ni (%)	Cu (%)	Co (%)	Ni (%)	Cu (%)	Co (%)	
Southwest Main Zone														
NK08-22		63.00	113.48	50.48	**0.37**	**0.09**	**0.02**	-	-	-	-	-	-	Upper
	including	63.65	73.09	9.44	-	-	-	**0.83**	**0.26**	**0.07**	-	-	-	Lower
		231.00	275.00	44.00	**0.64**	**0.16**	**0.03**	-	-	-	-	-	-	Lower
	including	235.00	244.95	9.95				**0.97**	**0.26**	**0.04**	-	-	-	Lower
NK08-23		91.00	154.70	63.70	**0.48**	**0.13**	**0.03**	-	-	-	-	-	-	Upper
	including	92.80	100.60	7.80	-	-	-	**0.92**	**0.23**	**0.06**	-	-	-	
	including	98.60	100.60	2.00	-	-	-	-	-	-	**1.23**	**0.19**	**0.08**	
		111.70	125.20	13.50	**0.85**	**0.24**	**0.05**	-	-	-	-	-	-	Upper
	including	119.70	125.20	5.50	-	-	-	-	-	-	**1.06**	**0.29**	**0.07**	
		317.70	326.70	9.00	**0.42**	**0.09**	**0.01**	-	-	-	-	-	-	Lower
NK08-30		107.37	120.92	13.55	**0.70**	**0.21**	**0.04**	-	-	-	-	-	-	Upper
	including	112.16	120.92	8.76	-	-	-	-	-	-	**1.01**	**0.31**	**0.07**	Upper
		132.00	157.00	25.00	**0.63**	**0.18**	**0.04**	-	-	-	-	-	-	Upper
NK08-24		153.33	198.75	45.42	**0.51**	**0.15**	**0.03**	-	-	-	-	-	-	Upper
	including	159.95	165.50	5.55	-	-	-	**0.89**	**0.32**	**0.06**	-	-	-	Upper
	and including	172.40	185.75	13.35	-	-	-	**0.91**	**0.28**	**0.06**	-	-	-	Upper
NK08-45		160.50	206.15	45.65	**0.40**	**0.08**	**0.01**	-	-	-	-	-	-	Upper
	including	196.35	200.50	4.15	-	-	-	**0.62**	**0.14**	**0.02**	-	-	-	

Figure 11-1: Vertical Section M472, B – B"



11.2 Koona Zone

The Koona Zone is located 500 m to the northeast of the Southeast Main Zone, along a sub-parallel trend (Figure 7-2). Mineralization is hosted within a norite sill that has been intersected in two historic holes and four Strongbow holes. Thicknesses of the norite in the Strongbow drill holes range up to 35 m but this is an apparent thickness and the exact sill attitude and therefore the true thickness is not known. The two historic holes did not intersect mineralization because they drilled underneath the sulphide-bearing zone, a relationship that places constraints on the "down-dip to the southwest" extent of mineralization. Table 11-4 shows a summary of the assay results.

Table 11-4: Assay Results from Four Drill Holes from Koona Zone

DDH			Length	0.1% Ni Cut-off			0.5% Ni Cut-off			1.0% Ni Cut-off		
Hole	From	To	(m)	Ni (%)	Cu (%)	Co (%)	Ni (%)	Cu (%)	Co (%)	Ni (%)	Cu (%)	Co (%)
NK08-53	23.00	27.73	4.73	0.83	0.17	0.026	-	-	-	-	-	-
Including	25.89	27.73	1.84	-	-	-	-	-	-	1.89	0.37	0.059
NK08-49	31.16	32.30	1.14	-	-	-	-	-	-	1.74	0.31	0.066
NK07-13	26.60	27.85	1.25	0.55	0.07	0.023	-	-	-	-	-	-
Including	26.60	27.08	0.48	-	-	-	-	-	-	1.26	0.11	0.048
NK08-50	40.00	46.00	6.00	0.83	0.20	0.032	-	-	-	-	-	-
Including	41.05	45.40	4.35	1.02	0.24	0.039	-	-	-	-	-	-
Including	41.05	42.44	1.39	-	-	-	-	-	-	1.53	0.35	0.059

To date, this zone has been tested over a strike length of 400 m and magnetic, as well as EM data, indicate that the zone could be up to 650 m long. It is unknown how the norite sill hosting the Koona Zone relates to the Upper and Lower Sills of the Main Zone.

11.3 Ring Area

The Ring area is situated 2 km to 3.5 km south of the Main Zone and is part of a ring-shaped structure 1.0 km in diameter, defined by airborne magnetic anomalies (Figure 7-2 and Figure 10-1). Six coincident magnetic and electromagnetic conductors have been drill tested within this circular structure; however, the South Ring Zone is the only area in which significant nickel, copper and cobalt mineralization has been encountered. The remaining targets were explained by thin, less than 5 m thick dyklets of norite with variable concentrations of pyrrhotite and graphite occurring along the intrusive contacts and within the adjacent paragneiss (drill holes NK07-17, NK08-31, NK08-47, NK08-56 and NK08-57).

Three drill holes testing the South Ring Zone (Figure 7-2) have defined mineralization hosted within a differentiated norite/peridotite intrusion, interpreted to be a small, plug-shaped body. A number of stacked, disseminated sulphide zones were intersected and range in thickness from 3 m to 30 m. Table 11-5 shows a summary of the intercepts from these holes.

Table 11-5: Assay Results for Three Drill Holes from the South Ring Zone

DDH Hole	From	To	Length (m)	0.1% Ni Cut-off			0.5% Ni Cut-off			1.0% Ni Cut-off		
				Ni (%)	Cu (%)	Co (%)	Ni (%)	Cu (%)	Co (%)	Ni (%)	Cu (%)	Co (%)
NK07-16	6.00	27.16	21.16	0.47	0.27	0.025	-	-	-	-	-	-
including	24.80	27.16	2.36	0.95	0.37	0.043	-	-	-	-	-	-
including	14.10	17.90	3.80	-	-	-	0.73	0.47	0.040	-	-	-
and including	24.80	25.20	0.40	-	-	-	-	-	-	2.30	0.31	0.102
and including	26.48	27.16	0.68	-	-	-	-	-	-	1.24	0.66	0.055
NK08-46	30.50	44.52	14.02	0.50	0.37	0.024	-	-	-	-	-	-
including	37.50	41.50	4.00	-	-	-	0.87	0.70	0.041	-	-	-
NK08-48	9.50	30.08	20.58	0.45	0.28	0.024	-	-	-	-	-	-
including	19.00	30.08	11.08	0.57	0.37	0.030	-	-	-	-	-	-
including	28.00	30.08	2.08	-	-	-	-	-	-	1.06	0.64	0.055
	51.51	81.50	29.99	0.22	0.14	0.016	-	-	-	-	-	-
including	78.32	80.89	2.57	-	-	-	0.64	0.58	0.031	-	-	-

Drill holes NK07-16 and NK08-46 were collared within 20 m of each other and NK08-48 was located to the southwest of drill hole 16 and establishes at least 100 m of strike length to the zone. The South Ring Zone represents a new discovery in the Nickel King project area and confirms the potential for additional mineralized norite sills or plugs on the property.

11.4 KIZAN AREA

A single drill hole (NK07-18) positioned three kilometres southwest of the Main Zone was completed in the summer of 2007 (Figure 7-2). Minor sulphides comprising pyrite and pyrrhotite, locally with minor disseminated graphite, were intersected within extensive thicknesses (up to 44 m) of norite. No significant assays were returned; however, the minor occurrences of graphite are not considered sufficient to explain the magnitude of the original EM conductor. This area remains a high potential target because of the strength of the unexplained geophysical conductor that is coincident with a magnetic anomaly and a norite intrusion. A moderate borehole conductor was documented off-hole, which further enhances the prospectivity of this area.

11.5 SOUTH KIZAN AREA

A single drill hole positioned 5 km southwest of the Main Zone was completed in the summer of 2007 (Figure 7-2). Similar to the Kizan area, a concentration of Fugro Megatem anomalies were evaluated by ground geophysics and drill tested. Drill hole NK07-19 encountered several thin intervals of norite having a maximum thickness of 7 m. Minor sulphides comprising pyrite and pyrrhotite with disseminated graphite were intersected, primarily within the paragneiss country rock. A 0.95 m thick interval of 25% sulphides was intersected within paragneiss, at the base of a 2 m thick norite sill. This interval returned 399 ppm Ni and 638 ppm Cu and indicate some remobilization of mineralization from off hole within one of the norite sills. Borehole geophysics was completed and a very encouraging off-hole electromagnetic anomaly (located within 20 m of the hole at a depth of 85 m) was identified and is recommended for follow-up.

11.6 JOE ISLAND TREND

The Joe Island trend is situated three to 4 km west of the Main Zone and consists of a discontinuous trend of sulphide mineralization within an extensive norite intrusion (Figure 7-2). The trend is divided into the Channel, Joe Island, and Joe Island North areas and has been tested by six Strongbow, and three historic drill holes. Drilling in both 2007 and 2008 returned disappointing results due to consistently low sulphide metal contents. Table 11-6 shows a summary of the results from two of the three areas presented from south to north (NK08-58 at the Joe Island North target returned no significant results).

Table 11-6: Assay Results for the Joe Island Trend

DDH Hole	From	To	Length (m)	0.1% Ni Cut-off			0.5% Ni Cut-off		
				Ni	Cu	Co	Ni	Cu	Co
NK08-44	8.34	17.00	8.66	0.27	0.10	0.037	-	-	-
Channel	-	-	-	-	-	-	-	-	-
NK08-59	7.01	9.50	2.49	0.33	0.12	0.049	-	-	-
Channel	-	-	-	-	-	-	-	-	-
NK08-59A	5.50	8.01	2.51	0.27	0.13	0.040	-	-	-
Channel	-	-	-	-	-	-	-	-	-
NK07-10	86.20	87.95	1.75	0.42	0.24	0.085	-	-	-
Joe Island	-	-	-	-	-	-	-	-	-
NK07-08	48.00	52.37	4.37	0.35	0.15	0.069	-	-	-
Joe Island	-	-	-	-	-	-	-	-	-
	53.74	58.23	4.49	0.27	0.12	0.054	-	-	-
	54.48	55.00	0.52				0.55	0.14	0.107
NK07-09	39.00	42.42	3.42	0.25	0.14	0.050	-	-	-
Joe Island	-	-	-	-	-	-	-	-	-
	43.85	45.50	1.65	0.26	0.13	0.052	-	-	-

Sulphide metal contents of the Channel (~2% Ni in 100% S) and Joe Island (~0.8% Ni in 100% S) are low compared to the Main Zone. Typically, the zones are 1.5 m to 4 m thick but range up to 8.5 m in NK08-44. The true thicknesses are unknown due to the limited amount of drilling. It is interesting to note that the area has the highest copper and cobalt values, as a proportion of the nickel values. The Ni:Cu ratio is typically 2.5:1 at Joe Island compared to the Main Zone which is approximately 4 to 5:1 range. The Ni:Co ratio at Joe Island is typically 5 to 7: 1 compared to the Main Zone which ranges between 15 and 25:1.

As stated earlier the Koona Zone, Ring Area, Kizan Area, South Kizan Area, Joe Island trend are not part of the resource model described in Section 17 of this report.

12.0 SAMPLING METHOD AND APPROACH

Strongbow adheres to a rigorous and detailed set of protocols for all samples collected during the Nickel King exploration programs. Drill core is transported directly from the drill site using a helicopter or ski-doo, at which time a basic geotechnical assessment is completed, followed by the core logging process. Once all samples are marked out by a geologist, half or quarter core is collected using a pneumatic splitter or a rock saw. Samples are sealed in bags, tabulated, and prepared for shipment. The chain of custody for the samples is monitored along the entire route from field site to analytical laboratory. The details of this process are as follows.

Geotechnical logging was conducted for all drill holes of the 2007 and 2008 drilling programs. Drill core in each box is reassembled and measured to ensure the accuracy of the run marker placements and note any significant core loss. A metal tag recording drill hole number, box number, and drill core intervals is stapled to the end of each box. Geotechnical data, including core recovery and magnetic susceptibility measurements, is recorded in an excel spreadsheet using a laptop computer. In general, core recovery from the Main Zone was excellent, approaching 100%. Geological logging follows geotechnical logging and involves the description of lithology, textures, structure, alteration, and mineralization.

All mineralization is hosted within norite, a variably fine to coarse-grained mafic intrusive rock that is typically massive and poorly foliated. This contrasts with the strongly foliated paragneiss country rocks. The paragneisses typically have a well-defined layering and foliation that is quite heterogeneous (alternating bands of biotite, muscovite, quartz, feldspar, and garnet) in composition. Identification and determination of individual sample intervals within the norite is based on visual characteristics of the rock, including geological boundaries and contacts, and type, intensity and changes in alteration or mineralization. Sample intervals vary from a minimum of 30 cm to a maximum of 150 cm and consist of one half split of NQ-2 core, except duplicate pairs, each of which comprises a quarter core split. Typically, continuous sampling is completed over intervals with >1% to 3% combined sulphide minerals (pyrrhotite, chalcopyrite ± pentlandite). In addition, “shoulder” samples of unmineralized rock are collected to bracket mineralized zones and to accurately define the full extent of the respective zone. Within the Upper and Lower Sills of the Main Zone, a typical drill hole will encounter a broad 30 m to 60 m thick zone of disseminated sulphides with a central 2 m to 14 m wide zone of net-textured to rare semi-massive sulphides. Prior to splitting, drill core is fitted together and rotated to represent symmetrical or representative halves in order to maintain sampling consistency. A continuous line is drawn on the top of the core to indicate where splitting should occur and a unique sample number, matching the assigned sample tags, is marked within the interval.

All drill core is digitally photographed before splitting.

During the 2008 drill program, samples were split using a manual core-splitter or a rock saw. In general, mineralized zones were sawn including all cores with greater than 5% sulphides. During the 2007 drilling programs, all cores was split using a manual core-splitter. Between each sample, the core-splitter was cleaned with a 3" paintbrush along the surface of the blades, as well as the surrounding area of the splitter and the stainless steel collection pan. The rock saw was cleaned between drill holes by spraying water over the work area and removing any rock debris or clay retained in the collection pan below the rock saw. For each sample, one-half of the core and one sample tag are placed in a numbered poly ore sample bag and closed with a non-reusable plastic tie. The remaining half core is returned to the core box in the correct orientation and refitted to ensure all pieces are present and located at the appropriate metre markers. A second tag bearing the sample number is stapled into the core box at the end of the sample interval providing clear record of each sample location within the core box.

Company geologists supervise the collection of all samples. Individual samples are catalogued and placed in groups of five to twenty samples into rice bags for shipment. Information on each sample shipment, including total number of rice bags, individual sample numbers within each rice bag and requested analytical methods are documented on a shipping form. One copy of the shipping form is placed in the top of the first rice bag of each shipment. A second copy is scanned and emailed to Strongbow's head office and the original filed for reference in the field office. Sample shipments are transported from Nickel King by chartered aircraft to Stony Rapids. Each shipment is received by Strongbow's expediting personnel in Stony Rapids and shipped by commercial flights to Saskatoon and subsequent land transport to ACME Analytical Laboratories (ACME) in Vancouver. For the rare times when there is a wait time between samples arriving in Stony Rapids and subsequent shipment on an available commercial flight, Strongbow rents a locked storage unit that serves as temporary storage for all the samples until transport is arranged. It is the authors' opinion proper industry-standard techniques were used during collection, preparation, and transport of all samples.

Since no drills were currently active during the site visit, PEG did not have the possibility of reviewing the core logging and core handling procedures described in this section; however, the description above coincide well with the cursory examination of the core stored at the Strongbow's field camp. This is especially true in the area of core markings, splitting, QA/QC tags, storage, and logistics.

13.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

13.1 SAMPLE PREPARATION AND TECHNIQUES

All samples collected during the Nickel King program were shipped to ACME for initial analysis. ACME is currently registered with ISO 9001:2000 accreditation. Sample pulps from a small number (less than 1%) of the total samples were sent to either Global Discovery Laboratories or to ALS Chemex Laboratories (both in Vancouver, British Columbia, Canada) for independent umpire analysis. The umpire labs were also used for an assessment of standard materials. ALS Chemex is registered with ISO 9001:2000 certification and Global Discovery participates in a proficiency testing program for mineral analysis (PTP-MAL), which operates in full accord with ISO/IEC Guide 43-1.

Rock sample preparation at ACME involves crushing each sample to 70% passing a 10 mesh (2 mm) screen. A 250 g split is then pulverized to 95% passing a 150 mesh (100 µm) screen. Coarse rejects are stored at the laboratory and pulp rejects are securely stored at Strongbow's storage unit in Vancouver.

The 2007 and 2008 drilling campaigns utilized different assay techniques (Table 13-1). Group 1 techniques (e.g., 1EX, 1DX) are performed on all samples submitted for assay and have an upper detection limit of 10,000 ppm. The Group 1DX method is a 36-element ICP-MS package using aqua regia digestion and a 0.5 g pulp sample size. The Group 1EX method is a 41-element ICP-MS technique using four-acid (total) digestion and a 0.25 g pulp sample size. Group 7 techniques (e.g., 7TD, 7AR) provide greater accuracy at upper detection limits for the elements of interest. Group 7AR is a 21-element package analyzed by ICP-ES, whereas Group 7TD has a 22-element package using a four-acid (total) digestion, also using an ICP-ES analysis. The Group 7 techniques were requested for intervals that contained sulphides visually estimated to be 5% or greater (in 2007), or 10% or greater (in 2008). Additionally, all samples returning greater than 5,000 ppm Ni using 1EX or 1DX, were automatically triggered for Group 7 analysis. Group 3B, a fire assay using a 30 g sample split and an ICP-ES finish, was requested for all high sulphide content samples (i.e., >10% visual sulphides) and tested for gold, platinum, and palladium. In addition, some samples were selected for whole rock and trace elements analysis by ICP (Group 4A and 4B).

Table 13-1: Sampling Techniques (based on ACME Lab Coding) Applied to 2007 and 2008 Drilling Campaigns

Year	Drill Holes	Group 1	Group 7
2007	NK07-01	1DX, 1EX	7TD
	NK07-02	1DX	7AR, 7TD
	NK07-03	1DX,1EX	7AR, 7TD
	NK07-04 to 06	1EX	7AR, 7TD
	NK07-07 to 10	1EX	7TD
	NK07-11 to 13	1EX	7AR, 7TD
	NK07-14 to 15	1EX	7TD
	NK07-16	1EX	7AR, 7TD
	NK07-17 to 19	1DX, 1EX	7TD
2008	NK08-20 to 63	1DX	7AR

In 2007, some samples were submitted for dual Group 1 (1 EX and 1 DX) or 7 (7TD and 7AR) analyses in order to compare the analytical results for nickel, copper, and cobalt using the aqua regia and four acid (total) digestion methods. A comparison of results for the 1DX-1EX and 7AR-7TD techniques indicated a near 1:1 linear correlation for the elements of interest. Therefore, 1DX and 7AR (i.e., only aqua regia analysis) were adopted as the primary analytical techniques for the 2008 program. In addition, a petrographic and electron microprobe study of the Nickel King norite indicated that very minimal nickel is hosted in silicate minerals, which further supported the use of aqua regia digestion.

13.2 ASSAY QUALITY CONTROL

QA/QC for the 2007 and 2008 drilling programs consisted of inserting analytical standards, blanks and field duplicates at regular intervals into the sample stream sent to ACME (Table 13-2). For every 100 series of sample numbers, blanks were inserted every 25 samples (numbers ending in 15, 40, 65, and 90), pre-packaged nickel standards were inserted using two 10 g packets for every 25 samples (numbers ending in 01, 26, 51, 76), and field duplicates were collected every 50 samples (numbers ending in 10/11 and 60/61). Blanks were inserted to monitor for potential contamination during sample preparation and analysis. Analytical standards (OREAS 72A or 73A purchased from Analytical Solutions of Toronto Ontario) are intended to measure the precision and accuracy of ACME's analyses. Field duplicates were inserted as a measure of reproducibility and precision of data. Each of the QA/QC samples was treated as a regular rock sample and assigned a unique sample number and placed in a labelled 8" x 12" poly ore bag with the appropriate sample tag. In addition to Strongbow's QA/QC program, ACME also maintains an internal QA/QC program that involves the insertion of their own analytical standards, sample and preparatory blanks, and pulp and core reject duplicates.

Table 13-2: QA/QC Samples Submitted for 2007 and 2008 Drilling Activities

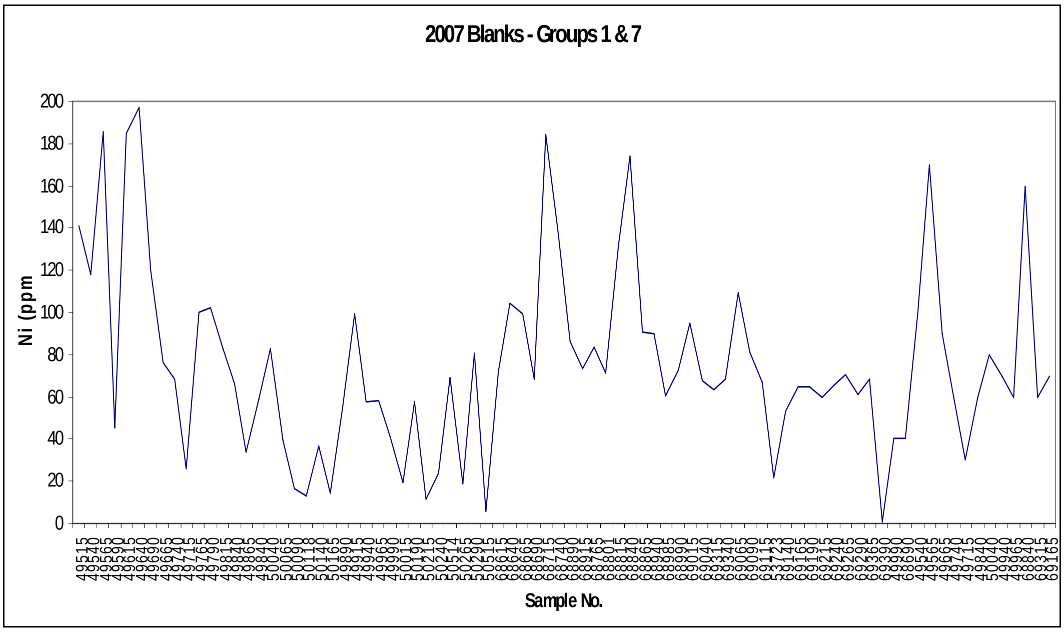
Sample Type	No. of Samples
2007 QA/QC Samples	
Blanks	
1EX	53
1DX	15
7TD	12
7AR	2
Analytical Standard OREAS 73A	10
1EX	7
1DX	25
7TD	1
7AR	
Analytical Standard OREAS 72A	
1EX	42
1DX	8
7TD	19
7AR	1
Duplicate Pairs	
1EX	25
1DX	4
7TD	13
7AR	1
2008 QA/QC Samples	
Blanks	
1DX	178
7AR	80
Analytical Standard OREAS 72A	
1DX	174
7AR	76
Duplicate Pairs	
1DX	87
7AR	29

ACME conducts Group 1 analyses in batches of thirty-four client (i.e., Strongbow) samples, referred to as an internal batch. In the event of a failure of one of Strongbow's inserted blanks or analytical standards, a rerun is requested for the entire internal sample batch associated with the failed blank or standard. For any requested reruns, one blank and analytical standard were resubmitted for each 34-sample internal batch. Acceptable results from the re-analysis of the pulps (in the case of a QA/QC failure) replaced the original failed samples in the database.

13.3 BLANKS

Blanks are inserted into the ACME sample stream to determine if contamination has occurred during sample preparation or analysis. In 2007, the inserted blanks consisted of four different materials, listed in order of use: (1) paragneiss sourced near the Main Zone, (2) sandstone sourced near the town of Stony Rapids, SK, (3) granite sourced from north of Thye Lake, and (4) silica chips purchased from a garden centre. Due to inconsistency in the composition of the blanks, the values for the elements of interest (nickel, copper, and cobalt) for Group 1 and Group 7 analytical methods are highly variable and therefore failure was not assigned to any given batch (Figure 13-1). Using 2008s failure criteria of 150 ppm Ni as a comparison (see below), and taking into account the variability of nickel results in 2007, the overall nickel values for the 2007 blank materials do not indicate significant contamination. Nickel values of 1 to 197.4 ppm (the range of results from the 2007 blank material analyses) represent a small percentage (<4%) of 5,000 ppm nickel, which, during 2007, was perceived as being the "significant" grade value for nickel mineralization in the Main Zone.

Figure 13-1: 2007 Blank Results for Nickel (ppm) using Group 1 and Group 7 Analyses



For 2008, blank material was comprised of silica rock chips purchased from a garden centre. During the initial part of the 2008 QA/QC program, 50 ppm Ni or 0.005% Ni was the first pass screen used to contamination for both the Group 1 and Group 7 methods. This value was selected as it approximately represents three times the background Nickel value (average of 15.6 ppm) reported for this blank material. The pass-fail value was later adjusted upwards to 150 ppm or 0.015% Ni as it represents a 3% value of 5,000 ppm Ni. Failure is typically associated with contamination from the preceding sample(s) within the batch, which, if identified, would trigger a re-analysis for the associated internal ACME batch. Blank reruns would also be requested if the Ni-standard material within the associated batches fails (see below).

Blanks submitted for both Group 1 and Group 7, or multiple Group 1 or Group 7 analyses returned comparable results for nickel, copper, and cobalt. Of all the blank samples submitted in 2008, 17 failed using the 50 ppm criteria. Subsequent reruns of these samples have been completed and pass the 150 ppm nickel criteria, with values ranging from background to 0.2 ppm to 106 ppm nickel (Figures 13-2 and 13-3).

During 2007, blank material was not sent to a secondary lab for check analysis. In 2008, as part of the umpire process, ten samples of blank material were analyzed at a secondary laboratory (ALS Chemex in Vancouver) using analytical method ME-ICP41, which is a nitric aqua regia digestion analyzed using ICP-AES for 35 elements on a prepared sample pulp of 0.5 g. This method was chosen because it was the analytical technique that was the most comparable to ACME's Group 7AR and it has a lower detection limit for nickel, copper, and cobalt of 1 ppm. All ten blank samples returned nickel, copper, and cobalt analyses near this lower detection limit. Figure 13-4 shows nickel results for these blank samples. As a comparison, most of the blank samples submitted to ACME during the 2008 program returned values in the 10 ppm to 50 ppm nickel range. However, it is the authors' opinion that differences on the 10 ppm to 50 ppm scale comprise a very small, and acceptable, percentage of the assigned significant nickel value of 5,000 ppm.

Figure 13-2: 2007 and 2008 Blank Results for Nickel (ppm) using Group 1 DX Analyses

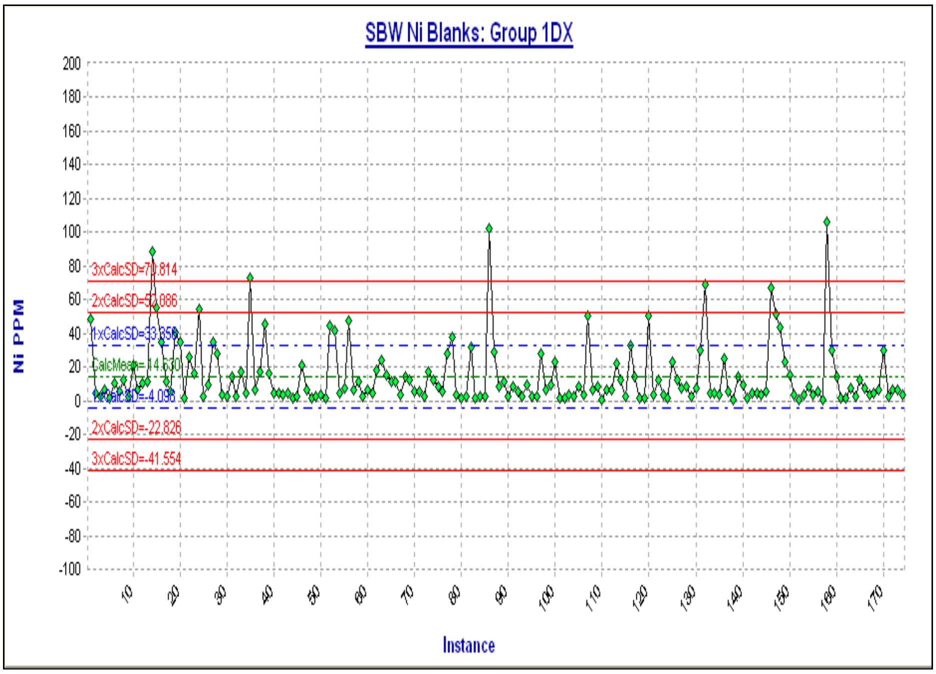


Figure 13-3: 2007 and 2008 Blank Results for Nickel (ppm) using Group 7AR Analyses

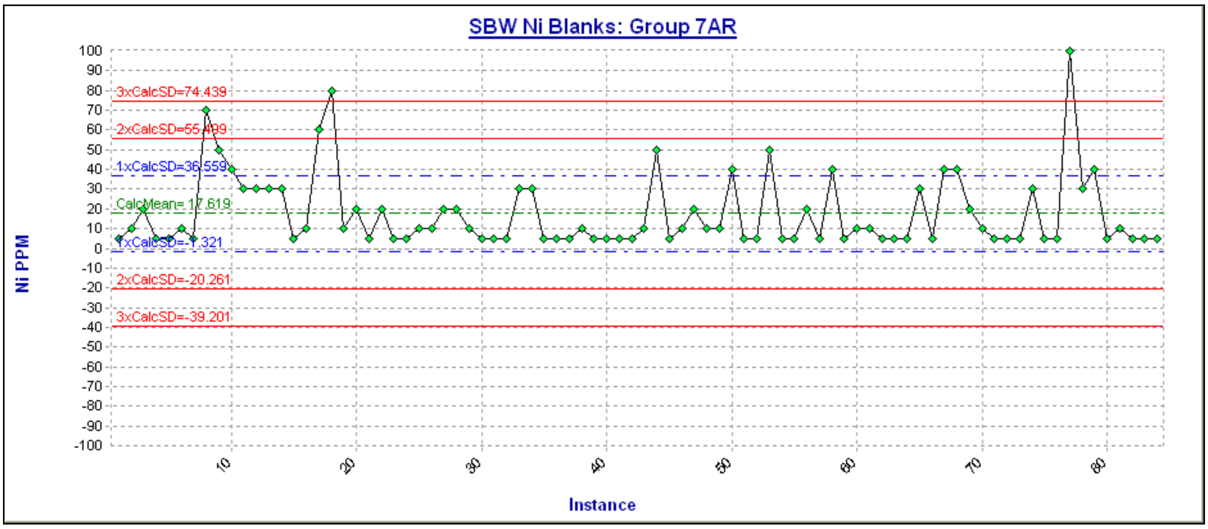
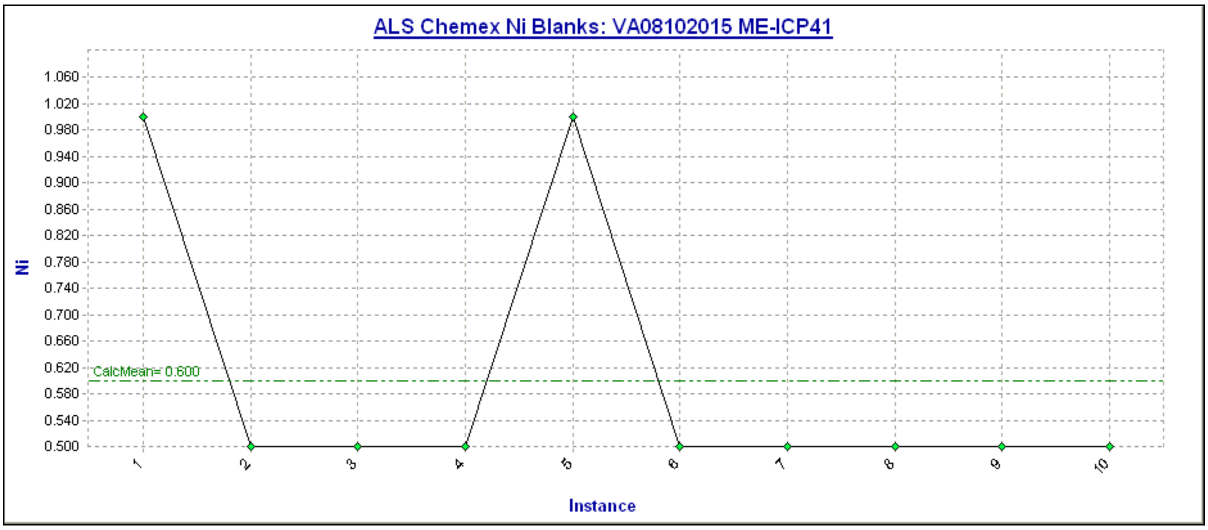


Figure 13-4: 2008 Blank Results for Nickel (ppm) using ME-ICP41 (ALS Chemex)



13.4 ANALYTICAL STANDARDS

Analytical standards OREAS 72A and 73A were purchased from Analytical Solutions Ltd. of Toronto, Ontario and used to assess laboratory precision and accuracy. The use of these standards throughout the Nickel King drilling programs also allows an evaluation of the degree of analytical precision over time. The reference materials comprise 10 g of homogenized rock powder packaged in laminated foil pouches. OREAS 72A is derived from a low grade nickel sulphide ore prepared from a blend of high grade nickel ore from the

Cosmos nickel mine, Western Australia, and barren alkali olivine basalt from Epping, Victoria, Australia. OREAS 73A is derived from a high grade nickel sulphide ore also prepared from the Cosmos nickel mine, Western Australia, and barren ultramafic material from the same mine. OREAS 72A was inserted into the sample stream during both the 2007 and 2008 drilling programs, whereas OREAS 73A was utilized only for 2007. In both cases, a four acid (total) digestion is recommended to achieve the expected values.

Table 13-3 shows the recommended mean values and upper and lower confidence values (for the third standard deviation) of OREAS 72A and 73A standard materials for nickel and copper. Equivalent values for both standard materials were calculated by ACME using Group 7 analyses and are also included in Table 13-3 for comparison. For the purposes of assessing the quality of data from the laboratory in 2007 and 2008, the recommended values provided by OREAS were used to pass or fail analytical results reported to Strongbow.

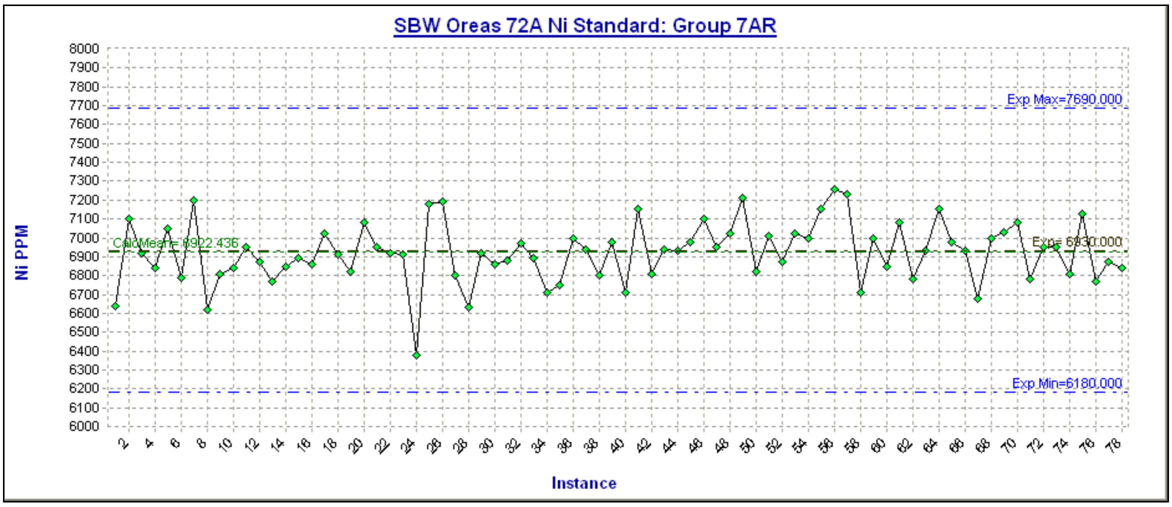
Table 13-3: Comparison of Ni, Cu, and Co Values in QA/QC Pulp Standards

Analytical Method	Mean Value (ppm) ± 3rd Standard Deviation	Lower Confidence Interval (ppm)	Upper Confidence Interval (ppm)
OREAS 72A Total digestion – Recommended Ni	6930 ± 760	6170	7690
OREAS 72A Total digestion – Recommended Cu	316 +43/-47	269	359
OREAS 72A Total digestion – Recommended Co	157 +36/-35	122	193
OREAS 72A ACME Group 7TD Calculated Ni	6681 ± 722	5959	7403
OREAS 72A ACME Group 7TD Calculated Cu	302 ± 34	268	336
OREAS 72A ACME Group 7TD Calculated Co	146 ± 32	114	178
OREAS 72A ACME Group 7AR Calculated Ni	6897 ± 622	6275	7519
OREAS 72A ACME Group 7AR Calculated Cu	243 ± 149	94	392
OREAS 73A Total digestion – Recommended Ni	14100 ± 700	13400	14800
OREAS 73A Total digestion – Recommended Cu	877 ± 74	803	951
OREAS 73A Total digestion – Recommended Co	286 ± 27	259	313
OREAS 73A ACME Group 7TD Calculated Ni	14093 ± 1305	12788	15398
OREAS 73A ACME Group 7TD Calculated Cu	864 ± 92	772	956
OREAS 73A ACME Group 7TD Calculated Co	282 ± 38	245	320

Group 1 analyses were not used to fail a batch because Group 1DX aqua regia does not completely digest OREAS 72A or OREAS 73A material and the expected value of OREAS 73A (14,100 ppm Ni) exceeds the detection limits of both Group 1DX and 1EX. If Group 7 analysis was requested for an OREAS 72A standard the Group 7 value would supersede the Group 1 value as a pass-fail criterion. Failure is assigned to Group 7 results for OREAS 72A and 73A if results fall outside of the established third standard deviation confidence levels, two consecutive standards in the same batch fall outside of the second standard deviation upper and lower margins, or the blank sample within its internal batch indicates contamination. Similar to blank reruns, the rerun results for standards would supersede the initial results if the standard and blanks passed the second evaluation.

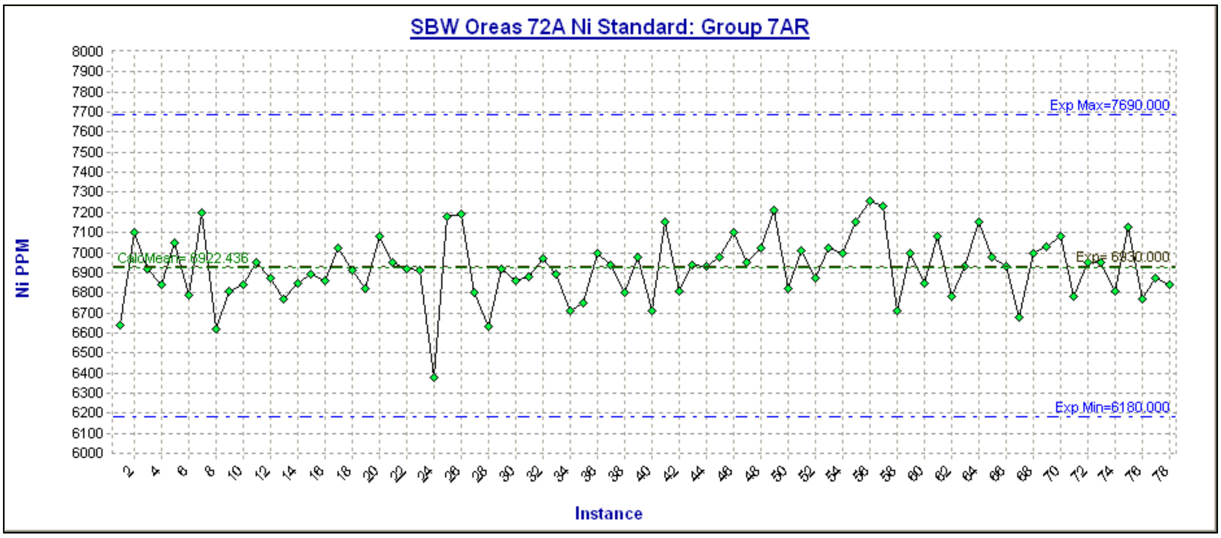
Figures 13-5 and 13-6 show the final 2007 nickel results for OREAS 72A and OREAS 73A standard using Group 7 analyses and Figure 13-7 shows the final 2008 nickel results for OREAS 72A standard. A single failure (13,260 ppm Ni) occurs for OREAS 73A analyzed with Group 7TD (Figure 13-7). The sample's failure was not rerun because there are five other standards in the same batch that fall within Strongbow's acceptable third standard deviation limits.

Figure 13-5: 2007 and 2008 Control Chart for OREAS 72A Nickel Values using Group 7AR Analyses



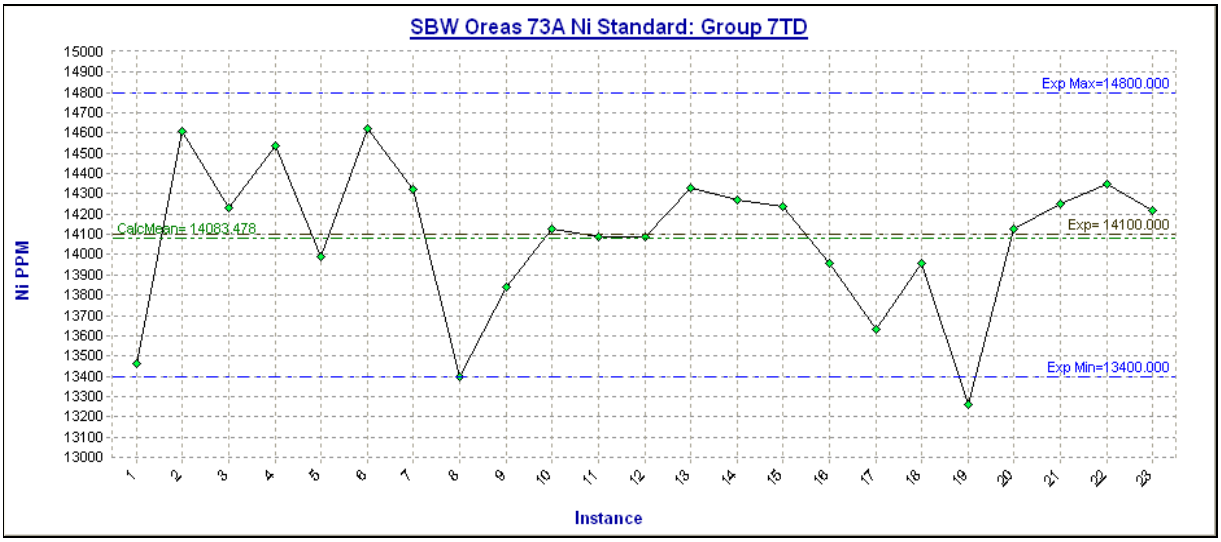
Note: The black line indicates the expected value as reported by OREAS; the blue maximum and minimum lines represent third standard deviation lines.

Figure 13-6: 2007 Control Chart for OREAS 72A Nickel Values using Group 7TD Analyses



Note: The black line indicates the expected value as reported by OREAS; the blue maximum and minimum lines represent third standard deviation lines.

Figure 13-7: 2007 Control Chart for OREAS 73A Nickel Values using Group 7TD Analyses



Note: The black line indicates the expected value as reported by OREAS; the blue maximum and minimum lines represent third standard deviation lines.

13.5 DUPLICATE SAMPLES

13.5.1 *FIELD DUPLICATES*

Field duplicate samples were collected in order to test the repeatability of analytical results. A total of 29 duplicate samples were submitted for Group 1 analyses and fourteen duplicates for Group 7 analyses in 2007. In 2008, 87 duplicate samples were submitted for Group 1DX analyses and 29 duplicate samples were submitted for Group 7AR analyses. Figures 13-8 and 13-9 show comparisons Group 1 (blue) and Group 7 (green) nickel analyses for 2007 and 2008 duplicate sample pairs, respectively. A 1:1 ratio line (solid black), a best-fit line (bold, solid red and black), and ± 10% regression lines (dotted red) are also shown. A very good correlation (R^2 values greater than 0.96) is shown between the duplicate samples suggesting that there is acceptable reproducibility between the duplicate pairs. A very minimal number of points occur outside of the ± 10% regression lines reflecting some sample heterogeneity.

13.5.2 *PREPARATORY DUPLICATES*

At regular intervals, ACME produces a second pulp from the remaining coarse reject material, which is submitted for analysis as a "preparatory duplicate." A total of 25 preparatory duplicate samples were submitted for Group 1DX and 1EX and five samples for Group 7AR and 7TD in 2007, and 109 preparatory duplicate samples were submitted for Group 1DX and 26 samples for Group 7AR in 2008. Regression plots of 2007 and 2008 original vs. preparatory duplicate pairs are shown in Figures 13-10 and 13-11. Bolded, solid red and black lines represent the best-fit linear trend line, solid black represents the 1:1 ratio line, and dotted red lines represent ±10% regression lines. Overall, preparatory duplicate pair results are interpreted to show a high degree of reproducibility (R^2 = 0.977 to 0.995). Only a small percentage of the overall points fall outside of the ±10% regression lines. Copper and cobalt comparisons show similar trends as nickel for Group 1 and Group 7 results.

13.5.3 *PULP DUPLICATES*

Pulp duplicates are samples that have been analyzed a second time using the original pulp material. A total of 49 pulp duplicates were processed for Group 1 and 20 pairs for Group 7 in 2007 and 44 pulp duplicates were processed for Group 1 and 39 pairs for Group 7 in 2008. Regression plots of 2007 and 2008 original vs. pulp duplicate pairs are shown in Figures 13-12 and 13-13. Bolded, solid red and black lines represent the best-fit linear trend line, solid black represents the 1:1 ratio line, and dotted red lines represent ±10% regression lines. In general, the pulp duplicates indicate good reproducibility with regression values averaging 0.998. Copper and cobalt trends compare well with nickel trends and indicate close to 1.0 regression values.

Figure 13-8: 2007 Nickel Assay Results of Original and Field Duplicate Pairs for Groups 1 and 7 Analyses

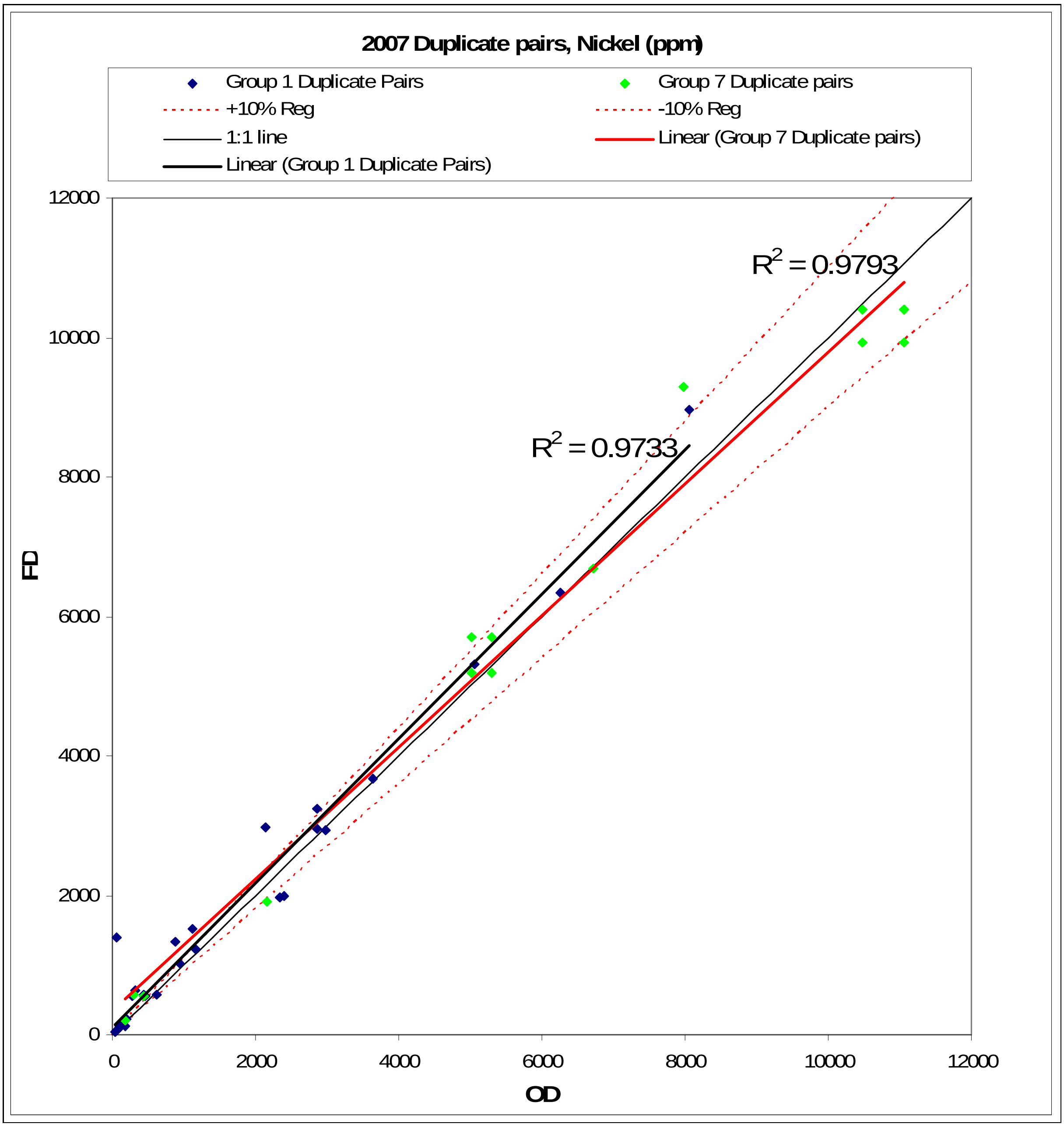


Figure 13-9: 2008 Nickel Assay Results of Original and Field Duplicate Pairs for Groups 1 and 7 Analyses

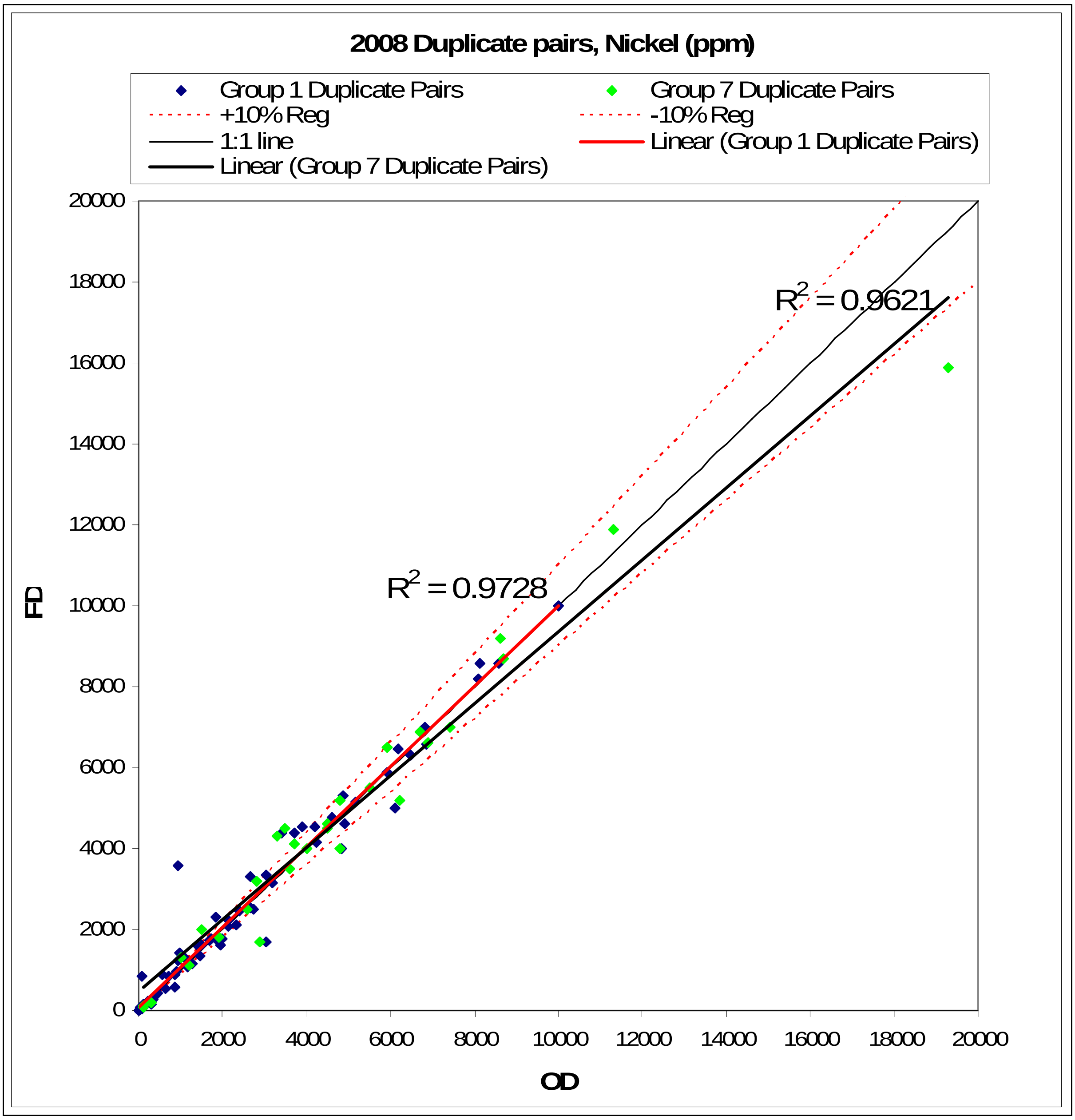


Figure 13-10: 2007 Regression Plot of Original vs. Preparatory Duplicate Pairs for Group 1 and 7 analyses

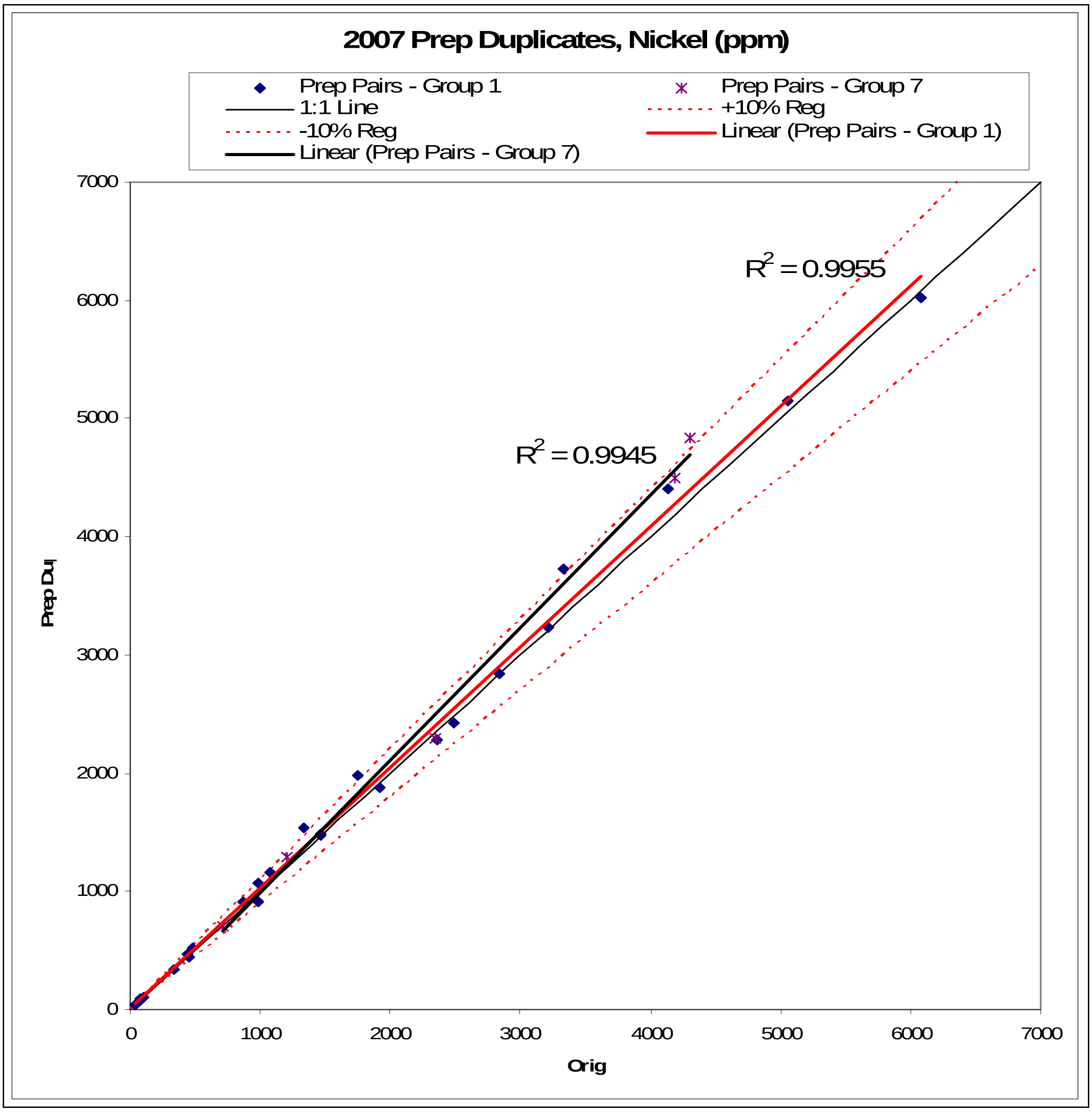


Figure 13-11: 2008 Regression Plot of Original vs. Preparatory Duplicate Pairs for Group 1 and 7 Analyses

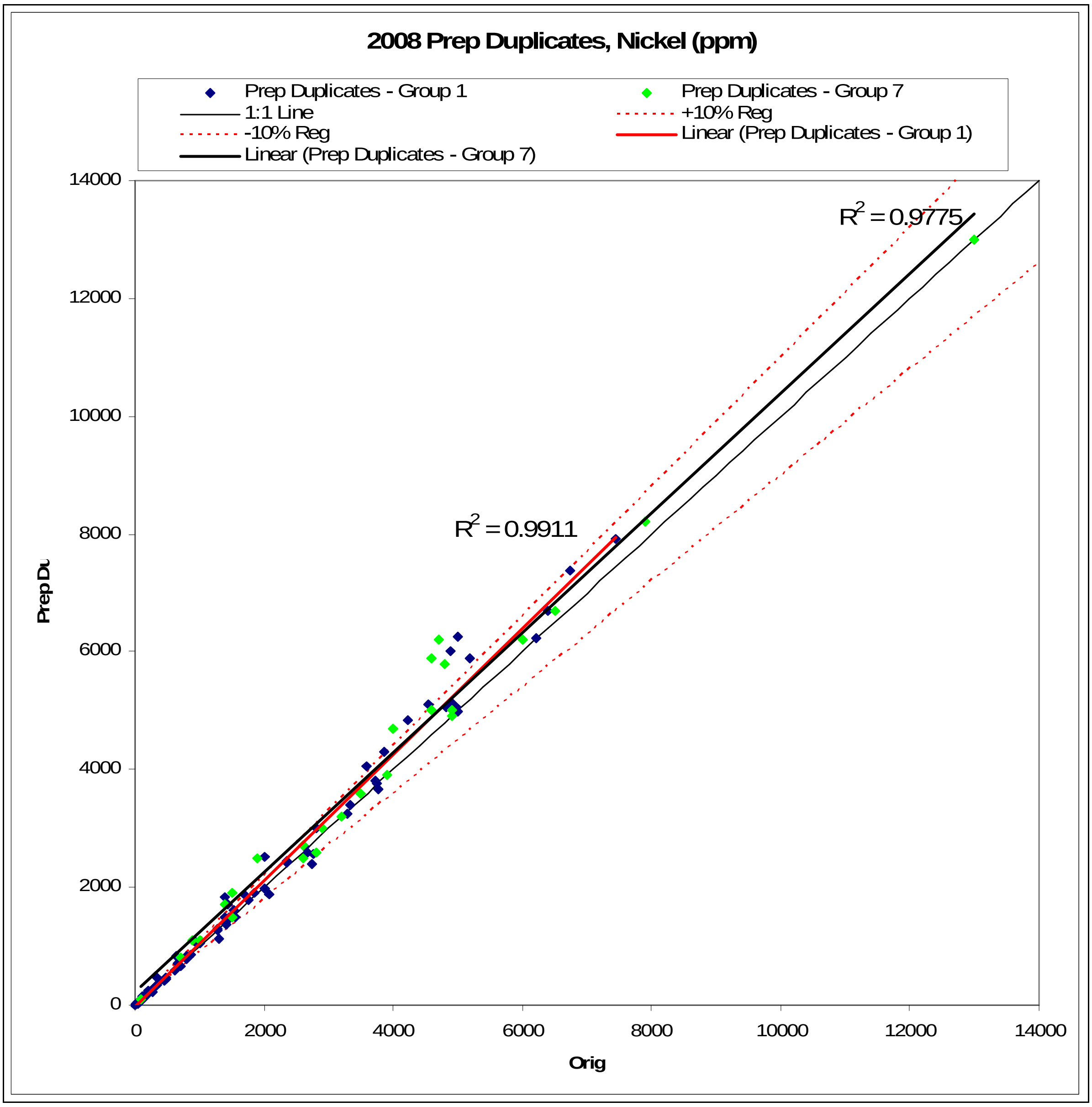


Figure 13-12: 2007 Regression Plot of Original vs. Pulp Duplicate Pairs for Group 1 and 7 Analyses

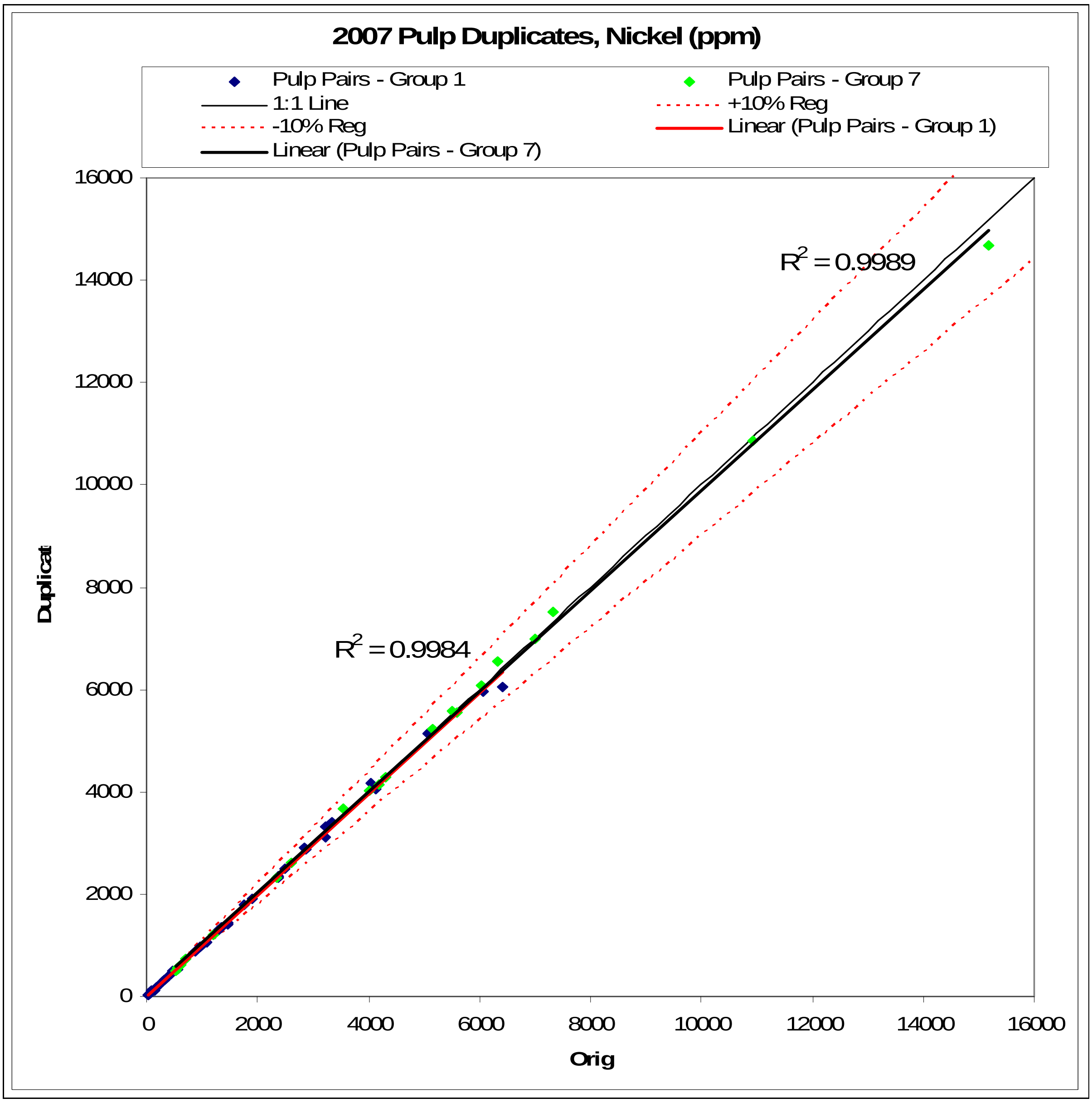
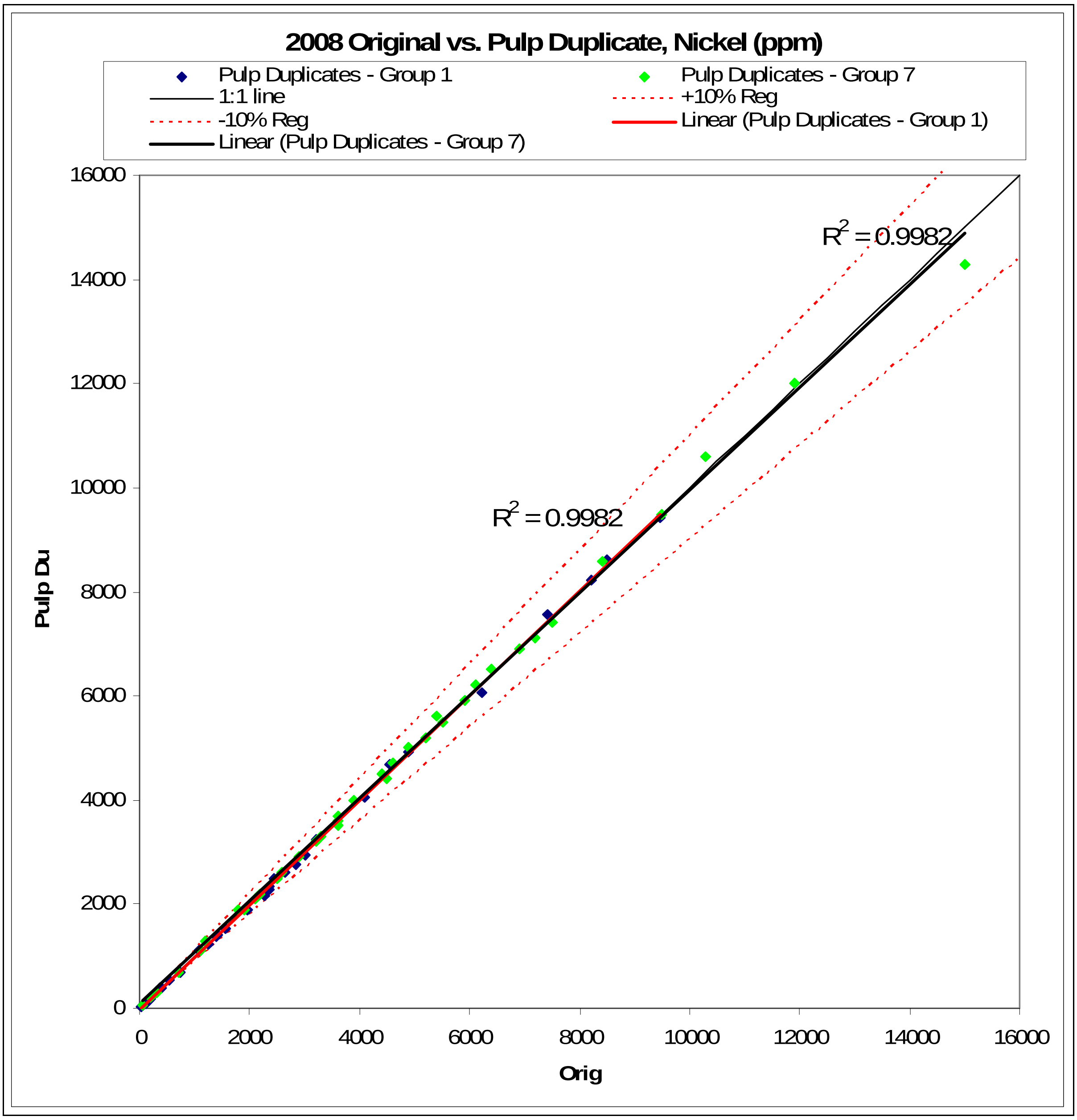


Figure 13-13: 2008 Regression Plot of Original vs. Pulp Duplicate Pairs for Group 1 and 7 Analyses



13.6 PEG ASSESSMENT OF THE QA/QC PROGRAM

PEG reviewed the results of the QA/QC program implemented by Strongbow and found that the guidelines used by Strongbow to determine the pass or fail status of a sample batch are consistent with industry standards. PEG is of the opinion that Strongbow was very pro-active following up QA/QC results as they became available and reacted quickly by re-submitting samples if a batch indicated a fail status.

PEG does have a few recommendations pertaining to the 2008 QA/QC program as follow:

1. Additional standards should be purchased to insert in the sample stream. Since the standard is delivered in pulp form to the laboratory and since it is the only sample of this type delivered, the single standard used in the 2008 drill program cannot be considered as a “blind” submission.

2. In the QA/QC charts, samples are ordered in sequence number. PEG recommends that Strongbow utilize charts and graphs that display the date along with the batch number and sample number. Control charts sorted by date assist in recognizing possible deteriorating trends in the analytical procedures.

3. While PEG consider the preparatory duplicate and pulp duplicate carried out by ACME as part of their internal QA/QC program to add value to Strongbow’s own program the samples process by the laboratory cannot be consider as blind submissions. To alleviate this concern, PEG recommend blindly inserting coarse rejects and pulps from earlier assays in the sample stream with a new tag number assuming that the logistics in relation to the rejects/pulp samples shipped back from the laboratory to the project site can be resolved. Obviously, the additional cost of adding this procedure to the QA/QC program needs to be weighed against the benefit obtained. One advantage gained with the re-submission of pulps is that the duplicate samples are of the same type as the standard and thus the primary laboratory receiving the samples no longer knows if they are processing a standard or a pulp duplicate.

4. Aside from the above recommendations, PEG believes that the current QA/QC program implemented by Strongbow both in the area of sample submission and monitoring is adequate to assess the precision and accuracy of the assays performed on the Strongbow core to the level that is needed to support the data used in the resource estimate.

14.0 DATA VERIFICATION

Strongbow geological staff have made a strong commitment to the geological and assay database and have, as far as is possible, produced a database that is complete and well documented.

Mr. Pierre Desautels visited the Nickel King deposit, accompanied by Dave Gale, Vice President, Exploration, between 31 March and 1 April 2009. No drill rigs were currently active during the site visit and the exploration camp was partly dismantled awaiting the next drill program.

The 2009 site visit entailed brief reviews of the following:

- Overview of the geology and exploration history Nickel King geology (presented by Strongbow geologist, Dave Gale)
- Current exploration program design (drill hole orientation, depth, number of holes, etc.)
- Surveying (topography and drill collar)
- Visit of the core logging facility and camp
- Discussion of the sample transportation and sample chain of custody and security
- Core recovery
- QA/QC program (insertion of standards, blanks, duplicates, etc.)
- Review of diamond drill core, core-logging sheets and core logging procedures. The review included commentary on typical lithologies, alteration and mineralization styles, and contact relationships at the various lithological boundaries
- Density sample collection.

Other than the Nickel King Main Zone Deposits, none of the other target areas previously explored by Strongbow were visited.

During the 2009 visit, PEG collected two half-core character samples and retained full custody of the sample from the Nickel King project site to Saskatoon, Saskatchewan where the samples were shipped to Activation Laboratories Ltd. located at 1428 Sandhill Drive, Ancaster, Ontario. The main intent of analysing these samples was to confirm the presence of nickel, copper, and cobalt in the deposit by an independent laboratory not previously used by Strongbow. These samples were analysed for total nickel, nickel sulphide, copper, and cobalt by digesting 0.5 g sample in aqua regia diluted volumetrically to 250 mL with 18 mega ohm water. Samples were then analysed on a Varian Vista 735 ICP.

For gold, platinum, and palladium, a 30 g sample is weighed, mixed with fire assay fluxes, and fused at 1,050°C for 1 hour. After cooling the lead button is separated from the slag and cupelled at 1,000°C to recover the Ag (doré bead) + Au, Pt, Pd. The Ag doré bead is digested in hot (95°C) HNO_3 + HCl. After cooling for two hours the sample solution is analyzed for Au, Pt, Pd, by ICP-MS. Smaller sample splits are used for high chromite or sulphide samples to ensure proper fluxing and metal recoveries.

From the assay results shown in Table 14-1, PEG concluded that the general range of values returned by the PEG samples correspond well with those reported by Strongbow.

Table 14-1: PEG Character Sample Results

	PEG	Strongbow	Difference	PEG	Strongbow	Difference
Sample Number	908	74,525	-	909	74,574	-
Total Nickel (ppm)	2,910	2,993.8	-83.8	14,200	12,960	1,240
Nickel Sulphide (ppm)	2,480	N/A	N/A	12,500	N/A	N/A
Copper (ppm)	600	635.6	-35.6	2,700	3,050	-350
Cobalt (ppm)	120	110.2	9.8	620	580	40
Gold (ppb)	12	13.4	-1.4	106	38.4	67.6
Platinum (ppb)	4.4	N/A	N/A	13.8	N/A	N/A
Palladium (ppb)	1.7	N/A	N/A	46.1	N/A	N/A

Strongbow uses a sample tag system consisting of three parts where the main part goes in the sample bag, another part is stapled to the core box, and the third part is retained. The insertion of purchased standard, pulp blank, coarse blank and pulp duplicate in the sampling chain could not be observed during the core logging facility visit. Sample tags for the QA/QC sample inserted in the sample chain were stapled to the core box.

Geologists responsible for logging the core can roughly estimate the (high/low) grade of the core in the field by the presence of sulphide. A Niton XL3T portable XRF analyser is sometimes used during core logging to estimate the grade of a sample interval.

The core is not continuously sampled in the norite sill. Intervals bearing no visual sulphide mineralization are commonly skipped with short check samples taken at regular intervals. During the site visit PEG noted a sample in hole NK08-060 that was not bracketed with a shoulder waste sample on the down hole side of the interval. PEG believes this was an exception; however, PEG recommends Strongbow sample the norite-mineralized horizon from top to bottom with at least one shoulder sample in the gneiss. Sampling outside the high-grade zone will provide an accurate grade assessment of mine waste and provide a realistic grade during the compositing process, which currently uses 0.00 for un-sampled intervals.

In proximity to the top and bottom contact of the Upper and Lower Sills, alternating layers of gneiss and norite can be seen over a length of 0.5 m or less. The transition is short and non-gradational. In the hole examined, the mineralization of the Upper Sill is finely disseminated. The dominant sulphide mineral is pyrrhotite with finely disseminated chalcopyrite and fine-grained pentlandite often visible without a hand lens.

Photo 1 displays a series of photographs taken during the site visit.

Photo 1: Site Visit Photos

Exploration camp

Core box markings – NK08-060

Hole NK08-035 and 036 casing cap

Specific gravity samples collected by Strongbow

NK08-060 at 111.0 m high grade mineralization

Cross stack core storage

14.1.1 DATABASE VALIDATION

Prior to the site visit and resource evaluation, PEG carried out an internal validation of the drill holes in Strongbow's Nickel King database used in the 25 February 2009, resource estimate. Holes were selected for validation according to the following criteria:

- highest nickel grade
- highest average grade
- crosscutting both the Lower and Upper Sills
- distribution in the deposit
- representative selection based on the drilling year.

A total of 10 holes were completely validated amounting to 917 individual samples out of a total of 4,845 that were either checked against paper copies of the original signed certificate or against the electronic version of the certificate provided by the issuing laboratory. The validation rate amounted to 16.1% of the Strongbow and Aber drill holes and 18.9% of the assays used in the resource estimate.

Table 14-2 shows a list of the holes selected for validation.

Table 14-2: Hole Selected for Validation

Hole-ID	Easting	Northing	Elevation	Length (m)	Assay Count
AN95-06	526541	6680025	414	203	46
AN95-11	525532	6679767	396	188	31
NK07-012	526309	6680149	403	232.81	124
NK07-015	527122	6679754	405	182	65
NK08-029	525871	6680032	396	312.27	144
NK08-033A	525994	6680133	396	289	119
NK08-035	526281	6680139	403	212.15	117
NK08-036	526281	6680139	403	200.2	110
NK08-051	526358	6680145	400	201.37	77
NK08-060	526302	6680175	400	249.05	84
Total	-	-	-	**2,269.85**	**917**

14.1.2 COLLAR COORDINATE VALIDATION

Collar coordinates were validated with the aid of a hand-held Garmin GPSmap model 60CSx. A series of collars were randomly selected and the GPS position was recorded. The difference with the Gems database was calculated in an X-Y 2D plane with the following formula:

$$X - Y\ difference = \sqrt{(\Delta \mathrm{East})^2 + (\Delta \mathrm{North}^2)}$$

As shown on Table 14-3, results indicated an average difference in the X-Y plane of 2.9 m for the 5 holes collars where the instrument was located on the top of the casing and 0.8 m in the Z-plane. The calculated differences in the X-Y plane are well within the accuracy of the hand held GPS unit used. The instrument reported “accuracy” between 2.5 m to 3.8 m at most field locations surveyed, which is typically influenced by the number of satellites seen on the time and day. Elevation difference is surprisingly good as the hand held GPS are notoriously inaccurate in elevation.

Table 14-3: Collar Coordinate Verification

Gemcom Database Entry				GPS Point Recorded During Site Visit				Differences between GEMS and GPS	
HOLE-ID	**East**	**North**	**El.**	**GPS Point**	**East**	**North**	**El.**	**X-Y Plane (m)**	**Z Plane (m)**
NK07-011	526157	6680110	397	D011	526159	6680111	396	2.236	1.000
NK08-035, 036	526281	6680139	403	D036	526282	6680141	403	2.236	0.000
NK08-040, 039	526072	6680107	398	D039	526075	6680108	395	3.162	3.000
NK08-043	526133	6680176	403	D043	526135	6680176	403	2.000	0.000
NK08-060	526302	6680175	400	D060	526307	6680176	400	5.099	0.000
Average Difference (m)								**2.947**	**0.800**

14.1.3 *DOWN-HOLE SURVEY VALIDATION*

Table 14-4 shows the down-hole survey data was validated by searching for large discrepancies between dip and azimuth reading against the previous reading. A total of 512 readings (entire database) were evaluated. Holes were separated whether they were collared vertically or at an angle and absolute differences were calculated along with the degree per meter change with the previous reading.

Only two holes were collared vertically (steeper than -87°). Forty-seven drill holes or 48% of the total number of holes in the database were collared sub-vertically between -85° and -87°.

Any azimuth with a difference exceeding 10° combined with change per meter in excess of 1° was considered suspect. The proportion of the sub-vertical drill holes in the database partially explains the relatively high incidence of large azimuth differences.

For dip measurement, any dip differences exceeding 1° combined with a change per meter in excess of 1° was flagged as suspect.

All suspect readings were forwarded to Mr. Dave Gale for review.

Table 14-4: Down-hole Survey Validation Results

For Angle Holes	Azimuth Diff.	Azimuth Diff. (ABS)	Dip Diff.	Azimuth Change/m	Dip Change/m
Min.	-27.8	-	-3.52	-	-
Max.	151	151	13	7.371	0.565
Average	1.415	3.252	0.315	0.152	0.016
First Quartile	-	0.1	-0.03	-	-
Median	0.241	0.57	0.01	0.029	0.003
Third Quartile	1.28	3.235	0.2	0.082	0.010
97 percentile	14.94	17.21	4.46	1.156	0.124
99 percentile	23.22	25.5	9.96	1.974	0.180

Results indicated 16 azimuths and 1 dip reading required further validation. Following review, one azimuth required correction and all others entries were considered as correct since the database entry coincided with the down hole survey instrument value.

14.1.4 *ASSAY VALIDATION*

Assay validation proceeded in two steps:

- validation of GEMS database entry against the PDF copies of the original signed laboratory certificates for the two selected Aber drill holes
- validation of the electronic version of the certificates against the Gems database entry.

The validation against the original signed PDF copies of the certificate consisted of two holes AN95-06 and AN95-11 with 77 assays re-typed in an XLS spreadsheet. The nickel and copper values on the certificates were rounded to the nearest 100th using the round-to-even method also known as bankers rounding. The cobalt were rounded to the nearest 10th using the same methodology. The average difference between the certificate and the database entry is less than 6 ppm and is considered negligible. For the nickel assay, Aber re-submitted 33 out of the 77 assays selected for validation because the value was approaching or exceeding the detection limit of the analytical procedure used. For these 33 assays, nickel grades from the analytical procedure yielding the highest precision were entered in the database. PEG found the error rate to be low (1 out of 77) for nickel and no errors were found for copper and cobalt. Sample 7100-07 was entered in the database as 0.94% Ni when the certificate value indicated 0.93%. Sample 7102-03 and 04 showed a discrepancy compared to others assays above or near the upper detection limit of the laboratory procedure used. For these two samples, the Ni ppm value was entered in the database when a Ni% value was available. Sample number 7100-07 showed a difference of 0.01 Ni% between the certificate value and the entry in the database. No errors were found for copper and cobalt.

The validation against the electronic version of the certificates consisted of comparing the values on the certificate against the GEMS database entry. Certificates for the holes in the validation list were requested from the ACME Analytical Laboratories Ltd (Strongbow's principal laboratory) in Vancouver, British Columbia, via David Gale of Strongbow. ACME issued the requested certificates as a series of text files in CSV format along with a similar file in XML format and the final, signed, PDF version of the certificate directly to the PEG e-mail address without Strongbow's interaction. A total of 903 assays results covering 14 drill holes were compiled from the certificates onto an Excel spreadsheet and matched against the sample number in the GEMS database. A total of 90 QA/QC assays did not find a matching sample number in the GEMS database with the remaining 813 samples number successfully matched.

Results show that out of the 813 samples reviewed none were entered erroneously in the database. Assay validation covers 15% of the entire database with a total of 18% coverage for the assays used in this resource estimate.

The error rate in the Strongbow drill database was found to be exceptionally low. The Qualified Person regards the sampling, sample preparation, security, and assay procedures as adequate to form the basis of resource estimation.

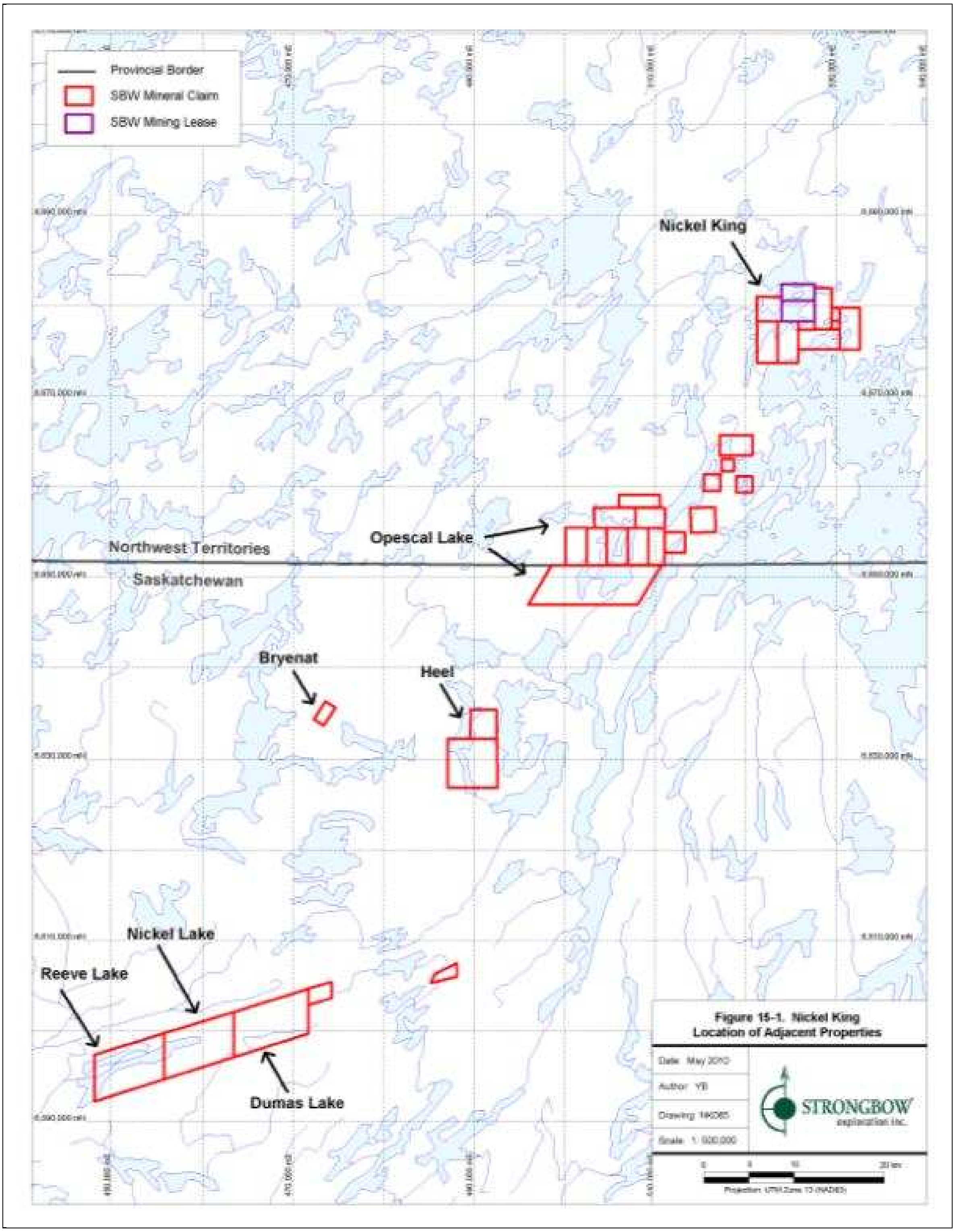
Table 14-5: Assay Validation Results

	Verified	Ni Errors	Cu Errors	Co Errors
Strongbow Assays	813	-	-	-
Aber Assays	77	3	-	-
Total	**890**	**3**	**-**	**-**
Total assays in DB	**5,993**	**-**	**-**	**-**
Percent Checked/Error	15%	0.05%	-	-

15.0 ADJACENT PROPERTIES

There are presently no third party exploration properties in the immediate vicinity of the Nickel King project. However, nickel-copper-cobalt mineralization has been identified to the southwest of Nickel King within the STZ. Strongbow is presently exploring the potential of the Opescal Lake, Heel, Dumas Lake, and Breynat properties as part of its Snowbird nickel project (Figure 15-1, overleaf). Strongbow has completed airborne magnetic and EM surveys, regional scale prospecting and limited mapping over these target areas and in each case has identified nickel-copper sulphide mineralization associated with mafic (norite, gabbronorite) and ultramafic (pyroxenite, peridotite) intrusions. Together, early results for each of these properties highlight the potential for nickel sulphide mineralization over a significant strike length of the southern STZ. More detailed documentation of the exploration results on these properties can be found in a separate technical report on the Snowbird project (Campbell and Gale, 2010). This report can be downloaded from Strongbow's profile at www.sedar.com.

Figure 15-1: Location of Adjacent Properties



16.0 MINERAL PROCESSING AND METALLURGICAL TESTING

16.1 SAMPLE PREPARATION AND CHARACTERIZATION

Approximately 140 kg of split core samples from drill hole NK08-035 of the Nickel King deposit were received at SGS Mineral Services on 21 April 2009. The samples were packed as ~1 m intervals contained in plastic bags, packed into six rice bags, packed into one wooden crate. The samples were inventoried and found to consist of sample IDs #69523 to #69587 inclusive, and represented a hole depth from 120 m to 186.5 m.

In consultation with the project team, it was decided that a single metallurgical composite would be prepared using the expected mineable portion of the hole based on the core log assays provided. This consisted of the entire length of the hole with the exception of a single sample at the top (#69523), and six samples at the very bottom (#69582 to #69587). These samples were deemed too low grade to be of economic interest.

The selected interval of samples was crushed to minus ½" and blended to generate 122 kg of composite. A 10 kg subsample of the composite was split out for head assay, primarily to determine whether dilution with waste rock should be considered. A large sub-sample was deemed necessary to mitigate the effects of sampling error resulting from the coarse (-½") particle size. The subsample was subsequently crushed to 100% passing 10 mesh (1.7 mm) and subsampled again for chemical analysis. Upon receiving this data (see “coarse sample” in Table 16-2), the project team decided that dilution was unnecessary. 40 kg was riffled out of the main composite and stored in SGS’s freezer for possible future work. The remaining 70 kg was submitted for testing as per the agreed scope.

16.1.1 CHEMICAL ANALYSIS

Summaries of the available head analysis data are presented in Table 16-1 and Table 16-2.

Table 16-1: Head Analysis for Cu, Ni, S

	% Ni	% Cu	% S
Core Log Estimate	0.69	0.15	4.92
Coarse Head Sample (-½")	0.72	0.16	5.21
Fine head sample (-10#)	0.61	0.16	4.63
Testwork Calculated Head	0.65	0.14	4.76

Table 16-2: Additional Assays Carried Out on the Coarse Ore Subsample

Element	Units	Result
NiS	%	0.62
Co	%	0.042
Ag	g/t	<2
Al	g/t	3,500
As	g/t	<30
Ba	g/t	130
Be	g/t	0.3
Bi	g/t	<20
Ca	g/t	31,000
Cd	g/t	<2
Cr	g/t	1,300
Fe	g/t	16,000
K	g/t	3,500
Li	g/t	<5
Mg	g/t	110,000
Mn	g/t	1,100
Mo	g/t	<5
Na	g/t	5,500
P	g/t	330
Pb	g/t	<50
Sb	g/t	<10
Se	g/t	<30
Sn	g/t	<20
Sr	g/t	98
Ti	g/t	1,100
Tl	g/t	<30
U	g/t	<30
V	g/t	96
Y	g/t	5.6
Za	g/t	120

Some variation is observed between the head analysis results presented in Table 16-1, particularly for the coarse head sample collected when the composite was only crushed to -½". This bias (~10%) is considered reasonable given the coarse nature and size of the sample. For reporting purposes, PEG believes that the average of the testwork calculated heads represents the best estimate of the composite head grade as it is based on the assay results from 10 individual flotation tests, all of which showed close agreement in calculated head grade.

The ICP scan presented in Table 2 did not highlight problematic concentrations of deleterious elements of concern in the head sample.

16.1.2 *PHYSICAL CHARACTERIZATION*

Coarse subsamples of the composite were submitted for rod and ball mill Bond Work Index. Results are presented in Table 16-3. Results are typical for this type of ore and do not suggest problems. Future test programs will develop this characterization and will add crushability and abrasion index information.

Table 16-3: Bond Ball and Rod Work Index Values for the Nickel King Composite

BWI_B	15.0 kWh/t
BWI_R	13.2 kWh/t

16.2 MINERALOGY

A test charge was ground to 80% passing ~110 mm (45 minute grind) before being submitted for mineralogical analysis. QEMScan and Electron Microprobe analyses were conducted on two polished sections and the average modal abundance of the minerals in these samples is summarized in Table 16-4.

Table 16-4: Modal Mineral Distribution for the Nickel King Composite

Mineral	Mass %
Pyrrhotite	12.6
Pentlandite	1.58
Chalcopyrite	0.48
Orthopyroxene	53.2
Cpx/Amphibole	7.76
Olivine	0.07
Talc	1.08
Quartz	0.70
Feldspars	13.7
Chlorite	0.40
Clays	2.42
Biotite	4.68
Magnetite	0.18
Al-Cr-Spinel	0.21
Apatite	0.22
Calcite	0.08
Dolomite	0.60
Other	0.10
Total	**100.0**

The analysis indicates the distribution of pyrrhotite, pentlandite, and chalcopyrite within a silicate matrix consisting mainly of orthopyroxene. Pyrrhotite is the dominant sulphide.

Electron Microprobe was used to identify the compositions of the sulphide and common nickel-bearing silicate minerals. Results are given in Table 16-5.

Table 16-5: Constituent Mineral Compositions as determined by Electron Microprobe

Mineral	Ni %	Fe %	Cu %	S %
Pentlandite	34.80	30.54	-	33.11
Pyrrhotite	0.47	60.28	0.01	39.18
Orthopyroxene	0.023	11.40	-	-

The modal abundance and nickel-bearing mineral composition data can be combined to give a summary of nickel distribution, as shown in Table 16-6.

Table 16-6: Calculated Nickel Distribution by Mineral for the Nickel King Composite

Mineral	Distribution %
Pentlandite	88.9
Pyrrhotite	9.12
Orthopyroxene	1.98
Total	**100**

Thus, almost 90% of nickel in the composite can be classed as “recoverable” (i.e., nickel in pentlandite) and this is typical of “good” low grade nickel ores. Silicate gangue (orthopyroxene) and sulphide gangue (pyrrhotite) must be effectively rejected by the flotation process in order that the concentrates are of sufficient grade for sale.

QEMScan software allows statistical analysis of particle maps to provide quantitative liberation data. Note that for this preliminary program, the robustness of data is affected by the sample size (two polished sections only). An estimate of mineral liberation for pentlandite and pyrrhotite is presented in Table 16-7. The results indicate that roughly 86% of the contained pentlandite is either liberated or sub-liberated at this grind size (80% - 110 µm). At the same time, ~10% of the pentlandite is associated with pyrrhotite, either as middlings or finely disseminated within the pyrrhotite matrix. Pyrrhotite liberation is similarly high, at roughly 88% classed as either liberated or sub-liberated.

Table 16-7: Liberation of the Principal Nickel Bearing Minerals (Pn and Po) in the Nickel King Composite

Mineral	Mass %
Liberated Pentlandite	83.4
Sub-Liberated Pn	2.57
Pn:Pyrrhotite	9.63
Pn:Chalcopyrite	0.00
Pn:Orthopyroxene	0.15
Pn:Cpx/Amphibole	0.02
Pn:Quartz	0.10
Pn:Feldspars	0.01
Pn:Biotite	0.00
Pn:Dolomite	0.25
Complex	3.86
Total (Pentlandite)	**100.0**

Mineral	Mass %
Liberated Pyrrhotite	81.5
Sub-Liberated Po	6.52
Po:Pentlandite	1.97
Po:Chalcopyrite	0.02
Po:Orthopyroxene	1.83
Po:Cpx/Amphibole	0.12
Po:Quartz	0.00
Po:Feldspars	0.24
Po:Biotite	0.02
Po: Apatite	0.02
Complex	7.74
Total (Pyrrhotite)	**100.0**

16.3 FLOTATION

16.3.1 *ROUGHER KINETICS FLOTATION TESTS*

In total, six rougher kinetics batch flotation tests were conducted. The purpose of the tests was to identify operating conditions such as grind size, reagent addition, and pH that would result in optimum recovery of copper and nickel while rejecting as much barren silicates and pyrrhotite as possible. Rougher kinetics flotation tests collect several increments of concentrate in order to obtain information regarding the rates of flotation for minerals of interest (pentlandite, pyrrhotite, chalcopyrite, and gangue).

Figure 16-1 and Table 16-8 show a summary of the results of the rougher kinetics tests.

Figure 16-1: Grade vs. Recovery and Mass vs. Recovery (Rougher Kinetics Tests)

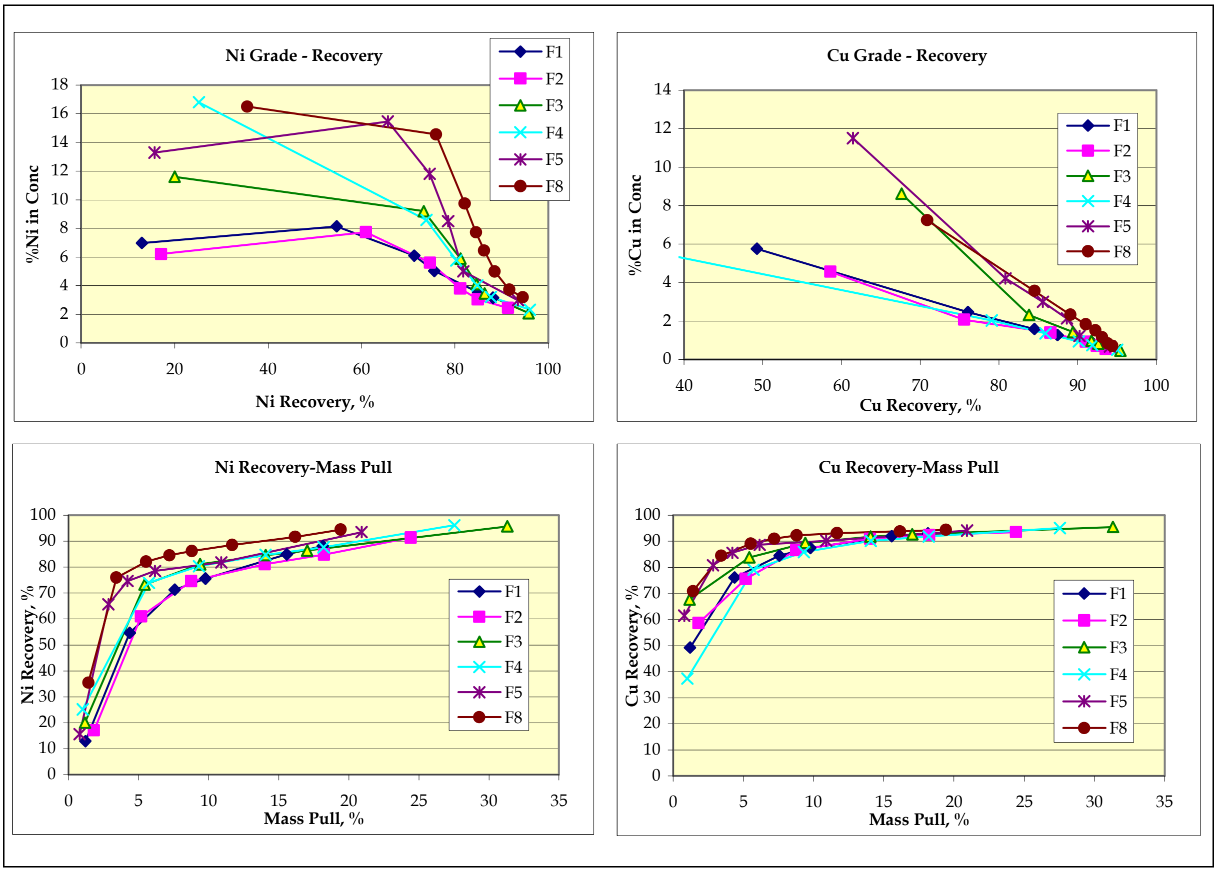


Table 16-8: Summary of Rougher Kinetics Flotation Test Results

Test Condition	Product	Wt %	Assays, %			Distribution %		
			Cu	Ni	S	Cu	Ni	S
F1	Ro Conc. 1	1.21	5.75	6.98	14.7	49.3	13.0	3.81
K_{80}=155 µm	Ro Conc. 1-2	4.36	2.46	8.14	13.4	76.1	54.7	12.5
pH9.5 Ro 1-4	Ro Conc. 1-3	7.58	1.57	6.11	12.4	84.5	71.3	20.1
pH8.0 Ro 5-6	Ro Conc. 1-4	9.76	1.26	5.03	11.3	87.4	75.5	23.7
SIPX 60 g/t	Ro Conc. 1-5	15.6	0.83	3.54	13.2	92.0	84.9	44.1
PAX 25 g/t	Ro Conc. 1-6	18.2	0.72	3.15	15.4	93.0	88.2	59.9
MIBC, CMC(75 g/t)	Rougher Tail	81.8	0.01	0.09	2.28	6.96	11.8	40.1
Head (calc.)		**100.0**	**0.14**	**0.65**	**4.66**	**100.00**	**100.0**	**100.0**
F2	Ro Conc. 1	1.81	4.56	6.20	12.2	58.6	17.1	4.85
K_{80}=101 µm	Ro Conc. 1-2	5.17	2.06	7.74	13.9	75.6	61.0	15.81
pH9.5 Ro 1-4	Ro Conc. 1-3	8.75	1.39	5.60	10.3	86.6	74.6	19.9
pH8.0 Ro 5-6	Ro Conc. 1-4	14.0	0.91	3.80	9.27	91.0	81.1	28.6
SIPX 60 g/t	Ro Conc. 1-5	18.2	0.71	3.05	9.86	92.5	84.9	39.6
PAX 25 g/t	Ro Conc. 1-6	24.4	0.54	2.46	13.4	93.6	91.4	71.9
MIBC, CMC(75g/t)	Rougher Tail	75.6	0.01	0.08	1.69	6.45	8.64	28.1
Head (calc.)		**100.0**	**0.14**	**0.66**	**4.55**	**100.00**	**100.00**	**100.0**
F3	Ro Conc. 1	1.18	8.61	11.60	22.2	67.6	20.0	5.38
K_{80}=74 µm	Ro Conc. 1-2	5.43	2.31	9.20	12.7	83.8	73.3	14.25
pH9.5 Ro 1-4	Ro Conc. 1-3	9.40	1.42	5.89	13.0	89.4	81.1	25.3
pH8.0 Ro 5-6	Ro Conc. 1-4	14.1	0.98	4.10	11.6	91.8	84.6	33.6
SIPX 60 g/t	Ro Conc. 1-5	17.0	0.81	3.46	11.3	92.6	86.3	39.7
PAX 100 g/t	Ro Conc. 1-6	31.3	0.46	2.08	14.8	95.4	95.8	95.5
MIBC, CMC(125g/t)	Rougher Tail	68.7	0.01	0.04	0.32	4.59	4.23	4.5
Head (calc.)		**100.0**	**0.15**	**0.68**	**4.85**	**100.00**	**100.00**	**100.0**
F4	Ro Conc. 1	1.00	5.48	16.80	22.1	37.4	25.2	4.54
K_{80}=95 µm	Ro Conc. 1-2	5.73	2.02	8.59	13.5	79.1	73.9	15.93
pH8.8 Ro 1-4	Ro Conc. 1-3	9.29	1.35	5.76	13.5	85.9	80.3	25.9
pH7.0 Ro 5-6	Ro Conc. 1-4	14.1	0.94	4.01	12.8	90.1	84.8	37.0
SIPX 60 g/t	Ro Conc. 1-5	18.2	0.74	3.21	13.9	91.9	87.7	52.1
PAX 50 g/t	Ro Conc. 1-6	27.5	0.51	2.33	16.9	95.0	96.1	95.8
MIBC, CMC(125g/t)	Rougher Tail	72.5	0.01	0.04	0.28	4.95	3.92	4.2
Head (calc.)		**100.0**	**0.15**	**0.67**	**4.86**	**100.00**	**100.00**	**100.0**
F5	Ro Conc. 1	0.79	11.50	13.30	28.8	61.4	15.7	4.72
K_{80}=150 µm	Ro Conc. 1-2	2.83	4.21	15.46	20.7	80.8	65.6	12.20
pH9.5 Ro 1-4	Ro Conc. 1-3	4.22	3.00	11.82	22.9	85.6	74.6	20.1
pH7.0 Ro 5-6	Ro Conc. 1-4	6.2	2.12	8.51	19.1	88.6	78.5	24.5
SIPX 60 g/t	Ro Conc. 1-5	10.9	1.22	5.01	17.1	90.2	81.8	38.8
PAX 50 g/t	Ro Conc. 1-6	20.9	0.66	2.98	20.3	94.1	93.5	88.2
MIBC, CMC(300 g/t)	Rougher Tail	79.1	0.01	0.06	0.72	5.89	6.51	11.8

Test Condition	Product	Wt %	Assays, % Cu	Assays, % Ni	Assays, % S	Distribution % Cu	Distribution % Ni	Distribution % S
Head (calc.)		**100.0**	**0.15**	**0.67**	**4.81**	**100.00**	**100.00**	**100.0**
F8	Ro Conc. 1	1.42	7.23	16.50	26.4	70.9	35.6	7.91
K_{80}=100 µm	Ro Conc. 1-2	3.42	3.56	14.56	26.6	84.5	75.9	19.26
Na_2CO_3	Ro Conc. 1-3	5.53	2.32	9.74	23.2	89.1	82.1	27.2
pH9.5 Ro 1-4	Ro Conc. 1-4	7.2	1.83	7.72	22.0	91.0	84.5	33.5
pH7.0 Ro 5-6	Ro Conc. 1-5	8.8	1.51	6.44	21.1	92.2	86.2	39.3
SIPX 60 g/t	Ro Conc. 1-6	11.7	1.15	4.98	21.9	93.1	88.5	54.2
PAX 50 g/t	Ro Conc. 1-7	16.2	0.84	3.72	21.6	93.8	91.7	73.9
MIBC, CMC(300 g/t)	Ro Conc. 1-8	19.4	0.70	3.20	21.3	94.4	94.5	87.4
	Rougher Tail	80.6	0.01	0.05	0.74	5.58	5.52	12.6
Head (calc.)		**100.0**	**0.14**	**0.66**	**4.73**	**100.00**	**100.00**	**100.0**

Tests F1 and F2

Tests F1 and F2 represented the first attempt at flotation of the Nickel King sample. The conditions chosen represent a typical starting point for low grade pentlandite-pyrrhotite-chalcopyrite systems:

pH	Lime added to adjust pulp pH to 9.5 for early stages of flotation. This has a tendency to depress the flotation of pyrrhotite. After 15 min of flotation, pH was lowered to 8.0 to encourage flotation of pyrrhotite (thus avoiding Po in rougher tails)
Collector	SIPX for first 15 minutes of flotation (more selective against pyrrhotite than some other collectors). PAX for later stages (+15 min) – again to encourage pyrrhotite flotation.
Depressant	CMC (cellulose-based) depressant added throughout the flotation process to depress the flotation of silicate gangue minerals.

For tests F1 and F2, the grind was varied (F1=80% -150 µm, F2=80% -100 µm), whereas the other conditions remained the same.

Results: Comparing F1 and F2 demonstrated that a finer grind improves flotation response. After 29 minutes of flotation, both tests recovered approximately 90% of the nickel into approximately 20% of the feed mass. Significant sulphur grade remaining in rougher tails was believed to be slow-floating pyrrhotite.

Tests F3 and F4

Tests F3 and F4 were designed to evaluate a finer grind (80% -75 µm), the natural pH, and the effect of additional CMC. In addition, the scavenging of pyrrhotite was continued for a further 20 minutes in an attempt to better understand the flotation rate of slow-floating pyrrhotite and gangue.

Results: The finer grind used in test F3 did not improve performance significantly as compared to test F4. For the purposes of this preliminary program, one can conclude that (~80% -100 µm) is a good target for effective flotation of this composite.

Test F4 can be compared to test F2 (both at ~100 µm grind). The natural pH (8.8) utilized in F4 increased the rate recovery of pyrrhotite, with almost 10% more Po recovery to concentrates 1-4 (cumulative); however, mass recovery to concentrates 1-4 was unchanged at 14%, meaning that the additional CMC dosed to F4 was effective at lowering gangue recovery.

Test F5

The conditions for test F5 were developed during a meeting between SGS and DRAA. Results from F1 – F4 were available at the meeting. The rationale behind test F5 was:

- grind to remain at 80% -100 µm (seen to be most effective)
- pH 9.5 (using lime) for early flotation to inhibit pyrrhotite flotation
- further increases in CMC addition throughout the rougher to inhibit flotation of silicate gangue
- improve rate of flotation of pyrrhotite after dropping pH to 7.0 by adding more PAX.

Results: Unfortunately, the sample was miss-ground in this test, resulting in a coarser than desired K_{80} of 171 µm. However, even at the coarse grind, the additional CMC improved rejection of silicates and thus allowed higher concentrate grades at similar recovery points.

The lower pH and higher PAX were observed to improve recovery of pyrrhotite at the back end of the circuit.

Test F8

In test F8 the conditions of F5 were repeated, except that a finer target grind of ~110 µm was used, and soda ash (Na_2CO_3) was used in place of lime. It was reasoned that soda ash, although more costly than lime, might improve rejection of silicates due to improved pulp dispersion.

Results: In Figure 16-1 it can be seen that the nickel grade recovery curve (top left graph) for test F8 demonstrates a significant improvement over the previous tests in this series. This is largely due to improved rejection of silicate gangue material, and hence reduced mass pull to the rougher concentrate (see lower left hand graph). The effect of this change is not as evident for copper grade vs. recovery or recovery vs. mass pull, as the kinetics of copper recovery are faster than those for nickel under all conditions tested.

16.3.2 BATCH CLEANER FLOTATION TESTS

In total, four batch cleaner tests were conducted on the Nickel King Composite. The purpose of the tests was to identify optimum conditions for upgrading the rougher concentrate to saleable grades while maintaining good metal recoveries. Variables investigated during the cleaner flotation tests included circuit configuration, regrind size, CMC addition, soda ash addition, and flotation time.

Test F6

The initial starting point for the cleaner tests consisted of a copper/nickel rougher followed by a separate pyrrhotite rougher. The resulting concentrates would be reground and cleaned independently, as shown in Figure 16-2. The copper/nickel circuit consisted of two stages of cleaning with a scavenger on the first cleaner tailings. The pyrrhotite circuit consisted of a single stage of cleaning only.

Results: This first cleaner test was, like test F5, mistakenly conducted at 162 µm primary grind, and with only 75 g/t CMC in the primary circuit. However, the results are still valid, and demonstrate that a saleable concentrate grade can be achieved, although at a particularly poor recovery (14.4% Ni grade in a final concentrate containing only 22% of total nickel).

Figure 16-2: Flowsheet for Cleaner Flotation Test F6

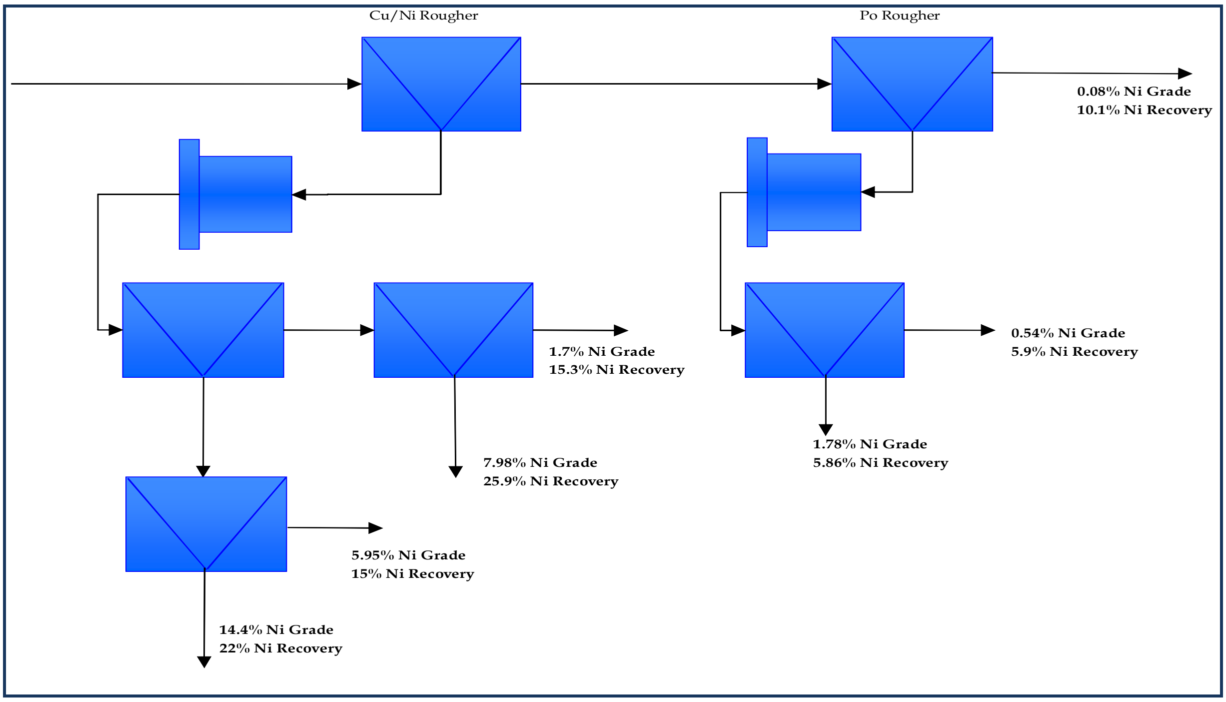


Table 16-9: Summary of Results for Cleaner Flotation Test F6

			Assays, %			% Distribution		
Test Condition	**Product**	**Wt %**	**Cu**	**Ni**	**S**	**Cu**	**Ni**	**S**
F6	Cu/Ni 2nd Cl. Conc. 1	0.40	19.60	6.07	29.2	57.3	3.7	2.40
K_{80}=162 µm	Cu/Ni 2nd Cl. Conc. 1-2	0.69	13.91	11.16	28.9	70.5	11.9	4.1
Cu/Ni Ro	Cu/Ni 2nd Cl. Conc. 1-3	0.99	10.27	14.43	28.5	74.5	22.0	5.8
pH 9.5	Cu/Ni 1st Cl. Conc.	2.63	4.06	9.14	16.2	78.2	36.9	8.8
40 g/t SIPX	Cu/Ni 1st Cl. + Scav. C.	4.7	2.36	8.63	15.3	81.7	62.8	14.9
75 g/t CMC	Cu/Ni Rougher Conc.	10.6	1.08	4.78	14.1	83.7	78.2	30.8
Reg K_{80}= 30 µm								
	Po Rej 1st Cl. Conc. 1	0.6	0.49	3.12	29.6	2.3	3.1	3.9
Po Rougher	Po Rej 1st Cl. Conc. 1-2	1.0	0.35	2.58	29.0	2.6	4.0	6.1
pH 7.0	Po Rej 1st Cl. Conc. 1-3	1.5	0.27	2.16	29.0	2.9	4.8	8.7
75 g/t PAX	Po Rej 1st Cl. Conc. 1-4	2.1	0.21	1.78	29.4	3.2	5.9	13.0
Reg K_{80}= 36 µm	Po Rougher Conc.	9.2	0.07	0.83	25.6	4.6	11.8	48.8
	Rougher Tail	80.1	0.02	0.08	1.23	11.72	10.1	20.3
Head (calc.)		100.0	0.14	0.65	4.85	100.00	100.0	100.0

When considering the poor circuit recovery, one should note the following:

- The coarse primary grind and low dose of CMC resulted in 78.2% Ni (see Table 16-9) recovery to the copper/nickel rougher concentrate. We would expect this to improve to 85% to 88% Ni with correct conditions.
- The copper/nickel cleaner circuit has two stages (see Figure 16-2). In each stage of cleaning, a tails stream is generated which, in a locked cycle test (or in an industrial plant), would be re-circulated within the circuit. The nickel reporting to these streams accounts for 56% recovery.

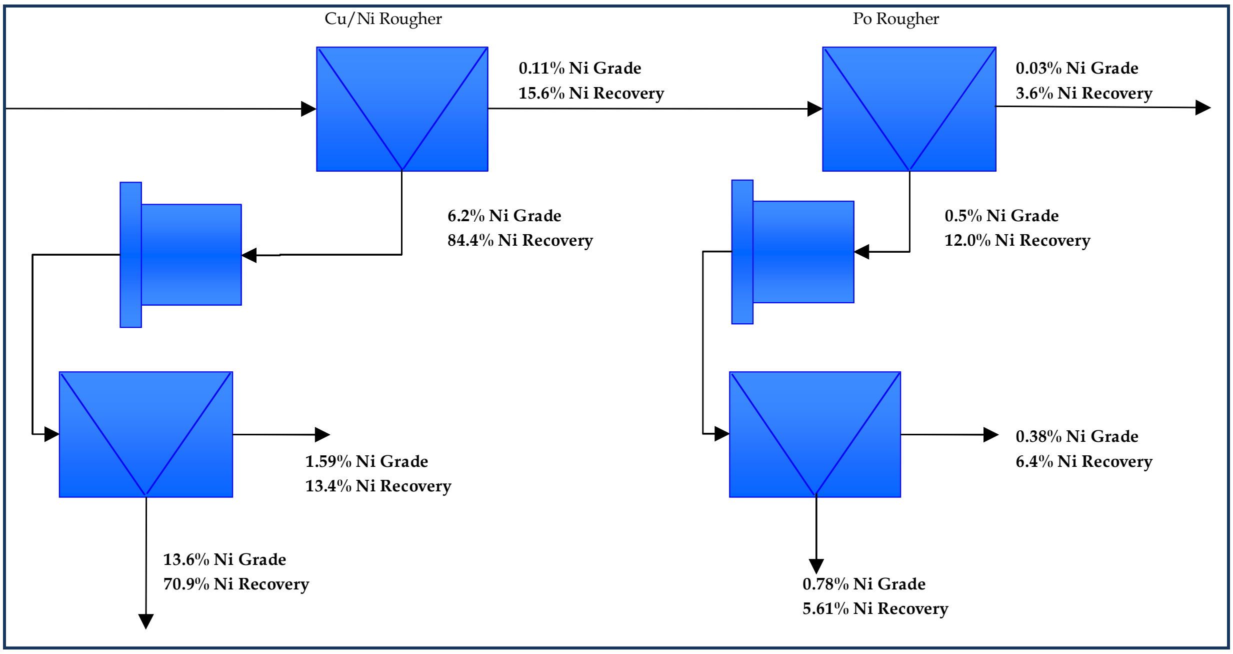
Test F7

The knowledge gained in F6 helped to develop the conditions for F7. This test was designed to achieve the following:

- primary conditions set to ensure ~85% nickel recovery to the copper/nickel rougher concentrate (finer grind, increased levels of CMC)
- improve recovery of nickel in the copper/nickel cleaners, while maintaining saleable grades
- improve nickel recovery and rejection of pyrrhotite in the pyrrhotite cleaner circuit (using high pH and sodium sulphite as a pyrrhotite depressant).

The flowsheet was simplified to a single stage of cleaning (Figure 16-3) in the copper/nickel circuit in order to focus on the flotation kinetics in the first stage.

Figure 16-3: Flowsheet for Cleaner Flotation Test F7



Results: In contrast to F6, a nickel recovery of 70.9% was achieved while maintaining a 13.6% Ni grade into the final copper/nickel concentrate.

The primary copper/nickel roughers recovered 84.4% Ni at a reasonable grade (6.2% Ni), primarily as a result of the additional CMC and finer grind.

Pyrrhotite roughers recovered most of the remaining nickel, although mineralogy shows that much of this is nickel in solid solution within pyrrhotite. The pyrrhotite cleaners were no more efficient than in test F6, with relatively poor selectivity between pentlandite and pyrrhotite; this will be an area for optimization in further tests.

Note that like F6, the copper/nickel cleaner circuit rejected a reasonable amount of nickel, and once more it must be remembered that this stream will recycle back to the regrind mill for reprocessing.

Table 16-10: Summary of Results for Cleaner Flotation Test F7

			Assays, %, g/t			% Distribution		
Test Condition	**Product**	**Wt %**	**Cu**	**Ni**	**S**	**Cu**	**Ni**	**S**
F7	Cu/Ni 1st Cl. Conc. 1	1.17	10.20	14.40	28.7	77.9	26.8	6.95
K_{80}=102 µm	Cu/Ni 1st Cl. Conc. 1-2	2.10	6.32	15.90	28.2	86.3	52.9	12.2
Cu/Ni Ro	Cu/Ni 1st Cl. Conc. 1-3	2.83	4.82	14.71	27.3	88.7	66.0	15.9
pH 9.5	Cu/Ni 1st Cl. Conc. 1-4	3.29	4.19	13.61	26.4	89.5	70.9	18.0
350 g/t CMC	Cu/Ni Rougher Conc.	8.6	1.63	6.18	18.1	91.4	84.4	32.3
Reg K_{80}= 51 µm								
	Po Rej 1st Cl. Conc. 1	1.8	0.22	1.00	21.5	2.6	2.9	8.1
Po Rougher	Po Rej 1st Cl. Conc. 1-2	2.9	0.16	0.89	22.6	3.0	4.1	13.6
pH 7.0, 200 g/t Na_2SO_3	Po Rej 1st Cl. Conc. 1-3	4.6	0.11	0.78	23.2	3.4	5.6	21.8
Reg K_{80} = 27 µm	Po Rougher Conc.	15.2	0.05	0.50	20.5	5.2	12.0	64.1
	Rougher Tail	76.2	0.01	0.03	0.23	3.47	3.62	3.61
Head (calc.)		100.0	0.15	0.63	4.85	100.00	100.0	100.0

Test F9

The kinetics of nickel recovery in the cleaner circuit of the initial cleaner tests were observed to be relatively slow with F7 requiring 10 minutes of flotation time to reach a stage Ni recovery of 84.0%. It was suggested that the early rougher concentrates (i.e., 0-3 minutes) were likely to be highly liberated, fast floating pentlandite that may not benefit from regrinding and could be "crowding" the slower floating material in the cleaner circuit. As a result, F9 divided the copper/nickel Roughers into separate "A" and "B" stages, with only the B concentrate reporting to the regrind mill (see Figure 16-4). The A concentrate was sent to a single stage of cleaning intended to reject any liberated, but entrained, silicates reporting to this stream. At the same time, the pyrrhotite flotation circuit was modified to substitute SIPX for PAX in an effort to improve the post-regrind separation of any liberated pentlandite in the cleaner circuit feed.

Figure 16-4: Flowsheet for Cleaner Flotation Test F9

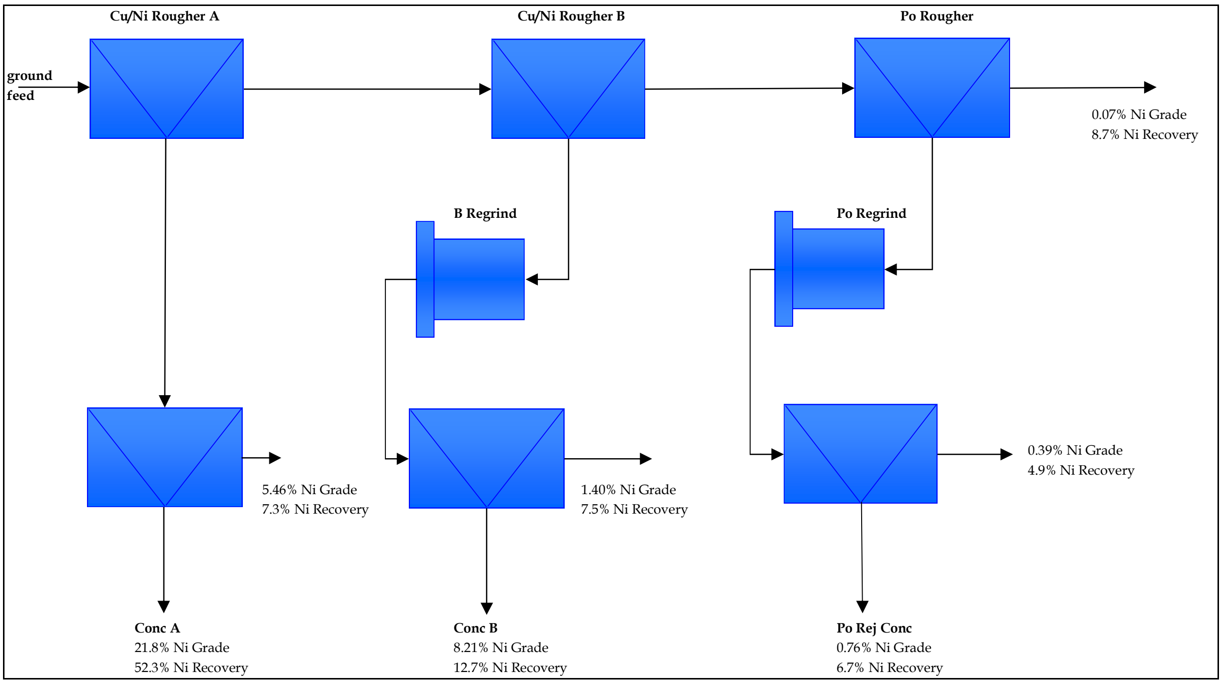


Table 16-11: Summary of Results for Cleaner Flotation Test F9

			Assays % (g/t)			Distribution %		
Test Condition	**Product**	**Wt %**	**Cu**	**Ni**	**S**	**Cu**	**Ni**	**S**
F9	Cu/Ni A 1st Cl. Conc.	1.45	7.09	21.80	30.4	79.0	52.3	9.62
K_{80} =100 µm	Cu/Ni A Ro. Conc. 1-2	2.26	4.73	15.96	23.5	81.9	59.6	11.6
Cu/Ni Ro	Cu/Ni B 1st Cl. Conc. 1	0.34	1.78	10.80	29.2	4.7	6.1	2.2
pH 9.5	Cu/Ni B 1st Cl. Conc. 1-2	0.71	1.13	9.24	29.2	6.1	10.8	4.5
350 g/t CMC	Cu/Ni B 1st Cl. Conc. 1-3	0.9	0.91	8.21	28.6	6.5	12.7	5.8
Reg K_{80} = 25 µm	Cu/Ni A+B 1st Cl. Conc.	2.4	4.67	16.47	29.7	85.5	65.0	15.4
(B con only)	Cu/Ni B Ro. Conc. 3-5	4.2	0.27	2.93	17.5	8.7	20.2	15.9
Po Rougher	Cu/Ni A-B Ro. Conc.	6.4	1.84	7.51	19.6	90.6	79.8	27.5
pH 7.0, 75 g/t SIPX	Cu/Ni A-B Ro. Tail	93.6	0.01	0.13	3.6	9.4	20.2	72.5
Reg K_{80} = 27 µm	Po Rej 1st Cl. Conc. 1	1.1	0.08	0.93	27.5	0.7	1.7	6.8
	Po Rej 1st Cl. Conc. 1-2	3.0	0.06	0.78	25.1	1.3	3.9	16.5
	Po Rej 1st Cl. Conc. 1-3	5.3	0.05	0.76	25.7	2.0	6.6	29.5
	Po Rougher Conc.	12.9	0.03	0.54	19.86	3.21	11.54	55.64
	Rougher Tail	80.7	0.01	0.07	0.96	6.19	8.67	16.88
Head (calc.)		**100.0**	**0.13**	**0.60**	**4.59**	**100.00**	**100.0**	**100.0**

Results: A superior concentrate grade of 21.8% Ni was realized from the A cleaner concentrate, giving clear indication that a regrind was not necessary for this fast floating concentrate. Stage recovery in the A cleaner was good at 87.8%, with the option of sending the A cleaner tails to the B cleaners where additional flotation time would improve recovery still further. The B cleaners demonstrated room for improvement, with only 62.9% Ni stage recovery. However, the B concentrate grade of 8.21% Ni was considered to be higher than it needed to be, given the grade of the A concentrate, so additional cleaner flotation time, and/or scavenging would be expected to improve recovery. In the pyrrhotite circuit the results were still poor, with no semblance of a saleable final product.

Test F10

Positive gains achieved in the cleaner circuit configuration in test F9 were combined with favourable rougher results observed in F8 (soda ash in place of lime) to generate the test conditions for F10. As shown in Figure 16-5, the Copper/nickel Roughers were again split into “A” and “B” stages, but the A cleaner tail was combined with the B cleaner concentrate after the regrind to serve as feed to the B 1st cleaner flotation. A second cleaner stage and a first cleaner scavenger were also added to the B circuit. For this test the pyrrhotite flotation circuit was dropped from the flowsheet, as the separation had not shown much promise in previous tests. As in F8, soda ash was substituted for lime throughout the test.

Results: Overall nickel recovery to the combined A + B concentrate was 75.1% at a grade of 17.9% Ni (see Table 16-12). At the same time, copper recovery to the final product was 89.2%. Improved rejection of silicates was observed in both the rougher and cleaner circuits due to the addition of soda ash.

Figure 16-5: Flowsheet for Cleaner Flotation Test F10

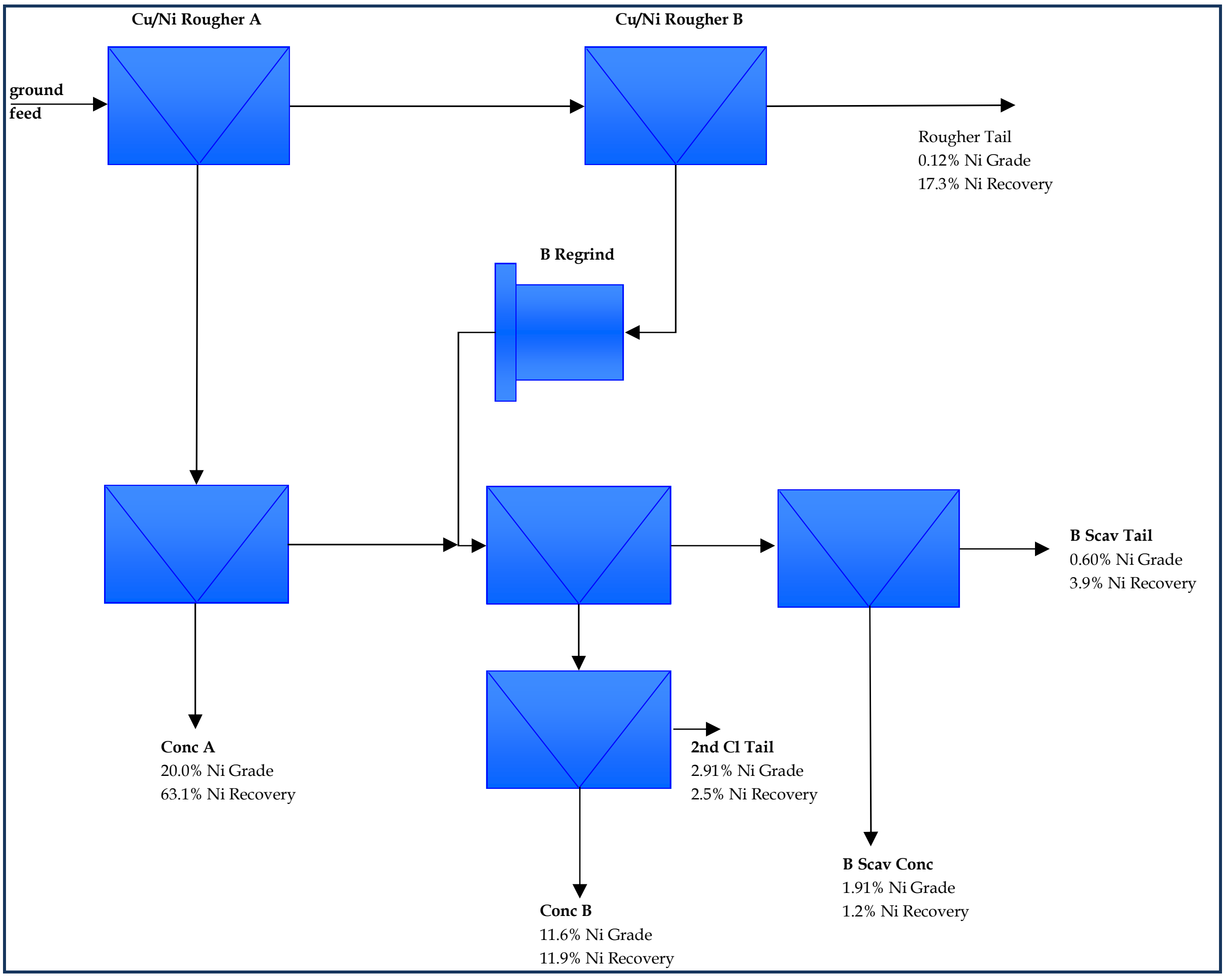


Table 16-12: Summary of Results for Cleaner Flotation Test F10

Test Condition	Product	Wt %	Assays % (g/t)			Distribution %		
			Cu	Ni	S	Cu	Ni	S
F10	Cu/Ni A 1st Cl. Conc.	2.02	5.72	20.0	30.8	82.4	63.1	12.7
K_{80}=100 µm	Cu/Ni B 2nd Cl. Conc.	0.65	1.56	11.6	29.4	7.3	11.9	3.9
Cu/Ni Ro	Cu/Ni B 1st Cl. Conc.	1.20	0.95	7.66	26.8	8.1	14.4	6.6
pH 9.5 (Na_2CO_3)	Cu/Ni B 1st Cl. Tail	4.58	0.09	0.72	13.4	2.9	5.1	12.6
350 g/t CMC	Cu/Ni A+B Combined Conc.	2.7	4.70	17.9	30.5	89.7	75.1	16.7
Reg K_{80}= 25 µm	Cu/Ni A+B Rougher Conc.	7.8	1.68	6.77	20.0	93.4	82.7	31.9
(B con only)	Rougher Tail	92.2	0.01	0.12	3.60	6.59	17.3	68.1
Head (calc.)		100.0	0.14	0.64	4.88	100.00	100.0	100.0

16.3.3 LOCKED CYCLE TEST

The optimised flowsheet used for test F10 was judged to be an adequate basis for the first locked cycle flotation test, LCT-1. The flowsheet was identical except that the B 2nd cleaner tails of each cycle were added to the B 1st cleaner feed of the next cycle, and the B 1st cleaner scavenger concentrate for each cycle was sent to the batch regrind of the next cycle. In addition, a pyrrhotite flotation step was added to the B Rougher tailings in the last two cycles in order to generate a low sulphur tailings product for environmental testing. The locked cycle test flowsheet is illustrated in Figure 16-6.

The test consisted of six consecutive cycles, with all products assayed for copper, nickel, and sulphur. In addition, final products from the last two cycles were assayed for cobalt, and the tailings from the last two cycles were submitted for size analysis. In general, the test demonstrated a good level of stability from the third cycle onwards. A metallurgical projection based on cycles C to F is presented in Table 16-13 (Note: low sulphur tailings stream is calculated from cycles E and F only).

The final (combined A and B) concentrate from the test graded 16.5% Ni at a recovery of 78.4%. Copper and cobalt recoveries were 89.1% and 63.5%, respectively. A low sulphur tailings stream was generated grading 0.65% S and representing ~79% of the overall mass of the sample. Nickel losses to the pyrrhotite tailings and scavenger tailings were 10.3% and 6.7%, respectively.

Figure 16-6: Flowsheet for Locked Cycle Test LCT-1

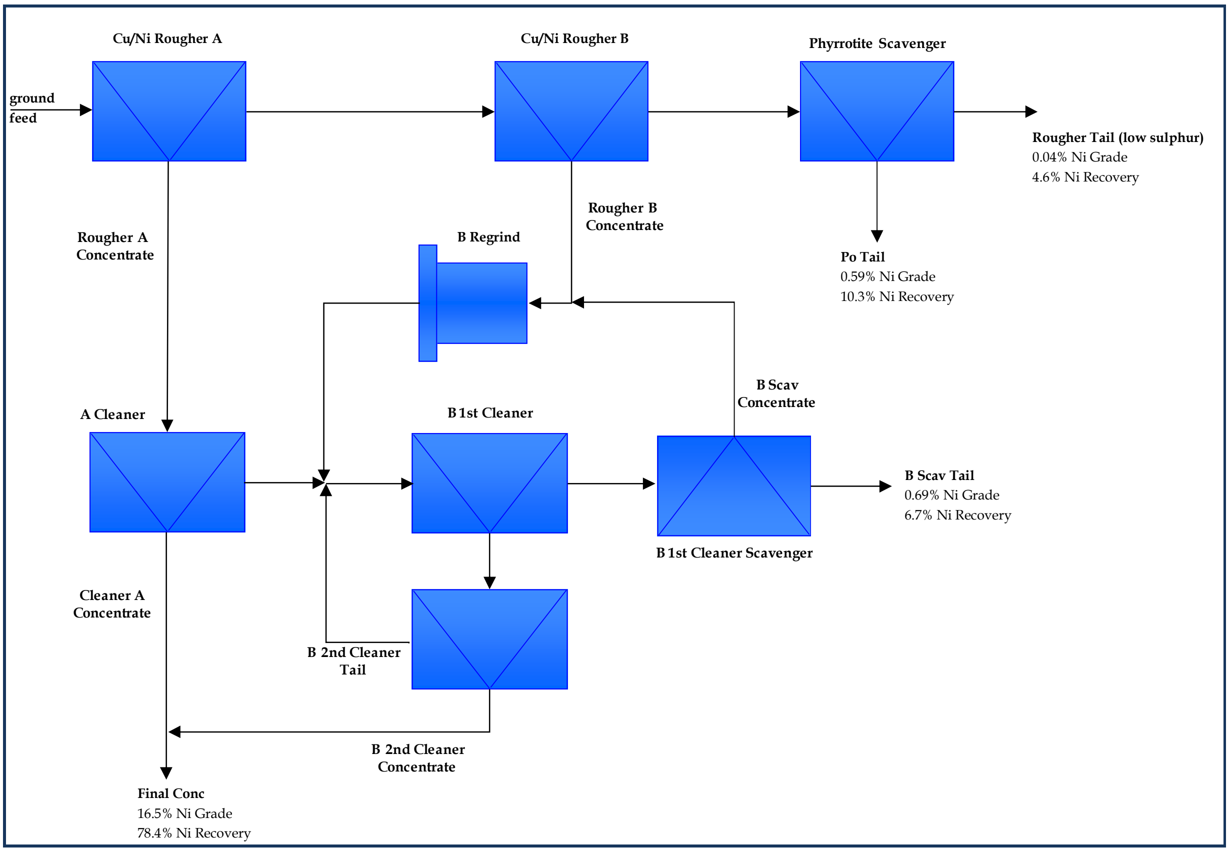
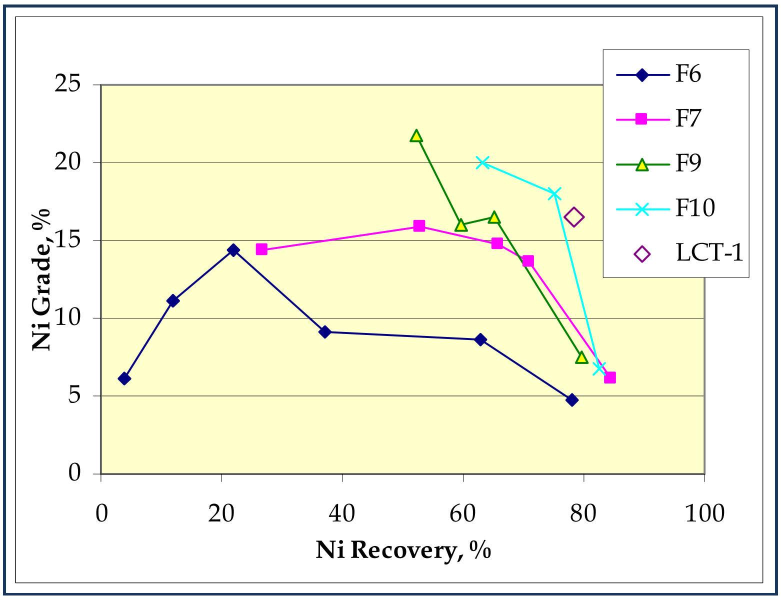


Table 16-13: Metallurgical Prediction Based on LCT-1

	Weight	Assays,%				Distribution%			
Product	%	Cu	Ni	S	Co	Cu	Ni	S	Co
Final Concentrate	3.0	4.21	16.5	28.3	0.740	89.1	78.4	18.5	63.5
Cleaner Scav. Tail	6.1	0.05	0.69	11.3	0.030	2.1	6.7	15.0	6.0
Pyrrhotite Tail	12.1	0.03	0.59	23.3	0.024	2.9	10.3	56.2	8.3
Low Sulphur Tail	78.9	0.01	0.04	0.65	0.010	5.9	4.6	10.3	22.2
Head (calc.)	100.0	0.14	0.63	4.62	0.035	100.0	100.0	100.0	100.0

In Figure 16-7 cleaner concentrate grade-recovery curves are plotted for the batch flotation tests. In comparison, a single point is plotted representing the projected performance from the locked cycle test. The graph indicates a slight improvement in nickel recovery over F10 with the recycling of the intermediate streams in the locked cycle test.

Figure 16-7: Comparison of Ni Grade vs. Recovery for Batch Cleaner and Locked Cycle Tests



In Figures 16-8 and 16-9 cleaner circuit separation curves are plotted for Pentlandite-Pyrrhotite and Pentlandite-Gangue, respectively. Superior results in the batch tests are realized in F10 where soda ash was substituted for lime in the rougher and cleaner circuits. The evident effect is one of improved dispersion and lower pulp viscosity resulting in reduced entrainment throughout the circuit. Operating points for the locked cycle test are included in both graphs and illustrate still further improvement with the recycling of the intermediate products and subsequent gains in overall nickel recovery.

Figure 16-8: **Comparison of Pn/Po Separation for Batch Cleaner and Locked Cycle Tests**

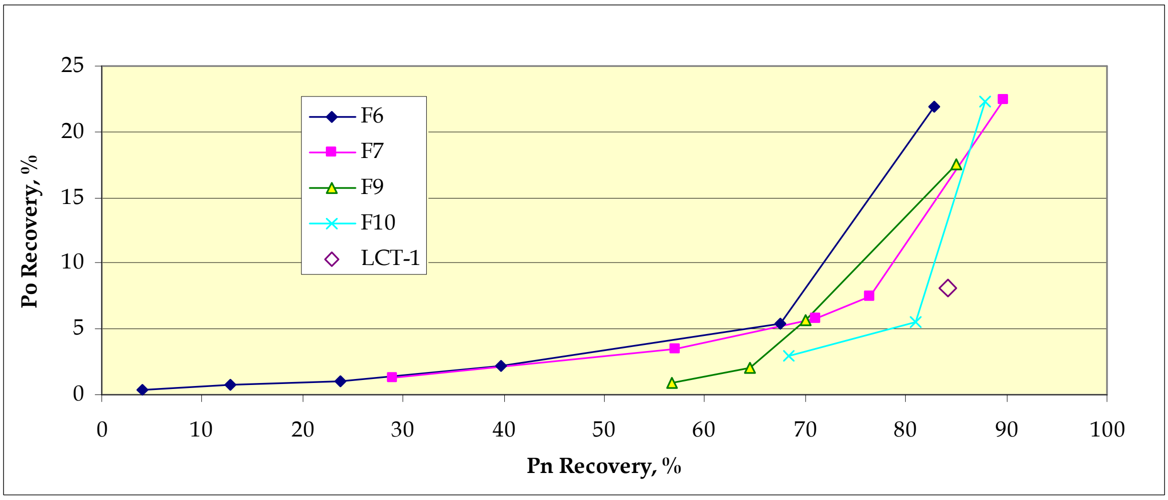
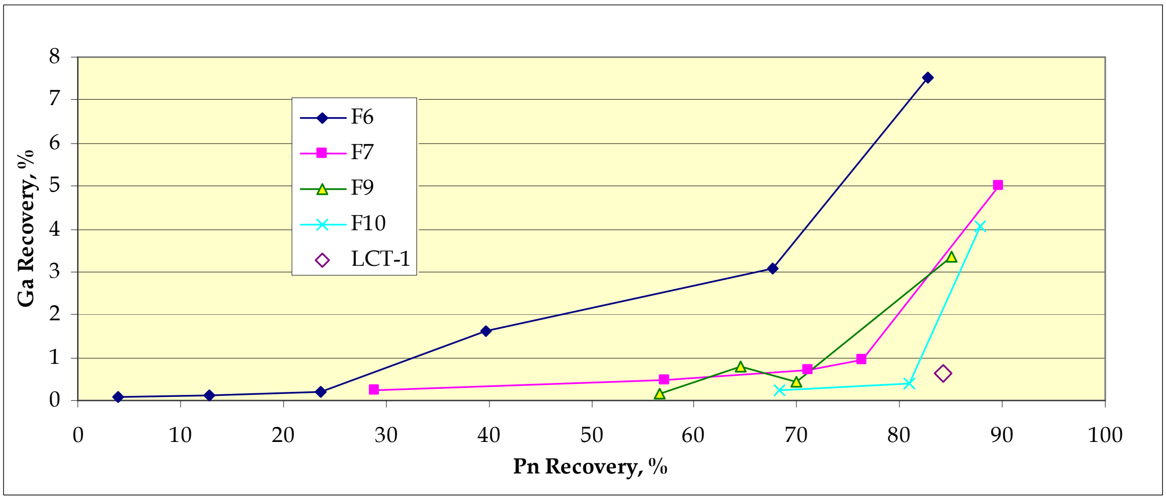
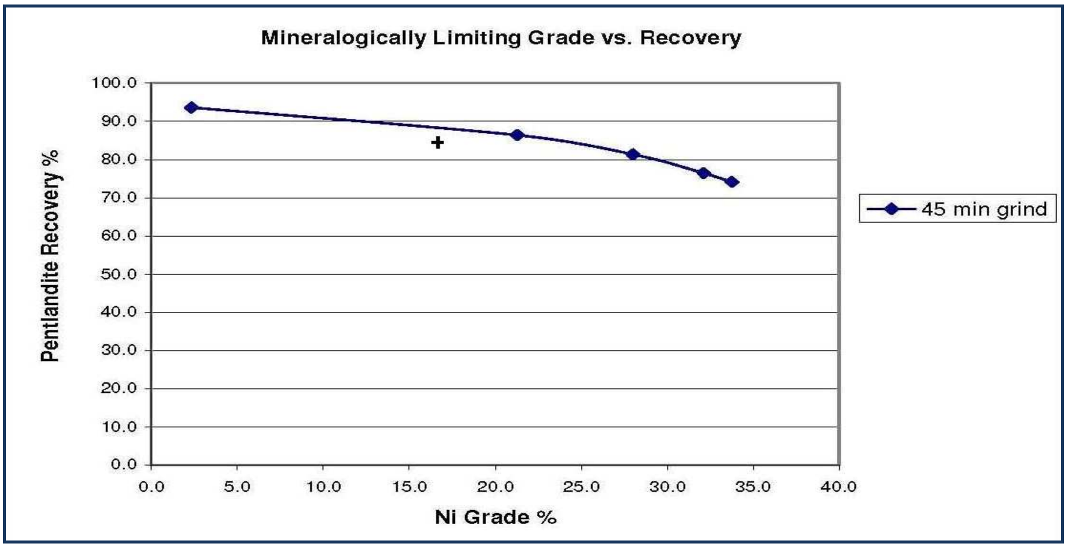


Figure 16-9: Comparison of Pn/Ga Separation for Batch Cleaner and Locked Cycle Tests



The mineralogically limiting grade-recovery curve from the mineralogy work on the head sample is presented in Figure 16-10. The line represents an estimate of a "best case scenario" pentlandite recovery, limited only by the liberation of the pentlandite mineral itself. A "+" has been added to the graph to indicate the metallurgical projection from LCT-1. The difference between the locked cycle operating point and the curve may indicate some opportunity for additional pentlandite recovery from the B 1st Cleaner Scavenger Tailings or the pyrrhotite Tailings stream. It should be noted, however, that the grade-recovery curve presented here is based on QEMScan analysis of only two polished sections, and as a result is likely to contain a certain degree of error based on the limited sampling size. Further mineralogical analysis is required to bring additional confidence to this result.

Figure 16-10: Optimum Grade-Recovery-Curve



16.3.4 *FINAL PRODUCT CHARACTERIZATION*

A sample of the combined final concentrate from cycle E of the locked cycle test was submitted for minor element analysis. The work consisted of a multi-element ICP scan coupled with individual assays for Cl, F, SiO_2, and Hg. Results of the analysis did not reveal any elevated concentrations of deleterious elements. Full details of the analysis are found in Appendix A.

Acid-Base Accounting testing of the Low Sulphur Tailings from the locked cycle test indicated a Neutralizing Potential/Acidifying Potential (NP/AP) ratio of 3.3 indicating a “Low” possibility of the tailings being acid generating. By comparison, and NP/AP ratio of 4.1 or higher is considered to have no possibility of being acid generating. Since earlier batch testwork indicated that tailings sulphur grades as low as 0.23% are achievable (compared to 0.65% S in this sample) it is reasonable to assume that an NP/AP ratio higher than 3.3 could be achieved, should it be required.

16.4 HEAVY LIQUIDS SEPARATION

A 5-kg sample of coarsely crushed (-½”) composite was submitted for heavy liquids separation testwork. This test is used to help metallurgists assess the extent to which a gravity-based preconcentration process will be economically viable.

Preconcentration processes, such as Dense Medium Separation, are used to reject less-dense material from diluted mill feeds. Where the “floats” fraction consists primarily of mining dilution and barren (or sub-economic) blocks, the process can often have a significant positive effect on process economics.

Results of this preliminary test are shown in Table 16-14.

Table 16-14: Summary of Heavy Liquids Testing on -1/2" Nickel King Composite

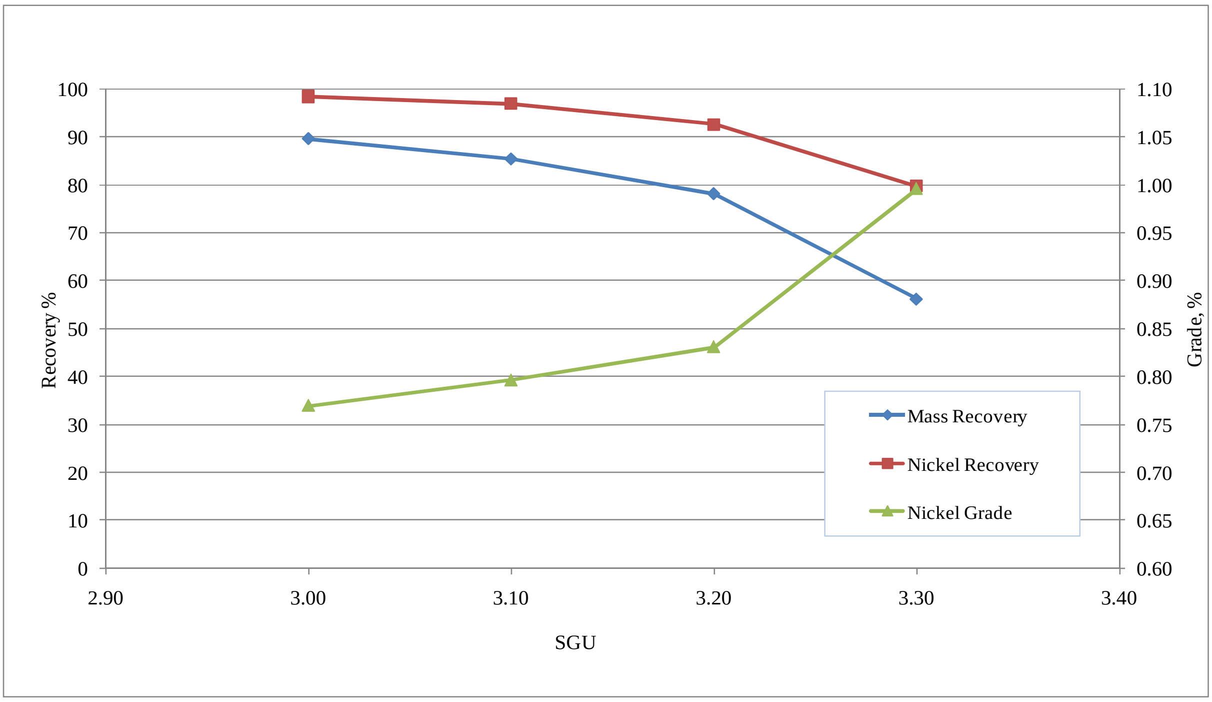
	Individual Data								Cumulative Data							
	Weight		Grade %			Recovery %			Weight		Grade %			Recovery %		
	kg	%	Cu	Ni	S	Cu	Ni	S	kg	%	Cu	Ni	S	Cu	Ni	S
-1 mm	761.5	15.2	0.2	1.01	7.50	20.1	21.9	22.1	761.5	15.2	0.20	1.01	7.50	20.1	21.9	22.1
3.3 Sink	2047.5	40.9	0.21	0.99	7.30	56.6	57.8	57.8	2809	56.1	0.21	1.00	7.35	76.7	79.7	79.9
3.2 Sink	1100.2	22.0	0.09	0.41	3.02	13.3	12.9	12.9	3909	78.1	0.17	0.83	6.13	90.0	92.6	92.8
3.1 Sink	362.3	7.2	0.12	0.42	2.97	5.7	4.3	4.2	4272	85.3	0.17	0.80	5.87	95.7	96.9	97.0
3.0 Sink	212.2	4.2	0.05	0.24	1.69	1.4	1.5	1.4	4484	89.6	0.16	0.77	5.67	97.1	98.4	98.3
3.0 Float	522.8	10.4	0.04	0.11	0.82	2.9	1.6	1.7	5007	100	0.15	0.70	5.16	100.0	100.0	100.0
Head	5006.5	100.0	0.15	0.70	5.16	79.9	78.1	77.9	-	-	-	-	-	-	-	-

The sample was first de-slimed at 1 mm. The -1 mm "fines" fraction was weighed and assayed (this stream would, in practice, bypass the preconcentration step directly to mill feed). The remaining +1mm material was submersed in heavy liquids, with the sinks fraction recovered for weighing and assay and the floats collected for subsequent tests (at lower densities). The charts below indicate the cumulative effect of these individual separations.

Looking at Figure 16-11 it can be seen that for this sample at a density cut point of 3.10, one could expect a mass rejection of 15% while maintaining 96.9% Ni and 95.7% Cu. The low-grade mass rejection effectively raises the mill feed grade by 14%, from 0.7% Ni to 0.8% Ni.

It should be noted that these results are obtained using the composite sample of 0.70% Ni grade and that current predictions for "average grade" are closer to 0.4% Ni. Given this, further work is encouraged and should attempt to correctly represent the additional dilution (i.e., inclusion of low grade blocks or normal mining dilution).

Figure 16-11: DMS Washability Curve for Nickel King Composite at -1/2"



17.0 MINERAL RESOURCE ESTIMATE

PEG has produced a resource estimate on the Nickel King Main Zone Deposit. Gemcom software GEMS 6.41™ was used for the resource estimate in combination with Sage 2001 for the variography. The metals of interest at Nickel King are nickel, copper, and cobalt. Minor amounts of palladium, platinum, gold, and silver are also present.

Strongbow provided a digital drill hole database in a series of Microsoft Excel spreadsheets (XLS) consisting of collar, survey, lithology, assay, and specific gravity information. Upper and lower contact points for the norite were also provided as a point file in XLS format. Topographical information was supplied in AutoCAD DXF format for contour lines, lakes, and rivers. All lines provided were exported from MapInfo at a 0 elevation. Strongbow provided an elevated point file for the lakes and contour line to facilitate the creation of the topographical surface in 3D. Additional information provided was in the form of PDF files for the surface geological interpretation of the two sills, and a drawing showing the location of the drill section lines and the drill-hole collar-location. A series of wireframes constructed by Strongbow in Discover3D™, representing the outline of the nickel mineralization at various nickel cut-offs, were also provided. The digital drill hole database included all holes drilled to the end of the 2008 winter/spring drill program for the Main Zone, Koona Zone, Ring area, Kizan area, South Kizan area, and Joe Island trend.

The Nickel King drill database includes 97 diamond drill holes totalling 19,258 m of core. During 2007 and 2008, Strongbow drilled 66 holes totalling 13,481 m or 70% of the drill database. The remaining holes consisted of historical drilling performed by the Canadian Nickel Company (CANICO) totalling 3,529 m in 18 holes (18%), and, more recently, holes from Aber Resources in 1995 totalling 2,249 m in 13 holes (12%). Of these, 70 drill holes were located within the Main Zone area and were used in the resource estimation, including 13 CANICO holes, which were used only to confirm continuity of mineralization but not for grade determination, due to the limited sampling/assay information. Table 17-1 shows a summary of the number of holes used in the resource estimate. The complete drill hole listings used in the resource are provided in Appendix A.

Table 17-1: Summary of Holes in the Nickel King Database

	No. of Holes	Total Metres	% of Total Database	Use
Strongbow Holes Located on Main Zone	49	11,047	57	Geology, grade & resource classification
Aber Resources Drilling on Main Zone	13	2,249	12	Geology, grade & resource classification
CANICO Drilling on Main Zone	14	3,033	16	Geology & resource classification
Subtotal	**76**	**16,328**	-	-
Strongbow Holes Outside Main Zone	17	2,434	13	Not used
CANICO Drilling outside Main Zone	4	496	3	Not used
Subtotal	**21**	**2,930**	-	-
Total in Digital Database	**97**	**19,258**	**100**	-

17.1 GEOLOGICAL INTERPRETATION

The 3D wireframes developed for the resource model to control the grade interpolation were based upon lithologies as opposed to nickel grade shells.

The geological wireframe for the Upper and Lower Sills utilized the drill hole intercepts with the norite horizons as provided by Strongbow. The wireframe also incorporates the current geological interpretation of the fold structure on the surface maps. The Upper and Lower Sill contacts were drawn on a set of radial sections fanning from west to east and originating from a central point located at approximately 526475E and 6679225N. The sections were designed to be about 50 m to 60 m apart in the central portion of the Lower Sill at lake elevation. Due to the radial configuration, the spacing for the Lower Sill was only about 40 m to 50 m. The wireframe construction was carried out in multiple steps as follows:

- The surface expression of the Upper and Lower Sills provided by Strongbow was digitized on plan view and elevated to the topographical surface.
- Lines describing the upper and lower contacts of the sills were digitized on the radial sections using the top and bottom intercept points of the drill hole lithological information as reference. The fold nose was modelled as part of the Lower Sill wireframe.
  The model was constructed using hanging wall/footwall surfaces trimmed to the topography in the up-dip extension and trimmed to the extent of the interpreted fold nose in the down-dip direction.
- The hanging wall and footwall surfaces of both sills were then stitched together into a 3D solid resulting in the Upper and Lower Sill lithological model of the Nickel King deposit.
- The main fault crossing the deposit on the east side was digitized on plan and interpolated vertically at depth.
- The Lower Sill east portion was modelled traditionally using rings and tie lines on a set of custom section lines drawn through the available drill holes. The Lower Sill east model was projected through the fault line.
- The Lower Sill east and west portion were subsequently trimmed to the fault surface.

The topography surface was constructed using the elevated contour lines provided by Strongbow. The bottom of the overburden surface was created by first flagging the holes drilled on ice versus the holes drilled on land. The difference between the elevation of the hole collar and the bottom of the casing indicated an average water + overburden depth of 13.61 m. The average overburden depth on land was 4 m. The bottom of overburden surface was created following these steps:

- A line was digitized in the middle of the lake at 382.39 m elevation (396 m water elevation minus the average water + overburden depth of 13.61 m).
- A second line was digitized on the crest of the hills at a slightly higher elevation than the last contour line on the slope of the hills.
- The bottom of casing X, Y, Z data point locations were added to the data set along with the lake edge topographical line.
- The surface was compared with the drill hole information on sections for fit. For localized areas, where the overburden surface protruded through the topography, the overburden surface was adjusted downward to a depth of 3 m below the topography.

The overburden/bedrock surface is considered a good approximation taking into account the data that was available.

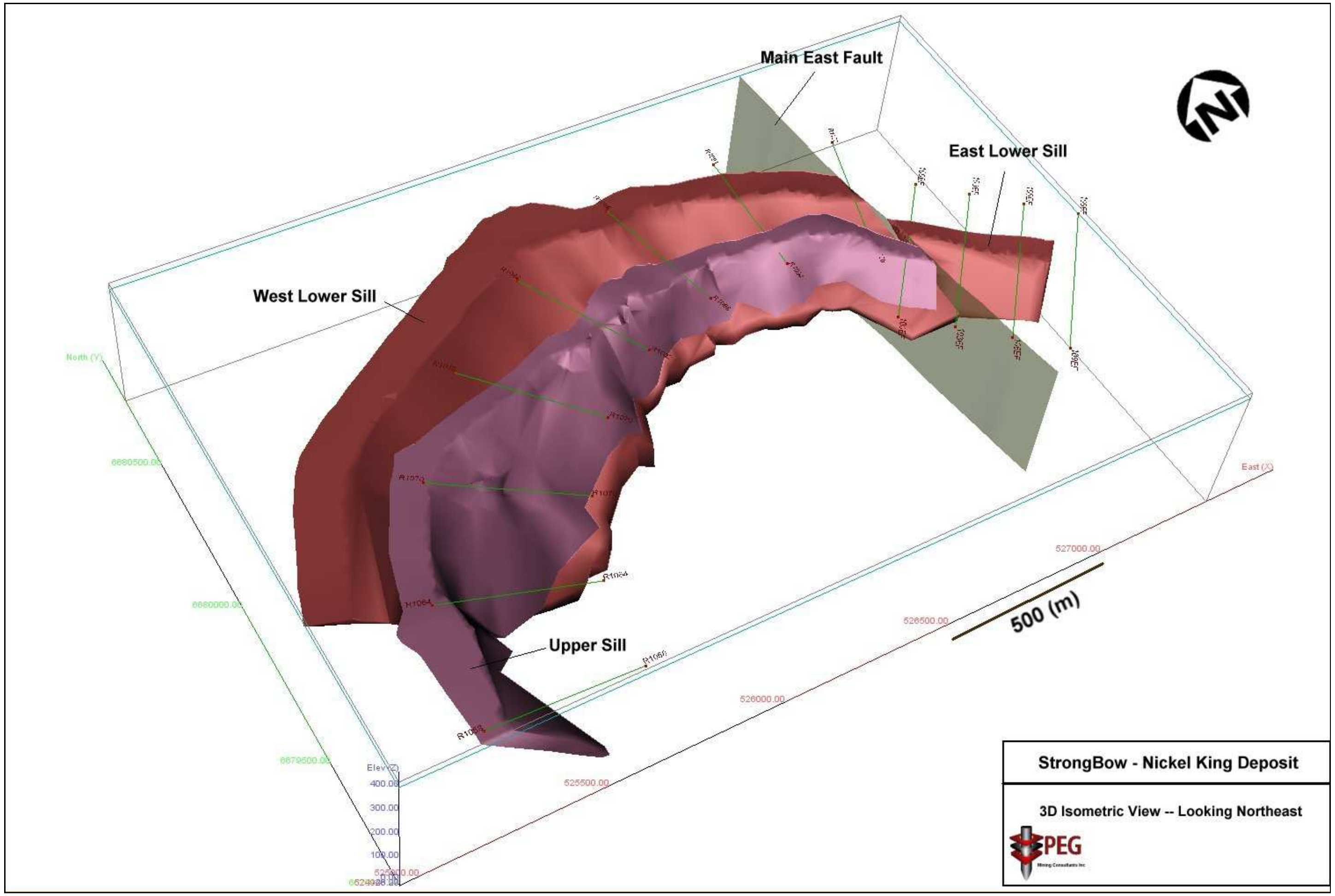
The lithological wireframes were clipped to the overburden surface for volume generation and display purpose. The average thickness for the Upper Sill and Lower Sill is 67 m and 57 m respectively (Figure 17-1). Table 17-2 shows the total volume of the wireframes.

Table 17-2: Wireframe Volume

Sill Name	Volume (M m^3)
Lower Sill – West	71
Lower Sill – East	3
Upper Sill	29
Total Volume	**102**

Figure 17-1 shows the completed geological models.

Figure 17-1: Geological Model with Section Lines Configuration Viewing North-East



17.2 EXPLORATORY DATA ANALYSIS

Exploratory data analysis is the application of various statistical tools to characterize the statistical behaviour or grade distributions of the data set. In this case, the objective is to understand the population distribution of the grade elements in the various units using such tools as histograms, descriptive statistics, and probability plots.

17.2.1 ASSAYS

PEG evaluated the raw assay statistics for the Main Zone separately for each sill. The statistics show that the wireframe captures most of the mineralization leaving very little material outside the Upper and Lower Sills.

The raw assay-statistics box-plots show that the Upper Sill is lower grade than the Lower Sill. The difference in grade is related to the higher values in the interpreted fold nose of the deposit, which form part of the Lower Sill model. Figure 17-3 provides the mean values for the sills.

Table 17-3: Raw Assay Statistics (excludes CANICO holes)

	Upper Sill			Lower Sill		
	Ni %	Cu %	Co (ppm)	Ni %	Cu %	Co (ppm)
Valid Cases	1,869	1,862	1,867	1,932	1,927	1,928
Mean	0.235	0.058	131.791	0.315	0.073	140.628
Variance	0.065	0.005	25,987.769	0.136	0.007	21,354.359
Std. Deviation	0.254	0.073	161.207	0.369	0.082	146.131
Variation Coefficient	1.085	1.252	1.223	1.170	1.113	1.039
rel. V. Coefficient (%)	2.509	2.901	2.831	2.663	2.535	2.367
Minimum	-	0.000	0.500	-	-	0.500
Maximum	1.817	0.542	920.000	3.872	0.624	1,510.000
1st percentile	0.004	0.002	5.000	0.004	0.002	5.000
5th percentile	0.014	0.004	19.300	0.015	0.004	17.200
10th percentile	0.022	0.006	23.800	0.020	0.005	20.190
25th percentile	0.056	0.013	40.800	0.044	0.010	34.900
Median	0.151	0.032	80.000	0.193	0.045	93.650
75th percentile	0.304	0.071	140.000	0.479	0.111	200.000
90th percentile	0.561	0.139	300.000	0.726	0.177	300.840
95th percentile	0.805	0.234	537.720	0.975	0.232	410.000
99th percentile	1.186	0.341	790.000	1.668	0.366	670.000

Frequency distribution shows a log normal distribution with 90% of the nickel values below 0.5 Ni% for the Upper Sill, and 90% of the nickel values below 0.75 Ni% for the Lower Sill. Complete raw assay statistics are provided in Appendix B.

Correlation tables show Ni-Cu correlation to be high with a correlation factor R^2 of 0.9. The Ni-Co correlation in the Lower Sill is very strong (R^2 of 0.99) while the Ni-Co correlation in the Upper Sill shows a bi-modal population, high Co/Ni ratio and low Co/Ni ratio. Since the Co is a minor component of the overall value of the Nickel King deposit the spatial distribution of these two populations have not been investigated further.

17.2.2 CAPPING

A combination of decile analysis and a review of probability plots were used to determine the potential risk of grade distortion from higher-grade assays. A decile is any of the nine values that divide the sorted data into ten equal parts so that each part represents one tenth of the sample or population. In a mining project, high-grade outliers can contribute excessively to the total metal content of the deposit.

Typically, in a decile analysis, capping is warranted if:

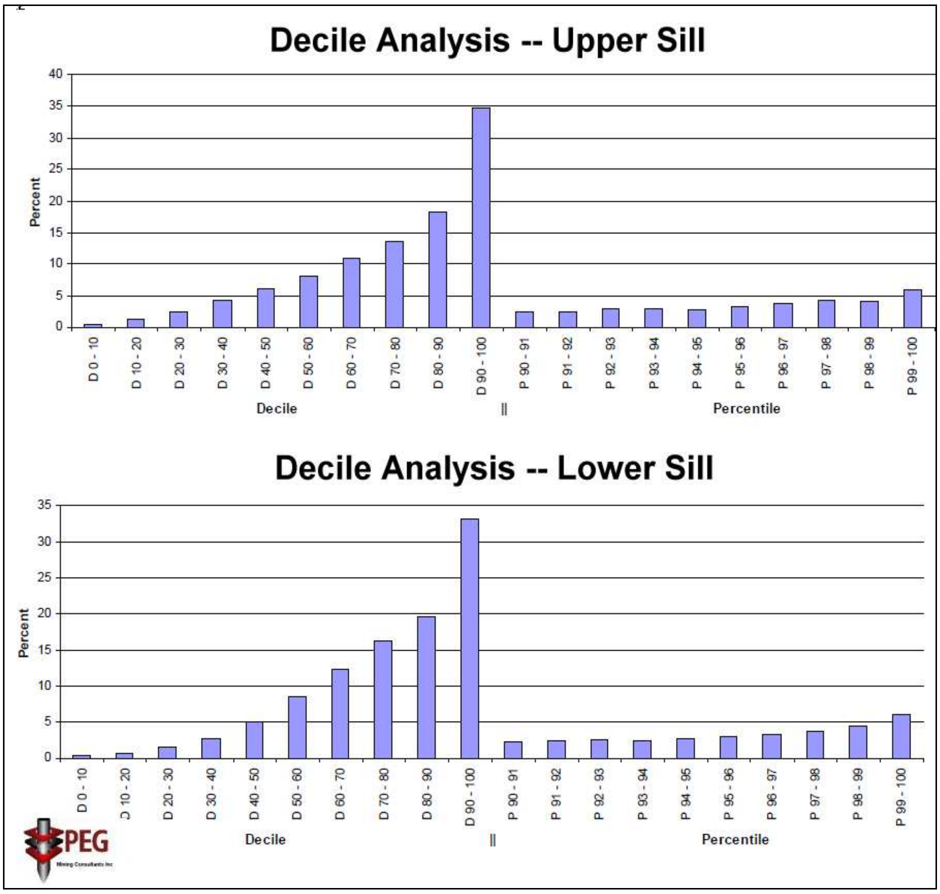
- the last decile has more than 40% of metal
- the last decile contains more than 2.3 times the metal quantity contained in the one before last
- the last centile contains more than 10% of metal
- the last centile contains more than 1.75 times the metal quantity contained in the one before last.

The decile analysis results shown in Figure 17-2 indicated that grade capping was not warranted for the nickel grade in any of the zones. For copper and cobalt, the Upper Sill assays were capped at 0.45% and 835 ppm respectively, while the Lower Sill assays were left uncapped. Table 17-4 shows the capping level used for the resource model. Results from the decile analysis are included in Appendix C.

Table 17-4: Capping Level Used

	Ni (%)	Cu (%)	Co (ppm)
Upper Sill	No Cap	0.45 Cap	835 Cap
Lower Sill	No Cap	No Cap	No Cap

Figure 17-2: Decile Analysis Results for Nickel in the Upper and Lower Sill



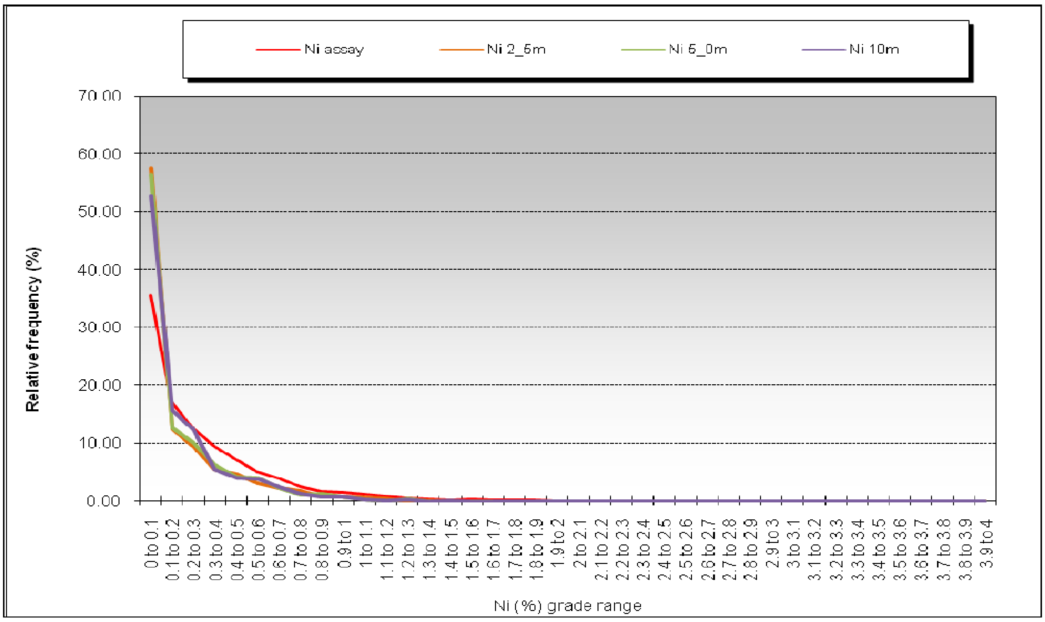
17.2.3 COMPOSITES

Sampling Length Statistics and Composites

Sampling intervals on the Nickel King property average 1.0 m. The upper third quartile of the sampling length shows a value close to 1.0 m. PEG evaluated composite intervals of 2.5 m, 5 m, and 10 m. Results show that beyond 0.25% Ni the composite grade has almost the same distribution regardless of the composite interval selected.

PEG elected to use a composite length of 2.5 m, generating about four data points per block in the 10 x 10 x 10 m block matrix selected while allowing grade variations to be represented. Assays were length-weighted averaged and any grade capping was applied to the raw assay data prior to compositing.

Figure 17-3: Relative Frequency Distribution for Grade Composited at Various Intervals



Composite intervals were created down from the collar of the holes toward the hole bottoms. The composite length was automatically adjusted by the software leaving no small remnants at the intersection of the wireframes. The final composite length averaged 2.54 m and 2.56 m for the Lower Sill and Upper Sill respectively. Complete composite statistics are provided in Appendix D.

17.2.4 EVALUATION OF THE ZERO GRADE COMPOSITES

During the compositing process, un-sampled drill core intervals were composited at zero grade. Upon examination of the composite data, a significant number of zero grade composite existed in the database; these were mostly associated with the old CANICO drill holes which were sporadically assayed only when there was significant sulphide mineralization encountered. Because of this, and in consultation with Strongbow's geological team, PEG elected to use the CANICO holes only to demonstrate continuity up-dip from the interpreted fold nose area, and to assist in the construction of the geological wireframe of the Upper and Lower Sills. No CANICO holes were used in the grade interpolation of the resource model.

Composite statistics are shown in Table 17-5 and details of the composite statistical analysis in Appendix D.

Table 17-5: Composite Statistics (excludes CANICO holes)

	Upper Sill			Lower Sill		
	Ni (%)	Cu (%)	Co (ppm)	Ni (%)	Cu (%)	Co (ppm)
Valid Cases	1137	1137	1137	1049	1049	1049
Mean	0.142	0.035	79.275	0.220	0.051	97.736
Variance	0.042	0.003	16,506.437	0.089	0.005	14,889.186
Std. Deviation	0.205	0.056	128.477	0.298	0.068	122.021
Variation Coefficient	1.442	1.606	1.621	1.358	1.333	1.248
Rel. V. coefficient (%)	4.278	4.763	4.806	4.193	4.115	3.855
Minimum	-	-	-	-	-	-
Maximum	1.319	0.384	917.373	1.874	0.447	793.462
1st percentile	-	-	-	-	-	-
5th percentile	-	-	-	-	-	-
10th percentile	-	-	-	-	-	-
25th percentile	-	-	-	0.005	0.001	4.749
Median	0.053	0.013	36.939	0.085	0.019	50.455
75th percentile	0.207	0.045	99.272	0.345	0.082	152.731
90th percentile	0.373	0.090	177.334	0.622	0.153	267.243
95th percentile	0.580	0.156	366.566	0.801	0.198	340.928
99th percentile	1.024	0.278	681.320	1.303	0.280	551.952

17.3 BULK DENSITY

The SG data provided by Strongbow consisted of 72 data points. Table 17-6 shows the mean value for the Upper and Lower Sills.

Table 17-6: SG Value for Upper and Lower Sill

Sill	Average Composite Grade Ni%	Number of Data Points	Mean SG Value (g/cm^3)
Lower Sill (LS)	0.220	24	3.53
Upper Sill (US)	0.142	39	3.33
Combined US+LS	0.179	53	3.43

The nickel scatter plots appear to indicate that using a regression to calculate the SG values based on the Ni content is possible. The R^2 is moderate at 0.675 for both zones combined. Individually the R^2 value for the Upper and Lower Sills are 0.52 and 0.75 respectively. A similar correlation on the scatter plots was also obtained using a combination of Ni+Cu+Fe assays with a R^2 of 0.63, as shown in Table 17-7, where the regression is defined by the following equation:

$$Y = A + B*X:$$

Table 17-7: Regression Results

Regression	R^2	A	B
Ni vs. SG	0.675	3.115	0.391
Cu vs. SG	0.510	3.121	1.595
Fe vs. SG	0.611	3.057	0.032
Ni+Cu+Fe vs. SG	0.627	3.053	0.030

The Ni vs. SG regression equation residuals were evaluated and showed a near normal distribution.

Final formulae selected to calculate the SG is:

SG in resource model = 3.11 + 0.391 * Ni% values in the Ordinary krige model

PEG elected to use a calculated SG based on the Ni grade in the resource model as opposed to using an average SG. This methodology is considered conservative since the average Ni grade of the composites is 0.179% for the Upper and Lower Sills, which based on the above equation, calculates to an SG of 3.18 g/cm^3. This is lower than the average SG of 3.43 g/cm^3 determined from all the data points.

The bulk density evaluation presented here is considered as a good approximation of the true bulk density using the data that was available; however, revision of the regression equation used will be necessary as more data points are collected during future drill programs.

Density outside the Upper and Lower Sills was set to a constant of 3.0 g/cm^3, equivalent to the textbook density of the norite.

17.4 SPATIAL ANALYSIS

17.4.1 VARIOGRAPHY

Geostatisticians use a variety of tools to describe the pattern of spatial continuity, or strength of the spatial similarity of a variable with separation distance and direction. The correlogram measures the correlation between data values as a function of their separation distance and direction. If we compare samples that are close together, it is common to observe that their values are quite similar and the correlation coefficient for closely spaced samples is near 1.0. As the separation between samples increases, there is likely to be less similarity in the values and the correlogram tends to decrease toward 0.0. The distance at which the correlogram reaches zero is called the range of correlation or simply the range.

The range of the correlogram corresponds roughly to the more qualitative notion of the range of influence of a sample; it is the distance over which sample values show some persistence or correlation. The shape of the correlogram describes the pattern of spatial continuity. A very rapid decrease near the origin is indicative of short scale variability. A more gradual decrease moving away from the origin suggests longer scale continuity.

Variography was conducted for the Upper and Lower Sills with a subset of the data between 525900 and 526425 E. The data was selected in an area were the drill density was high and the ore body relatively linear. For the purpose of this study, the variography conducted with this subset of the data is assumed representative of the entire Main Zone.

Using Sage 2001 software, directional sample correlograms were calculated for nickel, copper, and cobalt in each of the two statistical domains (Upper Sill and Lower Sill), along horizontal azimuths of 0, 30, 60, 90, 120, 150, 180, 210, 240, 270, 300, and 330 degrees. For each azimuth, a series of sample correlograms were also calculated at 30° dip increments. Lastly, a correlogram was calculated in the vertical direction. Using the complete suite of correlograms, an algorithm determined the best-fit model. This model is described by the nugget (C_0) which was derived using down hole variograms; one or two nested structure variance contribution (C_1, C_2) ranges for the variance contributions, and the model type (spherical or exponential). After fitting the variance parameters, the algorithm then fits an ellipsoid to all ranges from the directional models for each structure. The lengths and orientations of the axes of the ellipsoids give the final models of anisotropy.

All anisotropy models generated by SAGE 2001 were visually inspected in Gems to compare output with the expected geological controls on the mineralization.

Table 17-8 shows a summary of the variography results for the domains that returned a conclusive variogram. The traditional exponential range R in the tables is defined as Gam(3R) = 0.95 * Sill as defined by the first edition of GSLIB (Deutsch and Journel). Traditionally, the order and rotation parameters are derived from the variography. At the Main Zone deposit, because the variography was carried out on only a subset of the data, and due to the folded nature of the deposit, the order and direction of the rotations around the three axes are controlled by the search ellipsoid orientation for the various subdomains, as described in the interpolation plan in Section 17.6 of this report.

In general terms and within the selected area, the variogram model for the Upper and Lower Sills shows a preferred easterly orientation between 90° to 110° in azimuth with a relatively flat dip to the south. The combined C1 and C0 component axis oriented more or less with the mineralized zone for the data subset.

The variography was considered conclusive, allowing the model to be interpolated using ordinary kriging. Variography results for nickel are located in Appendix E.

Table 17-8: Variogram Parameters

	Component	Increment	Cumulative	Rotation	Rotation Angle	Range1	Range2	Range3
Lower Sill Nickel	nugget C0	0.1	0.1					
	exponential C1	0.703	0.803	ZXZ	Same as search ellipsoid	38.1	20.2	10.4
	exponential C2	0.197	1	ZXZ	Same as search ellipsoid	188.3	73.2	3.1
Lower Sill Copper	nugget C0	0.1	0.1					
	exponential C1	0.614	0.714	ZXZ	Same as search ellipsoid	138.5	19.8	8.3
	exponential C2	0.286	1	ZXZ	Same as search ellipsoid	241.4	60.9	5.6
Lower Sill Cobalt	nugget C0	0.1	0.1					
	exponential C1	0.796	0.896	ZXZ	Same as search ellipsoid	60.3	20.1	12.9
	exponential C2	0.104	1	ZXZ	Same as search ellipsoid	293.1	113.4	8.5
Upper Sill Nickel	nugget C0	0.11	0.11					
	exponential C1	0.772	0.882	ZXZ	Same as search ellipsoid	70.3	8.8	5.8
	exponential C2	0.118	1	ZXZ	Same as search ellipsoid	186.3	149.4	20.3
Upper Sill Copper	nugget C0	0.01	0.01					
	exponential C1	0.767	0.777	ZXZ	Same as search ellipsoid	50.8	19.6	5.7
	exponential C2	0.223	1	ZXZ	Same as search ellipsoid	173.8	161	2.9
Upper Sill Cobalt	nugget C0	0.3	0.3					
	exponential C1	0.492	0.792	ZXZ	Same as search ellipsoid	42.2	7.7	2.6
	exponential C2	0.208	1	ZXZ	Same as search ellipsoid	465.5	54.2	16.3

17.4.2 *SEARCH ELLIPSOID DIMENSION AND ORIENTATION*

The variogram is the key function in geostatistics, as it will be used to fit a model of the temporal/spatial correlation of the observed phenomenon, and ultimately sets the weights that will be applied to the samples during the grade interpolation. While it is common to use the variogram model *as a guide* to set the search ellipsoids range and attitude, the geologist modelling the deposit needs to consider the strike and dip of the mineralized horizon, and the drill hole spacing and distribution. PEG used the result of the variography as one of the guiding principles for setting the sample-search ellipsoid-dimension.

The first pass was sized to reach at least 1½ drill section spacing along the main axis of the mineralization as expressed by the variograms. A second and third multiplier was used for setting the subsequent search dimension for pass 2 and pass 3, leaving the ratio between the X, Y, and Z axes consistent with the results of the variography. The maximum range of the third pass search ellipsoid was set to approximate 90% the sill value on the best exponential practical variogram.

Due to the folded nature of the deposit, four subdomains were delineated with a separate fifth subdomain used for the east side of the fault. The subdomains allowed for the rotation of the search ellipsoid used for the data selection without resorting to any unfolding methodology as shown in Figures 17-4 and 17-5.

As stated earlier, for the Nickel King deposit the search ellipsoid attitude also controls the orientation of the variogram models used in the kriging equations. Table 17-9 lists the final values used in the resource model.

Figure 17-4: Upper Sill with Subdomains

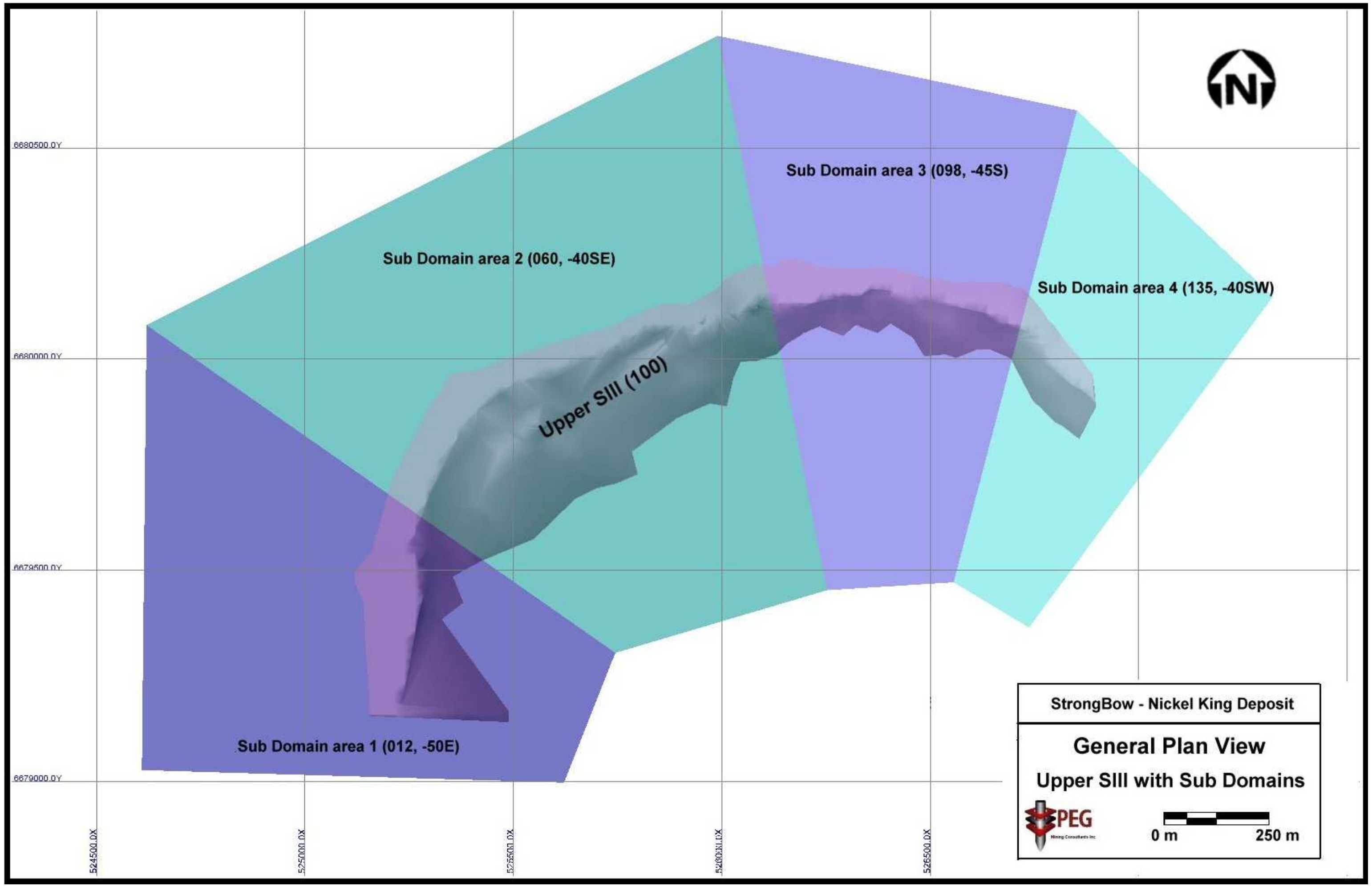


Figure 17-5: **Lower Sill with Subdomains**

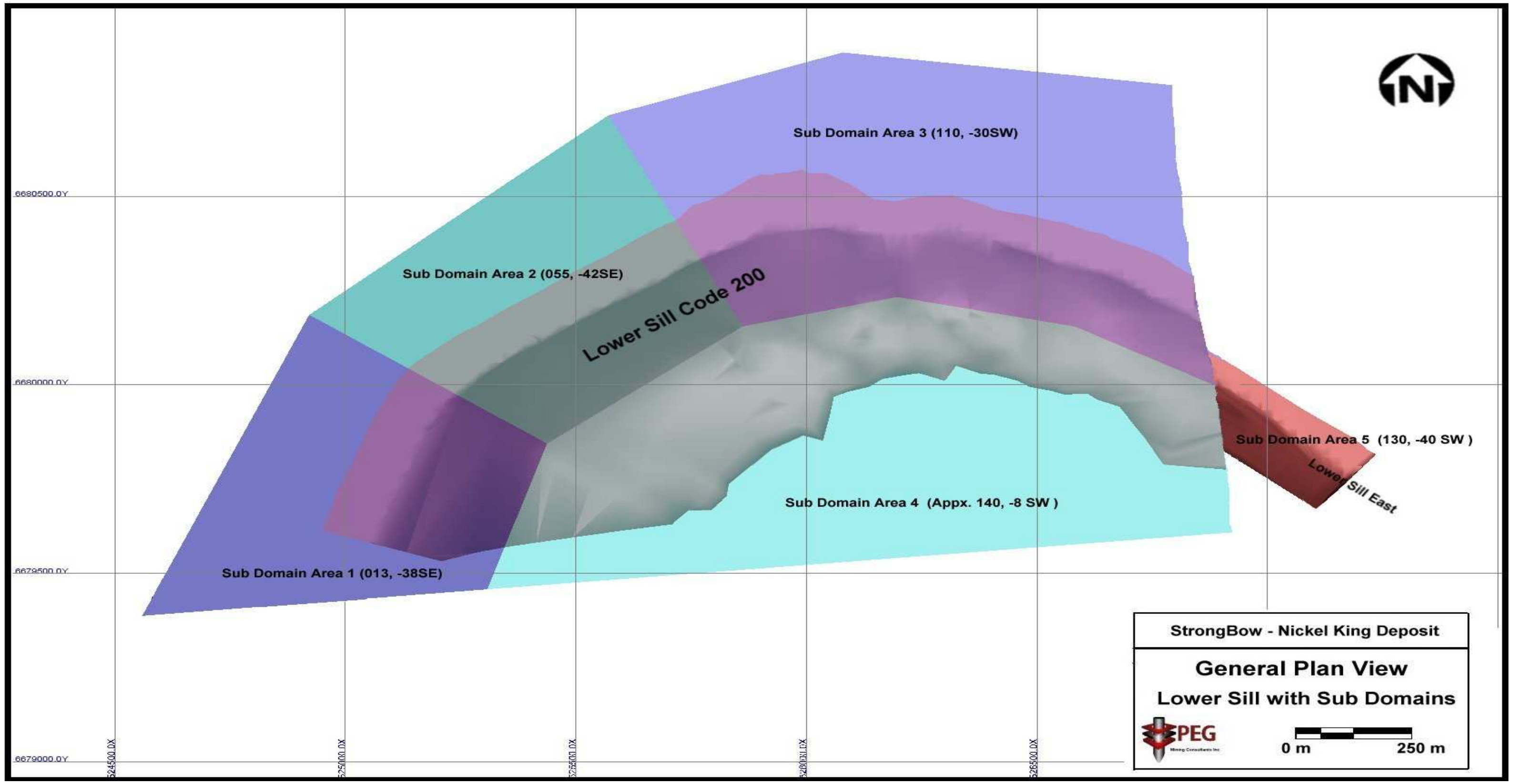


Table 17-9: Ellipsoid Search Parameters for the Upper and Lower Sills

Based on Variograms	Pass 1	Multiplier	Pass 2	Multiplier	Pass 3
Upper Sill – Search Parameters					
Range X	75	1.5	112.5	2	225
Range Y	45	1.5	67.5	2	135
Range Z	22.5	1.5	33.75	2	68
Upper Sill – Search Ellipsoid Orientation					
Anisotropy angles are defined by Rotation ZXZ	Sub Dom 1	Sub Dom 2	Sub Dom 3	Sub Dom 4	
Rotation about Z from X towards Y	78	30	-8	-45	
Rotation about X from Y towards Z	50	40	45	40	
Rotation about Z from X towards Y	0	0	0	0	
Lower Sill – Search Parameters					
Range X	90	1.5	135	2	270
Range Y	60	1.5	90	2	180
Range Z	20	1.5	30	2	60
Lower Sill - Search Ellipsoid Orientation					
Anisotropy angles are defined by Rotation ZXZ	Sub Dom 1	Sub Dom 2	Sub Dom 3	Sub Dom 4	LS East
Rotation about Z from X towards Y	77	35	-20	-20	-40
Rotation about X from Y towards Z	38	42	30	10	40
Rotation about Z from X towards Y	0	0	0	60	0

17.5 RESOURCE BLOCK MODEL

The block model was constructed in Gemcom's GEMS version 6.14™ software. A 10 x 10 x 10 m block size was selected based on open pit mining selectivity considerations and the density of the dataset.

The block model was defined on the project coordinate system (UTM – NAD 83) with no rotation. Table 17-10 lists the upper southeast corner of the model and is defined on the block edge.

The rock type model was coded by combining the geology model code with the subdomain code controlling the search ellipsoid orientation. The 100 series code represents the Upper Sill and the 200 series represents the Lower Sill. The subdomains were simply assigned a code of 1 to 5 from west to east. A block-model manipulation-script calculated the final rock type code by adding the subdomain code to the main geology code.

Table 17-10: Block Model Definition (block edge)

	Nickel King
Easting	524,750
Northing	6,679,120
Top Elevation	500
Rotation Angle	0
Block Size (X, Y, Z)	10 x 10 x 10
Number of blocks in the X direction	253
Number of blocks in the Y direction	151
Number of blocks in the Z direction	54

17.6 INTERPOLATION PLAN

The resource model was interpolated using ordinary kriging with inverse distance square and nearest neighbour check models.

The interpolation was carried out in a multi-pass approach with an increasing search dimension coupled with decreasing sample restriction, interpolating only the blocks that were not interpolated in the earlier pass.

Pass 1	used an octant search with 6 samples minimum, 15 maximum and a minimum of 3 octants and a maximum of 5 samples per octant
Pass 2	used an ellipsoid search with 6 samples minimum and 16 samples maximum
Pass 3	used an ellipsoid search with 4 samples minimum and 16 maximum.

A maximum of five samples per hole was imposed on the data selection forcing a minimum of two drill holes used in the interpolation of a block for pass 1 and 2.

The boundary between the Upper Sill and Lower Sill at depth was treated as a hard boundary, meaning that none of the composites from the Upper Sill were used in the interpolation of the Lower Sill and vice versa.

All subdomain boundaries within each sill were treated as soft boundaries, allowing samples from one subdomain to be used in the interpolation of the adjacent subdomain. PEG believes this is the correct methodology, since the subdomains were only used to control the orientation of the sample search ellipsoids, and do not correspond to any known lithological contact or fault.

17.7 MINERAL RESOURCE CLASSIFICATION

Several factors are considered in the definition of a resource classification:

- Canadian Institute of Mining (CIM) requirements and guidelines
- experience with similar deposits
- spatial continuity
- confidence limit analysis
- geology.

No environmental, permitting, legal, title, taxation, socio-economic, marketing, or other relevant issues are known to the author that may currently affect the estimate of mineral resources. Mineral Reserves can only be estimated on the basis of an economic evaluation that is used in a Pre-feasibility or Feasibility Study of a mineral project, thus no reserves have been estimated. As per NI 43-101 guidelines, mineral resources, which are not mineral reserves, do not have demonstrated economic viability.

Four confidence categories exist in the model. The usual CIM guidelines of Measured, Indicated and Inferred classes are coded 1, 2, and 3 respectively. A special code 4 called potential mineralization represents mineralization that was considered too far away from the existing drilling to be classified as inferred resource. As per NI 43-101 guidelines, the tonnage and grade for the potential mineralization is not included in this resource estimate, but has been discussed in Section 17.10, Potential Mineral Deposit. Typically, confidence level for a grade in the block model is reduced with the increase in the search ellipsoid size along with the diminishing restriction on the number of samples used for the grade interpolation. This is essentially controlled via the pass number of the interpolation plan described in the previous section. A common technique is to categorize a model based on the pass number and distance to the closest sample.

For the Lower Sill, variograms indicated that at 90% of the sill value the range varied between 135 m on strike and 97.5 m in the down-dip direction. At 60% of the sill value, range is relatively short, showing approximately 45 m. For classification purposes, PEG elected to use a distance to the closest sample of <45 m for the indicated category, and a distance of up to 100 m to set the inferred category.

For the Upper Sill, variograms indicated that at 90% of the sill value the range is about 75 m. At 60% of the sill value, range is shorter than the Lower Sill, showing approximately 40 m. For classification purposes, PEG elected to use a distance to the closest sample of <40 m for the indicated category, and a distance of up to 75 m to set the inferred category.

Resource blocks in the Potential category were converted to Inferred if their distance to an old CANICO drill hole was 75 m or less. This last manipulation affected mostly the Upper Sill and accounted for the increased confidence in the up-dip continuity of the mineralization as observed in the CANICO drilling.

Table 17-11 shows a summary of the classification parameters used for the Nickel King resource statement.

Table 17-11: Classification Parameters

Sill	Measured	Indicated	Inferred	Potential
Upper Sill	Not Used	<15 m distance to closest composite with block interpolated from 1 or more holes (pass 1, 2, or 3) OR < 40 m distance to closest composite with blocks interpolated with a minimum of 2 or more holes (pass 1)	≥40 m and ≤ to 75 m distance to closest composite with block interpolated from 1 or more holes (pass 1, 2, or 3) OR If block less than 75 m distance from an CANICO hole	>75 m distance to closest composite with block interpolated from 1 or more holes (pass 1, 2, or 3)
Lower Sill	Not Used	<15 m distance to closest composite with block interpolated from 1 or more holes (pass 1, 2 or 3) OR <45 m distance to closest composite with blocks interpolated with a minimum of 2 or more holes (pass 1)	≥45 m and ≤ to 100 m distance to closest composite with block interpolated from 1 or more holes (pass 1, 2, or 3) OR If block less than 75 m distance from an CANICO hole	>100 m distance to closest composite with block interpolated from 1 or more holes (pass 1, 2, or 3)

Based on the criteria outlined in Table 17-11, approximately 9% of the blocks estimated in the Nickel King Main Zone model are Indicated resources. Inferred resources accounted for 31% of the total volume outlined by the Upper and Lower Sills wireframe. The remaining 60% of the volume was either uninterpolated, bearing no grade, or interpolated, but classified as Potential mineralization. No resources were classified as Measured since the confidence necessary for Measured resource is generally predicated upon exposed working faces or very dense drilling

17.8 MINERAL RESOURCE TABULATION

Effective 25 February 2009, PEG has estimated the Mineral Resource for the Nickel King Main Zone deposit, utilizing approximately 12,296 m of diamond drill hole data. The resource estimate takes into account all drilling information for the Nickel King Main Zone deposit up to the end of the 2008 drill campaign.

The Nickel King resource estimate comprises Indicated and Inferred resources reported as Ni-Cu-Co mineralization with a base case cut-off grade of 0.20% Ni.

The base case cut-off grade chosen was determined by considering the characteristics of the deposit, envisioned open pit mining method, an assumed metallurgical recovery based upon a thin-section analysis of representative core, and cut-offs used at other similar deposits.

Table 17-12 shows a summary of the result of the resource estimate at the Nickel King Main Zone Deposit. The total Indicated Resource is 11.1 Mt grading at 0.40% Ni, 0.10% Cu and 0.018% Co, containing 97.7 Mlb of Ni, 23.5 Mlb of Cu, and 4.4 Mlb of Co. The total Inferred Resource is 33.1 Mt grading at 0.36% Ni, 0.09% Cu, and 0.017% Co, containing 262.4 Mlb of Ni, 63.9 Mlb of Cu, and 12.3 Mlb of Co. Table 17-13 shows the resource at various cut-offs with the 0.2% Ni base case highlighted.

Table 17-12: Nickel King Project – Main Zone Resource Estimate at a 0.2% Ni Cut-off (Base Case)

		Grade			Contained Metal		
Class	Tonnes (000s)	Ni (%)	Cu (%)	Co (%)	Ni (Mlb)	Cu (Mlb)	Co (Mlb)
Indicated	11,111	0.40	0.10	0.018	97.7	23.5	4.4
Inferred	33,061	0.36	0.09	0.017	262.4	63.9	12.3

Table 17-13: Nickel Project at Various Cut-offs (Base Case Highlighted)

Classification	Ni Cut-off (%)	Tonnes (000s)	Ni (%)	Cu (%)	Co (%)
Indicated	0.5	2,773	0.62	0.15	0.027
	0.3	7,340	0.48	0.12	0.021
	0.2	11,111	0.4	0.1	0.018
	0.15	14,082	0.35	0.08	0.016
Inferred	0.5	4,930	0.59	0.15	0.026
	0.3	19,609	0.44	0.11	0.02
	0.2	33,061	0.36	0.09	0.017
	0.15	43,974	0.31	0.08	0.015

17.9 BLOCK MODEL VALIDATION

The Main Zone grade models were validated by four methods:

- visual comparison of colour-coded block model grades with composite grades on section plots
- comparison of the global mean block grades for ordinary kriging, inverse distance, nearest neighbour models, composite, and raw assay grades
- comparison using grade profiles at 100 m spacing in the X and Y direction and 50 m spacing in the Z direction, looking for local bias in the estimate
- naïve cross validation test with composite grade versus block model grade.

17.9.1 VISUAL COMPARISON

The visual comparisons of block model grades with composite grades show a reasonable correlation between values. No significant discrepancies were apparent from the sections reviewed. The orientations of the estimated grades on sections follow more or less the projection angles defined by the search ellipsoid. Tweaking the search ellipsoid orientation by adding a few additional domains or the use of an unfolding technique may marginally improve the interpolation.

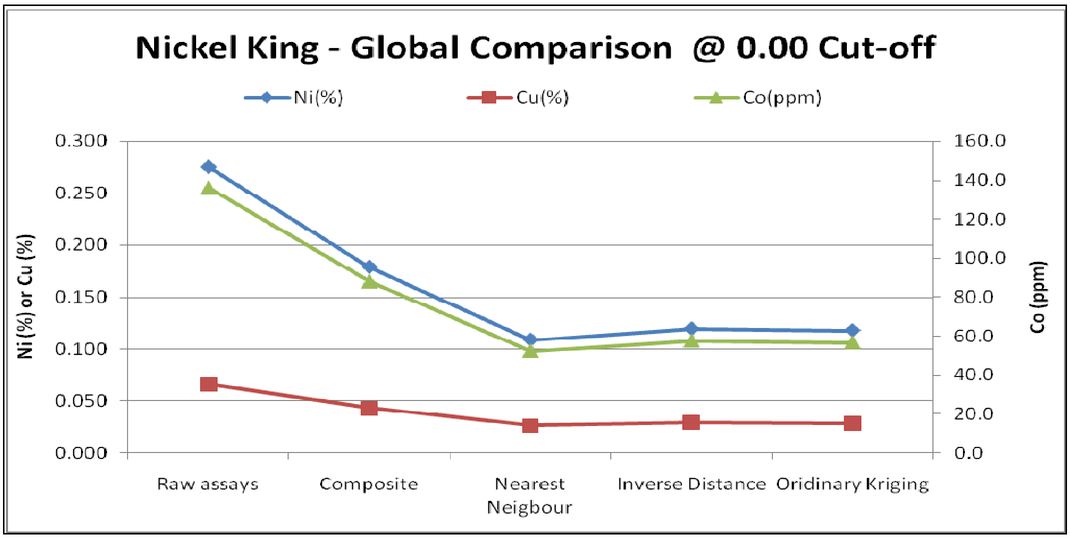
17.9.2 GLOBAL COMPARISONS

Table 17-14 shows the grade statistics for the raw assays, composites, ordinary kriging, nearest neighbour, and inverse distance models. Figure 17-6 shows the differences. Statistics for the composite mean grade compares well to raw assay grade, with a normal reduction in value partly due to the addition of zero grade assigned to the un-sample intervals during the compositing process, and also to smoothing related to volume variance. The block model mean grade when compared against the composites shows a normal reduction in values for all elements. More importantly, the grade of the nearest neighbour, inverse distance, and ordinary Kriging at 0.00 cut-off are all very close to each other, showing that no global bias was introduced from the interpolation method used.

Table 17-14: Global Comparisons – Grade at 0.00 Cut-off

Methodology	Ni %)	Cu %	Co (ppm)
Raw Assays	0.275	0.066	136.3
Composite	0.179	0.043	88.1
Nearest Neighbour	0.109	0.026	52.3
Inverse Distance	0.120	0.029	57.6
Ordinary Kriging	0.118	0.029	56.8

Figure 17-6: Global Comparison at 0.00 Cut-off



17.9.3 *LOCAL COMPARISONS – GRADE PROFILE*

The comparison of the grade profiles (swath plots) of the raw assay, composites, and estimated grade allows for a visual verification of an over- or under-estimation of the block grades at the global and local scales. A qualitative assessment of the smoothing and variability of the estimates can also be observed from the plots. The output consists of three swath plots generated at 25 m intervals in the X direction, 15 m in the Y direction, and 11 m vertically for nickel, the main grade element.

The kriged estimate should be smoother than the nearest-neighbour estimate, thus the nearest-neighbour estimate should fluctuate around the kriged estimate on the plots or display a slightly higher grade. The composite line is generally located between the assay and the interpolated grade. A model with good composite distribution should show very few crossovers between the composite and the interpolated grade line on the plots. In the fringes of the deposits, as composite data points become sparse, crossovers are often unavoidable. The swath size also controls this effect to a certain extent; if the swaths are too small, then fewer composites will be encountered, which usually results in a very erratic line on the plots.

Due to the folded nature of the deposit at Nickel King, there is no perfect orientation for the swath plots. Each X and Y orientation will perform an average along the strike of the deposit in one portion of the plot, which is not ideal. The swath plot in the Z-axis plane should show the best results for this model.

In general, the swath plots show good agreement with all three methodologies, with no major local bias. The resource model appears to return higher grade in the eastern part of the deposit as exhibited along the X-axis. The Z-axis shows a minor crossover between 206 m and 306 m, which could not be identified visually on plan view. Grade profiles are presented in Figures 17-7 to 17-9.

Figure 17-7: X Axis Swath Plots

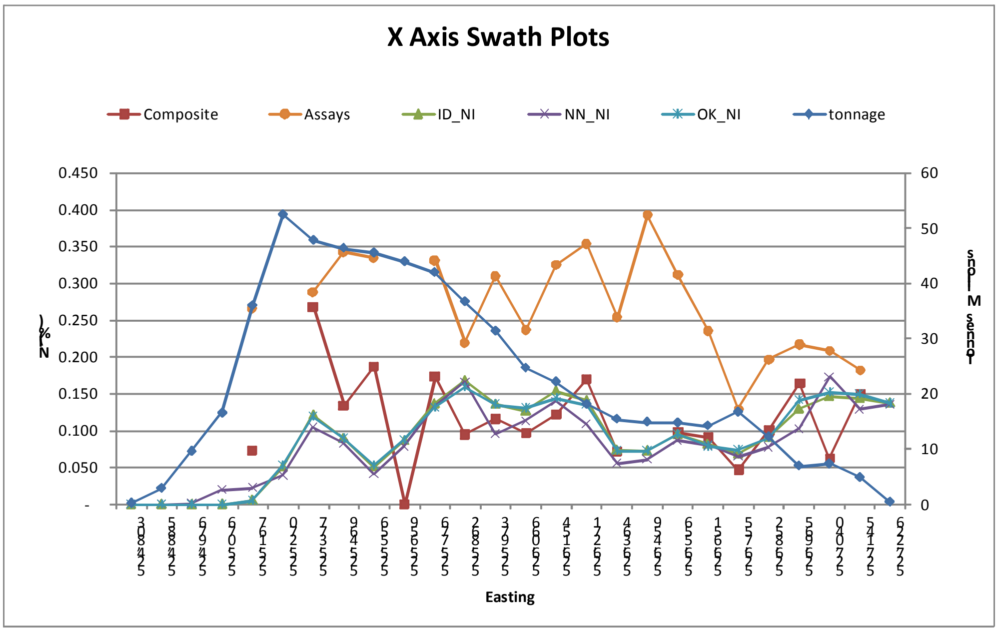


Figure 17-8: Y Axis Swath Plots

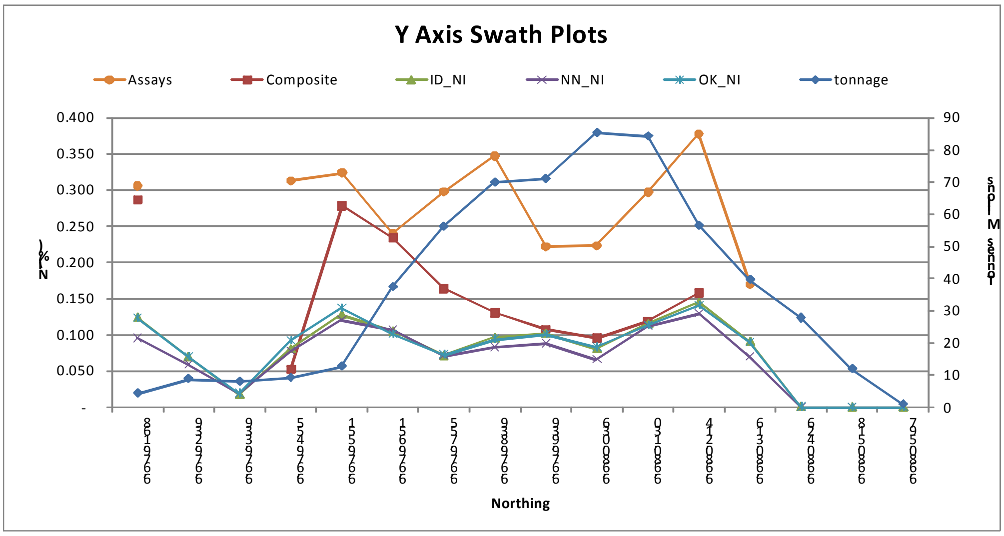
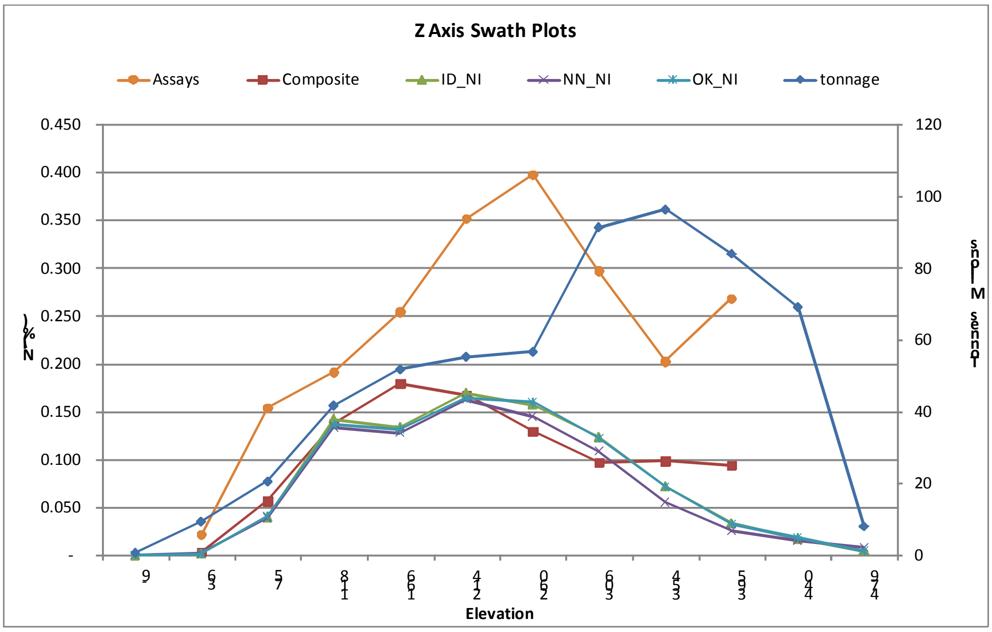


Figure 17-9: Z Axis Swath Plots



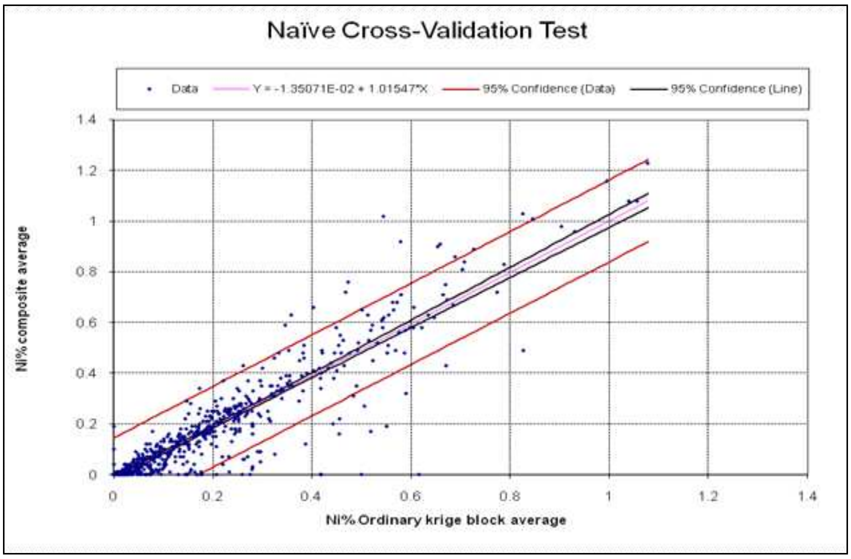
17.9.4 *NAÏVE CROSS-VALIDATION TEST*

A comparison of the average grade of the composites within a block with the estimated grade of that block provides an assessment of the estimation process close to measured data. Pairing of these grades on a scattered plot gives a statistical valuation of the estimates. This methodology differs from Jack Knifing, which replaces a composite with a pseudo block at the same location. Jack knifing evaluates and compares the estimated grade of the pseudo block against that of the composite grade.

It is anticipated that the estimated block grades should be similar to the composited grades within the block without being of exactly the same value, especially with ordinary Kriging, where the weights applied to the composite points are controlled by the spatial distribution in the data.

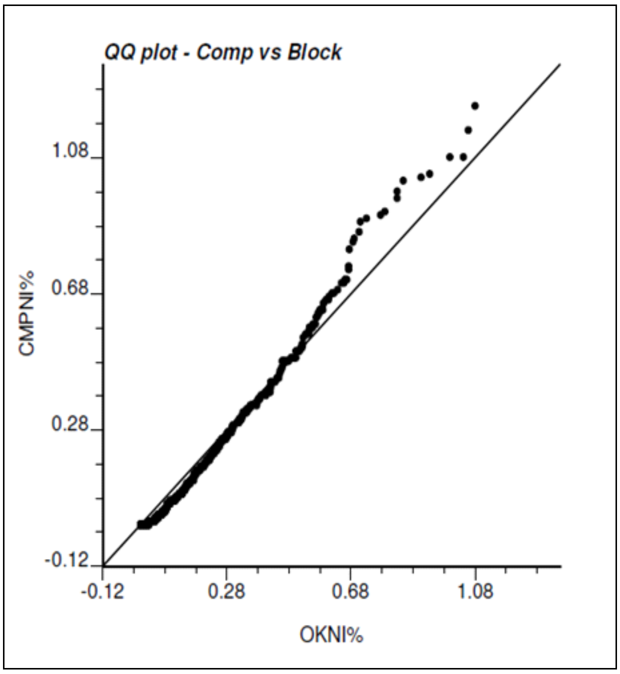
A high correlation coefficient will indicate satisfactory results in the interpolation process, while a medium to low correlation coefficient will be indicative of larger differences in the estimates and would suggest a further review of the interpolation process. Figure 17-10 presents the results from the pairing of the composited and estimated grades within blocks pierced by a drill hole. The R^2 value is 0.852.

Figure 17-10: Naïve Cross Validation Test Results



Plotting the same data on a QQ plot (Figure 17-11), revealed that the interpolation performs remarkably well in the grade ranging from 0% to 0.58% Ni. Above 0.58% Ni, the krige model appears to underestimate slightly the composite grade due to smoothing, which is a common observation with this type of interpolation.

Figure 17-11: Naïve Cross Validation Composite (CMPNi%) vs. Ordinary Kriging (OKNi%)



17.10 POTENTIAL MINERAL DEPOSITS

Strongbow has also identified PMD within the extent of the Nickel King Main Zone deposit. The PMD occupy gaps within the resource estimate where there is insufficient drilling to classify an inferred resource.

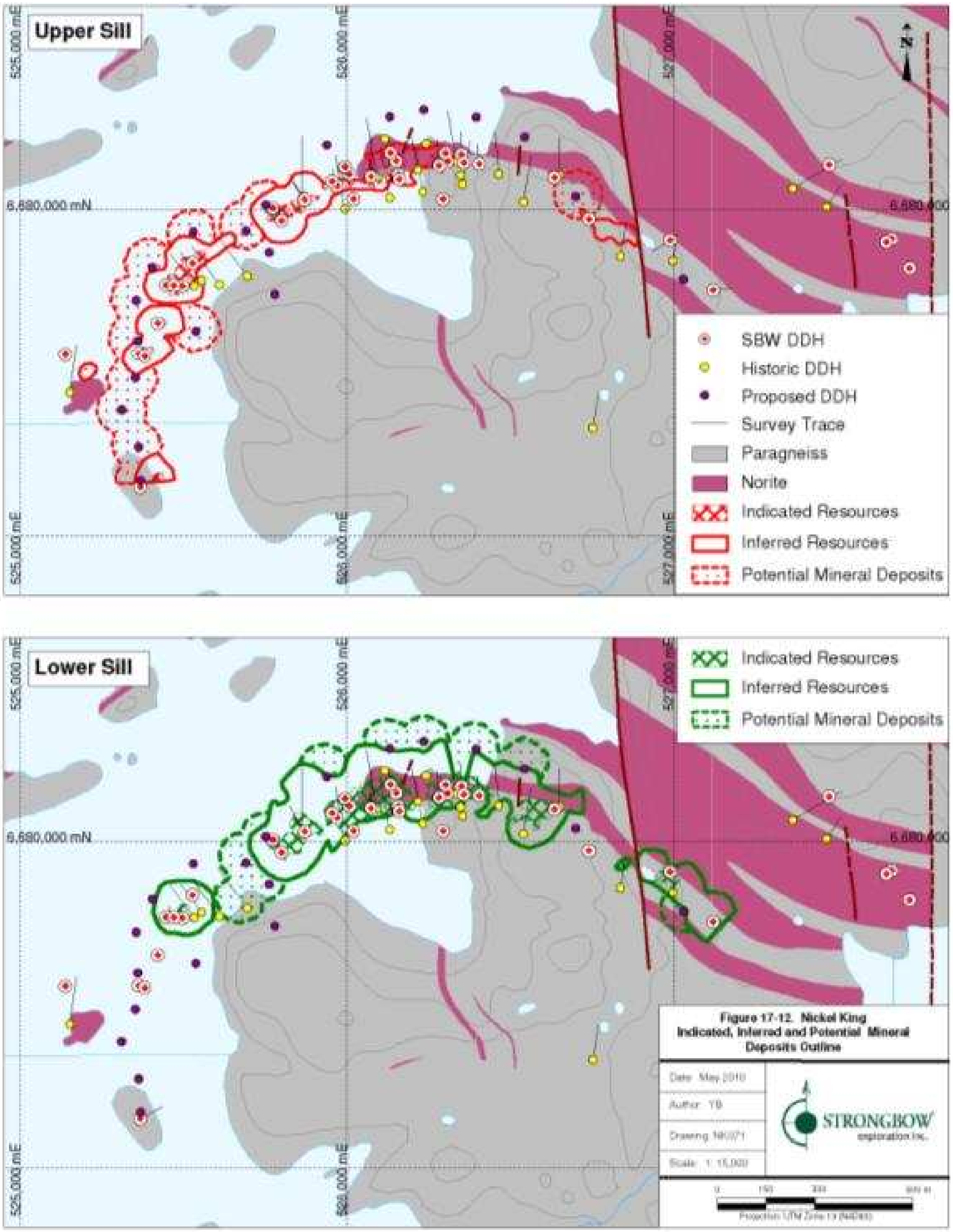
Modelled blocks falling within these gaps are either uninterpolated or have been classified with special code 4 called Potential mineralization (see Section 17.7 Mineral Resource Classification, above). Strongbow's estimate of PMD was determined by modelling the pierce points of 23 proposed drill holes through the resource block model, with the intent of evaluating targets that have the potential to convert the maximum number of blocks into Inferred resources.

The modelled drill holes represent approximately 5,200 m of drilling. PMD were estimated by adding together blocks that were classified by PEG as code 4 Potential mineralization, fall within a 75 m area of influence around each proposed drill hole, and had an interpolated grade exceeding 0.2% Ni cut-off. Based on this evaluation, Strongbow has estimated that between 10 and 27 Mt of PMD are located within the area of the block model that would be tested by the proposed drill holes. The 27 Mt maximum PMD includes all resource blocks that satisfy the three criteria above. The lower 10 Mt PMD was determined by making a reasonable estimate, based on geological and geophysical information, of the minimum extent of mineralization expected within the areas tested by these model drill holes.

Figure 17-12 shows location of the PMD, in plan view, with respect to the indicated and inferred resources of the Nickel King Main Zone deposit. The figure is split into two different views showing the plan view extent of the Indicated and Inferred resources in the Upper Sill (top image) and Lower Sill (bottom image). The figure also shows the proposed drill holes (note that some drill collar positions represent more than one proposed drill hole), and the approximate outlines of the 75 m area of influence around each proposed hole within which the PMDs have been estimated.

PEG has reviewed this estimate of PMD within the Nickel King Main Zone deposit and considers the estimate to be reasonable. PEG cautions that this PMD estimate is conceptual in nature and that there has been insufficient exploration to define a resource in these areas, and it is uncertain whether additional exploration drilling will be successful in delineating a mineral resource in these areas.

Figure 17-12: Indicated and Inferred Resource Outlines and Potential Mineral Deposits



18.0 INTERPRETATION AND CONCLUSIONS

18.1 GEOLOGY

The Nickel King project is located in the southeastern Northwest Territories, approximately 550 km southeast of Yellowknife and 145 km northeast of Stony Rapids, Saskatchewan.

The Main Zone deposit is situated on mining lease 4954 and is located at UTM 526200E and 6680160N (North American Datum 83, Zone 13). Strongbow is the registered owner of a 100% interest in all of the mineral claims and mining leases.

The Nickel King project area is subject to a sub-polar climate. The project area is remote with no direct road access. Ground exploration (prospecting, mapping, etc.) is generally limited to the summer season, between break-up and freeze-up (May to September). Drilling may be accomplished year-round once sufficient supplies, air support, and accommodation are in place.

CANICO commenced the exploration on the project area in 1952. This was followed throughout the years by Geological Survey of Canada, Highwood, Kizan, Aber (which transferred interest to Navigator Exploration Corp.), and more recently by Falconbridge. In May 2004, Navigator amalgamated with Strongbow Resources Inc. to form Strongbow Exploration Inc. Because of the amalgamation, Strongbow assumed the rights and obligations of Navigator, including the Nickel King project.

The Nickel King project lies within the Snowbird Tectonic Zone (STZ) in the northwestern part of the Canadian Shield. The property geology of the Nickel King area is dominated by late Archean or early Proterozoic paragneisses. Locally, mafic intrusive rocks (norite and pyroxenite) and pegmatite dykes intrude the paragneiss. Past workers have variably described the mafic/ultramafic intrusions in the Nickel King area as norites, gabbros, olivine gabbros, gabbronorite pyroxenites, amphibolites, and peridotites. Strongbow, however, has not identified olivine in any of these rocks, and therefore, based on mineralogy, refers to all mafic intrusions as norite and, locally, pyroxenite. Norite intrusions occur as medium to coarse-grained rocks comprised of enstatite, hornblende, plagioclase, and phlogopite.

The Nickel King project area contains nickel-copper-cobalt sulphide mineralization hosted in mafic (norite) intrusions, sharing many characteristics with a diverse class of mineral deposits commonly referred to as “magmatic sulphide deposits.”

Five areas of mineralization discovered to date within the Nickel King project are the Main Zone, Koona, and South Ring Zones, the Joe Island Trend and the Selwyn Intrusion. In each case nickel-copper sulphide mineralization is hosted within a norite intrusion, ranging from a discrete, plug-shaped intrusive (South Ring) to the extensive, >2,600 m long arcuate Main Zone norite which is the subject of the resource model feature in this technical report.

The Main Zone has been defined over a strike length of 2,600 m. Drilling and mapping surveys indicate that mineralization is hosted within the limbs of a recumbent, tightly folded norite sill which, in turn, was folded by a younger folding event that resulted in a south facing, arcuate, open fold geometry.

The Upper Sill of the Main Zone is exposed on the east shore of Thye Lake, trends approximately east-west, typically dips moderately (30° to 50°) to the south, and plunges gently (approximately 10° to 15°) to the southwest beneath Thye Lake. The Lower Sill is located beneath the Upper Sill, dips to the south at a somewhat shallower angle (10° to 40°), and has a similar gentle southwest plunge. Mineralization consists of disseminated to semi-massive iron-nickel-copper sulphides (pyrrhotite-pentlandite-chalcopyrite), concentrated close to the upper margins of both sills.

Almost 80% of the 66 drill holes completed by Strongbow have been focused on the Main Zone. Historical drilling done by CANICO and Aber supplemented the database by 31 holes. Strongbow adheres to a rigorous and detailed set of protocols for all samples collected during the Nickel King exploration programs.

PEG reviewed Strongbow logging and sample collection procedures, and is of the opinion that core logging is of sufficient detail to be utilized in this resource estimation. Core sampling procedures follow standard industry practices with a well implemented QA/QC program that meets or exceed industry standards. PEG is of the opinion that the assay results and QA/QC protocols are acceptable for resource estimation purposes.

Data verification was performed by PEG through a site visit, collection of independent character samples and a database audit prior to mineral resource estimation. PEG found the database to be exceptionally well maintained and error free, and usable in mineral resource estimation.

The in-situ bulk density of the rock types was based on 72 specific gravity determinations. While this is considered weak on an advanced project, it is adequate for the level of exploration at the Nickel King deposit. PEG is of the opinion that the specific gravity determinations constitute a good representation of the in-situ bulk density of the mineralized norite given the data available.

Mineral resources at Nickel King were classified using logic consistent with the CIM definitions referred to in NI 43-101. At Nickel King, the mineralization, density, and position of the drill holes satisfies sufficient criteria to be classified into the Indicated and Inferred categories.

This independent mineral resource estimate and review by PEG supports the 25 February 2009 disclosure by Strongbow of the mineral resource statement for the Nickel King deposit. Strongbow has also identified PMD within the extent of the Nickel King Main Zone deposit. The PMD occupy gaps within the resource estimate where there is insufficient drilling to classify an inferred resource. Strongbow has estimated that between 10 and 27 Mt of PMD with an interpolated grade exceeding 0.2% Ni cut-off are located within the area of the block model that would be tested by 23 proposed drill holes. PEG has reviewed this estimate of PMD within the Nickel King Main Zone deposit and considers the estimate to be reasonable.

18.2 METALLURGY

Based on the results presented herein, a number of conclusions are drawn:

- A metallurgical composite was prepared from a continuous interval of hole NK08-035 from the Nickel King deposit, which had an average head grade of 0.65% Ni, 0.14% Cu, 0.035% Co, and 4.76% S.
- Grindability testing of the composite indicated the sample was of a medium hardness, with a Bond Rod Work Index of 13.2 kWh/t and a Bond Ball Work Index of 15.0 kWh/t. Combined with a relatively coarse grind target of ~110 µm, the comminution equipment required for this process would be modestly sized.
- Preliminary mineralogical characterization has indicated that the contained nickel in the composite was associated primarily with pentlandite (88.9%), and to a lesser extent with pyrrhotite (9.2%) and silicates (1.9%).
- At the target grind of 110 µm the pentlandite was characterized as well liberated with 86% of the mineral said to be liberated or sub-liberated.
- This flotation test program indicated that a saleable grade bulk concentrate grading 16.5% Ni, 4.21% Cu, and 0.74% Co, could be generated with nickel, copper, and cobalt recoveries of 78.4%, 89.1%, and 63.5%, respectively.
- Depression of MgO minerals was achieved with moderate addition of CMC depressant and soda ash.
- Minor element analysis revealed no deleterious elements of concern in the locked cycle test concentrate products.
- ABA testing of a low sulphur tailings sample indicated a low possibility of being acid generating.

19.0 RECOMMENDATIONS

Following the completion of the mineral resource estimates of the deposit, and assuming a possible open pit scenario, PEG recommends the following:

- Future drill campaigns should focus on the following areas:
  - up-dip of the zones to assess continuity
  - drilling of potential land base targets in the southeast end of the deposit should be undertaken. Additional potentially mineable zones could be incorporated early on in a mine plan and provide waste rock for the construction of a dyke system. In this manner, the cost of the dyke would be offset by using productive material as opposed to a separate quarry strictly for building the dyke
  - infill drilling in the interpreted fold nose area concentrated in the poorly drilled sections of the deposit
  - resource upgrading to Inferred or Indicated mineralization should be a priority, as opposed to solely targeting high-grade mineralization.

- Collection of specific gravity data should be incorporated in the future drill program to supplement the existing data. SG determination should be carried out automatically at a rate of 1 sample every 5 m. This would ensure proper coverage in both high-grade and low-grade sections of the deposit. The SG data collection should also incorporate waste rocks for those areas encompassed by an open pit and around the perimeter of the deposit.
- The location of a potential winter/all season road to reach the Nickel King deposit coincides with the location of Strongbow's other nickel prospects in Saskatchewan. This offers excellent opportunities for shared costs for any deposits discovered on these properties.
- Assuming an open pit-mining scenario, the bathymetry of Thye Lake and the depth of lake sediments must be determined for proper estimation of fill requirements of a dyke and proper estimation of waste tonnages to be moved.
- With additional drilling, future resource models should revisit the Main Zone isoclinal fold interpretation since the structure is not well understood.
- Closely spaced grade data is required in sufficient quantity to enable modelling of well-structured directional variogram models. At Nickel King, due to the folded nature of the deposit, it is recommended that an unfolding technique be tested once sufficient infill drilling is available in the eastern and western fringe of the deposit.
- As part of the next drill campaign, PEG recommends the implementation of a comprehensive geotechnical data-collection program. Guidance regarding the proper

collection methodologies should be sought from a specialized firm to ensure the data will be usable in a preliminary economic assessment study.

Following the site visit, audit of the project database, and review of the QA/QC program, PEG recommends the following:

- Additional standards should be purchased to insert in the sample stream.
- Strongbow should evaluate the date of the laboratory analysis along with the batch number and sample number. This would allow Strongbow to produce control charts sorted by dates which can be use to locate deteriorating trend in the analytical procedures even though the QA/QC results may be above the fail mark.
- Coarse rejects and pulps from earlier assays should be inserted in the sample stream with a new tag number in order to incorporate a blind coarse and pulp duplicate procedure to the QA/QC protocol. This recommendation assumes that the logistics in relation to the rejects/pulp samples shipped back from the laboratory to the project site can be resolved. Obviously, the additional cost of adding this procedure to the QA/QC program needs to be weighed against the benefit obtained.

19.1 EXPLORATION

- Exploration drilling to test beyond the extent of the known Main Zone deposit. This should include single holes spaced 200 m to 300 m apart along the strike length of the deposit, both on the southwestern and southeastern margins. 2,500 m in 10 holes are required.
- Resource definition drilling of the Main Zone to establish the extent and continuity of mineralization within the deposit. Approximately 40 drill holes totalling 10,000 m are a preliminary estimate for this work. Assumptions made to support this estimate include: (i) the drill hole pattern should ensure a minimum of two drill holes on sections spaced no further than 100 m apart along the defined strike length of the Main Zone, plus an additional fence of two drill holes located 100 m beyond the current southwestern and southeastern extents of mineralization, and (ii) most drill holes are near vertical.
- Drill testing the strike extent of the Koona prospect at 100 m spacing (300 m in four holes).
- Drill testing of the Ring South Prospect (350 m in four holes).
- Bore hole TDEM surveys should be conducted in all drill holes including non-surveyed holes from 2007 and 2008 (where possible).

- Exploration drilling of untested targets in the, Kizan, Kizan South, Koona, Ring and South Duck areas, and along the Joe Island Trend (minimum 1,600 m in 6 drill holes).

19.2 PROPOSED BUDGET

Further drilling is required at Nickel King to test the full potential of the deposit. It is proposed that this work be broken down into three phases which will proceed based on successful results being returned from each phase.

The "all in" cost of drilling at the Nickel King project, based on the 2007 and 2008 programs, is approximately $400/m. This cost includes all drill support and bore hole geophysical surveys. The pricing of all future drilling programs is based on this all-in drilling cost.

Phase 1: The first phase should focus on the along strike potential of the Main Zone mineralization and is an exploration-focused program. This would include drilling on the southwestern and southeastern extensions of the deposit, with the focus on further expanding the currently defined 2,600 m strike length. This program is best suited for the winter as the majority of the targets on the southwestern extent of the deposit can most effectively be tested from the ice. This program is budgeted at 2,500 m in 10 drill holes for a total cost of $1 million.

Phase 2: This second phase is contingent on positive results from Phase 1, and is a resource definition program. It should be focused on expanding the Inferred and Indicated resources. 10,000 m in 40 drill holes will be required, at a cost of $4 million. This includes the 5,200 m of drilling described above in Section 17.10. The remaining balance (4,800 m) would be distributed along the trend of the deposit, infilling the current drill pattern on regular centres.

Phase 3: This third phase could be completed in conjunction with either Phase 1 or 2, or as a follow-up program pending the successful completion of the initial two phases. It includes 2,000 m of drilling in 10 holes and is an exploration-focused program. It will test all of the satellite targets around the Main Zone deposit included Koona, South Duck, South Ring, Kizan, South Kizan, and Joe Island. It is estimated to cost $0.8 million.

19.3 METALLURGY

The success of this first test program suggests that further investigation is warranted. Future programs should include:

- Variability sampling across the deposit. Specific attention should be given to any areas of the deposit that are mineralogically dissimilar, in order to evaluate how the current flowsheet handles areas of high or low pyrrhotite, copper, cobalt, etc. Testing would include grindability evaluation, flotation, and heavy liquid separation.

- A "Master Composite" sample should be generated that represents a best estimate of the deposit as a whole. In particular, the grade of the sample should resemble that of the resource estimate. Confirmatory testing of the present locked cycle flowsheet should be carried out to evaluate the circuit performance at a lower head grade.
- Heavy liquid separation testing should be further investigated once an accurate estimate of mining dilution has been made. Future HLS testwork is likely to focus on lower grade areas of the deposit that may be uneconomic to mill without a pre-concentration step. In certain circumstances, the inclusion of a preconcentration step allows the cut-off grade to be lowered.
- An opportunity may exist to improve overall nickel recovery by targeting any liberated pentlandite lost to the pyrrhotite tailings or B 1st cleaner scavenger tailings. Quantitative mineralogy such as QEMScan may be used to evaluate the potential increase in recovery prior to a detailed flotation program to target process improvement in this area.
- Once the present flowsheet has been confirmed on a lower grade sample, additional flotation work should be carried out to investigate the possibility of separating the bulk concentrate into separate copper and nickel concentrate products.

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21.0 CERTIFICATE OF QUALIFIED PERSONS

21.1 JOSEPH ROSAIRE PIERRE DESAUTELS, P.GEO

I, Joseph Rosaire Pierre Desautels of Barrie, Ontario, do hereby certify that as one of the authors of this updated technical report titled "Updated Technical Report for the Nickel King Main Zone Deposit; Northwest Territories, Canada dated June 2, 2010, I hereby make the following statements:

- I am a Principal Resource Geologist with PEG Mining consultants Inc. with a business address at 92 Caplan Avenue, Suite 610, Barrie, Ontario, Canada, L4N 0Z7.
- I am a graduate of Ottawa University (B.Sc. Hons., 1978).
- I am a member in good standing of the Association of Professional Geoscientists of Ontario (Registration #1362).
- I have practiced my profession in the mining industry continuously since graduation.
- I have conducted a site visit at the Nickel King Project between 31 March and 1 April 2009.
- I have read the definition of "qualified person" set out in NI 43-101 and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- My relevant experience with respect to resource modelling includes 30 years experience in the mining sector covering database, mine geology, grade control, and resource modelling. I was involved in numerous projects around the world in both base metals and precious metals deposits.
- I am responsible for the preparation of Sections 1.1, 2-15, 17, 18.1, 19.1, 19.2, and 20 of this updated technical report titled "Updated Technical Report for the Nickel King Main Zone Deposit; Northwest Territories, Canada" dated June 2, 2010.
- I have no prior involvement with the property that is the subject of the Technical Report.
- As of the date of this Certificate, to my knowledge, information, and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am independent of the Issuer as defined by Section 1.4 of the Instrument.
- I have read NI 43-101 and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Signed and dated this 2nd day of June 2010.

"Original Document signed and sealed by
Pierre Desautels, P.Geo."

Signature

21.2 ANDY HOLLOWAY, P.ENG., C.ENG.

I, Andy Holloway, of Peterborough, Ontario, do hereby certify that as one of the authors of this technical report titled “Updated Technical Report for the Nickel King Main Zone Deposit; Northwest Territories, Canada dated June 2, 2010; I hereby make the following statements:

- I am a Principal Process Engineer with PEG Mining Consultants Inc., with a business address at 92 Caplan Avenue, Suite 610, Barrie, Ontario, L4N 0Z7.
- I am a graduate of the University of Newcastle in Tyne, England, B.Eng. (Hons.), 1989, and I have practiced my profession continuously since then.
- I am a Professional Engineer licensed by Professional Engineers Ontario (Membership #100082475).
- I have read the definition of “qualified person” set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purpose of NI 43-101.
- My relevant experience with respect to mineral processing and metallurgy includes 19 years experience in the mining sector covering mineral processing, process plant operation, design engineering, and management. I have been involved in numerous projects around the world in both base metals and precious metals deposits.
- I am responsible for the preparation of Sections 1.2, 16, 18.2, 19.3, and 20 of this updated technical report titled “Updated Technical Report for the Nickel King Main Zone Deposit; Northwest Territories, Canada” dated June 2, 2010.
- As of the date of this Certificate, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information required to be disclosed to make the technical report not misleading.
- I am independent of the Issuer as defined by Section 1.4 of the Instrument.
- I have read NI 43-101 and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Signed and dated this 2nd day of June 2010.

“Original Document Signed and Sealed by
Andy Holloway, P.Eng. C.Eng”

Signature

APPENDIX A

DRILL HOLE LISTINGS

RESOURCE	ORIGIN	HOLE-ID	LOCATIONX	LOCATIONY	LOCATIONZ	LENGTH	Assay Count
Geol+Class	Inco	10231	526,242.5	6,680,203.6	398.4	72.8	15
Geol+Class	Inco	10232	526,115.8	6,680,219.5	396.6	156.7	65
Geol+Class	Inco	10236	526,092.0	6,680,101.0	397.0	253.9	26
Geol+Class	Inco	10237	526,217.0	6,680,123.2	397.5	135.3	11
Geol+Class	Inco	10238	526,336.0	6,680,148.0	399.0	218.3	9
Geol+Class	Inco	10239	526,232.8	6,680,056.3	397.0	233.2	30
Geol+Class	Inco	10240	526,353.8	6,680,079.8	399.0	232.3	23
Geol+Class	Inco	10241	526,464.7	6,680,111.4	408.0	225.9	22
Geol+Class	Inco	10242	526,130.3	6,680,037.2	396.0	271.6	18
Geol+Class	Inco	10243	525,993.3	6,680,003.7	396.0	281.6	11
Geol+Class	Inco	10244	525,862.4	6,680,021.9	396.0	288.3	2
Geol+Class	Inco	10245	526,348.3	6,680,107.3	399.0	237.1	11
Geol+Class	Inco	10246	526,652.0	6,680,109.0	422.0	194.5	13
Geol+Class	Inco	10247	527,366.2	6,680,067.2	426.5	231.3	8
Geol+Grade+Class	Aber Res	AN95-01	525,150.9	6,679,440.6	396.0	188.0	24
Geol+Grade+Class	Aber Res	AN95-02	526,998.8	6,679,844.1	401.0	116.7	30
Geol+Grade+Class	Aber Res	AN95-03	526,838.5	6,679,857.8	405.0	157.0	49
Geol+Grade+Class	Aber Res	AN95-04	527,470.5	6,680,009.5	420.0	119.0	12
Geol+Grade+Class	Aber Res	AN95-05	526,753.7	6,679,332.0	416.0	143.0	16
Geol+Grade+Class	Aber Res	AN95-06	526,541.0	6,680,024.6	414.0	203.0	46
Geol+Grade+Class	Aber Res	AN95-07	526,541.0	6,680,024.6	414.0	200.0	33
Geol+Grade+Class	Aber Res	AN95-08	525,609.0	6,679,770.7	396.0	201.0	30
Geol+Grade+Class	Aber Res	AN95-09	525,609.0	6,679,770.7	396.0	239.0	0
Geol+Grade+Class	Aber Res	AN95-10	525,694.2	6,679,797.1	396.0	197.0	0
Geol+Grade+Class	Aber Res	AN95-11	525,532.0	6,679,766.6	396.0	188.0	31
Geol+Grade+Class	Aber Res	AN95-12	525,532.0	6,679,766.6	396.0	151.0	5
Geol+Grade+Class	Aber Res	AN95-13	525,554.4	6,679,783.2	396.0	146.0	11
Geol+Grade+Class	StrongBow	NK07-001	526,350.3	6,680,170.3	399.1	242.0	139
Geol+Grade+Class	StrongBow	NK07-002	526,151.2	6,680,150.7	401.0	260.0	107
Geol+Grade+Class	StrongBow	NK07-003	525,970.0	6,680,074.0	396.0	296.0	111
Geol+Grade+Class	StrongBow	NK07-004	525,799.0	6,679,967.0	396.0	350.0	134
Geol+Grade+Class	StrongBow	NK07-005	525,445.0	6,679,770.0	396.0	232.5	80
Geol+Grade+Class	StrongBow	NK07-006	525,362.0	6,679,559.0	396.0	246.8	67
Geol+Grade+Class	StrongBow	NK07-007	525,496.0	6,679,767.0	396.0	223.4	47
Geol+Grade+Class	StrongBow	NK07-011	526,157.0	6,680,110.0	397.0	257.0	140
Geol+Grade+Class	StrongBow	NK07-012	526,309.0	6,680,149.0	403.0	232.8	124
Geol+Grade+Class	StrongBow	NK07-013	527,670.0	6,679,912.0	416.5	195.6	43
Geol+Grade+Class	StrongBow	NK07-014	526,742.0	6,679,972.0	415.0	194.0	63
Geol+Grade+Class	StrongBow	NK07-015	527,122.0	6,679,754.0	405.0	182.0	65
Geol+Grade+Class	StrongBow	NK08-020	526,160.0	6,680,094.0	396.0	261.9	98
Geol+Grade+Class	StrongBow	NK08-021	525,955.0	6,680,088.0	396.0	289.3	127
Geol+Grade+Class	StrongBow	NK08-022	525,774.0	6,679,999.0	396.0	326.1	140
Geol+Grade+Class	StrongBow	NK08-023	525,470.0	6,679,768.0	396.0	363.7	146
Geol+Grade+Class	StrongBow	NK08-024	525,381.0	6,679,553.0	396.0	209.7	79
Geol+Grade+Class	StrongBow	NK08-025	526,008.0	6,680,110.0	396.0	260.5	106
Geol+Grade+Class	StrongBow	NK08-026	525,527.0	6,679,837.0	396.0	185.3	111
Geol+Grade+Class	StrongBow	NK08-027	525,871.0	6,680,032.0	396.0	326.0	172
Geol+Grade+Class	StrongBow	NK08-028	525,871.0	6,680,032.0	396.0	313.0	130
Geol+Grade+Class	StrongBow	NK08-029	525,871.0	6,680,032.0	396.0	312.3	144
Geol+Grade+Class	StrongBow	NK08-030	525,420.0	6,679,653.0	396.0	244.5	148
Geol+Grade+Class	StrongBow	NK08-032	525,137.0	6,679,557.0	396.0	121.5	49
Geol+Grade+Class	StrongBow	NK08-033	525,993.0	6,680,130.0	396.0	57.7	30
Geol+Grade+Class	StrongBow	NK08-033A	525,993.6	6,680,132.9	396.0	289.0	119
Geol+Grade+Class	StrongBow	NK08-034	525,774.0	6,680,004.0	396.0	25.3	0

Geol+Grade+Class	StrongBow	NK08-034B	525,773.0	6,680,007.0	396.0	149.8	44
Geol+Grade+Class	StrongBow	NK08-035	526,281.0	6,680,139.0	403.0	212.2	117
Geol+Grade+Class	StrongBow	NK08-036	526,281.0	6,680,139.0	403.0	200.2	110
Geol+Grade+Class	StrongBow	NK08-037	526,282.0	6,680,136.0	403.0	230.4	70
Geol+Grade+Class	StrongBow	NK08-038	526,020.0	6,680,033.0	396.0	279.0	78
Geol+Grade+Class	StrongBow	NK08-039	526,072.0	6,680,107.0	398.0	247.0	143
Geol+Grade+Class	StrongBow	NK08-040	526,072.0	6,680,107.0	398.0	263.0	147
Geol+Grade+Class	StrongBow	NK08-041	526,071.0	6,680,106.0	398.0	274.9	155
Geol+Grade+Class	StrongBow	NK08-042	526,074.0	6,680,103.0	398.0	278.2	161
Geol+Grade+Class	StrongBow	NK08-043	526,133.0	6,680,176.0	403.0	237.0	120
Geol+Grade+Class	StrongBow	NK08-045	525,366.5	6,679,149.5	397.5	390.0	163
Geol+Grade+Class	StrongBow	NK08-049	527,653.0	6,679,901.0	416.0	99.0	19
Geol+Grade+Class	StrongBow	NK08-050	527,724.0	6,679,821.0	415.0	75.0	20
Geol+Grade+Class	StrongBow	NK08-051	526,358.0	6,680,145.0	400.0	201.4	77
Geol+Grade+Class	StrongBow	NK08-052	526,358.0	6,680,145.0	400.0	197.0	40
Geol+Grade+Class	StrongBow	NK08-053	527,478.0	6,680,139.0	428.0	45.0	26
Geol+Grade+Class	StrongBow	NK08-054	526,991.0	6,679,908.0	399.0	150.0	50
Geol+Grade+Class	StrongBow	NK08-057	526,636.0	6,680,100.0	421.0	195.1	90
Geol+Grade+Class	StrongBow	NK08-060	526,302.0	6,680,175.0	400.0	249.1	84
Geol+Grade+Class	StrongBow	NK08-061	526,405.0	6,680,142.0	400.0	174.0	56
Geol+Grade+Class	StrongBow	NK08-062	526,405.0	6,680,142.0	400.0	195.0	55
Geol+Grade+Class	StrongBow	NK08-063	526,295.0	6,680,033.0	401.0	207.0	14
Total			76.0			16,328.3	5,109.0

The following are holes outside the map area - Not used for resource estimate

Not used	Inco	10233	523,188.5	6,678,553.4	399.1	12.8	23
Not used	Inco	10234	524,248.3	6,679,532.2	396.6	115.2	
Not used	Inco	10235	523,192.6	6,678,595.0	399.1	152.7	
Not used	Inco	10248	523,213.2	6,678,589.5	397.5	215.2	7
Not used	StrongBow	NK07-008	523,269.0	6,678,541.0	406.0	146.0	31
Not used	StrongBow	NK07-009	523,322.0	6,678,628.0	416.0	113.0	24
Not used	StrongBow	NK07-010	523,232.0	6,678,470.0	402.0	143.0	39
Not used	StrongBow	NK07-016	526,722.0	6,677,223.0	400.0	108.3	72
Not used	StrongBow	NK07-017	526,980.0	6,678,267.0	405.0	295.4	182
Not used	StrongBow	NK07-018	524,409.0	6,677,727.0	406.0	136.9	53
Not used	StrongBow	NK07-019	523,469.0	6,676,094.0	419.0	152.0	55
Not used	StrongBow	NK08-031	526,714.0	6,676,870.0	398.0	250.0	70
Not used	StrongBow	NK08-044	522,841.0	6,677,634.0	397.0	141.2	33
Not used	StrongBow	NK08-046	526,700.0	6,677,226.0	400.0	138.0	74
Not used	StrongBow	NK08-047	526,914.0	6,676,939.0	400.0	120.0	31
Not used	StrongBow	NK08-048	526,652.0	6,677,150.0	400.0	111.0	75
Not used	StrongBow	NK08-055	527,686.0	6,677,387.0	420.0	172.0	48
Not used	StrongBow	NK08-056	527,672.0	6,678,059.0	428.0	101.0	24
Not used	StrongBow	NK08-058	523,709.0	6,679,547.0	398.0	238.9	30
Not used	StrongBow	NK08-059	522,853.0	6,677,655.0	397.0	11.7	5
Not used	StrongBow	NK08-059A	522,855.0	6,677,656.0	397.0	55.7	8
Total			21.0			2,930.0	884.0

APPENDIX B

RAW ASSAY STATISTICS

Descriptive Statistics

Both Sills combined

	NI	CU	CO	S	AG	FE	PB	ZN
Valid cases	3801	3789	3795	3548	3548	3524	3548	3548
Mean	0.275	0.066	136.280	1.926	0.545	5.526	22.799	48.565
Std. error of mean	0.005	0.001	2.496	0.045	0.024	0.086	0.505	0.829
Variance	0.102	0.006	23,647.102	7.165	2.053	26.063	905.919	2,440.505
Std. Deviation	0.320	0.078	153.776	2.677	1.433	5.105	30.098	49.401
Variation Coefficient	1.162	1.181	1.128	1.390	2.629	0.924	1.320	1.017
rel. V.coefficient(%)	1.885	1.918	1.832	2.333	4.413	1.556	2.216	1.708
Skew	2.657	2.189	2.649	3.023	44.842	1.618	1.685	2.252
Kurtosis	13.077	6.199	9.293	12.351	2,407.771	3.368	5.226	9.765
Minimum	-	-	0.500	0.020	0.050	0.170	0.300	2.000
Maximum	3.872	0.624	1,510.000	29.320	78.000	44.640	352.700	631.000
Range	3.872	0.624	1,509.500	29.300	77.950	44.470	352.400	629.000
Sum	1,046.773	249.976	517,183.700	6,834.210	1,934.000	19,474.560	80,891.900	172,309.000
1st percentile	0.004	0.002	5.000	0.020	0.050	0.642	0.800	6.000
5th percentile	0.015	0.004	18.000	0.100	0.050	0.850	1.100	8.000
10th percentile	0.021	0.005	21.900	0.140	0.050	1.050	1.400	9.000
25th percentile	0.049	0.011	38.600	0.300	0.100	1.680	2.100	13.000
Median	0.164	0.037	85.200	0.980	0.300	3.505	4.500	36.000
75th percentile	0.389	0.090	170.000	2.418	1.000	8.300	50.000	50.000
90th percentile	0.672	0.165	300.000	4.741	1.000	12.460	50.000	114.000
95th percentile	0.910	0.232	460.000	6.901	1.000	15.480	100.000	153.000
99th percentile	1.415	0.350	760.400	13.570	3.000	22.950	100.000	200.000
Geom. mean	----	----	82.456	0.868	0.271	3.644	7.742	30.705

Descriptive Statistics [Subset]

Both Sills combined

	NI	CU	CO
Valid cases	1932	1927	1928
Mean	0.315	0.073	140.628
Std. error of mean	0.008	0.002	3.328
Variance	0.136	0.007	21,354.359
Std. Deviation	0.369	0.082	146.131
Variation Coefficient	1.170	1.113	1.039
rel. V.coefficient(%)	2.663	2.535	2.367
Skew	2.623	1.923	2.646
Kurtosis	12.430	5.327	12.446
Minimum	-	-	0.500
Maximum	3.872	0.624	1,510.000
Range	3.872	0.624	1,509.500
Sum	608.405	141.481	271,130.800
1st percentile	0.004	0.002	5.000
5th percentile	0.015	0.004	17.200
10th percentile	0.020	0.005	20.190
25th percentile	0.044	0.010	34.900
Median	0.193	0.045	93.650
75th percentile	0.479	0.111	200.000
90th percentile	0.726	0.177	300.840
95th percentile	0.975	0.232	410.000
99th percentile	1.668	0.366	670.000
Geom. mean	----	----	85.054

Descriptive Statistics [Subset]

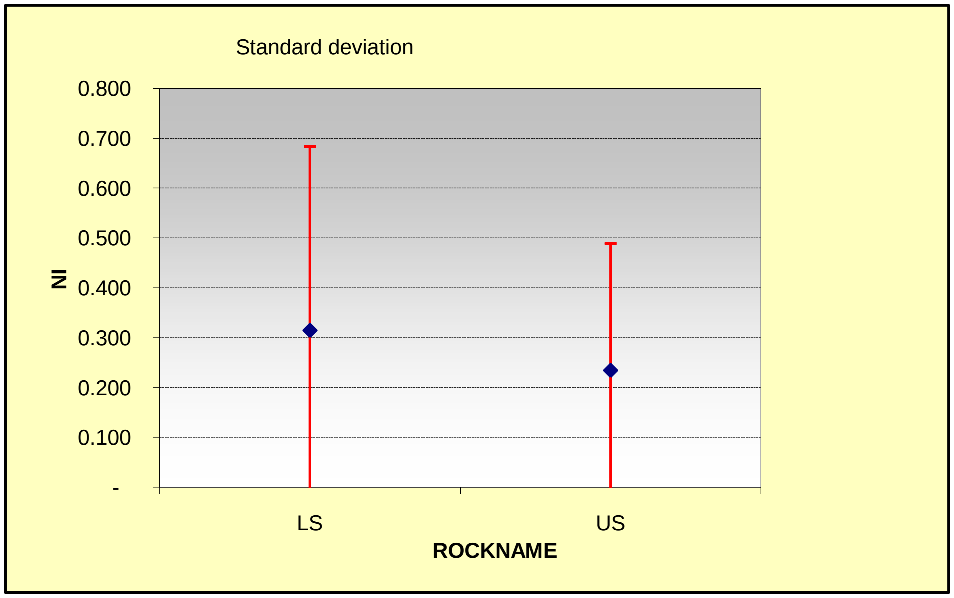
Both Sills combined

	NI	CU	CO
Valid cases	1869	1862	1867
Mean	0.235	0.058	131.791
Std. error of mean	0.006	0.002	3.731
Variance	0.065	0.005	25,987.769
Std. Deviation	0.254	0.073	161.207
Variation Coefficient	1.085	1.252	1.223
rel. V.coefficient(%)	2.509	2.901	2.831
Skew	1.998	2.548	2.658
Kurtosis	4.468	7.575	7.009
Minimum	-	0.000	0.500
Maximum	1.817	0.542	920.000
Range	1.817	0.542	919.500
Sum	438.368	108.495	246,052.900
1st percentile	0.004	0.002	5.000
5th percentile	0.014	0.004	19.300
10th percentile	0.022	0.006	23.800
25th percentile	0.056	0.013	40.800
Median	0.151	0.032	80.000
75th percentile	0.304	0.071	140.000
90th percentile	0.561	0.139	300.000
95th percentile	0.805	0.234	537.720
99th percentile	1.186	0.341	790.000
Geom. mean	----	0.030	79.857

Means

Variable: NI
grouped by: ROCKNAME

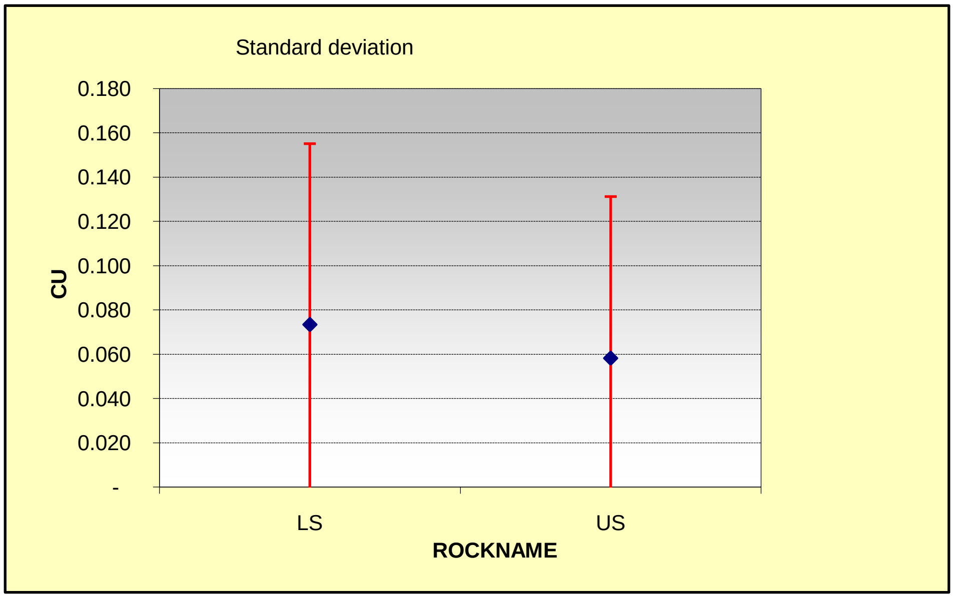
	N	Mean	95% Conf. (±)	Std.Error	Std.Dev.
LS	1932	0.315	0.016	0.008	0.369
US	1869	0.235	0.012	0.006	0.254
Entire sample	3801	0.275	0.010	0.005	0.320



Means

Variable: CU
grouped by: ROCKNAME

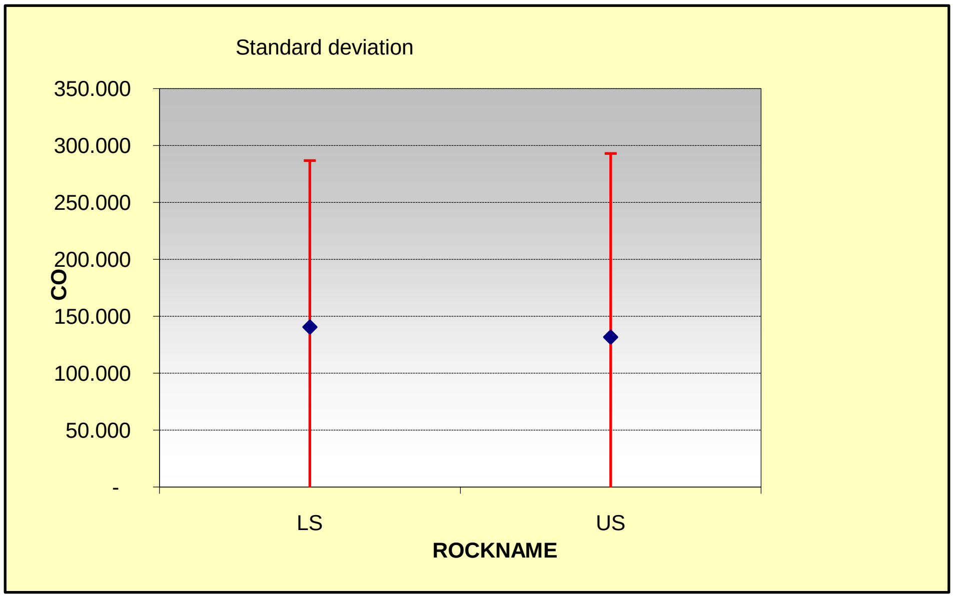
	N	Mean	95% Conf. (±)	Std.Error	Std.Dev.
LS	1927	0.073	0.004	0.002	0.082
US	1862	0.058	0.003	0.002	0.073
Entire sample	3789	0.066	0.002	0.001	0.078



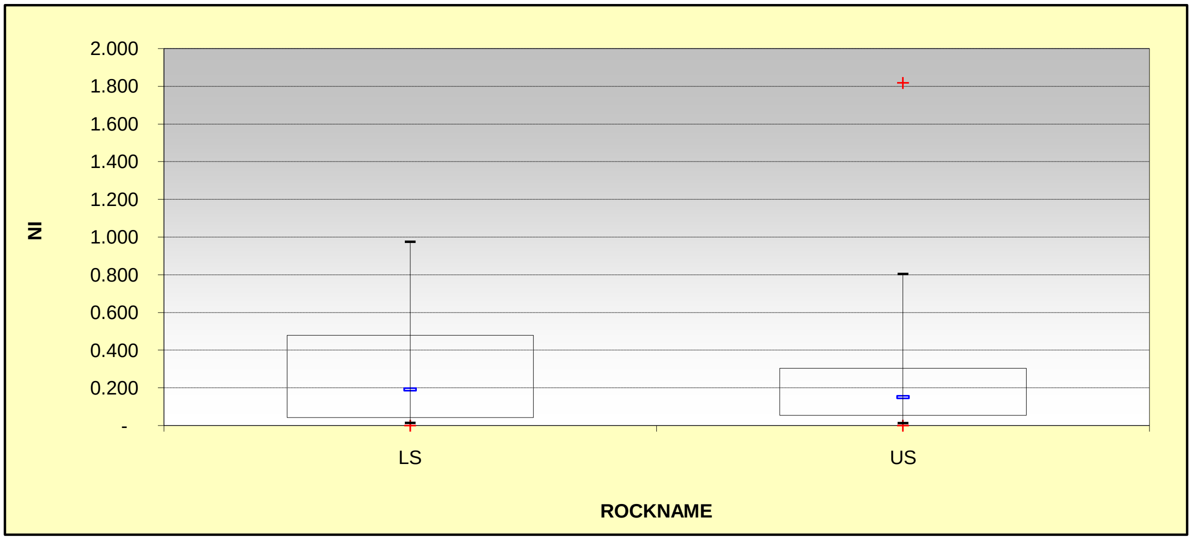
Means

Variable: CO
grouped by: ROCKNAME

	N	Mean	95% Conf. (±)	Std.Error	Std.Dev.
LS	1928	140.628	6.527	3.328	146.131
US	1867	131.791	7.317	3.731	161.207
Entire sample	3795	136.280	4.894	2.496	153.776

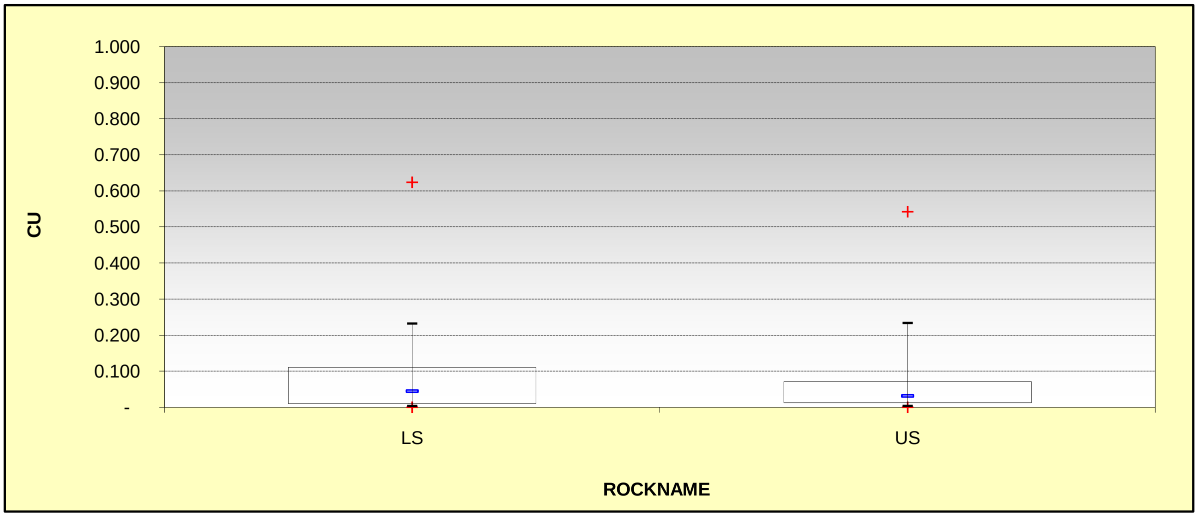


Box & Whisker



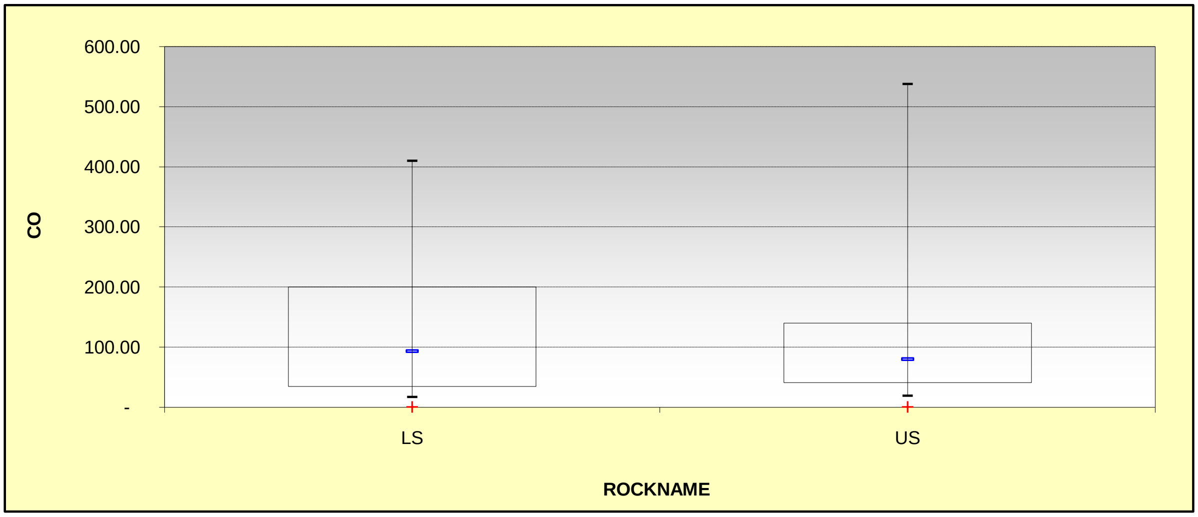
	Min	5%	25%	Median	75%	95%	Max
LS	-	0.015	0.044	0.193	0.479	0.975	3.872
US	-	0.014	0.056	0.151	0.304	0.805	1.817

Box & Whisker



	Min	5%	25%	Median	75%	95%	Max
LS	-	0.004	0.010	0.045	0.111	0.232	0.624
US	0.000	0.004	0.013	0.032	0.071	0.234	0.542

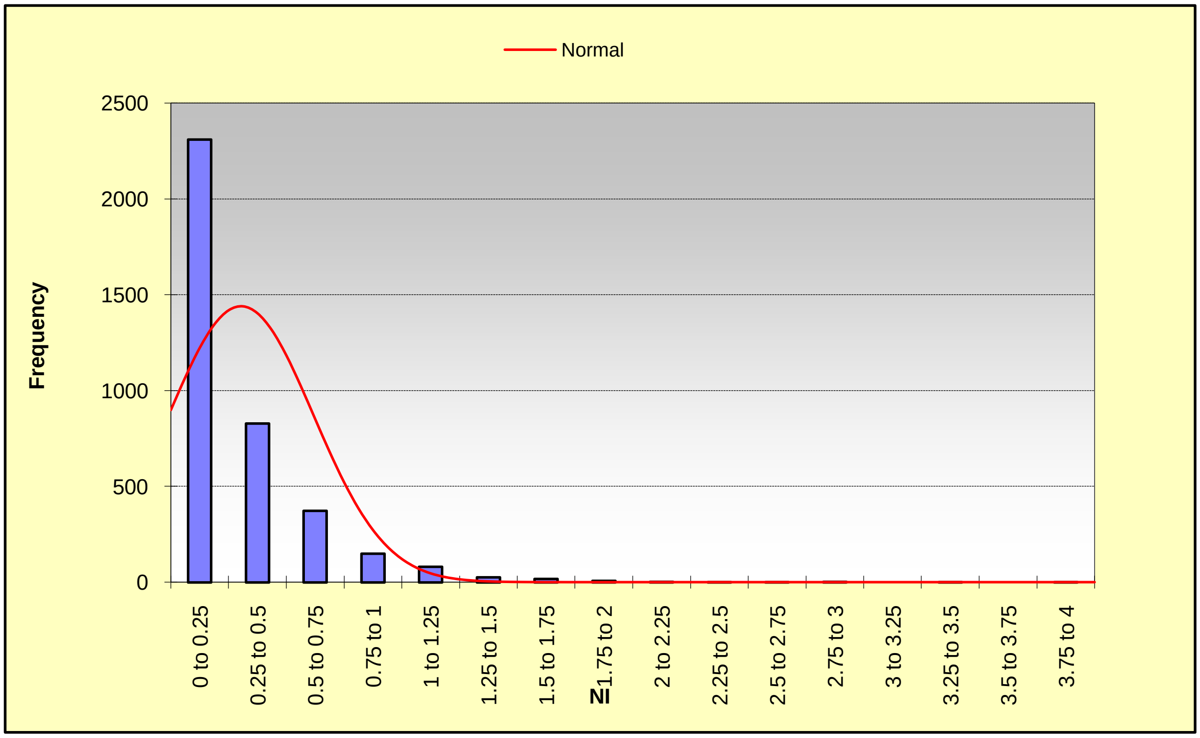
Box & Whisker



	Min	5%	25%	Median	75%	95%	Max
LS	0.50	17.20	34.90	93.65	200.00	410.00	1,510.00
US	0.50	19.30	40.80	80.00	140.00	537.72	920.00

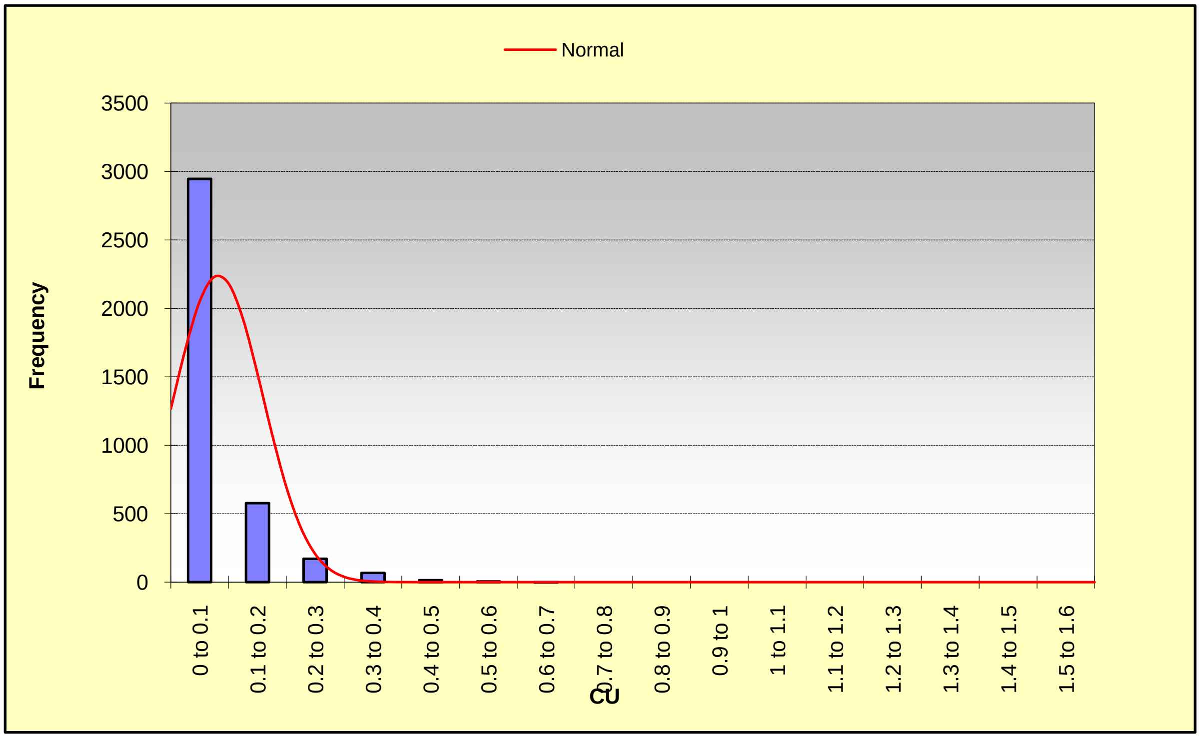
Frequencies

NI	Frequency	Percent	Cumulative Percent
0 to 0.25	2309	60.75	60.75
0.25 to 0.5	829	21.81	82.56
0.5 to 0.75	373	9.81	92.37
0.75 to 1	150	3.95	96.32
1 to 1.25	81	2.13	98.45
1.25 to 1.5	26	0.68	99.13
1.5 to 1.75	17	0.45	99.58
1.75 to 2	7	0.18	99.76
2 to 2.25	3	0.08	99.84
2.25 to 2.5	1	0.03	99.87
2.5 to 2.75	1	0.03	99.89
2.75 to 3	2	0.05	99.95
3 to 3.25	0	0.00	99.95
3.25 to 3.5	1	0.03	99.97
3.5 to 3.75	0	0.00	99.97
3.75 to 4	1	0.03	100.00



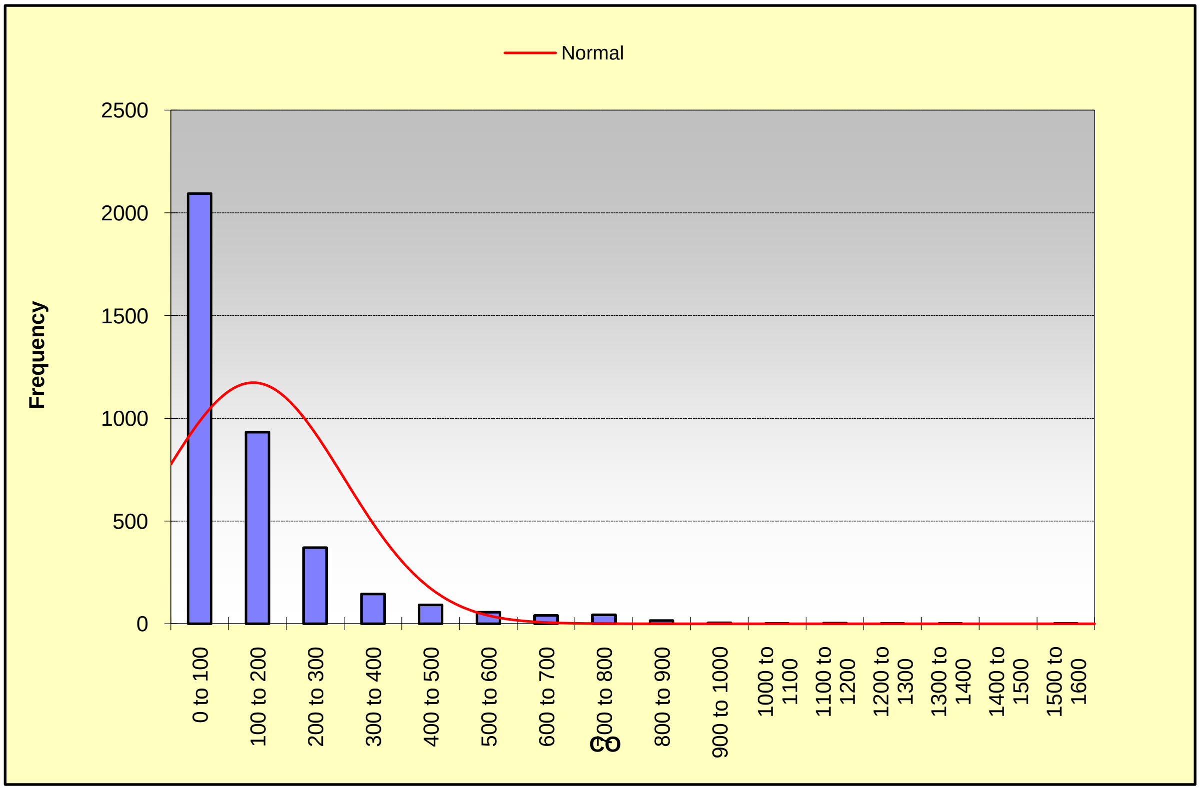
Frequencies

CU	Frequency	Percent	Cumulative Percent
0 to 0.1	2947	77.78	77.78
0.1 to 0.2	579	15.28	93.06
0.2 to 0.3	172	4.54	97.60
0.3 to 0.4	69	1.82	99.42
0.4 to 0.5	16	0.42	99.84
0.5 to 0.6	5	0.13	99.97
0.6 to 0.7	1	0.03	100.00
0.7 to 0.8	0	0.00	100.00
0.8 to 0.9	0	0.00	100.00
0.9 to 1	0	0.00	100.00
1 to 1.1	0	0.00	100.00
1.1 to 1.2	0	0.00	100.00
1.2 to 1.3	0	0.00	100.00
1.3 to 1.4	0	0.00	100.00
1.4 to 1.5	0	0.00	100.00
1.5 to 1.6	0	0.00	100.00



Frequencies

CO	Frequency	Percent	Cumulative Percent
0 to 100	2094	55.18	55.18
100 to 200	932	24.56	79.74
200 to 300	370	9.75	89.49
300 to 400	145	3.82	93.31
400 to 500	91	2.40	95.70
500 to 600	55	1.45	97.15
600 to 700	40	1.05	98.21
700 to 800	43	1.13	99.34
800 to 900	15	0.40	99.74
900 to 1000	4	0.11	99.84
1000 to 1100	1	0.03	99.87
1100 to 1200	2	0.05	99.92
1200 to 1300	1	0.03	99.95
1300 to 1400	1	0.03	99.97
1400 to 1500	0	0.00	99.97
1500 to 1600	1	0.03	100.00



Pearson Correlation

Upper and Lower sill

	NI	CU	CO
NI			
Correlation coefficient	1	0.896896521	0.937453131
valid cases	3801	3789	3795
one-sided significance	0	0	0
CU			
Correlation coefficient	0.896896521	1	0.892826419
valid cases	3789	3789	3789
one-sided significance	0	0	0
CO			
Correlation coefficient	0.937453131	0.892826419	1
valid cases	3795	3789	3795
one-sided significance	0	0	0

Cronbach's Alpha	0.008263081
Scott's Homogeneity-Quotier	1.069674829

Pearson Correlation

Lower sill

		NI	CU	CO
NI				
	Correlation coefficient	1	0.903022749	0.990640684
	valid cases	1932	1927	1928
	one-sided significance	0	0	0
CU				
	Correlation coefficient	0.903022749	1	0.896754468
	valid cases	1927	1927	1927
	one-sided significance	0	0	0
CO				
	Correlation coefficient	0.990640684	0.896754468	1
	valid cases	1928	1927	1928
	one-sided significance	0	0	0

Cronbach's Alpha	0.009198233
Scott's Homogeneity-Quotier	1.000752579

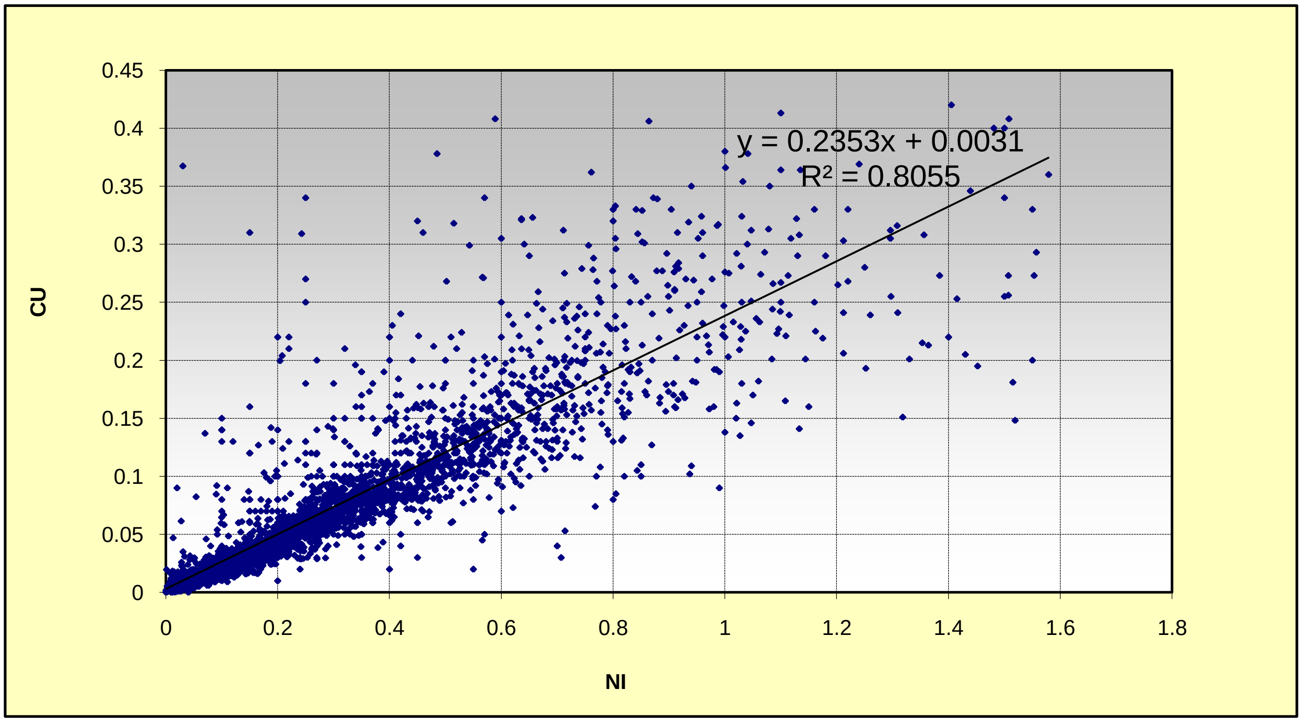
Pearson Correlation

Upper Sill

	NI	CU	CO
NI			
Correlation coefficient	1	0.902771552	0.935286777
valid cases	1869	1862	1867
one-sided significance	0	0	0
CU			
Correlation coefficient	0.902771552	1	0.902854681
valid cases	1862	1862	1862
one-sided significance	0	0	0
CO			
Correlation coefficient	0.935286777	0.902854681	1
valid cases	1867	1862	1867
one-sided significance	0	0	0

Cronbach's Alpha	0.007840861
Scott's Homogeneity-Quotier	1.293192762

Scatterplot [Subset]



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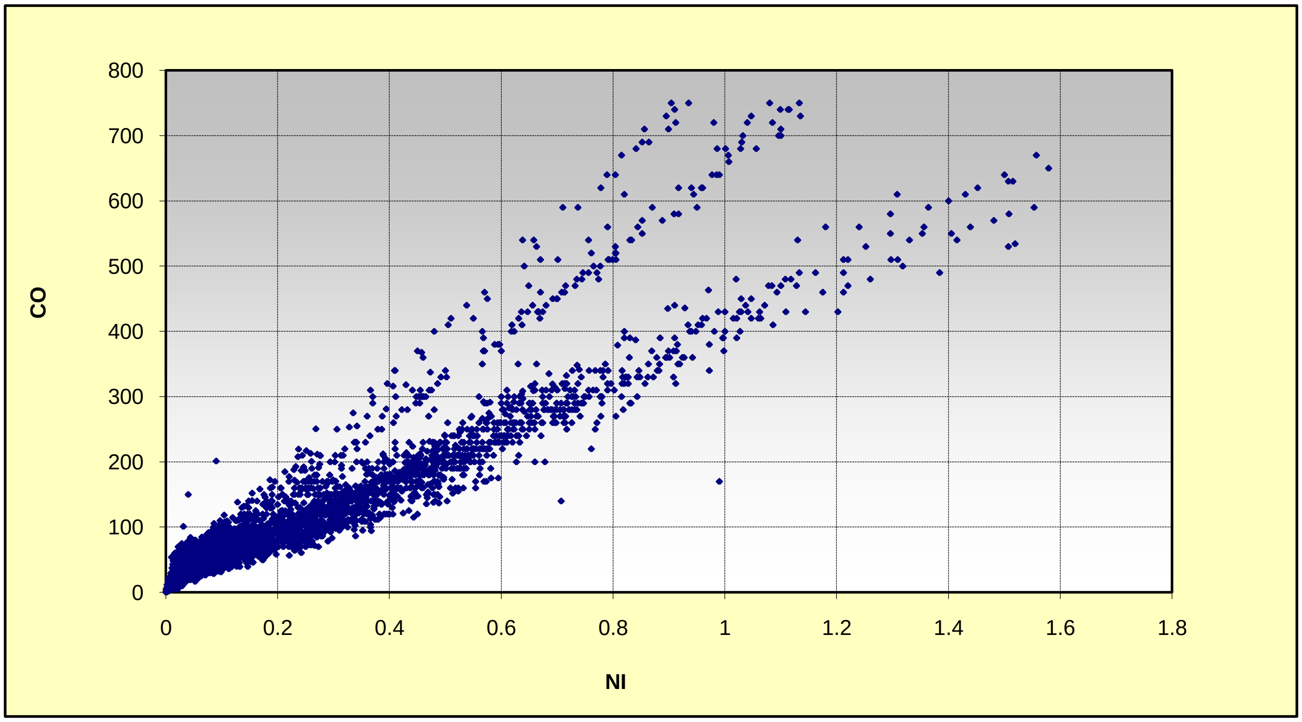

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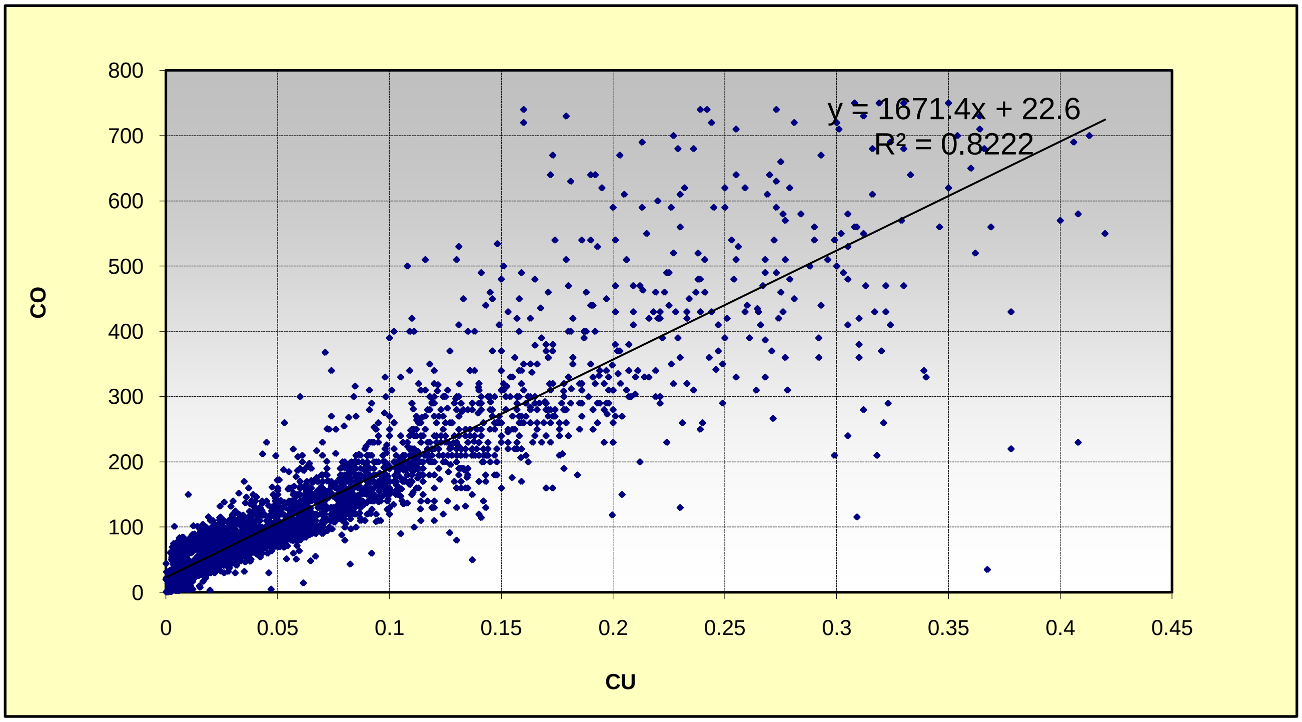

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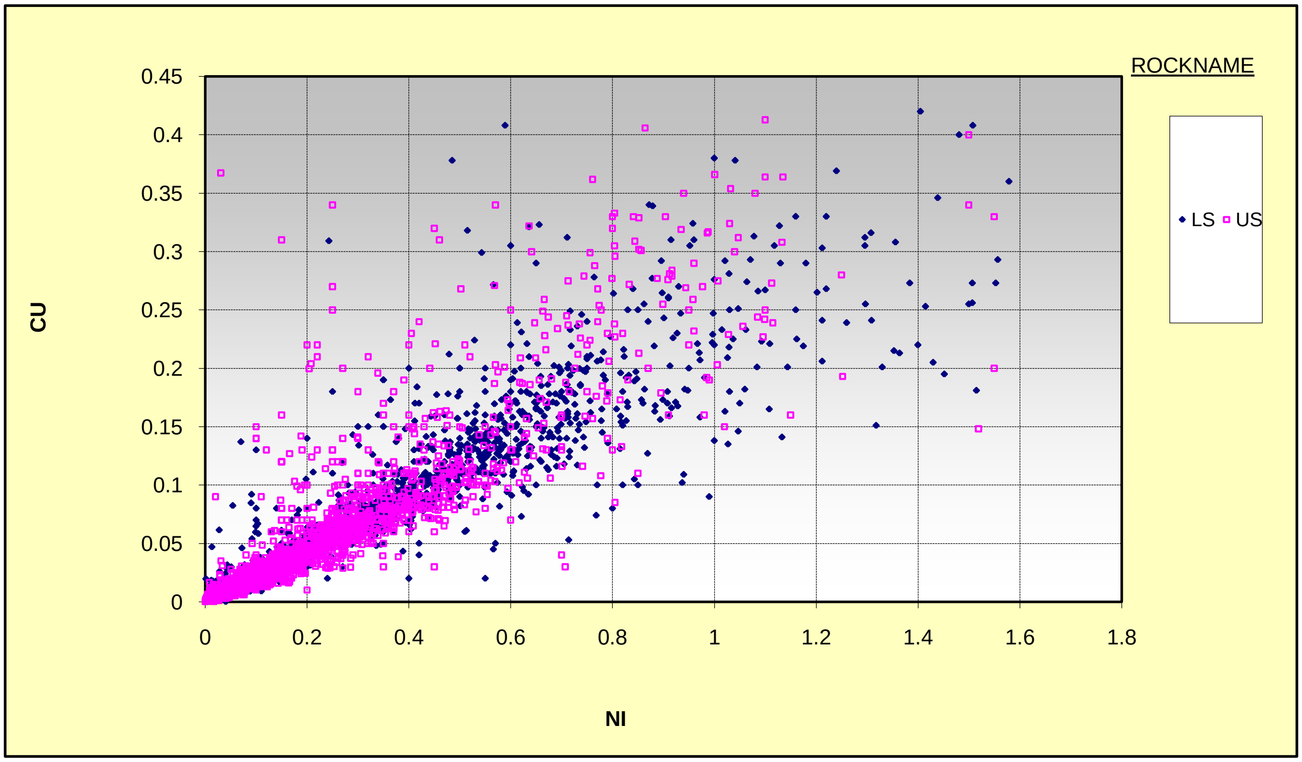

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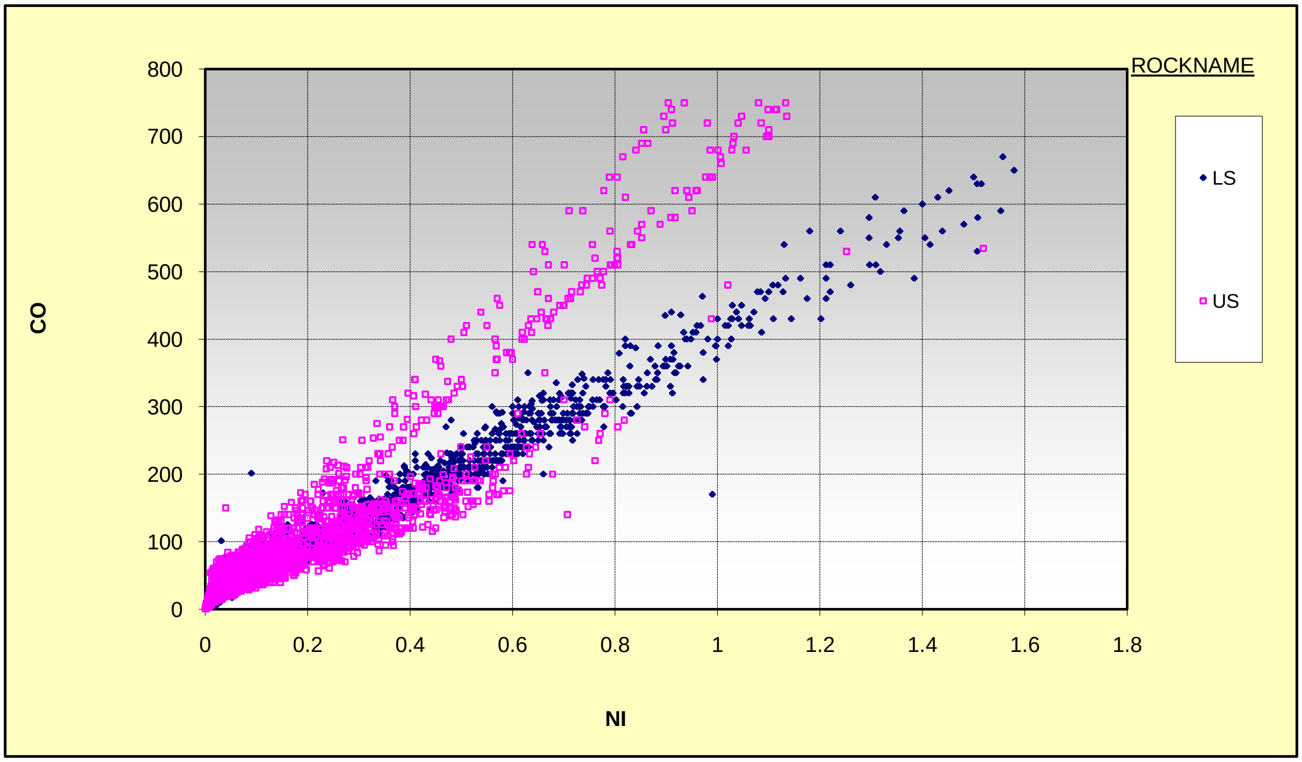


APPENDIX C

DECILE ANALYSIS RESULTS

Decile analysis Report and Capping study

Decile Analysis Nickel King - Upper Sill >>> Ni

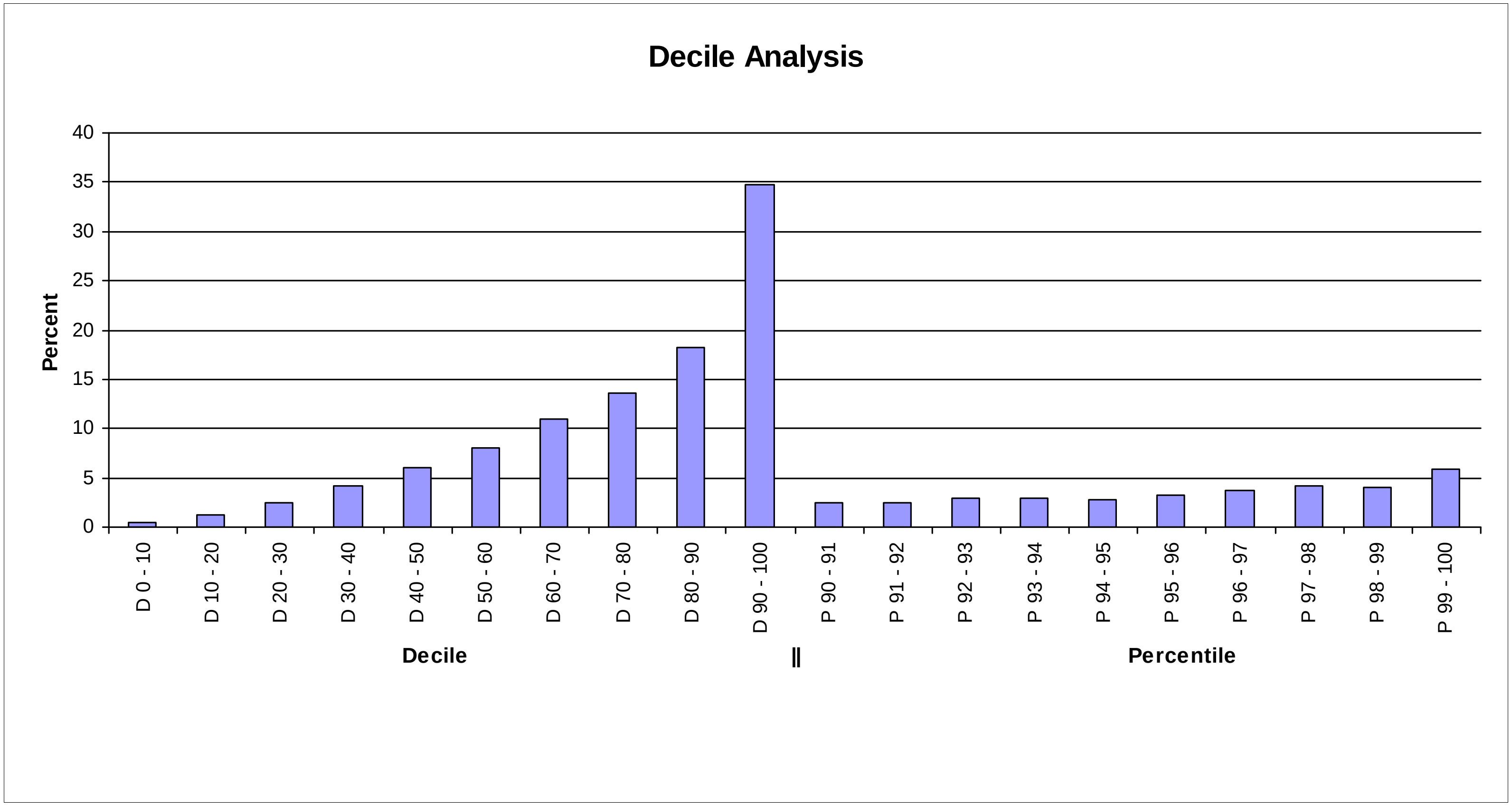
			------------ Element-------------			Total		
From	**To**	**Count**	**Mean**	**Min**	**Max**	**Metal**	**Percent**	**Capping Note**
Decile								
0	10	200	0.014	0.000	0.024	2.38	0.49	
10	20	200	0.035	0.024	0.047	6.32	1.30	
20	30	201	0.062	0.047	0.081	11.94	2.45	
30	40	200	0.101	0.081	0.119	20.41	4.20	
40	50	201	0.140	0.119	0.160	29.41	6.05	
50	60	201	0.190	0.160	0.223	39.19	8.06	
60	70	200	0.252	0.224	0.278	53.09	10.91	
70	80	201	0.322	0.279	0.374	66.33	13.64	
80	90	200	0.457	0.374	0.570	88.66	18.23	
90	100	201	0.914	0.573	3.000	168.72	34.69	<40 <2.3x -- <50 <3x
Percentile								
90	91	20	0.597	0.573	0.623	12.14	2.50	
91	92	20	0.647	0.627	0.667	12.15	2.50	
92	93	20	0.696	0.669	0.726	14.12	2.90	
93	94	20	0.756	0.732	0.778	14.29	2.94	
94	95	20	0.802	0.780	0.830	13.45	2.76	
95	96	20	0.880	0.833	0.933	15.98	3.28	
96	97	20	0.966	0.935	0.990	17.95	3.69	
97	98	20	1.052	1.001	1.100	20.09	4.13	
98	99	20	1.156	1.100	1.233	19.63	4.04	
99	100	21	1.556	1.239	3.000	28.91	5.94	<10 <1.75x -- <15 <2x
Total								
0	100	2005	0.249	0.000	3.000	486.44	100.00	

Interpretation notes:

Capping is warranted if

the last decile has more than 40 percent of metal; or,
the last decile contains more than 2.3 times the metal quantity contained in the one before last; or,
the last centile contains more than 10 percent of metal; or,
the last centile contains more than 1.75 times the metal quantity contained in the one before last.
---- Exception will be made if all following conditions are met:
the last decile has more than 50 percent metal; and,
the last decile contains more than 3 times the metal quantity contained in the one before last; and,
the last centile contains more than 15 percent of the metal; and,
the last centile contains more than 2 times the metal quantity contained in the one before last.

Nickel King - Upper Sill >>> Ni



Nickel King - Upper Sill >>> Ni

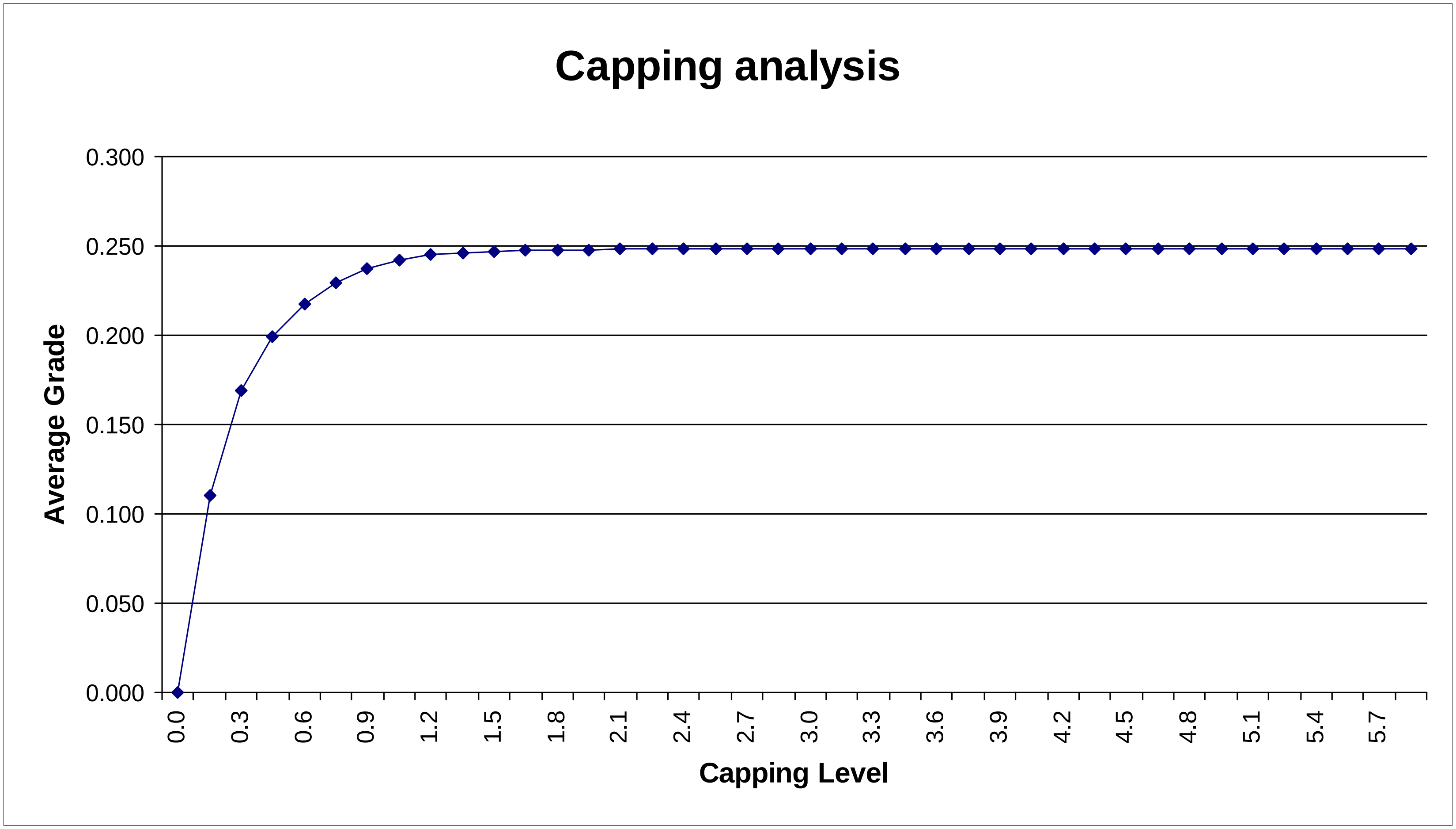
Capping Study Report

Element Basic Statistics

Count	2005	Min	0.000	Average	0.249	Max	3.000
Variance	0.077	Std Deviation	0.277	First Quartile	0.062	Third Quartile	0.317

Cap Level Bins	Average Grade	Number capped	Percent Nb capped	Percent Metal capped
5.85	0.249	0	0.000	0.000
5.70	0.249	0	0.000	0.000
5.55	0.249	0	0.000	0.000
5.40	0.249	0	0.000	0.000
5.25	0.249	0	0.000	0.000
5.10	0.249	0	0.000	0.000
4.95	0.249	0	0.000	0.000
4.80	0.249	0	0.000	0.000
4.65	0.249	0	0.000	0.000
4.50	0.249	0	0.000	0.000
4.35	0.249	0	0.000	0.000
4.20	0.249	0	0.000	0.000
4.05	0.249	0	0.000	0.000
3.90	0.249	0	0.000	0.000
3.75	0.249	0	0.000	0.000
3.60	0.249	0	0.000	0.000
3.45	0.249	0	0.000	0.000
3.30	0.249	0	0.000	0.000
3.15	0.249	0	0.000	0.000
3.00	0.249	1	0.050	0.000
2.85	0.249	1	0.050	0.007
2.70	0.249	1	0.050	0.015
2.55	0.248	1	0.050	0.022
2.40	0.248	2	0.100	0.070
2.25	0.248	2	0.100	0.124
2.10	0.248	2	0.100	0.179
1.95	0.248	2	0.100	0.233
1.80	0.248	4	0.200	0.291
1.65	0.247	5	0.249	0.403
1.50	0.247	10	0.499	0.547
1.35	0.246	12	0.599	0.801
1.20	0.245	25	1.247	1.253
1.05	0.242	50	2.494	2.238
0.90	0.237	88	4.389	4.158
0.75	0.229	135	6.733	7.113
0.60	0.217	191	9.526	11.636
0.45	0.200	309	15.411	18.385
0.30	0.169	551	27.481	30.374
0.15	0.110	1063	53.017	54.566
0.00	0.000	2005	100.000	100.000

Nickel King - Upper Sill >>> Ni



Decile analysis Report and Capping study

Decile Analysis Nickel King - Upper Sill >>> Cu

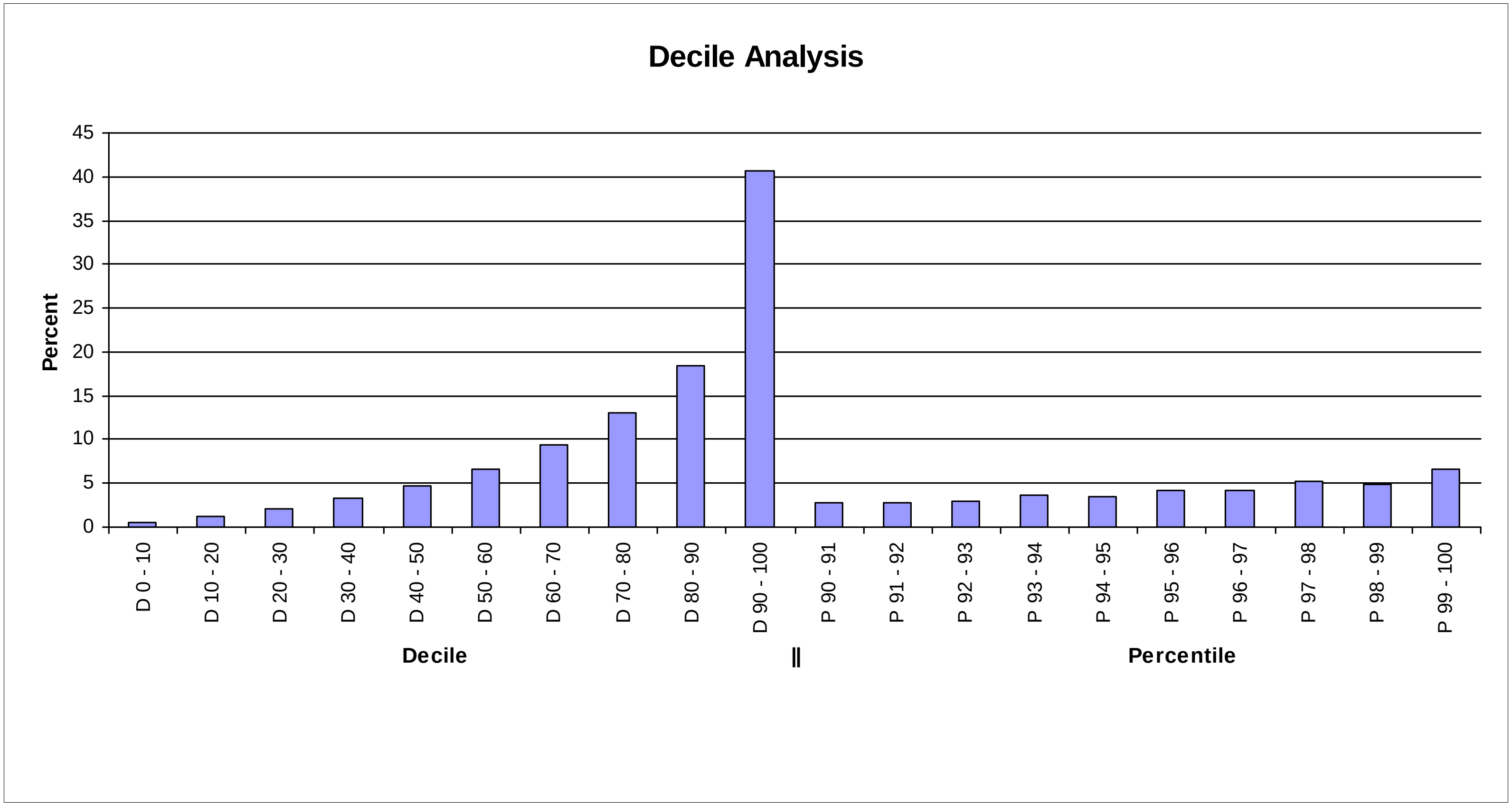
			------------ Element-------------			Total		
From	**To**	**Count**	**Mean**	**Min**	**Max**	**Metal**	**Percent**	**Capping Note**
Decile								
0	10	199	0.004	0.000	0.006	0.64	0.50	
10	20	200	0.008	0.006	0.011	1.60	1.24	
20	30	200	0.014	0.011	0.017	2.61	2.03	
30	40	199	0.022	0.017	0.026	4.26	3.32	
40	50	200	0.031	0.026	0.036	6.04	4.70	
50	60	200	0.043	0.036	0.051	8.55	6.66	
60	70	199	0.060	0.051	0.069	11.98	9.34	
70	80	200	0.079	0.070	0.092	16.78	13.07	
80	90	200	0.117	0.092	0.158	23.73	18.49	
90	100	200	0.267	0.158	1.160	52.16	40.64	>40 <2.3x -- <50 <3x
Percentile								
90	91	20	0.163	0.158	0.172	3.58	2.79	
91	92	20	0.182	0.173	0.190	3.66	2.85	
92	93	20	0.198	0.190	0.206	3.88	3.02	
93	94	20	0.217	0.209	0.227	4.75	3.70	
94	95	20	0.235	0.227	0.242	4.38	3.41	
95	96	20	0.254	0.243	0.270	5.29	4.12	
96	97	20	0.278	0.270	0.290	5.34	4.16	
97	98	20	0.307	0.293	0.319	6.62	5.16	
98	99	20	0.335	0.320	0.361	6.22	4.84	
99	100	20	0.499	0.362	1.160	8.45	6.59	<10 <1.75x -- <15 <2x
Total								
0	100	1997	0.065	0.000	1.160	128.35	100.00	

Interpretation notes:

Capping is warranted if

the last decile has more than 40 percent of metal; or,
the last decile contains more than 2.3 times the metal quantity contained in the one before last; or,
the last centile contains more than 10 percent of metal; or,
the last centile contains more than 1.75 times the metal quantity contained in the one before last.
---- Exception will be made if all following conditions are met:
the last decile has more than 50 percent metal; and,
the last decile contains more than 3 times the metal quantity contained in the one before last; and,
the last centile contains more than 15 percent of the metal; and,
the last centile contains more than 2 times the metal quantity contained in the one before last.

Nickel King - Upper Sill >>> Cu



Nickel King - Upper Sill >>> Cu

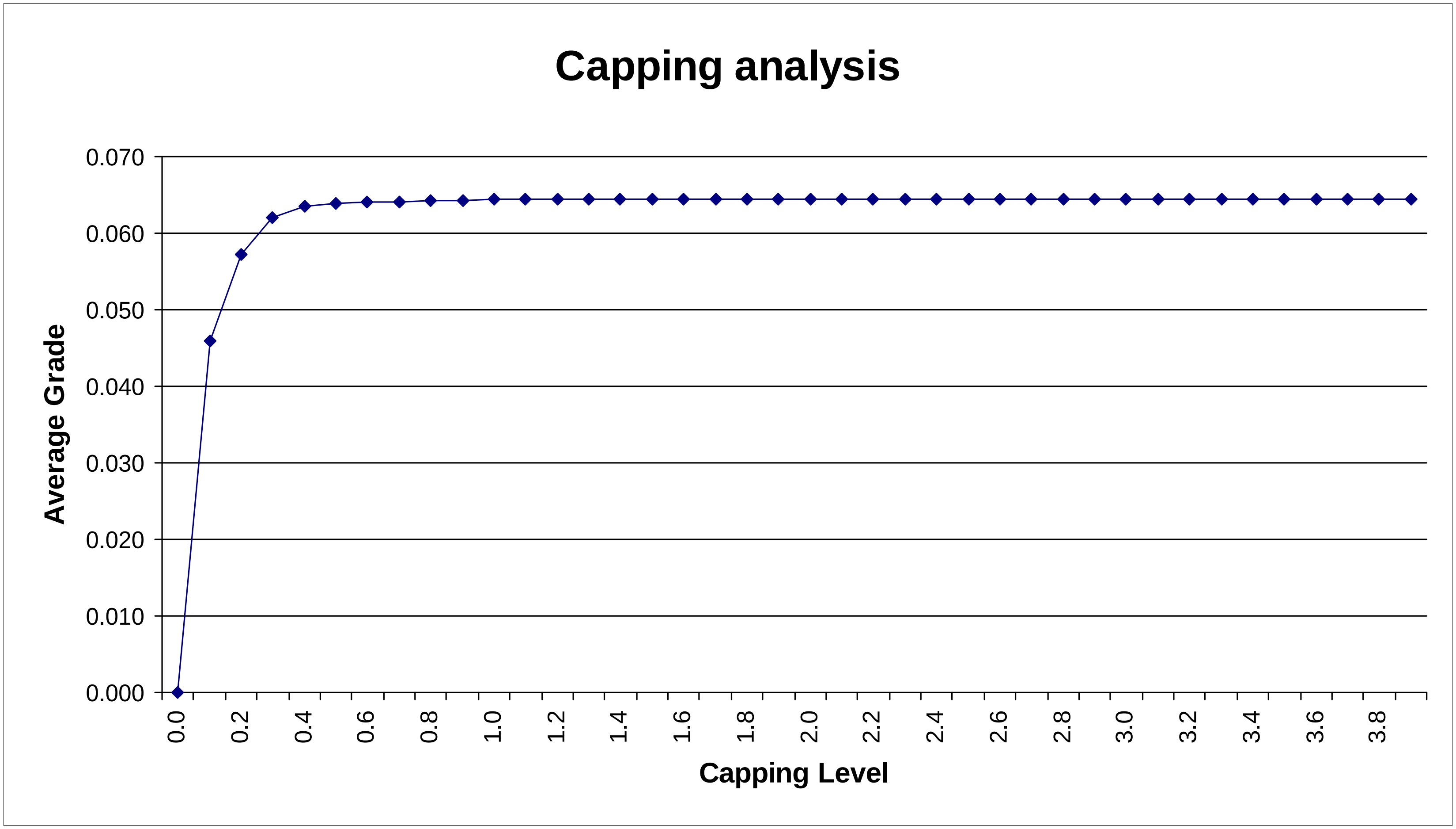
Capping Study Report

Element Basic Statistics

Count	1997	Min	0.000	Average	0.065	Max	1.160
Variance	0.007	Std Deviation	0.084	First Quartile	0.014	Third Quartile	0.080

Cap Level Bins	Average Grade	Number capped	Percent Nb capped	Percent Metal capped
3.90	0.065	0	0.000	0.000
3.80	0.065	0	0.000	0.000
3.70	0.065	0	0.000	0.000
3.60	0.065	0	0.000	0.000
3.50	0.065	0	0.000	0.000
3.40	0.065	0	0.000	0.000
3.30	0.065	0	0.000	0.000
3.20	0.065	0	0.000	0.000
3.10	0.065	0	0.000	0.000
3.00	0.065	0	0.000	0.000
2.90	0.065	0	0.000	0.000
2.80	0.065	0	0.000	0.000
2.70	0.065	0	0.000	0.000
2.60	0.065	0	0.000	0.000
2.50	0.065	0	0.000	0.000
2.40	0.065	0	0.000	0.000
2.30	0.065	0	0.000	0.000
2.20	0.065	0	0.000	0.000
2.10	0.065	0	0.000	0.000
2.00	0.065	0	0.000	0.000
1.90	0.065	0	0.000	0.000
1.80	0.065	0	0.000	0.000
1.70	0.065	0	0.000	0.000
1.60	0.065	0	0.000	0.000
1.50	0.065	0	0.000	0.000
1.40	0.065	0	0.000	0.000
1.30	0.065	0	0.000	0.000
1.20	0.065	0	0.000	0.000
1.10	0.064	1	0.050	0.071
1.00	0.064	2	0.100	0.201
0.90	0.064	2	0.100	0.338
0.80	0.064	2	0.100	0.475
0.70	0.064	2	0.100	0.612
0.60	0.064	2	0.100	0.749
0.50	0.064	5	0.250	0.921
0.40	0.063	15	0.751	1.481
0.30	0.062	57	2.854	3.295
0.20	0.057	151	7.561	10.590
0.10	0.046	368	18.428	28.029
0.00	0.000	1997	100.000	100.000

Nickel King - Upper Sill >>> Cu



Decile analysis Report and Capping study

Decile Analysis Nickel King - Upper Sill >>> Co

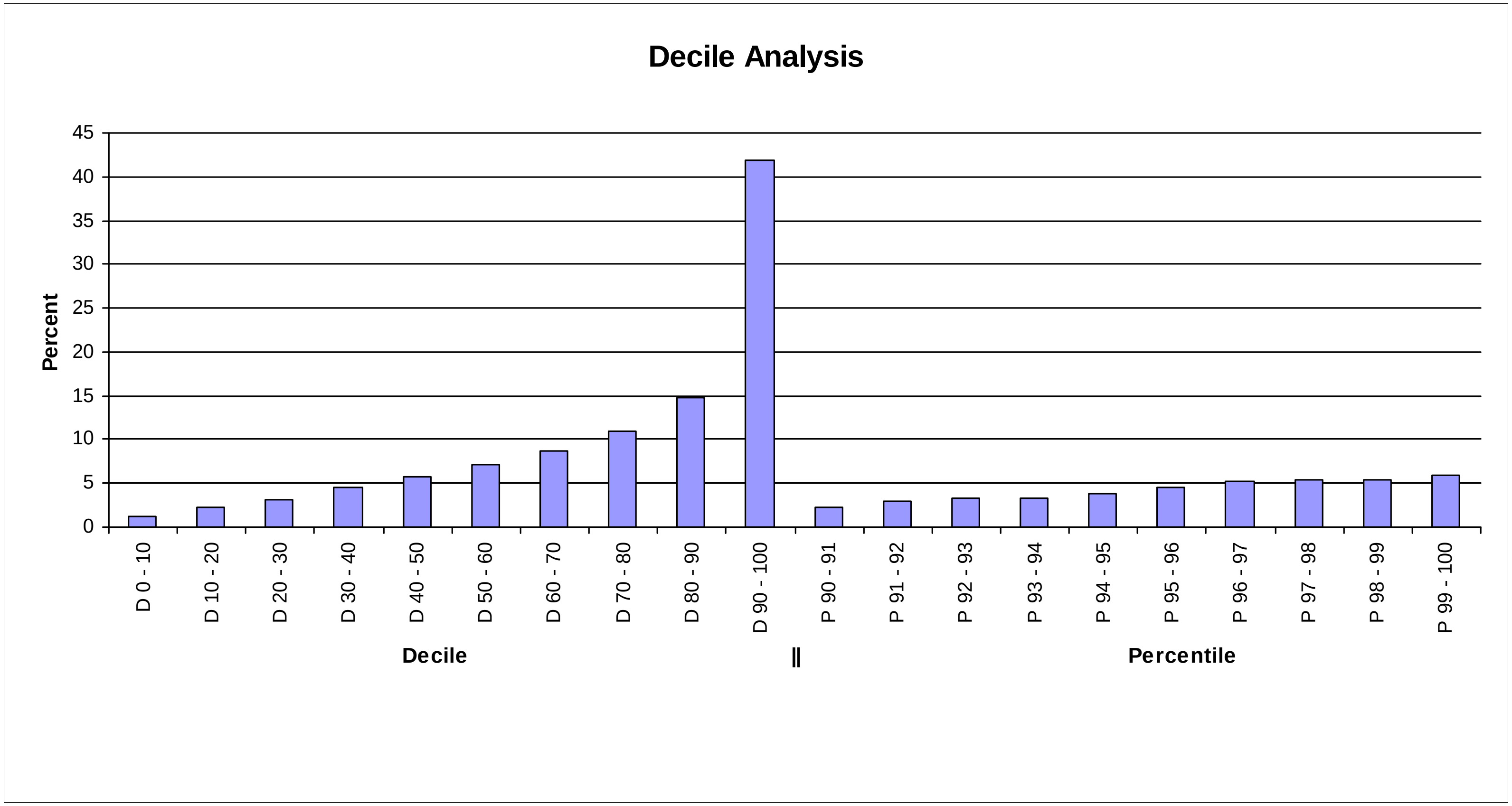
			------------ Element-------------			Total		
From	To	Count	Mean	Min	Max	Metal	Percent	Capping Note
Decile								
0	10	186	17.024	0.500	23.800	2,638.44	1.15	
10	20	187	28.845	23.800	34.500	5,019.39	2.19	
20	30	187	41.170	34.500	49.400	7,228.78	3.16	
30	40	186	57.048	49.400	64.600	10,260.61	4.48	
40	50	187	72.615	64.700	80.000	13,288.47	5.80	
50	60	187	89.320	80.000	100.000	16,244.52	7.09	
60	70	186	111.827	100.000	124.400	19,862.49	8.67	
70	80	187	141.409	124.700	160.000	24,929.48	10.88	
80	90	187	199.861	160.000	300.000	33,646.91	14.69	
90	100	187	557.666	300.000	920.000	95,915.94	41.88	>40 >2.3x -- <50 <3x
Percentile								
90	91	18	308.017	300.000	320.000	5,090.97	2.22	
91	92	19	359.221	330.000	390.000	6,635.20	2.90	
92	93	19	418.947	400.000	440.000	7,409.90	3.24	
93	94	18	460.000	440.000	480.000	7,367.00	3.22	
94	95	19	508.421	480.000	530.000	8,676.70	3.79	
95	96	19	567.595	534.300	610.000	10,446.60	4.56	
96	97	18	644.983	620.000	680.000	11,757.47	5.13	
97	98	19	703.158	680.000	730.000	12,366.80	5.40	
98	99	19	756.316	730.000	780.000	12,464.10	5.44	
99	100	19	836.316	790.000	920.000	13,701.20	5.98	<10 <1.75x -- <15 <2x
Total								
0	100	1867	131.791	0.500	920.000	229,035.03	100.00	

Interpretation notes:

Capping is warranted if

the last decile has more than 40 percent of metal; or,
the last decile contains more than 2.3 times the metal quantity contained in the one before last; or,
the last centile contains more than 10 percent of metal; or,
the last centile contains more than 1.75 times the metal quantity contained in the one before last.
---- Exception will be made if all following conditions are met:
the last decile has more than 50 percent metal; and,
the last decile contains more than 3 times the metal quantity contained in the one before last; and,
the last centile contains more than 15 percent of the metal; and,
the last centile contains more than 2 times the metal quantity contained in the one before last.

Nickel King - Upper Sill >>> Co



Nickel King - Upper Sill >>> Co

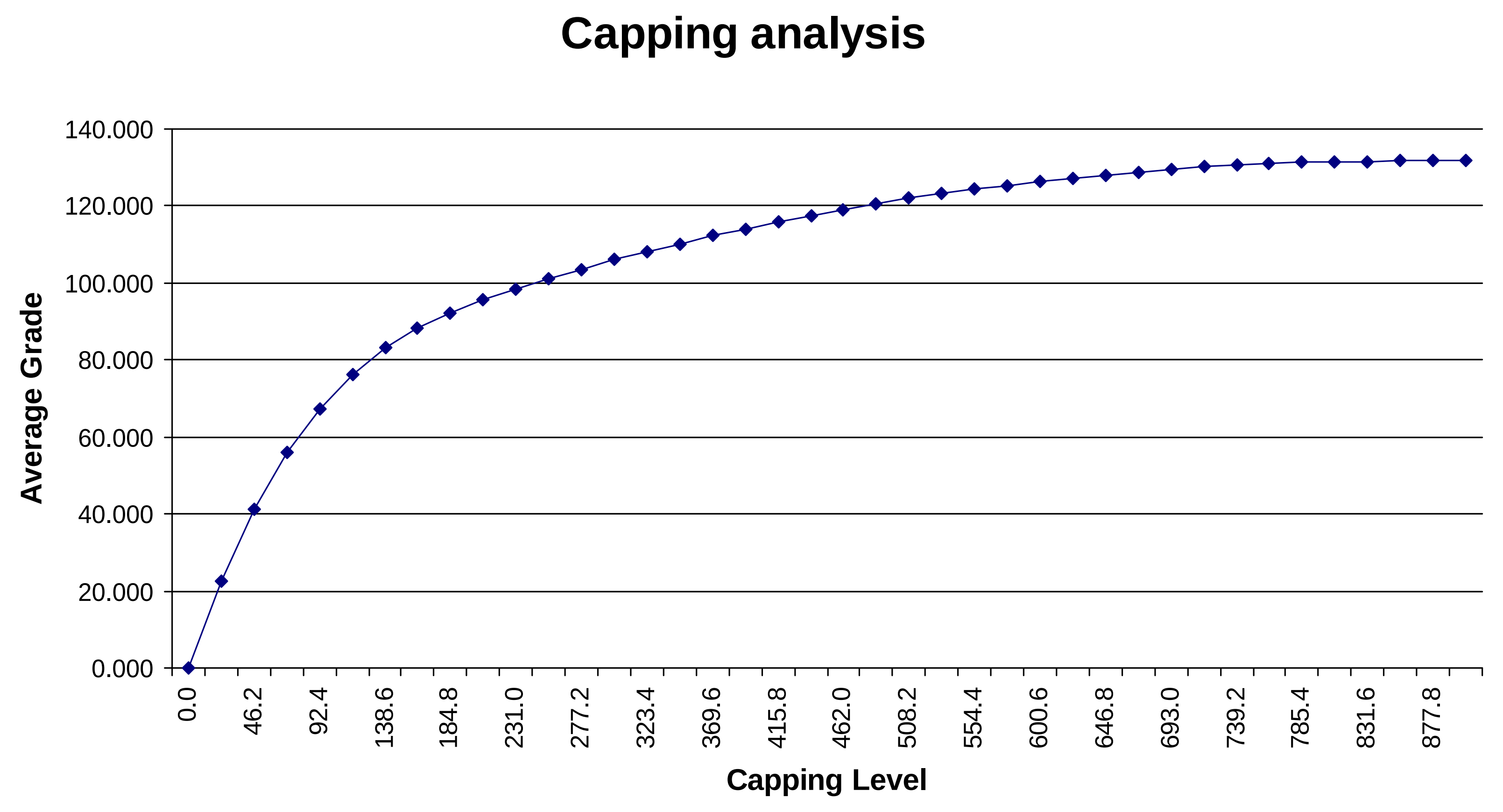
Capping Study Report

Element Basic Statistics

Count	1867	Min	0.500	Average	131.791	Max	920.000
Variance	5987.769	Std Deviation	161.207	First Quartile	40.700	Third Quartile	140.000

Cap Level Bins	Average Grade	Number capped	Percent Nb capped	Percent Metal capped
900.90	131.765	3	0.161	0.021
877.80	131.720	5	0.268	0.057
854.70	131.647	7	0.375	0.115
831.60	131.556	8	0.428	0.188
808.50	131.444	11	0.589	0.273
785.40	131.272	19	1.018	0.396
762.30	130.991	25	1.339	0.591
739.20	130.623	37	1.982	0.849
716.10	130.135	44	2.357	1.193
693.00	129.552	50	2.678	1.609
669.90	128.874	60	3.214	2.096
646.80	128.101	65	3.482	2.661
623.70	127.253	70	3.749	3.287
600.60	126.325	77	4.124	3.979
577.50	125.343	83	4.446	4.714
554.40	124.294	87	4.660	5.501
531.30	123.182	94	5.035	6.337
508.20	121.959	106	5.678	7.257
485.10	120.616	112	5.999	8.266
462.00	119.179	119	6.374	9.347
438.90	117.618	132	7.070	10.514
415.80	115.923	143	7.659	11.777
392.70	114.110	150	8.034	13.131
369.60	112.226	159	8.516	14.542
346.50	110.236	163	8.731	16.038
323.40	108.178	169	9.052	17.587
300.30	106.011	180	9.641	19.217
277.20	103.632	199	10.659	21.009
254.10	101.103	210	11.248	22.908
231.00	98.417	221	11.837	24.923
207.90	95.531	250	13.390	27.076
184.80	92.251	292	15.640	29.514
161.70	88.291	354	18.961	32.454
138.60	83.103	487	26.085	36.323
115.50	76.300	632	33.851	41.439
92.40	67.450	805	43.117	48.140
69.30	55.950	1083	58.007	56.964
46.20	41.029	1333	71.398	68.490
23.10	22.491	1700	91.055	82.809
0.00	0.000	1867	100.000	100.000

Nickel King - Upper Sill >>> Co



Decile analysis Report and Capping study

Decile Analysis Nickel King - Upper Sill >>> Cu (capped)

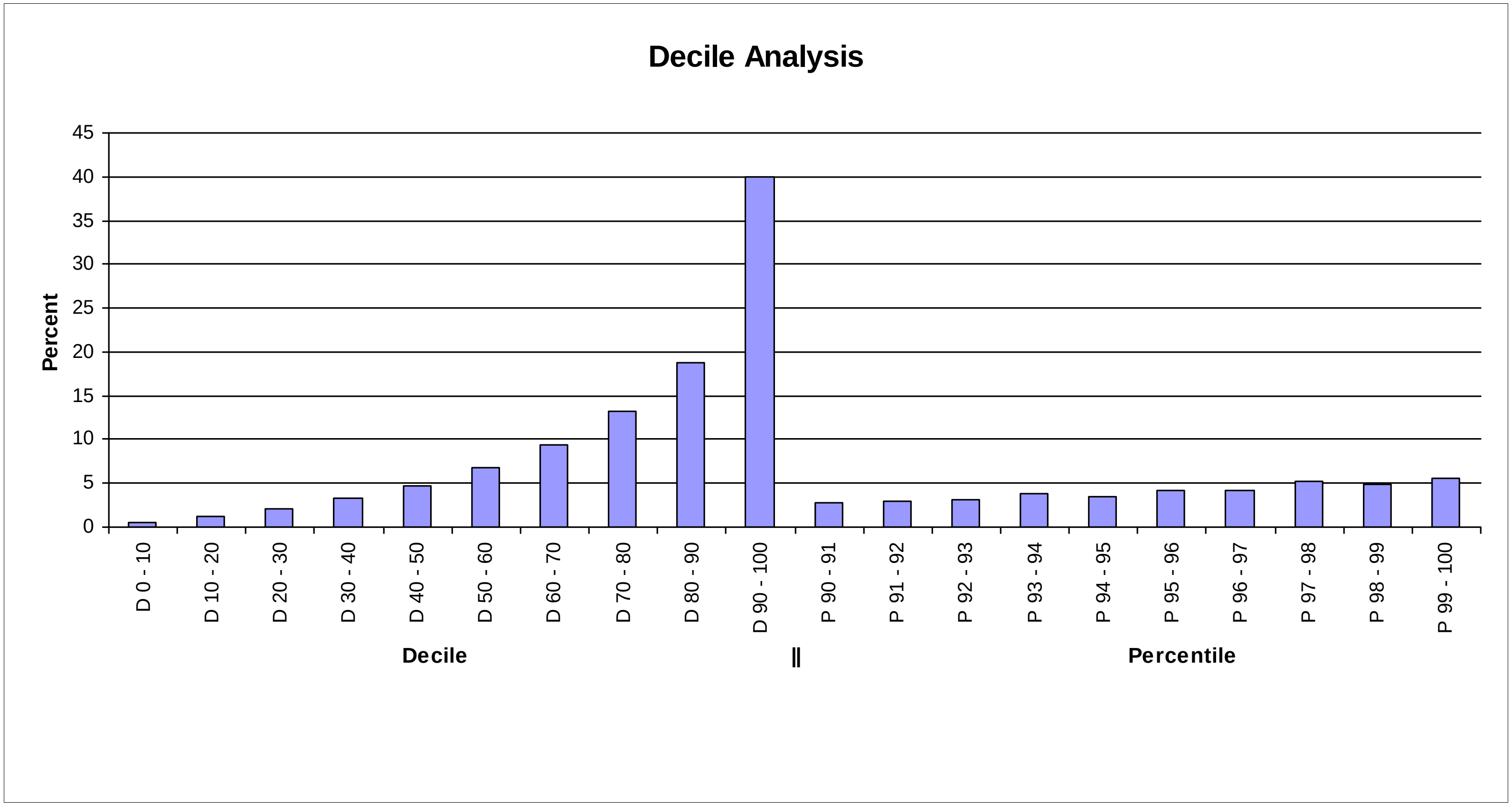
	From	To	Count	Element Mean	Element Min	Element Max	Total Metal	Percent	Capping Note
Decile									
	0	10	199	0.004	0.000	0.006	0.64	0.51	
	10	20	200	0.008	0.006	0.011	1.60	1.26	
	20	30	200	0.014	0.011	0.017	2.61	2.05	
	30	40	199	0.022	0.017	0.026	4.26	3.35	
	40	50	200	0.031	0.026	0.036	6.04	4.75	
	50	60	200	0.043	0.036	0.051	8.55	6.74	
	60	70	199	0.060	0.051	0.069	11.98	9.44	
	70	80	200	0.079	0.070	0.092	16.78	13.22	
	80	90	200	0.117	0.092	0.158	23.73	18.69	
	90	100	200	0.259	0.158	0.450	50.78	39.99	<40 <2.3x -- <50 <3x
Percentile									
	90	91	20	0.163	0.158	0.172	3.58	2.82	
	91	92	20	0.182	0.173	0.190	3.66	2.88	
	92	93	20	0.198	0.190	0.206	3.88	3.05	
	93	94	20	0.217	0.209	0.227	4.75	3.74	
	94	95	20	0.235	0.227	0.242	4.38	3.45	
	95	96	20	0.254	0.243	0.270	5.29	4.16	
	96	97	20	0.278	0.270	0.290	5.34	4.21	
	97	98	20	0.307	0.293	0.319	6.62	5.21	
	98	99	20	0.335	0.320	0.361	6.22	4.90	
	99	100	20	0.420	0.362	0.450	7.07	5.57	<10 <1.75x -- <15 <2x
Total									
	0	100	1997	0.064	0.000	0.450	126.96	100.00	

Interpretation notes:

Capping is warranted if

the last decile has more than 40 percent of metal; or,
the last decile contains more than 2.3 times the metal quantity contained in the one before last; or,
the last centile contains more than 10 percent of metal; or,
the last centile contains more than 1.75 times the metal quantity contained in the one before last.
---- Exception will be made if all following conditions are met:
the last decile has more than 50 percent metal; and,
the last decile contains more than 3 times the metal quantity contained in the one before last; and,
the last centile contains more than 15 percent of the metal; and,
the last centile contains more than 2 times the metal quantity contained in the one before last.

Nickel King - Upper Sill >>> Cu (capped)



Nickel King - Upper Sill >>> Cu (capped)

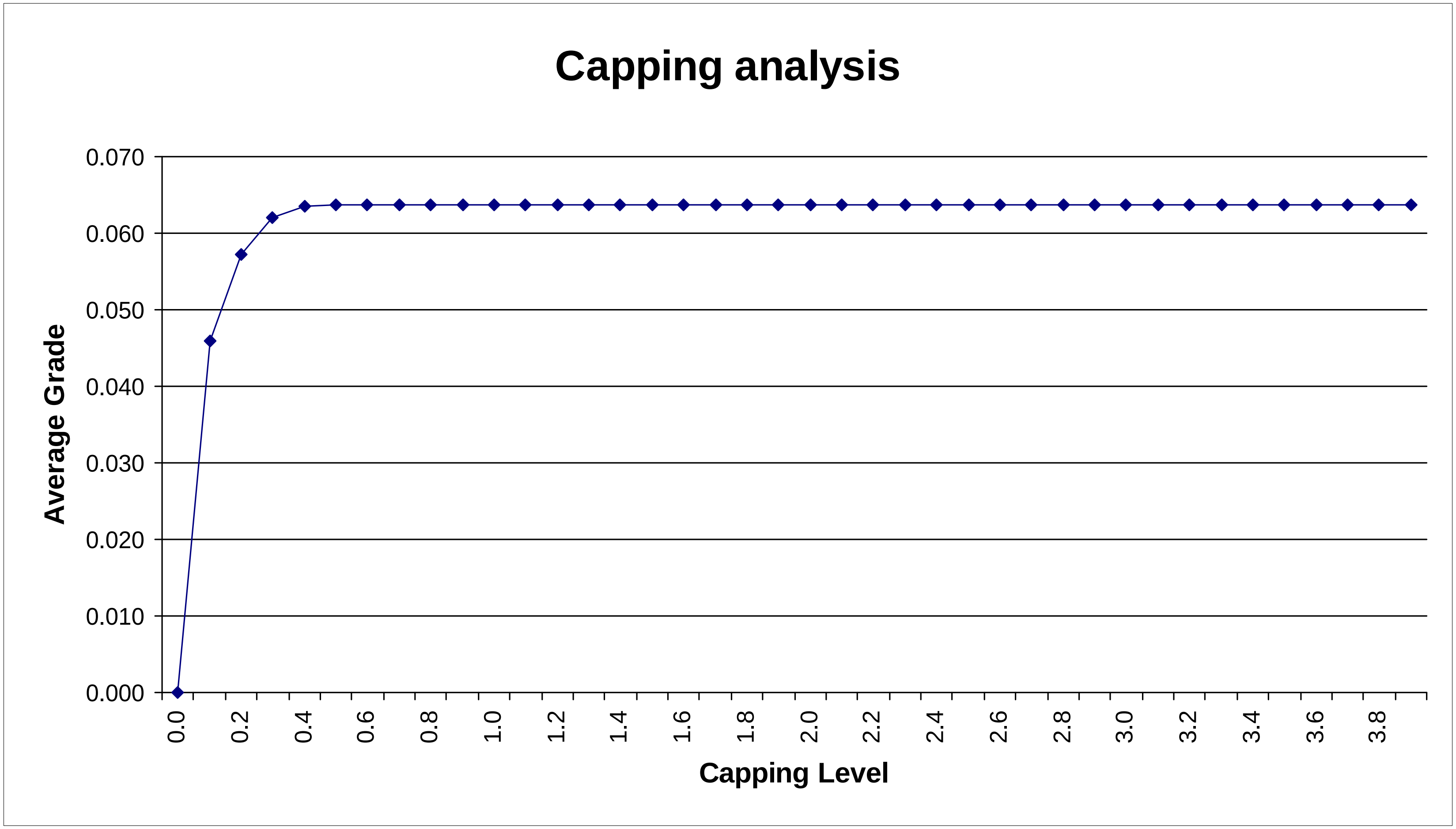
Capping Study Report

Element Basic Statistics

Count	1997	Min	0.000	Average	0.064	Max	0.450
Variance	0.006	Std Deviation	0.077	First Quartile	0.014	Third Quartile	0.080

Cap Level Bins	Average Grade	Number capped	Percent Nb capped	Percent Metal capped
3.90	0.064	0	0.000	0.000
3.80	0.064	0	0.000	0.000
3.70	0.064	0	0.000	0.000
3.60	0.064	0	0.000	0.000
3.50	0.064	0	0.000	0.000
3.40	0.064	0	0.000	0.000
3.30	0.064	0	0.000	0.000
3.20	0.064	0	0.000	0.000
3.10	0.064	0	0.000	0.000
3.00	0.064	0	0.000	0.000
2.90	0.064	0	0.000	0.000
2.80	0.064	0	0.000	0.000
2.70	0.064	0	0.000	0.000
2.60	0.064	0	0.000	0.000
2.50	0.064	0	0.000	0.000
2.40	0.064	0	0.000	0.000
2.30	0.064	0	0.000	0.000
2.20	0.064	0	0.000	0.000
2.10	0.064	0	0.000	0.000
2.00	0.064	0	0.000	0.000
1.90	0.064	0	0.000	0.000
1.80	0.064	0	0.000	0.000
1.70	0.064	0	0.000	0.000
1.60	0.064	0	0.000	0.000
1.50	0.064	0	0.000	0.000
1.40	0.064	0	0.000	0.000
1.30	0.064	0	0.000	0.000
1.20	0.064	0	0.000	0.000
1.10	0.064	0	0.000	0.000
1.00	0.064	0	0.000	0.000
0.90	0.064	0	0.000	0.000
0.80	0.064	0	0.000	0.000
0.70	0.064	0	0.000	0.000
0.60	0.064	0	0.000	0.000
0.50	0.064	0	0.000	0.000
0.40	0.063	15	0.751	0.409
0.30	0.062	57	2.854	2.242
0.20	0.057	151	7.561	9.617
0.10	0.046	368	18.428	27.246
0.00	0.000	1997	100.000	100.000

Nickel King - Upper Sill >>> Cu (capped)



Decile analysis Report and Capping study

Decile Analysis   Nickel King - Lower Sill >>> Ni

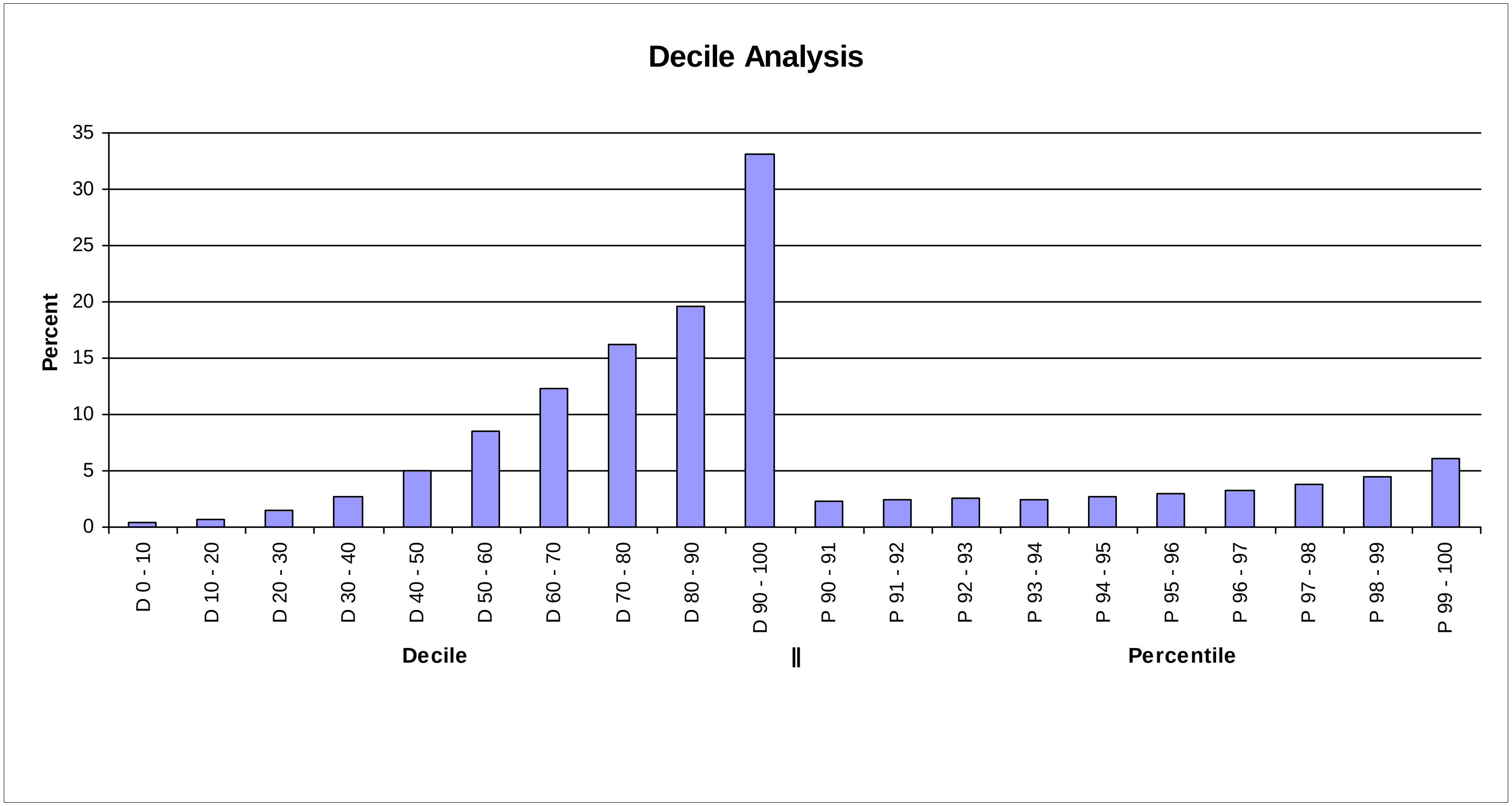
			------------ Element-------------			Total		
From	**To**	**Count**	**Mean**	**Min**	**Max**	**Metal**	**Percent**	**Capping Note**
Decile								
0	10	203	0.014	0.000	0.021	2.48	0.36	
10	20	204	0.028	0.021	0.037	5.11	0.74	
20	30	204	0.049	0.037	0.069	9.84	1.43	
30	40	203	0.093	0.069	0.123	18.58	2.70	
40	50	204	0.166	0.123	0.213	34.57	5.02	
50	60	204	0.270	0.214	0.324	58.52	8.49	
60	70	203	0.373	0.324	0.431	84.41	12.25	
70	80	204	0.495	0.431	0.555	111.87	16.23	
80	90	204	0.643	0.556	0.745	135.45	19.65	
90	100	204	1.160	0.747	3.872	228.34	33.13	<40 <2.3x -- <50 <3x
Percentile								
90	91	20	0.765	0.747	0.782	15.72	2.28	
91	92	21	0.814	0.786	0.830	16.96	2.46	
92	93	20	0.859	0.832	0.883	17.71	2.57	
93	94	20	0.911	0.884	0.934	17.12	2.48	
94	95	21	0.970	0.937	1.000	18.31	2.66	
95	96	20	1.032	1.000	1.064	20.83	3.02	
96	97	20	1.125	1.071	1.180	22.29	3.23	
97	98	21	1.285	1.200	1.400	26.50	3.84	
98	99	20	1.535	1.405	1.670	30.94	4.49	
99	100	21	2.272	1.676	3.872	41.96	6.09	<10 <1.75x -- <15 <2x
Total								
0	100	2037	0.329	0.000	3.872	689.17	100.00	

Interpretation notes:

Capping is warranted if

the last decile has more than 40 percent of metal; or,
the last decile contains more than 2.3 times the metal quantity contained in the one before last; or,
the last centile contains more than 10 percent of metal; or,
the last centile contains more than 1.75 times the metal quantity contained in the one before last.
---- Exception will be made if all following conditions are met:
the last decile has more than 50 percent metal; and,
the last decile contains more than 3 times the metal quantity contained in the one before last; and,
the last centile contains more than 15 percent of the metal; and,
the last centile contains more than 2 times the metal quantity contained in the one before last.

Nickel King - Lower Sill >>> Ni



Nickel King - Lower Sill >>> Ni

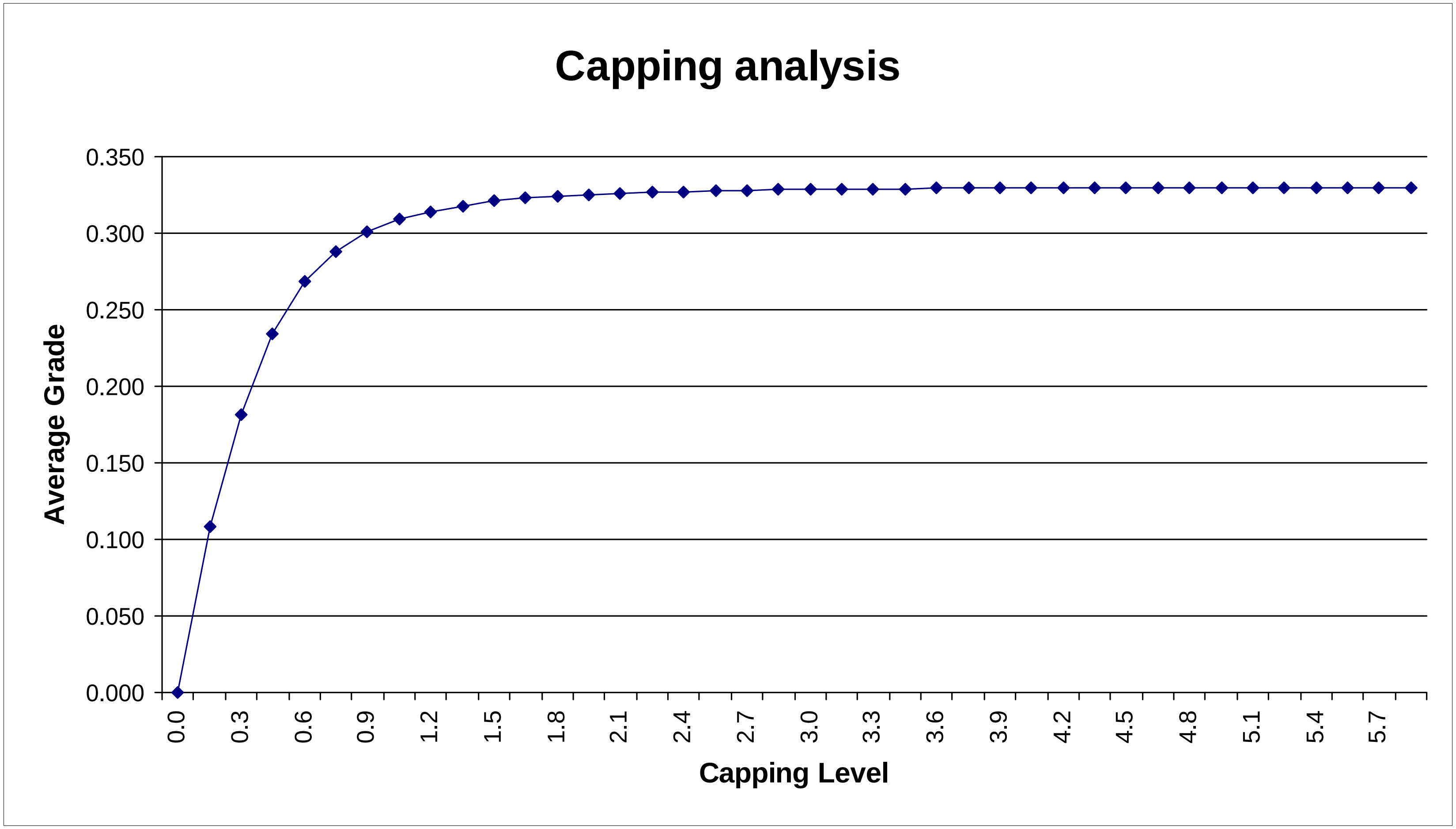
Capping Study Report

Element Basic Statistics

Count	2037	Min	0.000	Average	0.329	Max	3.872
Variance	0.141	Std Deviation	0.376	First Quartile	0.048	Third Quartile	0.496

Cap Level Bins	Average Grade	Number capped	Percent Nb capped	Percent Metal capped
5.85	0.329	0	0.000	0.000
5.70	0.329	0	0.000	0.000
5.55	0.329	0	0.000	0.000
5.40	0.329	0	0.000	0.000
5.25	0.329	0	0.000	0.000
5.10	0.329	0	0.000	0.000
4.95	0.329	0	0.000	0.000
4.80	0.329	0	0.000	0.000
4.65	0.329	0	0.000	0.000
4.50	0.329	0	0.000	0.000
4.35	0.329	0	0.000	0.000
4.20	0.329	0	0.000	0.000
4.05	0.329	0	0.000	0.000
3.90	0.329	0	0.000	0.000
3.75	0.329	1	0.049	0.007
3.60	0.329	1	0.049	0.016
3.45	0.329	1	0.049	0.024
3.30	0.329	1	0.049	0.033
3.15	0.329	3	0.147	0.059
3.00	0.329	3	0.147	0.104
2.85	0.328	4	0.196	0.161
2.70	0.328	6	0.295	0.234
2.55	0.328	6	0.295	0.337
2.40	0.327	7	0.344	0.450
2.25	0.327	7	0.344	0.576
2.10	0.326	9	0.442	0.730
1.95	0.325	10	0.491	0.912
1.80	0.324	16	0.785	1.177
1.65	0.323	24	1.178	1.567
1.50	0.321	35	1.718	2.138
1.35	0.318	46	2.258	2.950
1.20	0.314	62	3.044	4.029
1.05	0.309	86	4.222	5.551
0.90	0.301	138	6.775	7.887
0.75	0.288	202	9.917	11.432
0.60	0.268	356	17.477	17.251
0.45	0.234	581	28.522	27.511
0.30	0.182	863	42.366	43.817
0.15	0.108	1160	56.946	66.922
0.00	0.000	2037	100.000	100.000

Nickel King - Lower Sill >>> Ni



Decile analysis Report and Capping study

Decile Analysis Nickel King - Lower Sill >>> Cu

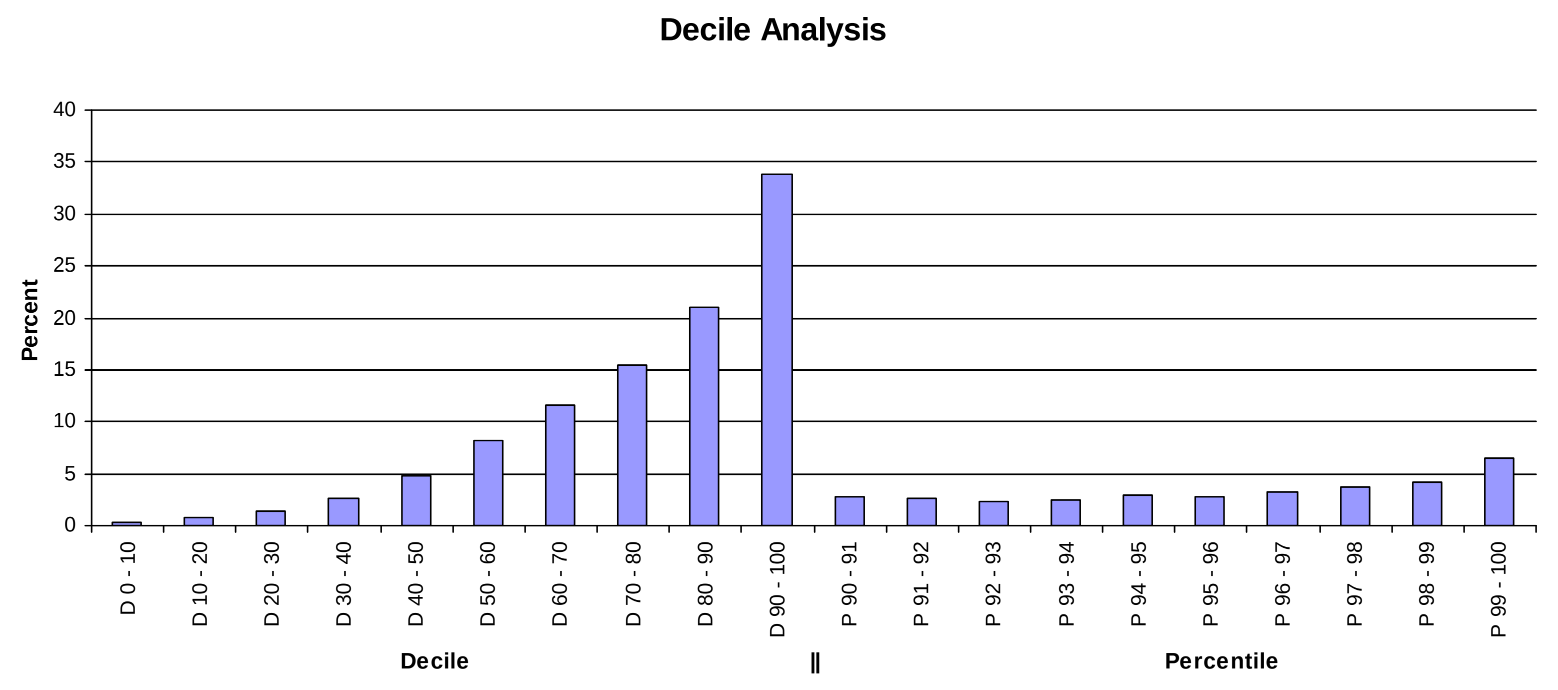
			------------ Element-------------			Total		
From	To	Count	Mean	Min	Max	Metal	Percent	Capping Note
Decile								
0	10	203	0.004	0.000	0.005	0.64	0.38	
10	20	203	0.007	0.005	0.009	1.31	0.79	
20	30	203	0.011	0.009	0.015	2.29	1.38	
30	40	204	0.022	0.015	0.030	4.28	2.59	
40	50	203	0.039	0.030	0.050	8.02	4.85	
50	60	203	0.063	0.050	0.076	13.48	8.16	
60	70	204	0.087	0.076	0.102	19.26	11.65	
70	80	203	0.118	0.102	0.135	25.48	15.42	
80	90	203	0.157	0.135	0.182	34.75	21.02	
90	100	204	0.278	0.182	1.570	55.79	33.75	<40 <2.3x -- <50 <3x
Percentile								
90	91	21	0.189	0.182	0.193	4.61	2.79	
91	92	20	0.198	0.194	0.200	4.46	2.70	
92	93	20	0.205	0.200	0.210	3.95	2.39	
93	94	21	0.218	0.211	0.224	4.02	2.43	
94	95	20	0.234	0.224	0.246	4.91	2.97	
95	96	20	0.255	0.247	0.265	4.68	2.83	
96	97	21	0.277	0.266	0.292	5.45	3.30	
97	98	20	0.307	0.292	0.318	6.04	3.65	
98	99	20	0.350	0.321	0.394	6.96	4.21	
99	100	21	0.547	0.400	1.570	10.71	6.48	<10 <1.75x -- <15 <2x
Total								
0	100	2033	0.079	0.000	1.570	165.30	100.00	

Interpretation notes:

Capping is warranted if

the last decile has more than 40 percent of metal; or,
the last decile contains more than 2.3 times the metal quantity contained in the one before last; or,
the last centile contains more than 10 percent of metal; or,
the last centile contains more than 1.75 times the metal quantity contained in the one before last.
---- Exception will be made if all following conditions are met:
the last decile has more than 50 percent metal; and,
the last decile contains more than 3 times the metal quantity contained in the one before last; and,
the last centile contains more than 15 percent of the metal; and,
the last centile contains more than 2 times the metal quantity contained in the one before last.

Nickel King - Lower Sill >>> Cu



Nickel King - Lower Sill >>> Cu

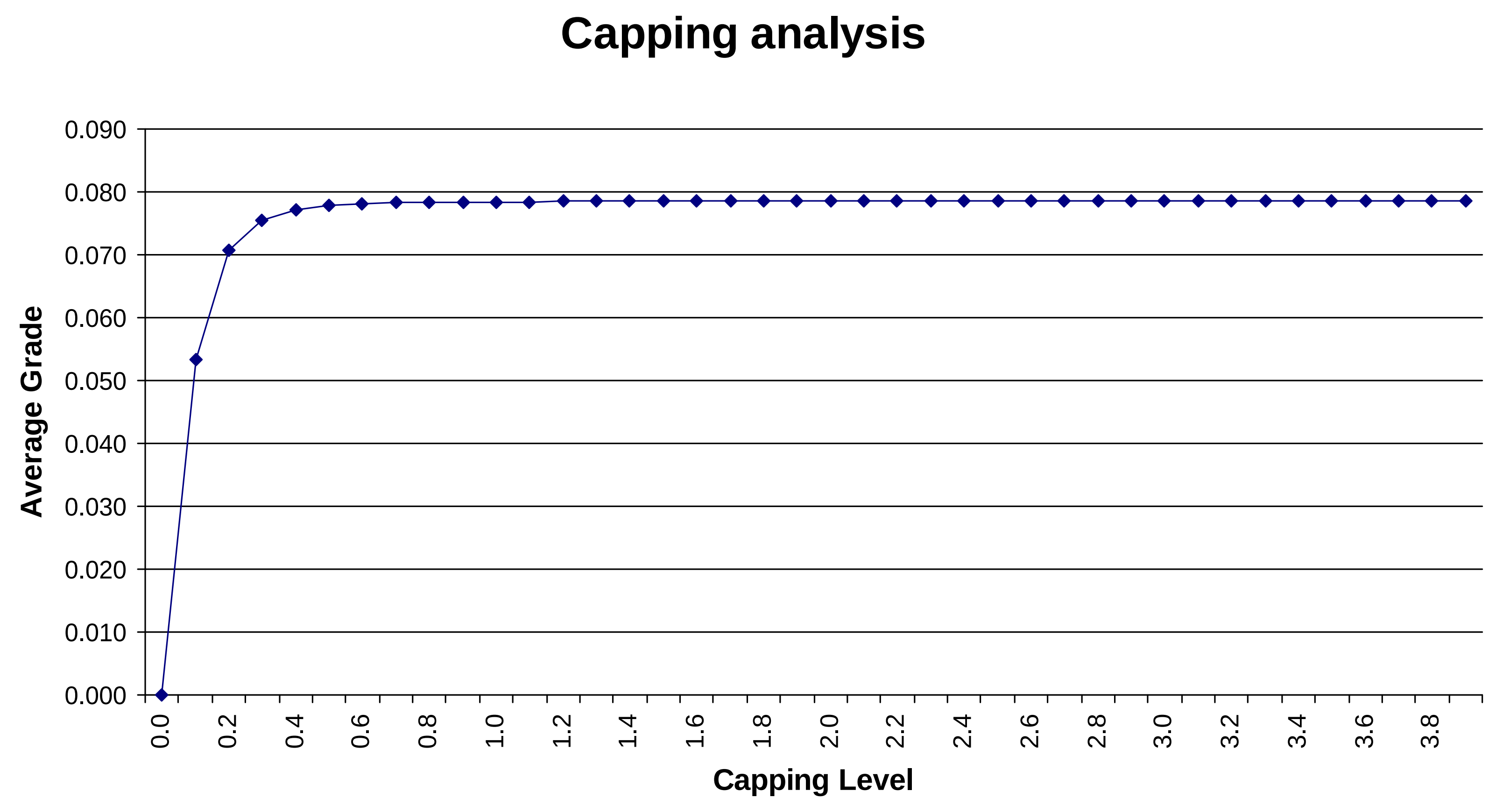
Capping Study Report

Element Basic Statistics

Count	2033	Min	0.000	Average	0.079	Max	1.570
Variance	0.009	Std Deviation	0.092	First Quartile	0.011	Third Quartile	0.117

Cap Level Bins	Average Grade	Number capped	Percent Nb capped	Percent Metal capped
3.90	0.079	0	0.000	0.000
3.80	0.079	0	0.000	0.000
3.70	0.079	0	0.000	0.000
3.60	0.079	0	0.000	0.000
3.50	0.079	0	0.000	0.000
3.40	0.079	0	0.000	0.000
3.30	0.079	0	0.000	0.000
3.20	0.079	0	0.000	0.000
3.10	0.079	0	0.000	0.000
3.00	0.079	0	0.000	0.000
2.90	0.079	0	0.000	0.000
2.80	0.079	0	0.000	0.000
2.70	0.079	0	0.000	0.000
2.60	0.079	0	0.000	0.000
2.50	0.079	0	0.000	0.000
2.40	0.079	0	0.000	0.000
2.30	0.079	0	0.000	0.000
2.20	0.079	0	0.000	0.000
2.10	0.079	0	0.000	0.000
2.00	0.079	0	0.000	0.000
1.90	0.079	0	0.000	0.000
1.80	0.079	0	0.000	0.000
1.70	0.079	0	0.000	0.000
1.60	0.079	0	0.000	0.000
1.50	0.079	1	0.049	0.052
1.40	0.079	1	0.049	0.125
1.30	0.079	1	0.049	0.199
1.20	0.078	1	0.049	0.273
1.10	0.078	1	0.049	0.347
1.00	0.078	1	0.049	0.421
0.90	0.078	1	0.049	0.494
0.80	0.078	1	0.049	0.568
0.70	0.078	1	0.049	0.642
0.60	0.078	2	0.098	0.723
0.50	0.078	11	0.541	0.989
0.40	0.077	21	1.033	1.816
0.30	0.076	57	2.804	3.692
0.20	0.071	170	8.362	9.454
0.10	0.053	627	30.841	31.435
0.00	0.000	2033	100.000	100.000

Nickel King - Lower Sill >>> Cu



Decile analysis Report and Capping study

Decile Analysis Nickel King - Lower Sill >>> Co

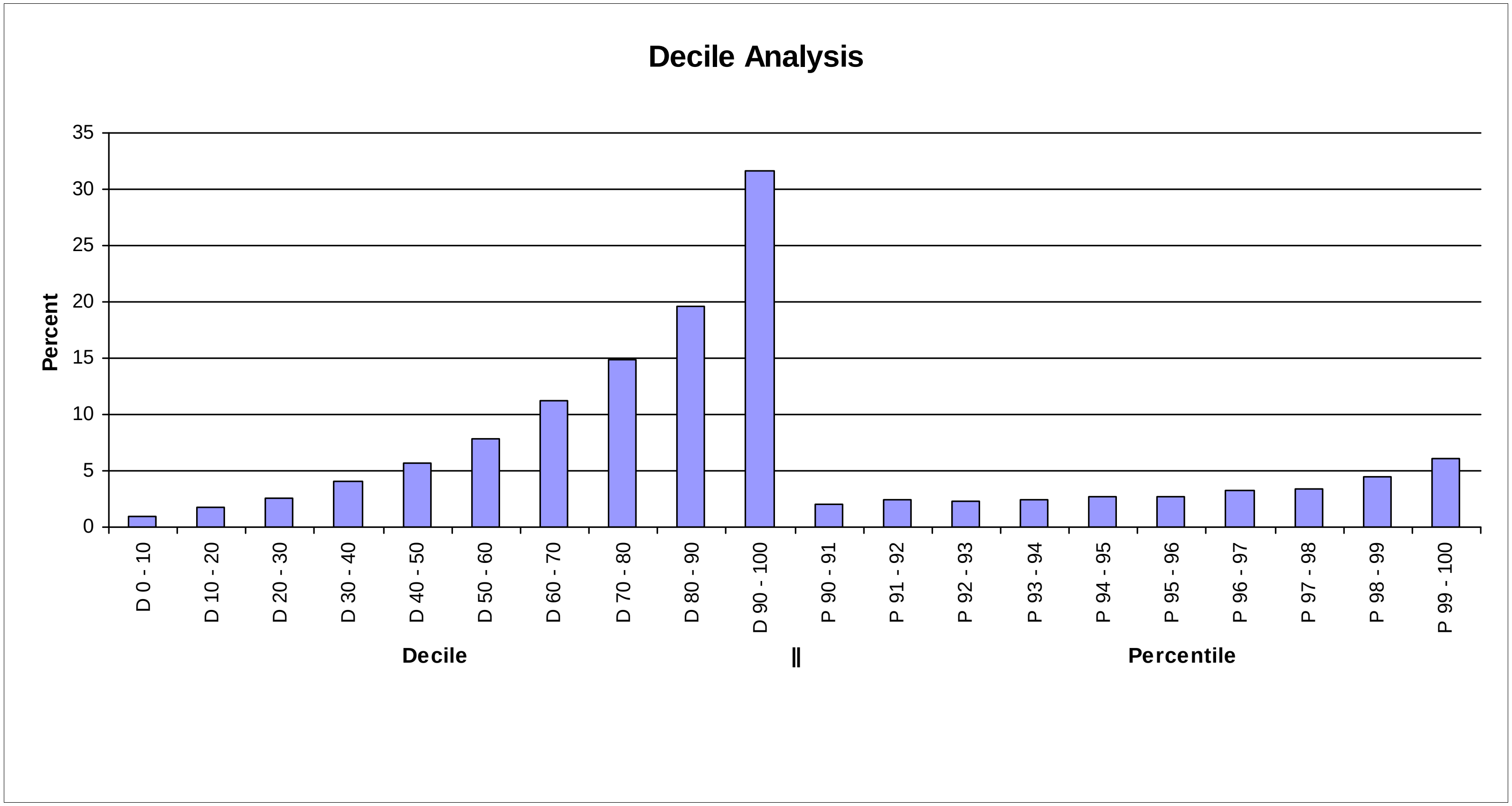
			------------ Element-------------			Total		
From	**To**	**Count**	**Mean**	**Min**	**Max**	**Metal**	**Percent**	**Capping Note**
Decile								
0	10	192	15.231	0.500	20.100	2,474.63	0.95	
10	20	193	24.436	20.200	29.000	4,448.21	1.70	
20	30	193	35.427	29.000	44.600	6,536.34	2.50	
30	40	193	56.122	44.600	68.300	10,486.64	4.02	
40	50	193	79.801	68.500	93.600	14,717.86	5.64	
50	60	192	110.811	93.700	130.000	20,528.94	7.86	
60	70	193	151.887	130.000	178.500	29,182.27	11.18	
70	80	193	202.283	179.800	230.000	38,763.84	14.85	
80	90	193	264.759	230.000	300.000	51,273.19	19.64	
90	100	193	464.720	300.500	1,510.000	82,617.75	31.65	<40 <2.3x -- <50 <3x
Percentile								
90	91	19	310.200	300.500	319.400	5,183.03	1.99	
91	92	19	323.158	319.800	330.000	6,200.80	2.38	
92	93	20	337.365	330.000	348.200	5,986.40	2.29	
93	94	19	358.947	350.000	370.000	6,466.10	2.48	
94	95	19	390.832	370.000	400.000	6,944.30	2.66	
95	96	19	422.105	410.000	430.000	6,985.60	2.68	
96	97	20	458.705	435.000	480.000	8,595.52	3.29	
97	98	19	521.579	490.000	560.000	8,906.70	3.41	
98	99	19	607.368	560.000	670.000	11,516.60	4.41	
99	100	20	901.000	670.000	1,510.000	15,832.70	6.07	<10 <1.75x -- <15 <2x
Total								
0	100	1928	140.628	0.500	1,510.000	261,029.68	100.00	

Interpretation notes:

Capping is warranted if

the last decile has more than 40 percent of metal; or,
the last decile contains more than 2.3 times the metal quantity contained in the one before last; or,
the last centile contains more than 10 percent of metal; or,
the last centile contains more than 1.75 times the metal quantity contained in the one before last.
---- Exception will be made if all following conditions are met:
the last decile has more than 50 percent metal; and,
the last decile contains more than 3 times the metal quantity contained in the one before last; and,
the last centile contains more than 15 percent of the metal; and,
the last centile contains more than 2 times the metal quantity contained in the one before last.

Nickel King - Lower Sill >>> Co



Nickel King - Lower Sill >>> Co

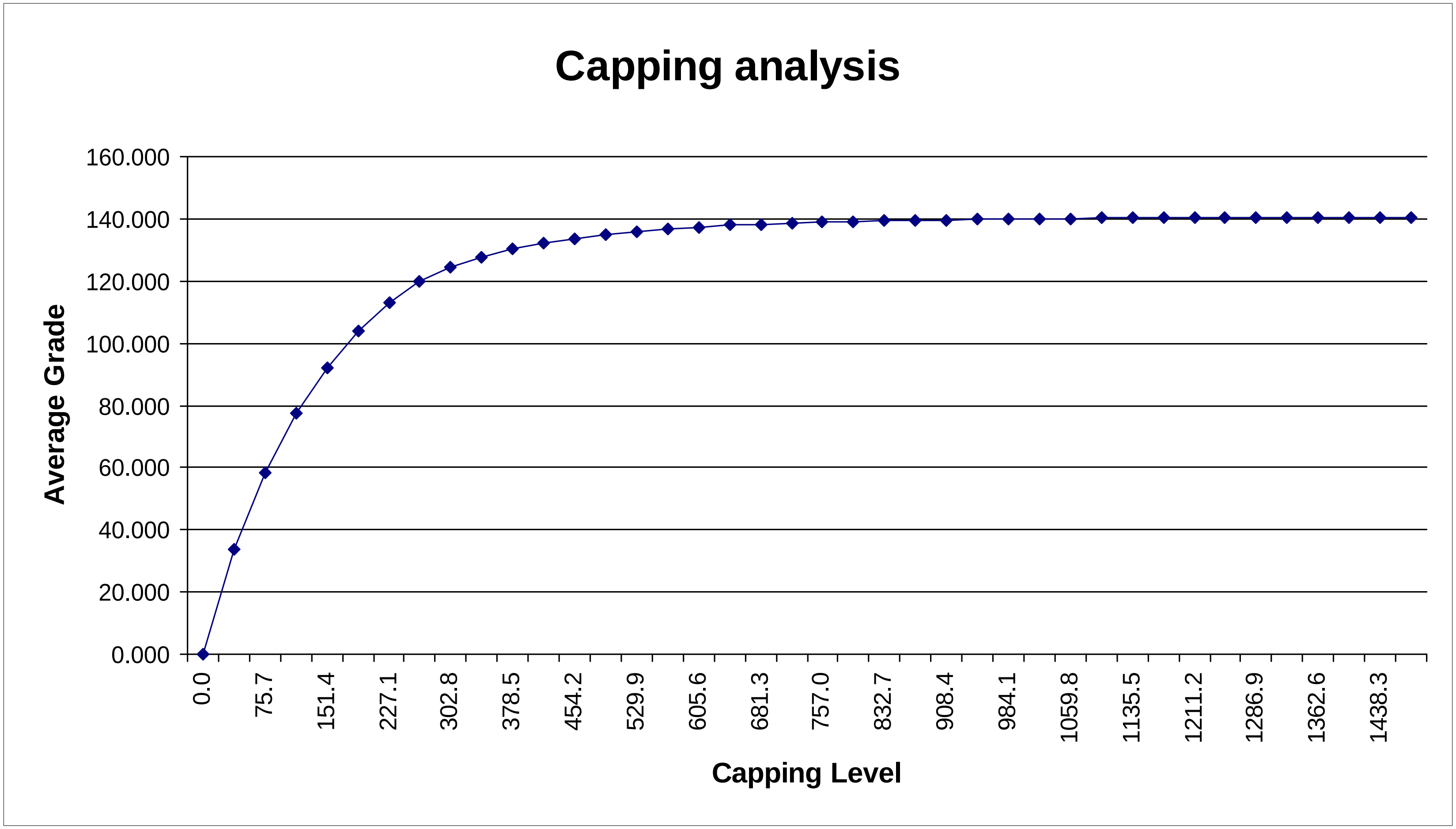
Capping Study Report

Element Basic Statistics

Count	1928	Min	0.500	Average	140.628	Max	1510.000
Variance	1354.359	Std Deviation	146.131	First Quartile	34.900	Third Quartile	200.000

Cap Level Bins	Average Grade	Number capped	Percent Nb capped	Percent Metal capped
1476.15	140.610	1	0.052	0.005
1438.30	140.591	1	0.052	0.011
1400.45	140.571	1	0.052	0.017
1362.60	140.552	1	0.052	0.023
1324.75	140.532	1	0.052	0.028
1286.90	140.505	2	0.104	0.036
1249.05	140.466	2	0.104	0.049
1211.20	140.422	3	0.156	0.064
1173.35	140.364	3	0.156	0.087
1135.50	140.305	3	0.156	0.110
1097.65	140.212	5	0.259	0.157
1059.80	140.114	5	0.259	0.208
1021.95	140.016	5	0.259	0.260
984.10	139.904	6	0.311	0.321
946.25	139.786	6	0.311	0.387
908.40	139.657	7	0.363	0.461
870.55	139.515	8	0.415	0.544
832.70	139.349	9	0.467	0.643
794.85	139.146	11	0.571	0.768
757.00	138.924	12	0.622	0.907
719.15	138.666	15	0.778	1.071
681.30	138.337	18	0.934	1.289
643.45	137.936	23	1.193	1.561
605.60	137.411	30	1.556	1.913
567.75	136.769	36	1.867	2.363
529.90	135.942	48	2.490	2.940
492.05	134.954	54	2.801	3.625
454.20	133.725	70	3.631	4.479
416.35	132.172	93	4.824	5.554
378.50	130.155	115	5.965	6.936
340.65	127.697	137	7.106	8.620
302.80	124.418	192	9.959	10.865
264.95	119.688	286	14.834	14.157
227.10	113.060	394	20.436	18.851
189.25	104.037	542	28.112	25.315
151.40	92.176	667	34.595	33.863
113.55	77.287	846	43.880	44.632
75.70	58.387	1088	56.432	58.267
37.85	33.907	1418	73.548	75.880
0.00	0.000	1928	100.000	100.000

Nickel King - Lower Sill >>> Co



Nickel King Capping Sensitivity

Lenses or zone	Method	Rec	Cap_Level	Average Grade	Total Samples	Number Capped	Percent capped	Percent Metal Capped	PEG Suggested level
US - Ni	Uncapped	NC							No Cap
	Hist Prob								
	Capping Chart								
	Decile @ 99 perc. Avg								
LS - NI	Uncapped	NC							No Cap
	Hist Prob								
	Capping Chart								
	Decile @99 perc. Avg								

Rec is the recommendation based on the Decile analysis

C	*Capping recommended Last decile and last centile have more than their share of metal*
M	*Marginal - OK in the decile however the last centile has more than its share on metal*
NC	*No capping recommended*

Nickel King Capping Sensitivity

Lenses or zone	Method	Rec	Cap_Level	Average Grade	Total Samples	Number Capped	Percent capped	Percent Metal Capped	PEG Suggested level
US - Cu	Uncapped	C	99999	0.06	1,997	0	-	0.00	0.45 Cap
	Hist Prob		0.55	0.06	1,997	2	0.10	0.82	
	Capping Chart		0.4	0.06	1,997	15	0.75	1.48	
	Decile @ 99 perc. Avg		0.49	0.06	1,997	5	0.25	0.95	
LS - Cu	Uncapped	NC							No Cap
	Hist Prob								
	Capping Chart								
	Decile @98.5 perc. Avg								

Rec is the recommendation based on the Decile analysis

C	*Capping recommended Last decile and last centile have more than their share of metal*
M	*Marginal - OK in the decile however the last centile has more than its share on metal*
NC	*No capping recommended*

Nickel King Capping Sensitivity

Lenses or zone	Method	Rec	Cap_Level	Average Grade	Total Samples	Number Capped	Percent capped	Percent Metal Capped	PEG Suggested level
US - Co	Uncapped	C	99999	131.79	1,867	0	-	(0.00)	835 Cap
	Hist Prob		900	131.76	1,867	3	0.16	0.02	
	Capping Chart		800	131.39	1,867	14	0.75	0.31	
	Decile @ 99 perc. Avg		835	131.57	1,867	8	0.43	0.18	
LS - Co	Uncapped	NC							No Cap
	Hist Prob								
	Capping Chart								
	Decile @98.5 perc. Avg								

Rec is the recommendation based on the Decile analysis

C	*Capping recommended Last decile and last centile have more than their share of metal*
M	*Marginal - OK in the decile however the last centile has more than its share on metal*
NC	*No capping recommended*

APPENDIX D

COMPOSITES STATISTICAL ANALYSIS

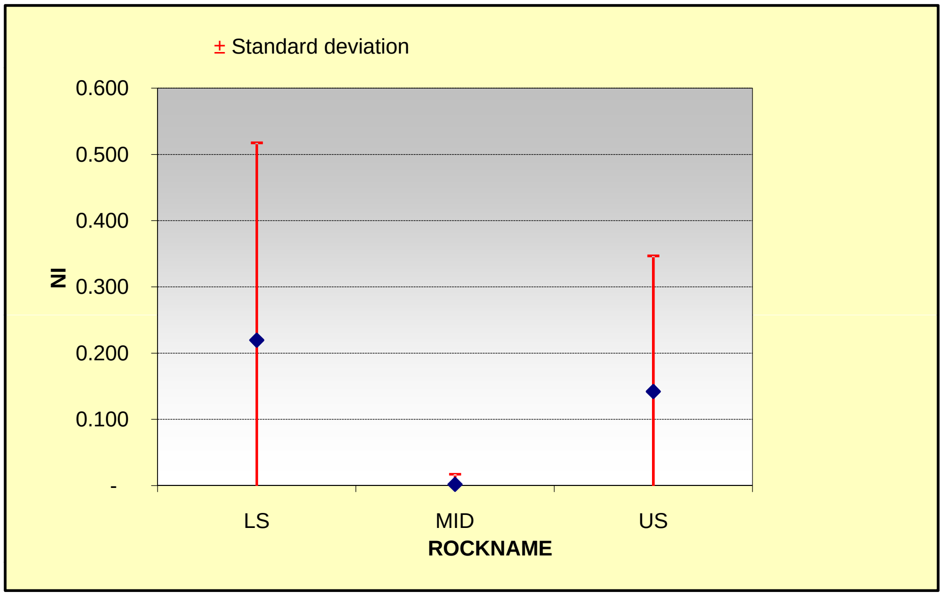
Composite Core Length Statistics after Compositing by 2.5 meters						
DOMAIN_T	Count	Avg	Min	Max	StDev	Var
LS	1291	2.54	2.20	2.60	0.03	0.00
MID	1248	2.54	2.51	2.86	0.04	0.00
US	1393	2.56	2.50	2.74	0.05	0.00
Total/Average	3932	2.55	2.40	2.73		

Ajusted manually hole NK08-032 as GEMS would not split the interval

Means

Variable: NI
grouped by: ROCKNAME

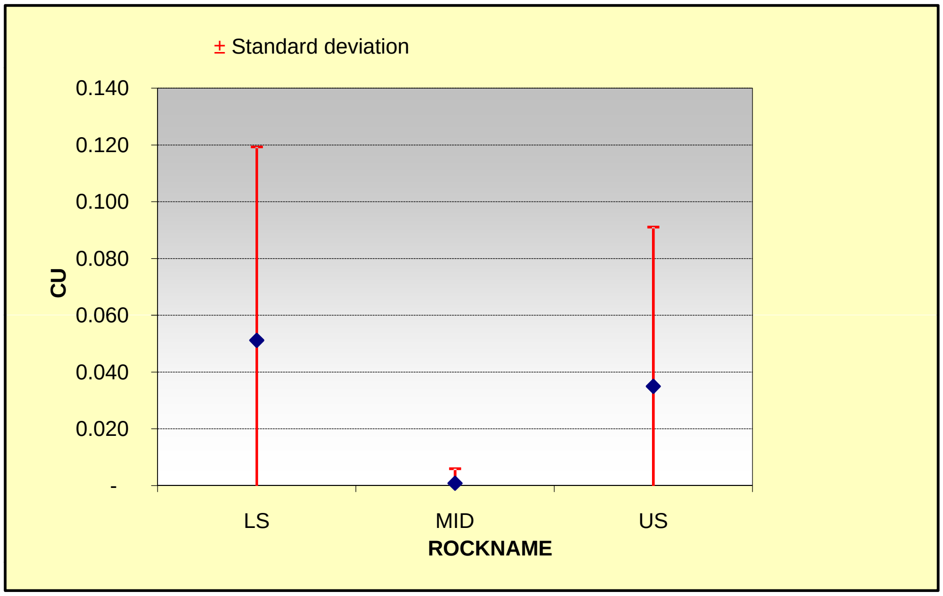
	N	Mean	95% Conf. (±)	Std.Error	Std.Dev.
LS	1,049.00	0.220	0.02	0.01	0.30
MID	972.00	0.002	0.00	0.00	0.02
US	1,137.00	0.142	0.01	0.01	0.20
Entire sample	3,158.00	0.125	0.01	0.00	0.23



Means

Variable: CU
grouped by: ROCKNAME

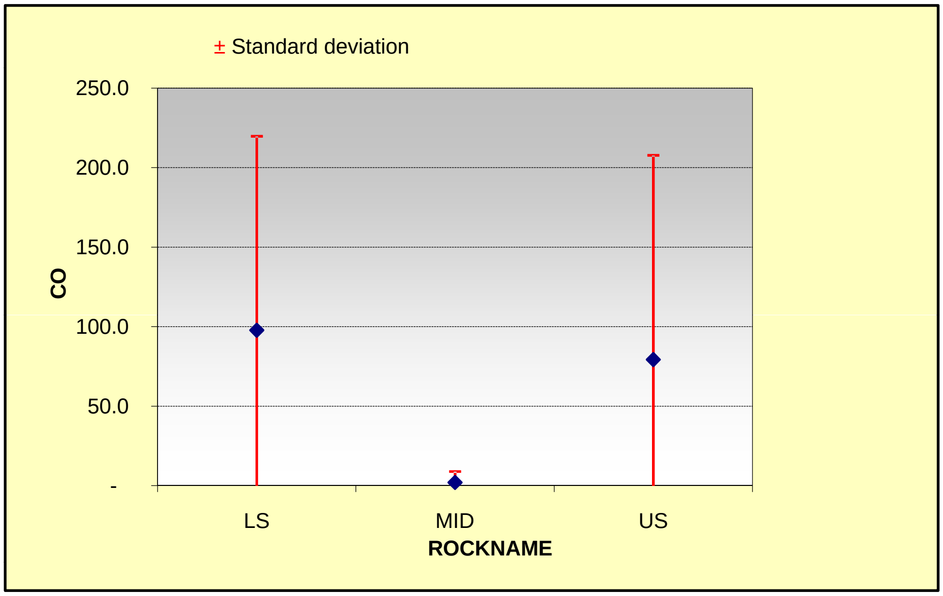
	N	Mean	95% Conf. (±)	Std.Error	Std.Dev.
LS	1049	0.051	0.004	0.002	0.068
MID	972	0.001	0.000	0.000	0.005
US	1137	0.035	0.003	0.002	0.056
Entire sample	3158	0.030	0.002	0.001	0.056



Means

Variable: CO
grouped by: ROCKNAME

	N	Mean	95% Conf. (±)	Std.Error	Std.Dev.
LS	1049	97.7	7.4	3.8	122.0
MID	972	2.0	0.4	0.2	6.9
US	1137	79.3	7.5	3.8	128.5
Entire sample	3158	61.6	3.9	2.0	112.0



APPENDIX E

NICKEL VARIOGRAPHY RESULTS

Lower Sill - Search parameters

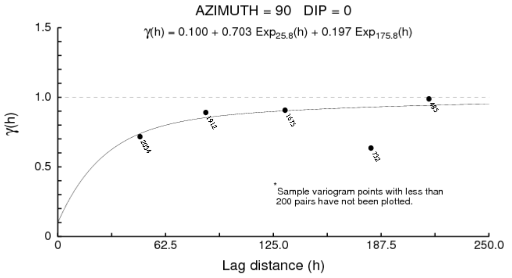
	Guidance by Vario		To reach next hole					
	Start	Multiplier	Pass 1	Multiplier	Pass 2	Multiplier	Pass 3
GEMS ZXZ Rot Angles	Various	Azimuth and dip varies by sub-domains					
Range X	180	0.5	90	1.5	135	2	270
Range Y	120	0.5	60	1.5	90	2	180
Range Z	40	0.5	20	1.5	30	2	60

Pass 3 target Range		Max (traditional * 3)/2
	Vario Max	Practical conversion
Strike	180	270
Dip	120	180
Across	40	60

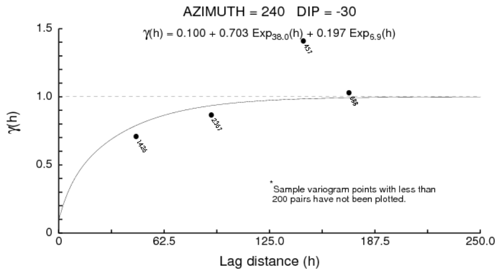
Lower Sill - Search ellipsoid orientation

Anisotropy angles are defined by Rotation ZXZ	Sub Dom 1	Sub Dom 2	Sub Dom 3	Sub Dom 4	LS East
Rotation about Z from X towards Y	77	35	-20	-20	-40
Rotation about X from Y towards Z	38	42	30	10	40
Rotation about Z from X towards Y	0	0	0	60	0

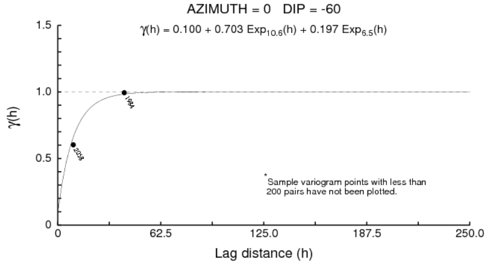
On Strike



Dip Direction



Cross dip -- short range



Upper Sill - Search parameters

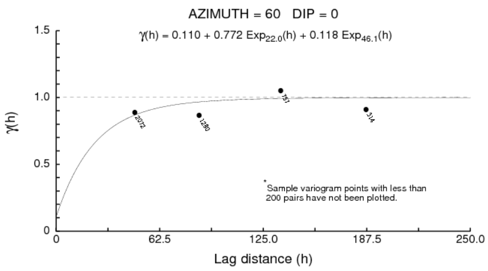
	Guidance by Vario		To reach next hole				
	Start	Multiplier	Pass 1	Multiplier	Pass 2	Multiplier	Pass 3
GEMS ZXZ Rot Angles	Various	Azimuth and dip varies by sub-domains					
Range X	150	0.5	75	1.5	112.5	2	225
Range Y	90	0.5	45	1.5	67.5	2	135
Range Z	45	0.5	22.5	1.5	33.75	2	67.5

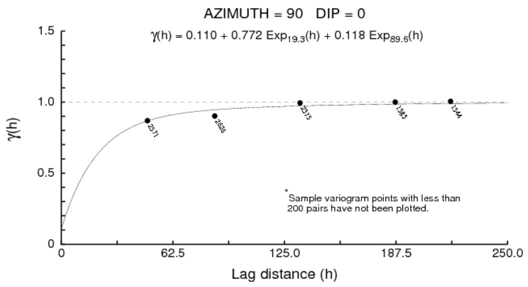
Pass 3 target Range		Max (traditional * 3)/2
	Vario Max	Practical conversion
Strike	150	225
Dip	90	135
Across	45	67.5

Upper Sill - Search ellipsoid orientation

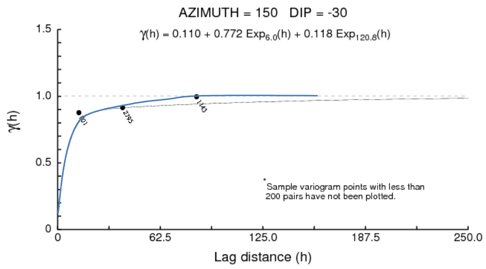
Anisotropy angles are defined by Rotation ZXZ	Sub Dom 1	Sub Dom 2	Sub Dom 3	Sub Dom 4
Rotation about Z from X towards Y	78	75	-8	-45
Rotation about X from Y towards Z	50	45	45	40
Rotation about Z from X towards Y	0	22.5	0	0

On Strike





Dip Direction



Cross dip -- short range