NI 43-101 Technical Report

Geovic Mining Corp.

Nkamouna and Mada Deposits

East Province of Cameroon, Africa

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Certificate of Author

Appendix B
Mining Decree

Appendix B
Variograms

Summary (Item 3)

Geovic Mining Corp. (Geovic) has engaged SRK Consulting (U.S.), Inc. (SRK) to prepare a Technical Report for the Nkamouna and Mada Deposits (or the Project) to meet the requirements of Canadian National Instrument 43-101 (NI 43-101) and for the further development and advancement of the Project.

This Technical Report is a summarized version of the *Nkamouna Co-Ni-Mn Project Feasibility Study* authored by Lycopodium Minerals Pty Ltd (Lycopodium) with contributions by SRK, Knight Piésold and Geovic Cameroon PLC (GeoCam).

Property Description and Location

The Nkamouna Cobalt-Nickel-Manganese Project (Project) includes greenfields development of an open-cut mine, process plant and associated project infrastructure at the Nkamouna site located in the Haut Nyong Division, East Province of Cameroon, Africa, approximately 400 km from the capital city of Yaounde. The Project is under evaluation by GeoCam which holds a mining permit that covers the Project site and surrounding areas.

Ownership

Ownership of the Project holder, Geovic Cameroon PLC, is held 60.5% by Geovic Ltd. Geovic Ltd. is a 100% held subsidiary of Geovic Mining Corporation (Geovic). The National Investment Corporation of Cameroon (SNI), holds 20% and 19.5% is held by other Cameroonian investors who are represented by SNI.

The right to mine held by GeoCam is realized through its Mining Permit. On April 11, 2003, a Mining Permit Decree was issued to GeoCam, covering an area of 1,250 km^2. The permit authorizes 1,250 km^2 within a 1,645 km^2 boundary.

Most of the Mining Permit lands are zoned 'mineral exclusive' lands, and assigns GeoCam exclusive rights to the cobalt, nickel and related materials within the Mining Permit.

Community Relations

GeoCam has integrated preventative engineering and mitigation strategies with respect to social and environmental programs since the exploration phase of the Project. Mining and production-related facilities are designed using industry-proven and/or laboratory demonstrated technologies, and are considered industry best practices. Also, international requirements such as Equator Principals and International Finance Corporation (IFC) norms have guided the preparation of the Environmental and Social Action Plan (ESAP), including activities to preserve the health and safety of the workforce and local communities.

The main potential impacts are summarized as follows:

- Loss of natural flora and fauna habitats due to land clearance for mining activities and the construction of processing facilities and support infrastructure for the mining operations;
- Alteration of land topography and form due to mining exploitation;
- Increase in soil erosion due to soil grading and the construction of mining facilities;
- Increase in metal and/or dissolved solid load in surface and ground water due to unexpected releases of process solutions or soil erosion;

- Increase of illegal hunting activities due to population influx;
- Dissemination of dust in the air due to intensive traffic of vehicles; and
- Increase in job opportunities for local resident population.

Minerals Resource Statement

The mineral resources for the Nkamouna and Mada cobalt-nickel deposits have been audited by SRK at 59.8 Mt grading an average of 0.24% cobalt, 0.68% nickel and 1.37% manganese classified as Measured Mineral Resources with an additional 60.8 Mt grading an average of 0.22% cobalt, 0.62% nickel and 1.32% manganese classified as Indicated Mineral Resources. An additional 202.5 Mt grading an average of 0.20% cobalt, 0.59% nickel and 1.20% manganese is classified as Inferred Mineral resources. The resource is stated above a 0.12% cobalt cut-off for ferralite and a 0.23% cobalt cut-off for breccias, constrained above the bedrock surface.

The mineral resources are reported in accordance with CSA NI 43-101 and have been estimated in conformity with generally accepted CIM 'Estimation of Mineral Resource and Mineral Reserves Best Practices' guidelines (Table 1).

Table 1: SRK Mineral Resource Statement for the Nkamouna and Mada Cobalt-Nickel-Manganese Deposits, October 14, 2009

Lithology Measured	Resource Category	Cut-off (%Co)	Tonnes (kt)	Average Grade		
				Co (%)	Ni (%)	Mn (%)
Nkamouna						
Breccia	Measured	0.23	6,527	0.47	0.55	2.60
Ferralite	Measured	0.12	53,277	0.22	0.69	1.22
Subtotal	***Measured***		***59,805***	***0.24***	***0.68***	***1.37***
Mada						
Breccia	Measured	0.23	-	-	-	-
Ferralite	Measured	0.12	-	-	-	-
Subtotal	***Measured***		-	-	-	-
Total	**Measured**		**59,805**	**0.24**	**0.68**	**1.37**
Indicated						
Nkamouna						
Breccia	Indicated	0.23	672	0.43	0.49	2.53
Ferralite	Indicated	0.12	20,247	0.19	0.68	1.07
Subtotal	**Indicated**		***20,918***	***0.19***	***0.67***	***1.12***
Mada						
Breccia	Indicated	0.23	6,625	0.38	0.53	2.21
Ferralite	Indicated	0.12	33,251	0.21	0.60	1.28
Subtotal	***Indicated***		***39,876***	***0.23***	***0.59***	***1.43***
Total	**Indicated**		**60,794**	**0.22**	**0.62**	**1.32**
Total	**M+I**		**120,599**	**0.23**	**0.65**	**1.35**
Inferred						
Nkamouna						
Breccia	Inferred	0.23	766	0.39	0.49	2.19
Ferralite	Inferred	0.12	19,163	0.18	0.66	1.05
Subtotal	***Inferred***		***19,929***	***0.19***	***0.65***	***1.09***
Mada						
Breccia	Inferred	0.23	14,790	0.40	0.53	2.47
Ferralite	Inferred	0.12	167,831	0.18	0.59	1.10
Subtotal	***Inferred***		***182,621***	***0.20***	***0.58***	***1.21***
Total	**Inferred**		**202,551**	**0.20**	**0.59**	**1.20**

Note: Mineral resources are not mineral reserves and do not have demonstrated economic viability.
All figures have been rounded to reflect the relative accuracy of the estimates.
Reported at cut-off grades of 0.12 and 0.23% cobalt contained within Ferralite and Breccia, respectively.

SRK has reviewed the sample preparation and analytic methods utilized during the various drilling and sampling campaigns conducted at the project, and is of the opinion that the results are generally of high quality and suitable for use in resource estimation. SRK has also conducted a thorough audit of the 2009 resource model that forms the basis for current Mineral Resources, and is of the opinion that the model has been constructed using industry accepted methods, and is suitable as a basis for a Feasibility level of study.

Mining

The two ore types breccia and ferralite are targeted for physical upgrading and mined at approximately 10,000 to 20,000 t/d using small excavators and articulated dump trucks (ADT's). Both breccia and ferralite will be transported from the pit face to run-of-mine (ROM) stockpiles located near the plant site in Nkamouna, the southern extent of Mada and an emergency stockpile between the two. From the stockpile fingers holding desired grade, the ore types are transported to the physical upgrade (PUG) plant using front end loaders and haul trucks. Waste is removed through the use of bulldozers and side casting of rehandle with long boom excavators.

Table 2: Nkamouna and Mada Reserves by Rock Type (as of December 31, 2010)

Ore Type	Ore Tonnes (000's)	Co Grade (%)	Mn Grade (%)	Ni Grade (%)
Ferralite Ore	57,097	0.23	1.30	0.69
Breccia Ore	11,035	0.42	2.37	0.54
Total Proven and Probable	**68,132**	**0.26**	**1.48**	**0.66**

Notes:
Reserves are based on a Co price of US$57,761/t (US$26.20/lb) Ni price of US$19,208/t (US$8.713 /lb) and a Mn price of US$1,360/t (US$0.544/lb).
Please note that the BFS states that the $MnCO_3$ price is US$1,306/t, based on 40% of a Mn metal price of US$3,000/t. Refer to Table 20.2.1
Full mining recovery is assumed.
Mine reserves are not diluted.
Cut-off grades are not representative of internal or break-even calculations but rather stockpile grade bin classification above 0.12% Co for ferralite and 0.2% Co for breccia.
In-situ Co, Mn and Ni grade does not include average metallurgical recovery of 58.66% Co, 16.43% Ni and 53.06% Mn.

SRK is of the opinion that an appropriate level of geological modeling, mine planning, metallurgical test work, metallurgical modeling, infrastructure design, tailings design, environmental planning, cost modeling and economic analysis to support a feasibility level study and associated resource and reserve statement.

Metallurgy

The metallurgical evaluation centered on the selected process route and the objective was to establish the design parameters in order to engineer the process. This was achieved by conducting small scale batch experiments for each of the unit processes followed by combination of the unit processes in the four pilot plant campaigns.

The extensive testwork provided confirmation of the selected process flowsheet and acceptable recoveries and product qualities were achieved during the continuous pilot plant campaigns at expected reagent consumptions.

The metallurgical evaluation provided all the required data to design the process plant and auxiliaries. The reagent consumptions for the operating cost estimate were derived from the metallurgical testwork campaigns.

Process and Plant

The process plant consists of two process circuits namely the physical upgrade (PUG) plant, more commonly known as a wash plant, and the leach and recovery circuit.

Physical Upgrade Plant

The objective of the physical upgrade (PUG) plant is to remove most of the non-economic material, thereby eliminating the requirements to construct a leach and recovery circuit to be able to treat the whole ore. This is achieved by utilizing the physical characteristics of the harder, coarser and denser cobalt, nickel and manganese-bearing minerals and that of the fine gangue material.

The objectives of the individual unit operations within the PUG circuit are presented in Table 3.

Table 3: Objectives of the PUG Circuit

Sub Circuit	Objective
Primary and secondary crushing	To accept mined ore via the ROM bin and to reduce the ore to minus 100 mm and minus 6 mm.
Paddle mixer and screen	To slurry the minus 100 mm and minus 6 mm feed material from the crushing circuit and to screen the slurried material to remove plus 6 mm particles.
Attritioning and hydrosizer classification	To attrition the material thereby liberating the coarse cobalt rich Asbolane material from the fine gangue material. To separate the coarse cobalt rich Asbolane particles from the fine gangue material.
Concentrate stockpiling	To create a live stockpile for surge capacity between the PUG circuit and the concentrate grinding circuit. To create dead stockpiles of concentrate that may be used to blend the concentrate thereby creating a constant grade to the downstream leach and recovery plant .

Leach and Recovery Plant

The leach and recovery plant's objective is to recover cobalt and nickel as a mixed sulfide product and manganese as a manganese carbonate product from the PUG concentrate.

The objectives of the individual circuits within the leach and recover circuits are presented in Table 4.

Table 4: Objectives of the Leach and Recovery Circuit

Sub Circuit	Objective
Concentrate grinding	To grind the PUG concentrate in an open circuit configuration to a nominal product size (P_{80}) of 106 µm
Leach	To leach cobalt, nickel and manganese from the PUG concentrate
Primary purification	To precipitate iron and aluminum from the pregnant leach solution (PLS)
Counter current decantation (CCD)	To recover leach solution containing cobalt, nickel and manganese from barren solids
CCD tailings processing	To collect and pump tailings to CCD tailings facility
Leach and purification area scrubber	To scrub off-gasses from the leach and primary purification tanks
Secondary purification	To precipitate more iron and aluminium from the PLS stream To remove precipitated Fe and Al from the PLS stream
Sulfide precipitation	To precipitate cobalt and nickel from the PLS as a mixed sulfide To recover the precipitated sulfide from the barren stream To filter and wash and package the mixed sulfide product to remove entrained process solution from the sulfide product
Tertiary purification	To precipitate the remaining iron and aluminium from the PLS stream To remove the precipitated iron and aluminium from the PLS stream
Manganese carbonate precipitation - 1	To precipitate 95% of the leached manganese as manganese carbonate To remove the precipitated manganese carbonate product from the solution To filter and wash and package the manganese carbonate product
Manganese carbonate precipitation - 2	To precipitate the remaining 5% of the leached manganese as manganese carbonate To remove the precipitated manganese carbonate product from the solution To filter the manganese carbonate precipitate

Tailings Storage Facilities

The Tailings Storage Facilities (TSFs) will provide separate storage for PUG tailings and CCD tailings. The TSFs will be built within the Napene Creek drainage basin and immediately to its north. Stored tailings will essentially fill the upper end of the Napene Creek basin. The location reduces offsite run-on to the facilities thus limiting the need to handle excess waters over the project life.

Financial Analysis

The financial analysis results, shown in Table 5, indicate an $NPV_{8\%}$ of US$669 million with an IRR of 22% (after estimated taxes). The estimated payback will be in 41 months (2Q 2017) from the start of production in 2014. Table 5 provides the basis of the LOM plan and economics:

Table 5: Financial Model Results

Description	Units	Value	Unit Cost (US$/lb-Co)
Production			
ROM Ore Processed	kt	68,132	-
Cobalt Produced	klb	229,843	-
Nickel Produced	klb	163,483	-
$MnCO_3$ Produced	klb	2,478,929	-
Estimate of Cash Flow			
Cobalt Price	US$/lb	$26.20	-
Nickel Price	US$/lb	$8.71	-
$MnCO_3$ Price	US$/lb	$0.54	-
Gross Revenue	**US$000's**	**7,635,149**	**$22.267**
Freight & Marketing	US$000's	(282,651)	($0.824)
Net Revenue	**US$000's**	**7,352,497**	**$21.442**
Gross Income	**US$000's**	**7,352,497**	**$21.442**
Operating Costs			
Mining	US$000's	329,067	$0.960
PUG Plant	US$000's	288,272	$0.841
Hydromet Plant	US$000's	1,920,910	$5.602
Process G&A	US$000's	473,140	$1.380
G&A	US$000's	521,987	$1.522
Ad Valorem Tax	US$000's	152,703	$0.445
Operating Costs	**US$000's**	**3,686,079**	**$10.750**
Cash Costs			$11.574
Operating Margin		**3,666,418**	**$10.692**
Capital			
Mine Equipment	US$000's	112,944	-
PUG & Hydromet Plants	US$000's	565,775	-
TSF	US$000's	97,586	-
Owners Costs	US$000's	11,168	-
Mine Closure	US$000's	51,252	-
Total Capital	**US$000's**	**838,725**	**-**
Total Tax	US$000's	726,463	-
Cash Flow	**US$000's**	**2,139,019**	**-**
Present Value @ 8%	US$000's	669,579	-
IRR		22%	-

1 Introduction (Item 4)

1.1 Terms of Reference and Purpose of the Report

Geovic Mining Corp. (Geovic) has engaged SRK Consulting (U.S.), Inc. (SRK) to prepare a Technical Report for the Nkamouna and Mada Deposits (or the Project) to meet the requirements of Canadian National Instrument 43-101 (NI 43-101) and for the further development and advancement of the Project.

The Project is Co-Ni-Mn greenfields deposit located in the Haut Nyong Division, East Province of Cameroon, Africa, approximately 400 km from the capital city of Yaounde.

This technical report is a summarized version of the *Nkamouna Co-Ni-Mn Project Feasibility Study* authored by Lycopodium Minerals Pty Ltd (Lycopodium) with contributions by SRK, Knight Piésold and Geovic.

This Technical Report is prepared using the industry accepted Canadian Institute of Mining, Metallurgy and Petroleum (CIM) "Best Practices and Reporting Guidelines" for disclosing mineral exploration information, the Canadian Securities Administrators revised regulations in NI 43-101 (Standards of Disclosure For Mineral Projects) and Companion Policy 43-101CP, and CIM Definition Standards for Mineral Resources and Mineral Reserves (December 11, 2005).

1.2 Reliance on Other Experts (Item 5)

Given the large contribution of work from multiple consulting firms, the following summary illustrates main areas of responsibility in the feasibility study used as a basis of this report:

Lycopodium

- Process design criteria based in part and in fact on historical metallurgical information and testwork results provided by GeoCam and the pilot plant testwork undertaken at Hazen and Ammtec;
- Mass and energy balance;
- Process flow diagrams (PFDs);
- Marked up instrumented PFDs (rather than full P&IDs);
- Mechanical and electrical equipment lists;
- Process control philosophy and instrumentation list;
- Compilation of process operating costs, based on cost certain inputs provided by GeoCam and independent reagent suppliers;
- Process plant layout and equipment sizing, based on the process design criteria;
- General arrangement drawings associated with the process plant, support facilities and services and infrastructure;
- Preliminary design and layout of the following infrastructure facilities:
    - Accommodation village and construction camp,
    - Main administration office, first aid and security buildings, and

- Site buildings.

- Mine services and infrastructure based on SRK and Geovic specifications;
- Design and layout of all services associated with the plant and accommodation village;
- High voltage distribution powerlines;
- Communication systems;
- Edjé River water pumping station and fresh water delivery pipeline to the plant site;
- PUG and CCD tailings distribution pipelines; and
- Capital cost estimate, excluding the TSF, mining, owner's costs and mine closure costs.

Geovic/GeoCam

- Preparation of the consolidated Physical Upgrade (PUG) report;
- Provision of Owner's costs;
- Establishment of operations organization structure and manning levels;
- Provision of general and administration (G&A) costs;
- Scoping and costing of the main road upgrade from Abong Mbang to the mine site;
- Management of SRK and Knight Piésold;
- Management of the testwork program at Hazen and Pocock;
- Permitting and environmental approvals;
- Aggregate quarry scope and costing; and
- Marketing.

SRK

- Resource and reserve estimate;
- Mine design;
- Mining equipment selection;
- Mine capital costs;
- Mine operating costs;
- Preparation of the financial model for the Project; and
- Preparation of 43-101 Technical Report.

Knight Piésold

- Plant site geotechnical studies;
- Design and costing for the PUG and CCD tailings storage facilities (TSFs), including decant return water pontoon and pumping system and underdrain and embankment

drainage systems. Earthworks costing unit rates were provided by SRK and Geovic/GeoCam;

- Design and costing for the Glauber's salt storage pad (GS Pad) and alternative Glauber's salt storage facility (Alternative GSF), including decant return and underdrain and embankment drainage systems. Earthworks costing unit rates were provided by SRK and Geovic/GeoCam;
- Permitting and environmental reporting, with assistance from Rainbow Environmental Consulting;
- Erosion and Sediment Control Plan;
- Preparation of the Environmental and Social Assessment Mine Reclamation and Closure Plan and associated supporting documents (ESAP, etc.); and
- Conceptual design of the CCD Tailings Effluent and Edjé River Mixing Model.

1.2.1 Sources of Information

SRK's opinion contained herein is based on information provided to SRK by Geovic, Lycopodium, Knight Piésold and GeoCam throughout the course of SRK's investigations.

The following two reports contained a significant amount of geological information used in the preparation of this Technical Report:

- Pincock, Allen & Holt, (March 12, 2007), *NI 43-101 Technical Report, Nkamouna and Mada Cobalt Projects, Cameroon* (the 2007 PAH Report); and
- Pincock, Allen & Holt, (January 18, 2008), *NI 43-101 Technical Report, Nkamouna Cobalt Project Feasibility Study* (the 2008 PAH Report).

For all other information, data has been extracted from the following feasibility volumes:

- 637-STY-001_E S1 Executive Summary.doc;
- 1637-STY-001_E S10 Process and Process Plant Description.doc;
- 1637-STY-001_E S11 Support Facilities and Services.doc;
- 1637-STY-001_E S12 Infrastructure.doc;
- 1637-STY-001_E S13 Tailings Storage Facilities.doc;
- 1637-STY-001_E S14 Reagent and Product Transport.doc;
- 1637-STY-001_E S15 Marketing.doc;
- 1637-STY-001_E S16 Project Implementation.doc;
- 1637-STY-001_E S17 Operations Plan.doc;
- 1637-STY-001_E S18 Operating Cost Estimate.doc;
- 1637-STY-001_E S19 Capital Cost Estimate.doc;
- 1637-STY-001_E S2 Project Summary.doc;
- 1637-STY-001_E S20 Financial Analysis.doc;

- 1637-STY-001_E S21 Threats and Opportunities.doc;
- 1637-STY-001_E S3 Permitting and Environmental.doc;
- 1637-STY-001_E S4 Community Relations and Social Mgt.doc;
- 1637-STY-001_E S5 Land and Legal.doc;
- 1637-STY-001_E S6 Geology.doc;
- 1637-STY-001_E S7 Mineral Resource and Reserves.doc; and
- 1637-STY-001_E S8 Mining.doc.

Additional sources of information include data and reports supplied by Geovic as well as documents cited in Section 21.

SRK used its experience to determine if the information from published feasibility volumes was suitable for inclusion in this Technical Report. The level of detail used in this report was appropriate for a feasibility level of study.

1.3 Qualifications of Consultants

SRK

The SRK Group is comprised of over 1,100 staff, offering expertise in a wide range of engineering disciplines. The SRK Group's independence is ensured by the fact that it holds no equity in any project and that its ownership rests solely with its staff. This permits SRK to provide its clients with conflict-free and objective recommendations on crucial judgment issues. SRK has a demonstrated record of accomplishment in undertaking independent assessments of Mineral Resources and Mineral Reserves, project evaluations and audits, technical reports and independent feasibility evaluations to bankable standards on behalf of exploration and mining companies and financial institutions worldwide. The SRK Group has also worked with a large number of major international mining companies and their projects, providing mining industry consultancy service inputs.

SRK are not insiders, associates, or affiliates of Geovic. The results of this Technical Report are not dependent upon any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings between Geovic and SRK. SRK will receive a fee for its work in accordance with normal professional consulting practice.

Lycopodium Minerals Pty Ltd

Lycopodium Minerals Pty Ltd (Lycopodium) is part of the Lycopodium Ltd group, a highly respected, Australian-based, leading international engineering and project management consultancy specializing in the area of extractive metallurgy and the design and construction of mineral processing plants and associated infrastructure. Lycopodium provides a complete service to the mining, mineral processing and infrastructure sectors from feasibility phase through to commissioning and handover of an operating plant.

Over the years Lycopodium has established a reputation for providing technically innovative and cost effective engineering solutions for mining and resource projects across the globe. Their

resume of projects, often undertaken in challenging environments, reflects diversity in not only commodity, but client background, technology, scale of operation and geographical location.

Lycopodium believes its success is a function of the success of its client's projects and is focused on always leaving a positive legacy in respect to the environment, safety and community in which they work.

Knight Piésold

Knight Piésold is an international consulting company providing engineering and environmental services to the public and private sectors in the areas of mining, power, water, transportation, and construction over the past 90 years. Each technical discipline is managed by a team of professionals working together to design projects to the highest standards. The integrated efforts of over 650 experienced professionals in our offices around the globe enables us to deliver high quality specialized services and innovative solutions that respect social, environmental, and economic responsibilities.

Knight Piésold has extensive experience in siting, design, permitting, construction support, monitoring, and closure of tailings storage facilities for a wide variety of climatic and seismic conditions. We have been involved in the design of tailings management facilities for over 400 mining projects and have extensive experience with all types of tailings management systems, and have pioneered the development of alternative tailings management technologies.

Understanding and protecting the environment is a fundamental responsibility of our business. Knight Piésold undertakes environmental baseline studies, environmental and socio-economic impact assessments, permitting, and construction monitoring for all types of resource development. We work with our clients so that their corporate policies and involved capital lending bodies are satisfied with respect to their own environmental requirements. Social and Environmental Assessments are typically carried out in accordance with standards set out in the Equator Principles, the International Finance Corporation, and host country regulations. Studies undertaken incorporate social and environmental mitigation measures to achieve project designs that maximize project benefits, reduce impacts, and achieve a fusion of development with environmental protection and sustainability.

Qualified Persons

The individuals who have provided input to this technical report have extensive experience in the mining industry and are members in good standing of appropriate professional institutions. Mr. Jeffrey Volk, Mr. Bret Swanson and Mr. Brett Crossley are the Qualified Persons (QP) for this Technical Report. Mr. Volk is responsible Sections 4,5,6,7,8,9,10,11,12,13,15. Mr. Crossley is responsible for Section 14. Mr. Swanson is responsible for mineral reserves and the compilation and editing of all Sections of the report.

The key project personnel contributing to this report are listed in Table 1.3.1. By virtue of the education and relevant past experience of Jeffrey Volk, Bret Swanson and Brett Crossley are all QP's as this term is defined in NI 43-101. Mr. Volk is a Principal Resource Geologist based in the SRK Denver office. Mr. Swanson is a Principal Mining Engineer based in the SRK Denver office and Mr. Crossley is Principal Metallurgist for Lycopodium Minerals Pty Ltd based in Perth, Australia.

The Certificate of Author for Jeffrey Volk, Bret Swanson and Brett Crossley are provided in Appendix A.

Table 1.3.1: Key Project Personnel

Name	Title	Discipline
Jeffrey Volk	Principal Resource Geologist	Resources/QP
Bret Swanson	Principal Mining Engineer	Mine Reserves/Mine Planning/QP
Brett Crossley	Principal Metallurgist	Metallurgy/QP
Dorinda Bair	Senior Geologist	Geology QA/QC
Herman Schwarzer	Lead Process Engineer	Process Plant/Infrastructure
Norbert Peyfuss	Executive Project Manager	Tailings/Geotechnical
Jaye Pickarts	Executive Vice President	Environmental

1.4 Site Visit

Jeffrey Volk visited the property on November 10, 2009 for 1 day. Bret Swanson visited the site on November 10, 2009 and September 17, 2010. Lycopodium sent appropriate project personnel to visit the site from January 25 to February 5, 2010. These Lycopodium representatives negated the need for Brett Crossley to visit the site. The Lycopodium site visit was conducted by Lycopodium's Director of Projects and a Senior Project Engineer to review all infrastructure, site access and layout requirements for the project.

The site visit consisted of an inspection of the geology and technical services offices, sample preparation areas, sample storage facilities and the main trench and pitting areas of the deposit. Tours of proposed infrastructure locations and access routes were also conducted. SRK also reviewed Geovic's drilling and sampling practices, and observed sampling and manual physical upgrading of material collected from the main trench during the visit and conducted a preliminary dozer push evaluation.

Site visits were conducted by Knight Piésold engineering staff in 2006 to observe proposed tailings storage areas and plant site among others, and by Knight Piésold environmental staff, in 2010 and 2011 to oversee baseline sample collection and support in-country consultants in the preparation of these documents.

1.5 Effective Date

The effective date of the resource and reserve estimate is December 31, 2010.

1.6 Units of Measure

The metric system is used throughout this report, except where otherwise stated. All currency is stated in US dollars.

2 Property Description and Location (Item 6)

The Nkamouna and Mada deposits are located in southeastern Cameroon, (Figure 2-1) approximately 640 road kilometers east of the port city of Douala and 400 road kilometers east of the capital of Yaounde.

The Nkamouna deposit is one of seven separately-named laterite plateaus forming a crescent-shaped array which extends 80 km north-south and 45 km east-west. The Mada deposit is a second laterite plateau that is contiguous with the Nkamouna deposit and extends approximately 10 km to the north. The irregularly-shaped laterite plateaus comprise approximately 337 km^2 within the 1,250 km^2 Mine Permit area. The named laterite plateaus are Nkamouna, Mada, Rapodjombo, North Mang, South Mang, Messea and Kondong. The subject of this technical report is the Nkamouna (pronounced Ka-moon-ah) and Mada areas.

Administratively, the project is located in the Haut Nyong Division of the East Province. Nkamouna-Mada and the other laterite plateaus (except Kondong) lie within the Lomie Subdivision. All the laterite plateaus except Kondong lie on the drainage divide between the Dja River to the south and west, and the Boumba River to the north and east. Both rivers are tributaries to the Congo (Zaire) River, which lies 600 km to the southeast.

Lomie is the administrative center of the Subdivision that hosts the project and has been the staging area for GeoCam activities. Lomie has about 3,500 inhabitants, a limited local electrical supply, and very basic services and supplies. There is no telephone service, airstrip, or approved heliport, and only rudimentary medical facilities are present. GeoCam's field operations are based from the Kongo Camp, a fully-contained compound near the village of Kongo. The compound has adequate working and sleeping quarters for approximately 40 people, a diesel generator, satellite-phone facilities, diesel fuel storage, a kitchen with refrigerators, repair shop and sample preparation and storage facilities.

General geographic coordinates for the Nkamouna and Mada project area are approximately: Longitude N-3º 20´ and Latitude E-13º 50´.

2.1 Property Location

The closest town to the Project site is Lomie, located approximately 26 km to the west-southwest. The closest railroad transport to the Project is located in the town of Belabo, a distance of approximately 250 km to the Project site. Transportation from Yaounde to the Project site is by paved highway to Ayos, an improved public road to Abong Mbang and via private logging roads or public roads to the project site.

International airports and modern telecommunication facilities exist at Yaounde and Douala. Suitable shipping and receiving facilities exist at the international seaport of Douala.

2.1.1 Location of Mineralization

The Nkamouna and Mada deposits are located within the boundaries of the Mining Permit area, as shown on Figure 2-2.

2.2 Mining Convention and Mineral Title

The Mining Code of 16 April 2001 governs mining activities in the Republic of Cameroon. It establishes the requirements for mining permits, which are granted by decree of the President of

the Republic. The Mining Code establishes the provisions regarding mineral title, operations, quarrying of materials, asserts the rights and obligations of operators, financing provisions, technical and administrative supervision of mining operations, penalties and other provisions.

Prior to obtaining a Mining Permit, a Mining Convention may be concluded between the holder of an exploration permit and the government. GeoCam signed a Mining Convention with the Republic of Cameroon on July 31, 2002. The Mining Convention lays out the obligations of GeoCam with respect to administrative monitoring and inspections, the employment of Cameroonian personnel, social infrastructure, and the use of Cameroonian equipment, services and insurance. It confers the right on GeoCam to create the infrastructure necessary for the mining operation. The commitments of the State, including the stability of the legal and tax regime, are also laid out in this Convention.

Subsequent to the signing of the Convention, the procedure for delivery of a Mining Permit was commenced and on April 11, 2003, the President of Cameroon issued the Mining Permit to GeoCam which is valid for an initial period of 25 years. The Mining Permit and Convention provide exclusive title to the minerals and are renewable every ten years thereafter until the depletion of resources.

2.2.1 Policy

Fifty-nine percent of the East Province is dominated by forests zoned 'multiple-use'. Over 64 logging concessions are designated in the province that surround GeoCam's 'mineral exclusive zone' pursuant to the provisions of the Mining Permit. A significant portion of the area is also dedicated to protected forests, wildlife reserves, and general 'evergreen forest' habitat (22%) that are located well away from planned operations. A small portion of the district is zoned for mineral development (1.6%), part of which includes 'mineral exclusive lands' (0.35%). Indigenous community lands, dominated by subsistence farming and 'community forest' developments, form the remainder of the district lands, which covers about 18% of the province. These lands are located principally along the main access routes developed when the province was first opened to plantation farming in the late 19th Century.

Lands held within the Mining Permit are designated 'multiple-use', where the principal mineralized areas are set aside for 'exclusive mine' development. Mining Permit lands have been specifically established to exclude village lands and avoid conflicts with local communities.

Specific sites that will be impacted by mining and mine related activities will be 'land leased' and will have 'site specific' environmental plans designed and approved by governing agencies prior to mining. This process requires local government approval, following a review of each site by district leaders. The closest village to the Nkamouna site is Kongo village, which is well removed from most day-to-day operational noise and activities.

Any new facility, including the staff housing facility, will have to be land leased or purchased. Port storage and facilities required for transportation to and from the Project site will also have to be land leased, sublet, or otherwise legally obtained from existing companies operating in Douala.

2.2.2 Tenure

The boundary of the Mining Permit has been surveyed by Global Positioning Satellite (GPS) carried out by a certified government surveyor. The irregular 18-corner polygon-shaped Mine Permit area is monumented with red painted cement markers that stand about 1 m high and are

described in the Mining Permit. Geographic references defining the Mining Permit boundary are cited in Table 2.2.2.1. These geographic references have been converted to UTM coordinates and the area contained within the perimeter has been calculated as 1,645 km^2.

Table 2.2.2.1: Mine Permit Boundary

Points	Geographic References	
	East	North
1	13°59’43.7’’	3°15’00.3’’
2	13°47’03 .4’’	3°14’49.6’’
3	13°47’02.5’’	3°29’57.7’’
4	13°58’37.6’’	3°29’56.0’’
5	14°09’21.3’’	3°22’46.6’’
6	14°13’45.9’’	3°22’46.8’’
7	14°13’57.7’’	2°53’46.1’’
8	14°08’11.9’’	2°53’40.0’’
9	13°56’54.8’’	2°49’19.6’’
10	13°56’54.5’’	2°42’41.9’’
11	13°51’41.3’	2°42’24.9’’
12	13°51’05.7’’	2°49’21.6’’
13	13°56’55.3’’	2°49’22.5’’
14	14°08’12.4’’	2°53’41.9’’
15	14°08’10.1’’	3°01’32.3’’
16	14°05’25.0’’	3°05’43.9’’
17	14°05’23.3’’	3°19’27.3’’
18	13°59’43.1’’	3°24’33.7’’

A copy of the mining decree is presented in Appendix B.

2.2.3 Surface Rights and Compensation

The Mining Permit grants GeoCam the exclusive right to mine the minerals within its boundary. A Land Lease authorizes the lease holder the right to use the surface of the land for mining and/or other developmental uses. A fee is charged for the use of the land on a per-hectare basis. The Land Lease configuration was approved by the Cameroon Prime Minister on August 21, 2008 and is awaiting final signature by the President of Cameroon. Once the Land Lease is signed by the President, lease payments will commence for the utilization of the land surface for mining and support activities for the Project.

The Land Lease boundary has an area of 2,489 hectares (ha). Geographic references defining the Land Lease boundary are cited in Table 2.2.3.1. As mining and concurrent reclamation activities progress, certain areas within the Land Lease boundary will no longer be required to support the Project activities and other areas will need to be added. These boundary modifications will be accommodated as a normal part of the ongoing permitting and compliance process for the Project.

Table 2.2.3.1: Land Lease Boundary

Points	Geographic References	
	East	North
B1	372 931.36	364 611.04
B2	373 007.16	359 946.13
B3	370 573.09	359 398.45
B4	367 979.97	359 332.56
B5	368 014.10	364 421.71

The Land Lease boundary is the nearly square boundary enclosing the plant, most of the mine, and most of the tailings storage facilities (Figure 2-3).

2.2.4 Land Ownership, Administration and Governance

GeoCam has 100% of the rights to mine the cobalt-nickel and associated materials within its Mining Permit. GeoCam is 60.5% owned by Geovic Ltd. Geovic Ltd. is a 100% held subsidiary of Geovic, a publicly held company, the shares of which are listed and posted for trading on the Over the Counter Traded Bulletin Board (OCTBB – Symbol GVCM) and the Toronto Stock Exchange (TSX – Symbol GMC). The remaining 39.5% interest is owned or represented by the National Investment Corporation of Cameroon (SNI), an arm of the Cameroon government. SNI owns 20% and provides a full carry for it's ownership and the remaining 19.5% shareholding which is held by private Cameroonian investors under separate agreement.

The right to mine held by GeoCam is realized through its Mining Permit. On April 11, 2003, a Mining Permit Decree was issued to GeoCam, covering an area of 1,250 km^2. The permit authorizes 1,250 km^2 within a 1,645 km^2 boundary as defined by the coordinates.

2.2.5 Tenement Holding

The Company has constructed an exploration camp near Kongo village. This camp includes offices, small stores, sleeping quarters and a commissary. Plans are to develop temporary and permanent housing for some of the staff near the Project site. There exists sufficient land area within the pending Land Lease boundary to accommodate Project housing requirements. However, the current plan is to locate future housing to the north of the Land Lease boundary. While land tenure for this area is not yet secured, there exists sufficient land within the Land Lease boundary at nominally the same construction cost as would be required in the current preferred location to the north.

2.2.6 Project Development and Approvals

The principal remaining permit required before the initiation of construction of the Nkamouna Project is the Land Lease for development sites. A title deed in favor of the State for the 2,489 ha permit area was created and signed by the Prime Minister in 2008, wherein a rental of maximum US$400,000/y was negotiated. This lease is awaiting signature by the President of Cameroon.

A Certificate of Conformity for the Environmental and Social Assessment (ESA) was received on 29 May 2007. The Ministry of Environment and Protection of Nature is requiring that the 2007 ESA, the 2010 ESA Update, and the outcomes of this Feasibility Study Update be consolidated into a single document. The Company plans to consolidate these documents in the first half of 2011. The Environmental Permit details the impact mitigation measures, plans and

monitoring. A land rehabilitation surety will be required based on the details presented in the updated Mine Closure and Reclamation Plan.

GeoCam will have the right to utilize lands, build roads, remove vegetation, and mine and process cobalt, nickel and related substances once the land is freed, in accordance to the Geovic Mining Convention of August 1, 2002.

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Figure 2-1
General Location Map for the Nkamouna Cobalt Project

Scale:	NTS
Date:	02 Nov. 2009
Dwg:	2-1 General

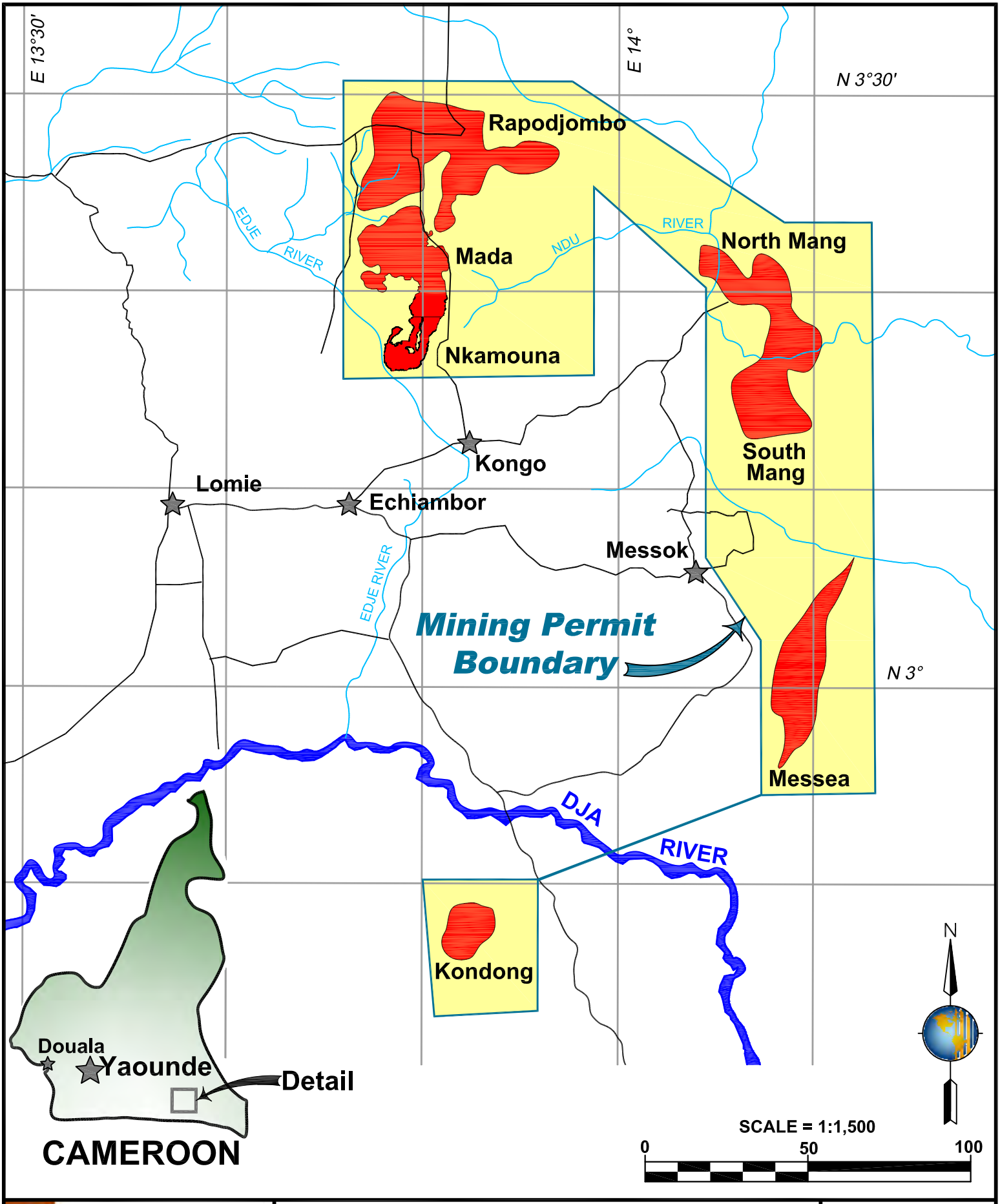

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Figure 2-2
Location of Laterites
Mining Permit Boundary

Scale:	1:1,500
Date:	02 Nov. 2009
Dwg:	2-2 Reserves

N

1 : 40,000
Scale in Meters

0 400 800 1,200 1,600 2,000

STAFF HOUSING

CCD

CCD

GSF

PUG TSF

PUG Plant

Stock Piles

To Kongo Camp

Legend

Schist Boundary

Rivers/Streams

Lease Boundary

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Fax: +235.22.21.1802

Nkamouna and Mada Mine Site Layout

System:	UTM 33 WGS 84	Prepared For:	Brian Briggs
Date:	22 December 2010	Drawn by:	Andrew Lovato
File:	Catchments Plan	Approved:	.
Scale:	1:40,000	Sheet No:	1 of 1

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

3.1 Topography, Elevation and Vegetation

The vegetation in the plateau areas is typical of a lowland equatorial evergreen forest, characterized by diverse endemic plant species. The forest area is stratified in three layers, including the 40 m tall tree canopy characterized by broad-crown diameters and straight limbless trunks; shorter, more slender, fast-growing, narrow-crown-diameter, fragile trees form the intermediate layer; and the scanty undergrowth layer consisting of vines, brush and ferns. Trees of local economic importance include Ayos, Sapelli, Wengive, Iroka, Bubinga, Azobe, and Obeche. Other diverse species occur in swamplands and patches of dense wet-substrate dominated valley floors.

Recent logging has occurred over most of the mineralized zones within the Mine Permit. The extent of this logging is documented on satellite images and by ground surveys. These logging activities are independent of GeoCam's operations and were part of pre-existing timber leases within the Mine Permit area.

3.2 Climate and Length of Operating Season

The climate of the region is classified as an Equatorial Guinea sub-type characterized by two main seasonal types, the main wet season and main dry season, and two minor seasonal types designated as mini wet and mini dry. The Project area is located on the northwestern margin of the Congo River tropical zone.

Annual maximum monthly temperature ranges from 24°C to 33°C. The lowest daily minimum temperature recorded is 12°C, but temperatures normally do not fall below 18°C.

The average annual precipitation over a 32-year period is 1,580 mm and both the humidity is and evaporation rates are typically high on an annual basis. Maximum annual precipitation measured to date totals 2,200 mm. The main wet season occurs between September and early November, and the main dry season occurs from November to May. The mini wet season lasts about eight weeks from March to May, and the mini dry season extends from June to mid-September. Limited amounts of rainfall occur throughout the year, except during the months of December and January. The average number of rain-free days at the Project was 229, and days receiving a total of at least 25 mm of precipitation at Nkamouna and Mada average 28 mm/y. Average monthly evaporation rates exceed rainfall during the two dry seasons. Data through 2004 show total precipitation of 1,820 mm and total evaporation of 1,951 mm, resulting in a net evaporation of 131 mm. The prevailing wind direction is from the south and southwest, and wind speeds averages less than 4 km/h. Wind gusts rarely exceed 8 km/h and are commonly undetectable beneath the tree canopy near the proposed Plant site.

3.3 Physiography

The central part of the cobalt-nickel mineral district is dominated by a series of rolling upland plateaus that are dissected by several river systems that feed into the main Congo River drainage basin. Elevations in the province range from about 450 m along the lower Dja River to 927 m above sea level at Mount Guimbiri, located east of Abong Mbang. The local upland plateau in the vicinity of the Nkamouna and Mada deposit areas is at an elevation of about 700 m.

The Nkamouna and Mada deposits are relatively flat and have an average depth of 15 m. The majority of the Nkamouna deposit is situated downslope from the process plant and has a natural grade of approximately 5% with upper elevations around 760 m and lower elevations near 610 m. The Nkamouna deposit is a crescent shape about 4 km from east to west and 2 km from north to south. The Mada deposit exhibits an arcuate shape, with maximum dimensions of approximately 10,500 m north-south and 9,000 m in an east-west direction. The process plant is adjacent to the proposed Nkamouna mining area and near the top of a saddle at an approximate elevation of 700 m above sea level.

The following satellite images, vertical aerial photographs and topographic coverage of the Province are available:

- TOPOGRAPHIC SHEETS:
  - Abong Mbang, Medoum, Mintom, Ngoila & Nkamouna sheets (1:200,000 scale), and
  - GeoCam coverage of the Nkamouna (1:5,000 scale) area and Mada (1:10,000 scale) area.
- BLACK AND WHITE PHOTOGRAPHS:
  - Vertical aerial photographs (1:200,000 scale). Flight lines are numbered on the back of 1:200,000 scale topographic sheets and are not comprehensive (1953-54 coverage).
- SATELLITE IMAGES:
  - Landsat, USA (1 m, 15 m and 30 m pixel resolution), and
  - SPOT, French (30 m pixel resolution).

3.4 Access to Property

Access to the Project site is from the seaport of Douala by a well-maintained provincial highway via Yaounde and Ayos. After travelling eastward through Ayos and across the Nyong River, the highway to the Central African Republic deteriorates rapidly to a well-traveled two-lane gravel road to Abong Mbang, however, this road segment has been widened and is being surfaced with asphalt. Turning south from Abong Mbang towards Lomie, the road narrows and is frequented by log and lumber trucks over the next 127 km distance to Lomie. The road from Lomie to Kongo village supports heavy log and lumber transports, as does the road from Kongo village to the project site.

From Lomie, the road passes east through the village of Echiambot, where it branches northeast to the Edjé River and Kongo village. The first mineralized zone at Nkamouna is located 10 km north of this village. The trip from Yaounde to Kongo village takes about 8 hours by 4-WD vehicle.

Transport infrastructure in Lomie includes the Haut Nyong Express that carries people four times per day to Yaounde (18 per bus) and 10 busses per day to Abong Mbang. Motorcycle taxis transport individuals in the Lomie area. Transportation routes to the project are shown in Figure 3-1.

3.5 Local Resources and Infrastructure

The Project site is located in a remote tropical setting that is characterized by an expanding economy driven primarily by the production of forestry related products and minor production of cocoa and palm oil. Government jobs and non-governmental organizations also contribute to the local economy. Several companies conduct substantial logging operations throughout the region and major sawmills are operated within 125 km of the Project site at Lomie, Messamena and Mindourou. Approximately 300 truckloads of logs are hauled per day in the East Province by contract trucking companies. The local logging industry uses large, modern equipment and employs numerous equipment operators and maintenance personnel. In addition, the industrial center and port city of Douala has several equipment, service and supply companies that support Cameroon's industries as well as much of land-locked West Africa.

The town of Abong Mbang, with a population of approximately 30,000, is located at the entrance to the district. It is the provincial headquarters of the Prefect and main administrative and commercial center for the Hyaut Nyong Division. The town hosts a local trade school, service stations (Texaco and Total-Elf), hotels, restaurants and rental phone service. It is the main administrative center for the Ministry of Environment and Protection of Nature, and the Ministry of Mining & Technological Development. The nursing school at Lyos, west of Abong Mbang, is the main training center for local nurses.

Lomie is the closest town to the Nkamouna and Mada Project sites. At present, it takes about one hour to drive the 40 km between Lomie and the Project site. The economy of Lomie is largely undeveloped, except for a large sawmill and surrounding timber harvesting operations. Local businesses include the Lomie Subdivision's government headquarters of the Subprefecture, police station, hospital (two doctors and eight nurses), parochial schools, shops, general mercantile stores and a motel. Most business activity centers around logging and the local saw mill that is located east of town. Other activities include road maintenance, palm oil production, subsistence and local supply agricultural and general commerce. Lomie's municipality has provided diesel electric power (200 kW) to those who can afford it, since 1997. Lomie is the site of a number of domestic and international NGO's that monitor the World Heritage Dja Biosphere reserve and other reserves within the region.

Within the Lomie Subdivision, the number of children in the Lomie primary school district total just over 3,000 pupils, representing over 60% of school age children in the Subdivision. Primary schools in the Subdivision total 25, headed by approximately 60 teachers of whom about 60% are government paid the rest being privately paid (38-government paid/18-private). The local technical school (SAR.SM) typically enrolls over 70 students who are trained principally in rudimentary building skills.

Messok is the second largest town in the Subdivision and hosts a medical aid station, police station, slab-wood constructed motel and a Belgian-based mission school. Kongo village, approximately 2 km from the site of GeoCam's present base camp, is located 32 km east of Lomie. The village population in 1998 totaled about 150 (all names are on the GeoCam land lease). The village has grown much larger since GeoCam began pre-construction and exploration and development activities in the current decade, as is evidenced by the growth of the GeoCam funded parochial school.

Mindourou and Messamena are two sawmill centered towns located in close proximity to the overall project area. These towns have been expanding rapidly, both in population and local infrastructure, as has Lomie and Abong Mbang.

3.5.1 Power Supply

All power will be generated on site with no connection to the utility. It is expected that the power supply for the overall Project will be supplied utilizing high speed diesel powered generators.

Power will be generated in a central power plant located at the main plant site and reticulated to the accommodation village, construction camp, river water harvesting and other isolated load centers via HV overhead transmission lines. The maximum demand for the site has been calculated to be in the order of 18.5 MVA (16 MW).

3.5.2 Water Supply

The Edjé River is the only supply of fresh water makeup to the Project. The water abstraction point currently planned is from a location approximately 3 km West North West from the plant. There is an elevation rise of approximately 135 m from the river to the plant site and two stage pumping into a minor fresh water dam located in the plant site is planned. This water will be filtered and treated in a number of stages to produce filtered, demineralized and potable water supplies for the plant.

The PUG tailings storage facility is designed to accommodate surge capacity of excess rain accumulated during the wet seasons. Operation of the PUG during the dry seasons where a net water input is required generates a drawdown of this surge capacity thereby reducing the seasonal flow variations that would otherwise exist in the fresh water harvesting requirement.

The PUG tailings storage facility is designed to be the prime source of process water and to store process waters required during prolonged dry periods. During such times and when water cannot be taken from the Edjé River for the process, process waters for the PUG and CCD circuits, fresh water to the Plant, and cooling requirements will be provided from reclaim water from the PUG tailings storage facility. This operational provision is intended to eliminate dependency on the Edjé River flow during prolonged dry periods as currently modeled.

Potable water for the accommodation village will be pumped from the main process plant water treatment facility. The water supply for the construction camp will be obtained from an upgraded Edjé River abstraction point currently located adjacent to the Kongo camp.

3.5.3 Buildings and Ancillary Facilities

Land clearing and preliminary construction commenced on a limited basis during 2009. Additional pre-construction activities are planned and budgeted for 2010 with more focused construction activates planned in 2011. The existing exploration camp is located 5 km from the proposed mining activities whereas the expected final camp site is located centrally in the Nkamouna area.

3.5.4 Camp Site

The existing Kongo exploration camp located approximately 12 km south of the main process plant currently feeds and houses 30 to 40 people, and is currently staffed with over 60 resident and day workers though as many as 200 have worked at the site during peak exploration periods.

A further 37 rooms will be constructed to house the mining department who will mobilize early in the construction phase to undertake earthworks on the TSF and bulk plant earthworks.

The Kongo exploration camp will be used to house some of the construction personnel for the initial plant construction and will also be used to house junior staff in the operational phase of the Project.

A separate accommodation village is also planned to be constructed at a location approximately 5 km NNE of the plant. This will accommodate approximately 276 permanent operations personnel. It is planned that the 600 person construction camp will be located near the accommodation village to make use of common facilities and services and to provide additional accommodation for operations personnel post construction.

3.5.5 Manpower

Technical and scientific staff would be recruited largely from the cities of Douala and Yaounde, where several universities offer internationally recognized degrees in specializing in geology and mining. The majority of mine laborers are expected to be sourced regionally. Training of workers is expected to be a critical component to the overall success of the project.

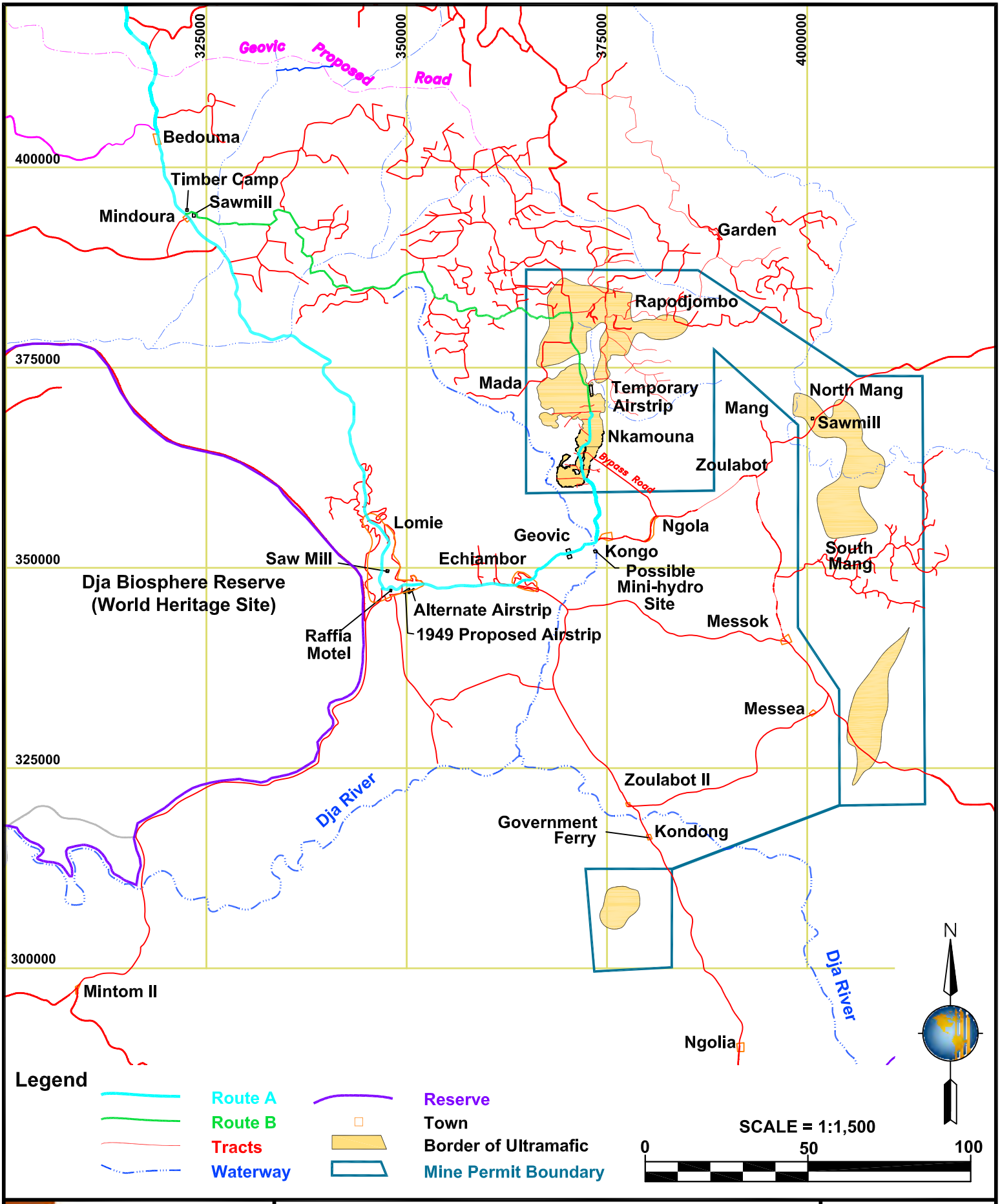

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Figure 3-1
Site Transportation Routes

Scale:	1:1,500
Date:	02 Nov. 2009
Dwg:	3-1 Access

4 History (Item 8)

The following section is excerpted from the 2008 PAH Report. Changes to standardizations have been made to suit the format of this report. Edits to text are annotated by the use of square brackets or reference to SRK.

4.1 Past Exploration and Development

The Nkamouna and Mada deposits are contiguous zones, and together comprise an enriched cobalt-nickel-manganese-iron laterite deposit located within an extensive mineral province in southeastern Cameroon, Africa. Nkamouna and several other nickelferous laterite deposits in southeast Cameroon were first discovered and investigated by the United Nations Development Programme (UNDP) during 1981-1986, in a cooperative project with the Cameroon Ministry of Mines, Water and Energy to evaluate mineral potential in southeastern Cameroon (UNDP Project CMR/81/005). Following a regional stream sediment geochemical survey which indicated the likely presence of laterite nickel mineralization, the UNDP project drilled eleven core holes in the Nkamouna area, which was the most accessible laterite area at that time.

Several of the UNDP holes at Nkamouna intersected laterite and saprolite with interesting nickel and cobalt values. The first hole, KG-S-1, traversed 56 m of lateritic profile and fresh serpentinite, with Ni values up to 1.00% and Co values up to 0.19%. Due to the remote location and the low nickel prices at the time, the discovery did not draw much attention.

No further exploration took place on the property until geologist William Buckovic became aware of the nickel discovery in 1988, subsequent to submitting a proposal in 1986 to explore for minerals to the Cameroon Ministry of Mines. No recorded exploration or mining had taken place on the property since the UNDP work. After assaying samples he was able to obtain from the area, Mr. Buckovic noted in 1994 the higher than typical Co:Ni ratio that characterizes the Cameroon deposits. This high ratio was confirmed by the assay results from the UN coring program. Mr. Buckovic was also aware of recent advances in Australia and elsewhere in the hydrometallurgical processing of previously sub-economic nickel laterite deposits. As a result, in 1995 he helped form a new company, GeoCam, to investigate this unusual but potentially promising occurrence.

A government-issued Prospecting License covering 19,600 km^2 was granted in 1995. In 1999, an Exploration Permit, PDR 67, was granted on a reduced area of 4,876 km^2. A Mining Convention was entered into between GeoCam and the Republic of Cameroon in 2002. In 2003, Mine Permit 33 was issued by decree granting an exclusive right to GeoCam to exploit the deposits within the permitted 1,250 km^2 area. GeoCam program initially was based entirely on manually-dug test pits, and later incorporated drilling and limited trenching. The program began at Nkamouna and was later extended to other laterite plateaus, which were identified by satellite images and air photos. Geologists from the Cameroon Ministry of Mines, Water and Energy participated in the work to provide government oversight as well as training.

By 2004, GeoCam had largely completed the reconnaissance sampling and had undertaken pitting and drilling patterns of varying densities at Nkamouna where access was greater due to recent logging operations, with an eye toward defining deposit parameters for an eventual feasibility study. Between 1995 and 2003, Geovic/GeoCam carried out extensive pitting at Mada. During the period 2005-2009, GeoCam completed significant infill drilling and pitting at

both the Nkamouna and Mada, deposits and the additional 2008-2009 data form the basis for the resources.

Historical studies at Nkamouna and Mada has been performed by GeoCam employees and various consultants. Geological and sampling oversight was provided by Mintec, Inc., of Tucson, Arizona, and by geologist and Qualified Person Nikolai R. Burcham during 2003. Metallurgical and other historical testing was performed from time to time by: Bateman Engineering, Inc. of Tucson, Arizona; METCON Research, Inc. of Tucson; Pittsburgh Metallurgical and Environmental Inc. (PMET) of Pittsburgh, Pennsylvania; Hazen Research of Golden, Colorado; Knight Piésold & Co. of Denver, Colorado; [WGI] and others.

The Nkamouna and Mada deposits are undeveloped, as are those of the adjacent laterite plateaus.

4.2 Historic Mineral Resource and Reserve Estimates

The reported historical mineral resource and mineral reserve inventories have been reported as compliant under CIM guidelines, and economic parameters used to derive the estimates accurately reflected the economics of exploiting these deposits using the assumptions and level of detailed work at the time. Procedures and data used for these estimates have been reviewed and verified by a Qualified Person as designated by Pincock, Allen and Holt (PAH), and therefore have been classified and reported as mineral resources and mineral reserves under NI 43-101. SRK has not conducted a review of previously declared mineral resources and mineral reserves, and notes that the previously calculated reserves are not current with the updated resource estimates, which are the subject of this report.

PAH produced resource estimates for the Nkamouna and Mada deposits, and a reserve estimate for the Nkamouna deposit in 2007. These previous estimates are provided in Tables 4.2.1 through 4.2.5.

PAH also contributed to a feasibility study which included updated resource and reserve estimates for Nkamouna in 2008, based on additional drilling and pitting information collected during 2007-2008.

Table 4.2.1: PAH Historic Resource Estimate for the Nkamouna Deposit, 2007*

Lithology	Resource Category	Cut-off (%Co)	Tonnes (000's)	Average Grade		
				Co (%)	Ni (%)	Mn (%)
Upper Laterite	Measured	0.12	18	0.411	0.395	2.332
Breccia	Measured	0.23	5,189	0.461	0.538	2.328
Ferralite	Measured	0.12	24,186	0.228	0.680	1.175
Total	**Measured**		**29,393**	**0.269**	**0.655**	**1.379**
Upper Laterite	Indicated	0.12	17	0.362	0.309	1.182
Breccia	Indicated	0.23	2,897	0.413	0.460	1.990
Ferralite	Indicated	0.12	28,798	0.204	0.672	1.037
Total	**Indicated**		**31,712**	**0.223**	**0.652**	**1.124**
Total	**Meas + Ind**		**61,105**	**0.245**	**0.654**	**1.247**
Upper Laterite	Inferred	0.12	75	0.140	0.152	0.641
Breccia	Inferred	0.23	538	0.384	0.434	1.612
Ferralite	Inferred	0.12	13,975	0.178	0.614	0.905
Total	**Inferred**		**14,588**	**0.185**	**0.605**	**0.930**

* (Pincock Allen and Holt, 2007). Resources stated above a 0.12% Co cutoff grade for laterite and ferralite, material, and above a 0.23% Co cutoff grade for breccias material. Resources are inclusive of reserves.

Table 4.2.2: Historic Resource Estimate for the Mada Deposit, PAH, 2007*

Lithology	Resource Category	Cut-off (%Co)	Tonnes (000's)	Average Grade		
				Co (%)	Ni (%)	Mn (%)
Inferred NN Resource Before Adjustment for Volume-Variance Effects						
Breccia	Inferred	0.28	13,200	0.43	0.56	2.87
Ferralite	Inferred	0.12	118,000	0.21	0.49	1.08
Total	**Inferred**		**131,200**	**0.23**	**0.50**	**1.26**
Inferred NN Resource After Adjustment for Volume-Variance Effects						
Breccia	Inferred	0.28	14,300	0.38	0.53	2.53
Ferralite	Inferred	0.12	130,800	0.20	0.48	1.00
Total	**Inferred**		**145,100**	**0.21**	**0.48**	**1.15**

* (Pincock Allen and Holt, 2007); Sample Spacing up to 1500 m, Maximum Extrapolation 420 m. Resources stated above a 0.12% Co cutoff grade for ferralite, material, and above a 0.28% Co cutoff grade for breccias material.

Table 4.2.3: Historic Reserve Estimate for the Nkamouna Deposit, PAH, 2007*

Classification	Mineralized Zone Tonnes	Average Grade			Interburden Tonnes	Overburden Tonnes	Total Tonnes
		Co (%)	Ni (%)	Mn (%)			
Proven	26,280,570	0.256	0.720	1.331			
Probable	26,433,743	0.218	0.718	1.109			
Total	**52,714,314**	**0.237**	**0.719**	**1.220**	**2,116,982**	**80,120,403**	**132,834,717**

* (Pincock Allen and Holt, 2007); Reserves stated above a US$12.00/tonne net revenue. Resources are inclusive of reserves

Table 4.2.4: PAH Historic Resource Estimate for the Nkamouna Deposit, 2008*

Lithology	Resource Category	Cut-off (%Co)	Tonnes (000's)	Average Grade		
				Co (%)	Ni (%)	Mn (%)
Upper Laterite	Measured	0.12	42	0.301	0.318	1.569
Upper Breccia	Measured	0.23	229	0.468	0.49	2.19
Ferricrete Breccia	Measured	0.23	1,447	0.527	0.55	2.689
Lower Breccia	Measured	0.23	2,905	0.448	0.545	2.228
Ferralite	Measured	0.12	26,839	0.226	0.689	1.178
Total	**Measured**		**31,462**	**0.263**	**0.667**	**1.352**
Upper Laterite	Indicated	0.12	44	0.272	0.291	1.371
Upper Breccia	Indicated	0.23	157	0.326	0.401	1.812
Ferricrete Breccia	Indicated	0.23	604	0.461	0.474	2.242
Lower Breccia	Indicated	0.23	1,588	0.426	0.48	2.059
Ferralite	Indicated	0.12	27,475	0.207	0.673	1.087
Total	**Indicated**		**29,869**	**0.224**	**0.657**	**1.166**
Total	**M+I**		**61,331**	**0.244**	**0.662**	**1.262**
Upper Laterite	Inferred	0.12	67	0.158	0.207	1.091
Upper Breccia	Inferred	0.23	4	0.286	0.426	1.817
Ferricrete Breccia	Inferred	0.23	10	0.459	0.497	2.486
Lower Breccia	Inferred	0.23	215	0.393	0.445	1.423
Ferralite	Inferred	0.12	17,117	0.177	0.556	1.057
Total	**Inferred**		**17,412**	**0.18**	**0.553**	**1.063**

* (Pincock Allen and Holt, 2008). Resources stated above a 0.12% Co cutoff grade for laterite and ferralite, material, and above a 0.23% Co cutoff grade for all breccias material. Resources are inclusive of reserves.

Table 4.2.5: Historic Reserve Estimate for the Nkamouna Deposit, PAH, 2008*

Classification	Mineralized Zone Tonnes	Average Grade			Interburden Tonnes	Overburden Tonnes	Total Tonnes
		Co (%)	Ni (%)	Mn (%)			
Proven	28,867,610	0.264	0.690	1.406			
Probable	25,874,014	0.230	0.683	1.250			
Total	**54,741,624**	**0.248**	**0.688**	**1.331**	**4,326,540**	**98,231,134**	**157,299,298**

* (Pincock Allen and Holt, 2008); Resources are inclusive of reserves.

5 Geological Setting (Item 9)

The following section is excerpted from the 2008 PAH Report. Changes to standardizations have been made to suit the format of this report. Edits to text are annotated by the use of square brackets or reference to SRK.

5.1 Regional Geology

Southeastern Cameroon lies within a region of metamorphosed Proterozoic rocks ranging in age from 2,500 to 600 My and extending across parts of several west-central African countries. In southeastern Cameroon, several assemblages of such metamorphic rocks have been mapped and named (Cameroon Direction des Mines et de la Geologie, undated). Due to the metamorphosed nature of the rocks and poor exposures, there is some uncertainty in distinguishing and dating various lithologic units.

The Nkamouna/Mada project area is primarily underlain by rocks of the Intermediate Series, which includes the Mbalmayo-Bengbis Series. These rocks are principally chloritic and sericitic schists and quartzites. Also included in the Intermediate Series are extensive metamorphosed felsic, mafic volcanic and volcaniclastic rocks. These rocks are post-Eburnean (i.e. younger than 1,800 My) and are cut by basic dikes. The original depositional age of the sediments was probably 1,800 to 1,400 My, with metamorphism to almandine-amphibolite facies occurring about 1,200 My ago, likely coincident with the Kibaran Orogeny.

The schists and quartzites contain inliers of ultramafic rock, which were probably emplaced long after deposition of the original sedimentary rocks. Due to poor exposures, the contact relations are unclear, but the ultramafic bodies appear to be emplaced along north-trending regional fractures, which apparently allowed emplacement of ultramafic rocks of a deep-seated origin (Figure 5-1).

5.1.1 Regional Metallogeny

The region within a 300 km radius of the GeoCam Project Area in Cameroon, Gabon, Congo, and Central African Republic has few producing mineral deposits and few with near-term production potential. Most of this region of west-central Africa is underlain by Proterozoic granite-gneiss-schist terrains, broadly similar to the rocks in the Project Area. Within the region, ultramafic rocks, the original source of the cobalt and nickel, are confined to the project area. There has been no previous production of minerals from the project area.

Alluvial gold is exploited on a small scale from stream gravels in various parts of Cameroon, Gabon, Congo, and Central African Republic. Few statistics are available because all production in the region is from artisanal sources. However, the U.S. Geological Survey's 2002 estimate for total gold production from all four countries combined is less than 1,600kg, or less than 50,000oz/y. In the southwest part of the Central African Republic, alluvial gold is accompanied by small quantities of alluvial diamonds in streams which drain Cretaceous sandstone and conglomerates exposed further east. The Cretaceous formations do not extend into Cameroon.

Small amounts of alluvial tin and rutile are extracted from streams in the region, also in quantities that are locally important to village economies but are not industrially significant. Artisanal production of sapphire is also locally important.

Deposits of iron ore are reported to exist in south-central Cameroon, north of the Gabon border, but little information is available about these deposits. The UN development program also evaluated several iron ore and limestone deposits. At Belinga in northeast Gabon, a stratiform iron deposit contains several hundred million tonnes of 64% Fe, but with high phosphorus content (+0.1% P). This deposit has not been exploited on a commercial scale.

Limestone deposits occur in the Proterozoic rocks, about 50 km southeast of Lomie. These deposits were drilled by the UNDP in 1981, but they have not been exploited on a large scale. Building stone, quartzose river sand, clays, flagstone, and pozzolana (volcanic ash) are produced at by small scale artisanal miners for local use in Cameroon. With the above exceptions, the mining of non-fuel minerals in Cameroon is in its infancy. Occurrences or resources of bauxite occur in northern and western Cameroon, but an aluminum smelter near Douala processes only imported alumina. There is little in the way of a mining culture or infrastructure in the country at present.

5.2 Local Geology

The Cameroon laterite profiles are similar to those observed elsewhere in humid tropical environments and show a strong vertical zonation, which reflects the transition from unweathered host rock at the base, to highly-leached residues at the surface. The Cameroon laterites depart from the norm somewhat, in possessing two layers of iron-rich laterite separated by an iron-rich ferricrete breccia. The laterite under the breccia includes the limonitic ferralite and underlying saprolite zones, which are more typical of humid tropical laterite profiles.

5.2.1 Local Lithology

The typical sequence of discernable horizons in the weathering profile at Nkamouna and Mada is described below in Table 5.2.1.1 (modified from Geovic report data) and illustrated in Figures 5-2 and 5-3. The terminology and abbreviations of these units have varied somewhat since 1995, with the currently used terminology being shown in the first column. Further descriptive details about these units as adopted by SRK are provided in the paragraphs below, modified slightly from the original Geovic descriptions.

Table 5.2.1.1: Laterite Stratigraphy

Horizon	Alternate/Former Names	Lithology	Dominant Mineralogy	Approx % Fe	Co Content	Ni Content	Comments
Organic Soil	- -	Mainly organic	- -	low	near zero	near zero	usually < 15 cm, not logged
Upper Laterite (UL)	Upper Clay, Upper Limonite (UL), Granular Zone (GL)	Red, fine-grained, pulverulent to pisolitic	limonite, hematite, maghemite	40-50	low	low	partly magnetic
Ferricrete Breccia (UB, LB,HP)	Upper Ferricrete Breccia (UB), Lower Ferricrete Breccia (LB), Hardpan (HP), Breccia (FB)	Iron oxides; indurated, brecciated, vuggy	limonite, hematite, maghemite, some asbolane	40-50	high near base	low	may contain schist fragments & gibbsite
Ferralite (FL)	Lower Limonite (LL)	Brown, fine-grained,	clays, limonite, hematite, some asbolane	20-35	high near top	high near base	may contain schist fragments & gibbsite
Silcrete (SI)	Transition Zone	Hard, platy silica	quartz, chalcedony, limonite,	10-20?	low	low	usually absent, occurs in LL or SP; old water table?
Saprolite (SP)	Saprolite Clay	Weathered serpentinite, clays, silica	clays, serpentine, silica	10-20	low	high	
Serpentinite Bedrock	Serpentine	Sheared, soft rock	serpentine, silica, talc?	5-10	very low	very low	schist, if present

Upper Laterite (UL). A purplish-red, highly magnetic, powdery clay-like soil. Ubiquitous, normally 4 to 8 m thick, ranging up to 19 m, except where removed by erosion at the borders of the laterite plateaus. This unit will be easy to excavate for completing test shafts and for mining.

Ferricrete Breccia. Beneath the Upper Laterite is a nearly ubiquitous horizon of ferruginous concretions, ranging in size from pisolites 1 to 2 cm across, to blocks larger than a meter across. Large blocks have complex structures, characterized by multiple stages of brecciation, with vesicular, tubular structures, and amoeboid shaped cavities. They are composed of agglutinated pisolites and angular ferricrete fragments, with some limonitic matrix. Ferricrete fragments are typically dark red outside and varicolored on fresh surfaces. Where the blocks were large enough to impede deepening of the test pits, the ferricrete breccia was formerly referred to as "Hardpan" (logging unit HP). The ferricrete breccia averages 6 to 8 m thick, and was often divided into two or three units by project geologists.

The Upper Ferricrete Breccia (UB) is typically pisolitic and relatively low in Co and Ni except locally where stained with black manganese (Mn) oxides.

Hardpan (HP) is the most highly-cemented ferricrete breccia and is very difficult to penetrate with hand tools. It forms outcrops in some areas, particularly at the borders of the lateritic plateaus, and averages 2 m thick. Where present, it grades upward and downward into UB and LB, respectively.

The Lower Ferricrete Breccia (LB) consists of reddish concretions, with abundant black Mn oxides, texturally similar to UB, with a matrix of Ferralite (FL). It is typically 1 to 2 m thick,

and >2% Co may occur at the base, especially where concretion-like aggregates of asbolane occur. It is hard to dig with hand tools.

Ferralite (FL). Ferralite is a limonitic laterite, sometimes pulverulent, mottled, with varied shades of black, yellow, brown, red. It is often foliated, reflecting relict serpentinite textures. Local maghemite occurs near the top. Thickness varies from a few meters to tens of meters, averaging near 8 m. It is consistently mineralized with good metal grades near the top where black Mn zones occur, moderate to low Co grades lower in the unit. The magnesium oxide (MgO) content is very low, averaging about 0.5%, part of which is present as non-reactive MgO in spinel (i.e., magnesian chromite). This unit is easy to moderately easy to excavate for completing test pits and for mining.

Silcrete (SI). This highly-discontinuous unit may lie at the boundary between the Ferralite and the upper Saprolite. It is composed of subhorizontal plates of white to grey silica, intercalated with varicolored clays. Usually 0.5 m thick or less, and often absent. Commonly has low metal contents and is very hard to dig. It is generally interpreted to mark a former water table, and often occurs just above the current water table.

Saprolite zone (SP). This zone is composed of green, sticky clay with less than 50% fragments of partly weathered serpentinite, grading downward into foliated, fractured serpentinite. It may have silica-filled steep fractures. It averages 1.5 m thick, is relatively poor in Co, but can be rich in Ni, and is easy to moderate to dig. Saprolite typically contains less than 40% Fe and elevated MgO (15-30%).

Serpentinite (SE). The serpentinite bedrock is olive green to dark green and may be fractured and fissile, with silica-filled fractures. It has uniformly low metals grades except in rare cases where garnierite-like nickelferous silicates fill fractures. It is relatively hard and rarely encountered in test pits, but sometimes in drillholes. Magnesium grades are typically greater than 35% and iron contents are usually less than 10%.

5.2.2 Alteration

The Nkamouna and Mada deposits have been formed by intense tropical weathering, resulting in deep supergene lateritic alteration of ultramafic rocks. The dominant alteration mineralogies are clays, limonite, and hematite, with oxides of manganese commonly observed. The laterite section is underlain by sheared and altered serpentinite, characterized by the presence of serpentine, silica and talc.

The serpentinitic ultramafic rocks that are the source the Cameroon mineralization originally contained the mineral olivine. The olivine in serpentinites of lower-crust origin typically contain 0.3 to 0.4% nickel and near 0.01% cobalt, in partial substitution for magnesium in the olivine solid solution series $(Mg,Fe,Ni)_2SiO_4$. Upon serpentinization and weathering of the olivine-bearing rock, cobalt and nickel are liberated as the olivine is destroyed, but are usually incorporated into other minerals formed at the same time, such as asbolane, nontronite clays, garnierite, and others. Other factors tending to localize mineralization include the permeability of the host rock, foliation and fracturing of the basement rocks, and the water table(s).

The other major constituents of serpentinites, MgO, SiO_2, and FeO, are also liberated by weathering. The FeO is oxidized to Fe_2O_3 which is highly insoluble, and remains as an iron-rich surficial laterite soil. The MgO and SiO_2 are usually mobilized several meters in the percolating

water, and are redeposited at varying distances as clays, silica, and other minerals. Their specific behavior depends on the chemistry and pH of the interacting water and rock.

Resistant minerals such as chromite and micas if present (e.g., phlogopite and chlorite) tend to remain intact in the weathering profile, and to some degree are concentrated in the weathered residuum, as other constituents are selectively removed. Mafic rocks not containing appreciable amounts of olivine, such as gabbro and pyroxenite, do not normally contain much nickel and cobalt, and thus do not give rise to significant nickel-cobalt enrichment during weathering.

The cobalt-nickel-bearing weathering profile has been partially preserved due to the resistant ferricrete breccia cap and low topographic relief. The surface relief within the Nkamouna and Mada areas is generally less than 100 m except at the outer margins, where active erosion is occurring and surface elevations locally fall below 700 m elevation. Resistant quartzite ridges are the exception, and may exceed 800 m in elevation.

5.2.3 Structure

The bedrock geology at Nkamouna and Mada has been mapped by GeoCam geologists through a combination of natural exposures, soil mapping, and, most importantly, observation of weathered or fresh rock encountered in pits and drillholes. Mapping of detailed structures, attitudes of foliation or fractures, etc. is generally not practical except in the deeper pits. Rock from pits, drillholes, and rare exposures indicate that the fresh underlying rock at Nkamouna is a pervasively-sheared serpentinite. Most serpentinites form from parental ultramafic rocks, as a result of hydration and shearing at moderate temperatures, either during emplacement of the ultramafic or during post-emplacement tectonism. At Nkamouna and Mada, petrographic evidence suggests that the parent rock to the serpentinite was probably a dunite (rock containing >90% olivine). Chrysotile asbestos, an accessory mineral in many serpentinites, has been reported from only one test pit in the Mada area, 4 km north of Nkamouna.

At Nkamouna, it is apparent that schist, meta-volcanics, and related rocks occupy the borders of the serpentinite and also occur as tectonic slivers within the serpentinites. Locally, lateritic soils with abundant schist fragments rarely overlie serpentinite bedrock. These anomalies may be due to the gravity-induced movement of soils down-slope.

5.3 Project Geology

5.3.1 Deposit Geology

The cobalt-nickel deposits are hosted in residual laterites which have formed by prolonged tropical weathering of serpentinites. Large areas of mineralized laterite, each several square kilometers in extent, have been preserved on low-relief mesas or plateaus underlain by ultramafic rocks that stand above the surrounding dissected lowlands. Nkamouna and Mada are two such plateaus. Most of the plateaus are underlain by ultramafic rocks, with some areas of schist, phyllite, and quartzite. The surrounding lowlands are underlain by schists, phyllites, quartzites, and meta-volcanics of the Intermediate Series.

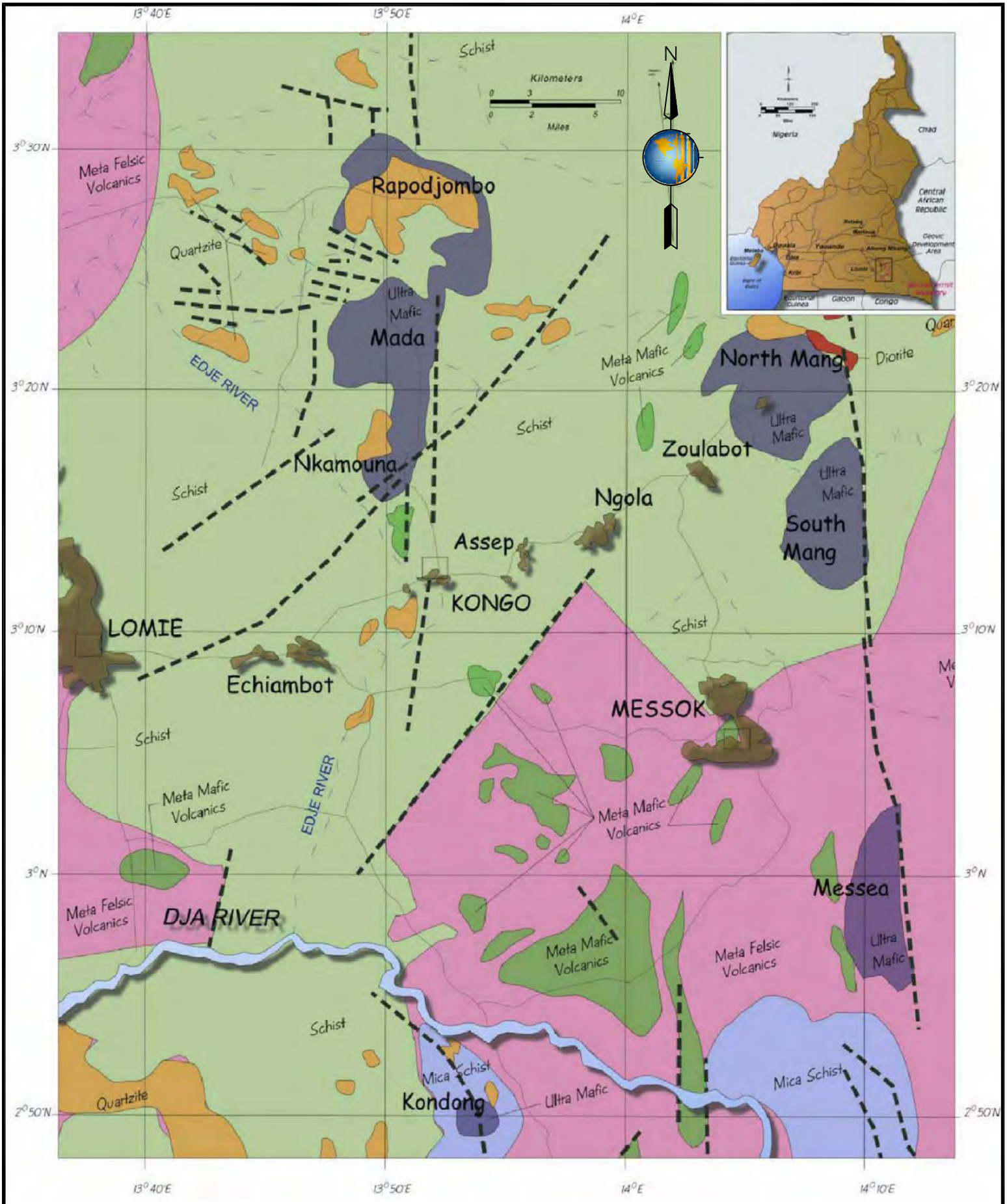

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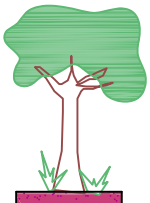
Figure 5-1
Regional Geology

Scale:	.
Date:	18 Nov. 2009
Dwg:	5-1 Reg Geo



Depth		Lithologies
GR 4m	UL	**Upper Laterite:** Dusky purplish-red, powdery clay texture, highly magnetic. it is typically barren.
UB 2m	UB	**Upper Ferricrete Breccia:** Composed of dusky red sub-spherical ferruginous concretions with cores of yellow, brown, and black mottling. Contains blocks of harder ferricrete breccia in a matrix of UL, especially toward bottom, where it may constitute "Hardpan" is usually barren of Co and Ni.
LB 2m	LB	**Lower Ferricrete Breccia:** Reddish, with shade of black MnO. Concretionary, gradational with the overlaying breccias, may be very hard ("Hardpan" toward top). Has a matrix composed of Upper Laterite. Generally mineralized with Co, Ni, Mn.
FL 15m	LL	**Lower Limonite / Ferralite:** Underlies the ferricrete with sharp contact. Mottled, with varied shades of black, yellow, brown, and red. The texture is foliated. Manganese-rich and ferrous iron-rich zones are dark gray color. Upper part often mineralized with Co, Ni, Mi, with higher Ni content near base.
SI 0.5m	SI	**Silcrete:** Made up of platy secondary opaline and crystalline silica, which are intercalated with clay (green, black, yellow or red). The presence of this zone is often indicative of a nearby water table.
SP 3m	SP	**Saprolite:** Green and brownish, sticky clay, with fragments of partly weathered serpentinite. Grades into fresh serpentinite below.
SE >300m	SE	**Serpentinite:** Olive green to dark green, very hard, often foliated. it is the primary ultramafic source of cobalt and nickel.

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Figure 5-2
Local Lithologies

Scale:	NTS
Date:	02 Nov. 2009
Dwg:	5-2 Local Lith

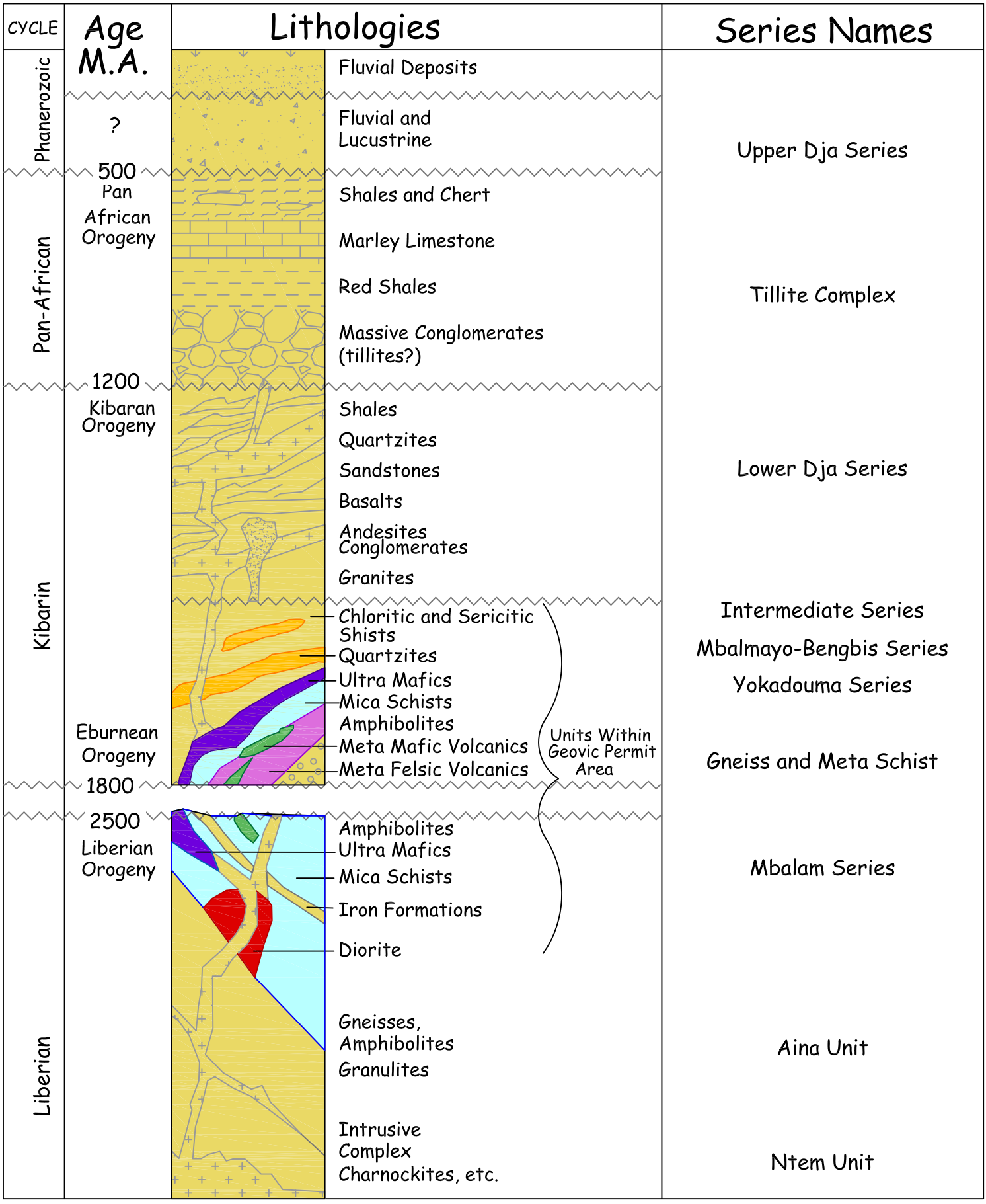

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Figure 5-3
Regional Stratigraphy
Modified from
Cameroon Direction des Mines et de la Geologie

Scale: NTS
Date: 02 Nov. 2009
Dwg: 5-3 Regional

6 Deposit Type (Item 10)

The following section is excerpted from the 2008 PAH Report. Changes to standardizations have been made to suit the format of this report. Edits to text are annotated by the use of square brackets or reference to SRK.

6.1 Laterite Deposits

The Nkamouna and Mada laterite deposits were formed by prolonged tropical weathering of minerals present in serpentinitic rocks. Laterization is largely a chemical process whereby ground water and biological processes interact with surficially exposed serpentinites, resulting in the concentration of certain elements in the soil profile (e.g., Fe, Al, Co, Ni, Cr, Mn) while dissolving and removing other elements (e.g., Mg, Ca, Si). The intensity of weathering is a function of time, climate, and bedrock characteristics (composition, degree of fracturing, etc.). Sulfide minerals are typically not common in serpentinites, and did not play a significant role, if any, in the formation of the enriched cobalt-nickel profiles. This is in stark contrast to many other cobalt and nickel deposits such as in the African Copper Belt (Zaire and Zambia), Sudbury, Thompson and Raglan (Canada), Norilsk (Russia), Bou Azzer (Morocco), and others, which were formed by magmatic and/or hydrothermal processes where the presence of sulfide minerals is of key importance.

Iron is less soluble in lateritic environments, and is enriched in the zone above the saprolite. The degree of iron enrichment and leaching of silica and magnesia results in the eventual formation of limonite layers rich in goethite that eventually collapse under their own weight to form the capping ferricrete breccias and overlying granular laterite zones. Only in the saprolite and transition zones is iron largely in a reduced state. Clay minerals may form from the weathered serpentinites and adjacent schists at Nkamouna and Mada. Secondary silica and nickel minerals may occur within fractured portions of the serpentinite (e.g., opal and garnierite).

Nkamouna and Mada are unusual laterite deposits with respect to the abundance and coarseness of asbolane; the low Ni:Co ratio, the low MgO content of the Lower Limonite, the abundance of kaolinite instead of smectite, and the presence of a well-developed ferricrete breccia horizon sandwiched between the Upper Laterite and Lower Limonite.

These features are consistent with long-lived and episodic formation of the Nkamouna and Mada profiles. Weathering of the ultramafic rocks in Cameroon most likely occurred during Cenozoic time, although precise dating is not possible due to the absence of Cenozoic rocks in the region other than very young alluvium. In any case, while the age of laterization is not well-constrained, the laterite characteristics at Nkamouna and Mada suggest a long period of laterization. Encroachment and incision of the Zaire (Congo) River tributaries (e.g., Sangha, Dja and Boumba Rivers) may have lowered the erosional base levels, as likely reflected in the episodic formation and destruction of the ferricrete and other laterite units.

Though unusual by having several distinctly different features, the Cameroon laterites share similarities with other nickel-cobalt laterites found around the world (e.g., Western Australia, New Caledonia, Indonesia, Philippines, and Cuba). The Cameroon deposits are unusual in their low magnesium content, high cobalt to nickel ratio, coarsely aggregated asbolane mineralization, abundance of maghemite, and widespread occurrence of ferricrete breccias. Also significant is the concentration of most of the cobalt mineralization in the lower ferricrete breccia and upper

portion of the ferralite zone. In other laterite deposits, cobalt is usually concentrated in the lower-most portion of the ferralite and upper saprolite zones.

6.2 Alluvial Deposits

Neither Geovic nor SRK are aware of any concentrations of valuable metals in streams draining the ultramafic massifs. While some fine-grained magnetite and black MnO concretions were observed in some streams draining the massif, along with chromite, these occurrences are not known to be present in economic concentrations or tonnages. There is no known panning of gold or platinum-group metals in the region.

7 Mineralization (Item 11)

The following section is excerpted from the 2008 PAH Report. Changes to standardizations have been made to suit the format of this report. Edits to text are annotated by the use of square brackets or reference to SRK.

7.1 Laterites

Cobalt-nickel mineralization in the Nkamouna and Mada deposits occurs within the weathering profile overlying sheared and serpentinized ultramafic rocks of probable dunite parentage. The lateritic weathering profile averages about 20 m thick, but in instances extends to 40 m in thickness adjacent to ridge crests or within fracture zones. The mineralized laterite forms lenses which are generally greater than 10 m thick. The lenses typically lie roughly parallel to the rolling topography of the Nkamouna plateau. They are relatively flat on top with irregular, where weathering has penetrated downward into fractures and shear zones within the underlying serpentinite.

Most of the economic mineralization occurs at a single level, with an average of about 1 m of ferricrete breccia underlain by approximately 4 m of ferralite. The ore types are characterized geologically by their mineral content, bulk composition, and texture, as described below. The deposits have an unusually large concentration of the coarsely aggregated ore mineral asbolane, thick ferricrete breccia, and abundant maghemite.

Water Table. The depth to the water table was recorded in numerous GeoCam reverse-circulation drillholes. In all but seven of these holes, the water table was between 12 and 25 m below surface, and was usually within the Ferralite or at the upper limit of Serpentinite. The top of the water table varies seasonally on an average of 4 to 5 m.

The Upper Laterite, Ferricrete Breccia, and Ferralite units contain fragments of schist, or minerals derived from schist, where the laterite is developed over schist inliers within and lateral to serpentinite bedrock. The Saprolite unit contains weathered schist fragments when schist forms the local bedrock, making it easy to distinguish schist-derived laterite from ultramafic-sourced laterite.

7.1.1 Laterite Mineralogy

The minerals of economic interest in the Nkamouna and Mada laterites are shown on Table 7.1.1.1, which incorporates data from various Geovic reports and the PMET report (2002). Of the minerals listed in Table 7.1.1.1, most occur in the majority of nickel-cobalt laterites worldwide, in proportions which vary widely from one laterite horizon to another, and from one deposit to another. In general, these minerals occur at Nkamouna and Mada as fine-grained clay-like or concretionary masses, and are only occasionally identifiable as discretely visible mineral specimens. One exception is gibbsite, which may occur as mammilary masses or vug-fillings of radiating transparent to milky white crystals several millimeters long. Of great significance is the size of the asbolane agglomerates and wad that host the cobalt and almost all of the manganese.

Table 7.1.1.1: Selected Minerals in Laterite Profile

Name	Mineral Type	Formula	Origin	Typical %Co	Abundance in Co-rich Material	Maximum Occurrence
asbolane ("wad")	hydrated oxide-hydroxide	$(Co,Ni)_{1-y}\ (MnO_2)_{2-x}(OH)_{2-2y+2x}.nH_2O$	weathering	10-15	6%	LB, LL
ernienickelite	hydrated oxide	$NiMn_3O_7 \cdot 3(H2O)$	weathering	low Co, high Ni	reported*, not confirmed	?
limonite	oxide-hydroxide	fine-grained mixtures of goethite FeO.OH, lepidocrocite FeO.OH, hematite Fe_2O_3; others	weathering	0.0X-0.3	60-70%	UL, LL
goethite		see "limonite", above			see limonite	see limonite
hematite	oxide	Fe_2O_3	weathering	0.00X	3% incl maghemite	UL, UB, LB, LL
maghemite	oxide	gamma $-Fe_2O_3$ (ferromagnetic)	weathering	0.00X	incl in hematite	UL, UB, LB, LL
carolite, kerolite, garnierite	silicates, mineraloids	approx formulas $(Mg,Ni)SiO_3.nH_2O$	weathering (rare)	0.0X Co, high Ni	reported*, not confirmed	saprolite
nontronite	silicate clay (smectite gp.)	$Na_{0.33}(Fe,Ni)_2\ (Si,Al)_4O_{10}(OH)_2.nH_2O$	weathering (rare)	0.X, 0.X to X Ni	reported*, not confirmed	saprolite
montmorillonite	silicate clay (smectite gp.)	$Na_{0.33}(Al,Mg)_2\ Si_4O_{10}(OH)_2.nH_2O$	weathering (rare)	low	reported, not confirmed	saprolite
kaolinite	silicate clay	$Al_2Si_2O_5(OH)_4$,	weathering of adjacent schists	0	9% in bulk sample	all
gibbsite	hydroxide	$Al(OH)_3$	weathering of adjacent schists	0	1%	UL, UB, LB, LL
silica, quartz	silicate	SiO_2	weathering, remobilization	0	5%	saprolite, serpentinite
serpentine	silicate	$(Mg,Fe,Ni)_3Si_2O_5(OH)_4$	late magmatic	0.01 Co, 0.3 Ni	<1%	saprolite, serpentinite
talc	silicate	$Mg_3Si_4O_{10}(OH)_2$		0	<1%	saprolite, serpentinite
magnetite	oxide	Fe_3O_4 (ferromagnetic)	magmatic	0.00X	8% (incl. chromite)	all, max in UL
chromite	oxide (spinel)	$FeCr_2O_4$	magmatic	0	See magnetite	all, max in UL
olivine	silicate	$(Mg,Fe,Ni)_2SiO_4$	magmatic	0.01 Co, 0.3 Ni	<<1%	rare in serpentinite

Asbolane

Absolane is the key mineral in the Nkamouna and Mada deposits, which hosts the cobalt, most of the manganese, and a significant part of the nickel. Between one-third and one-half of the deposit's nickel is hosted in asbolane (see PMET Mineralogical Report, 2002). This mineral is sometimes referred to as "asbolan" or "asbolite" in the scientific literature, or "wad" or "cobalt wad" as field terms. Asbolane is widespread in nickelferous laterites, but elsewhere is usually present in very small amounts and is normally inconspicuous as black blebs on fractures. Individual asbolane crystals have hexagonal symmetry, a Mohs hardness of 6, and are very dark in color. Typically, individual crystals are rarely visible to the naked eye or a hand lens; rather the mineral forms blackish patches or crusts on fractures and cavities. The asbolane occurrence at Nkamouna is unusual in that it occurs as both discrete platy crystals and in larger and coarser crystal aggregates and fine-grained wad up to 5 cm in diameter, sometimes as concretion-like nodules with chromite and goethite. It also occurs as a fine intergrowth with Cr and Fe oxides and hydroxides (PMET Report, November 2002).

There is no theoretical fixed Co or Ni content for asbolane because the formula of $(Co,Ni)_{1-y}MnO_2)_{2-x}(OH)_{2-2y+2x}.nH_2O$ allows for substitution among Co, Ni, and Mn to reflect the chemistry of formation waters, and to achieve charge balance. The Co:Ni ratio in asbolane at Nkamouna ranges from 1:1 to 10:1 (expressed as stoichiometric oxides) and averages about 2:1 (PMET, 2002). "Asbolite" from Soroako, Indonesia (Evans, et al., page 46) shows a range of compositions ranging from 2.5% Co to 5.8% Co, and averaging (in the limonite zone) Co:Ni 3.3:1. The Soroako and PMET analyses also report large amounts of Fe, so it is possible that the samples were not pure asbolane. Analyses of asbolane samples from Nkamouna range from 6.3 to 19.5% CoO. The asbolane component of the Nkamouna bulk mineralogy modal analysis expressed as a weight percent ranges from 3.2 to 7.2% (see PMET Report, 2002).

Asbolane is critical to the project economics, because it occurs as coarser aggregates of microscopic crystals, the aggregates being separable by crushing and wet screening from the pulverulent iron-oxide minerals and clays. The resulting coarse fraction contains most of the Co and Mn, and a significant portion of the Ni in the raw material, resulting in a significantly upgraded concentrate (3.1x for ferralite unit) prior to leaching.

Clay Minerals

Kaolinite is the most abundant clay mineral at Nkamouna (Table 7.1.1.1). The PMET report (2002) presents bulk samples of potentially processable material contain 9% kaolin on average. Although kaolinite normally occurs throughout the weathering profile (Gideon, 2005) the abundance of kaolinite is not typical of nickel-cobalt laterites, which normally give rise to magnesia-rich, alumina-poor smectites in at least small quantities. Smectite clays were not reported by PMET or by Gideon. Kaolinite, by contrast, does not usually form by weathering of ultramafic rocks, which have very low alumina contents (see Gleeson, et al., 2003, and articles in Evans, et al., eds., 1979). It is possible that the kaolinite at Nkamouna arose from weathering of the adjacent Proterozoic schists, and was laterally transported so as to become admixed with typical ultramafic-laterite minerals. The kaolinite does not contain cobalt and does not appear likely to influence mining or metallurgical processes.

Other Minerals

Serpentinites and other ultramafic rocks typically have very low aluminum content. At Nkamouna and Mada the weathering of aluminum-bearing minerals in the schists - principally micas and feldspars - gave rise to mobile aluminum ions which migrated laterally and vertically and precipitated to form gibbsite, $Al(OH)_3$. Gibbsite is locally abundant at Nkamouna but also occurs in many ultramafic-derived laterites worldwide. Gibbsite composes on average <1% of the bulk mineralogy at Nkamouna (PMET, 2002).

Comparison with Other Deposits

While Nkamouna and Mada are clearly laterite deposits formed by in-situ weathering of ultramafic rock, several features distinguish Nkamouna and Mada from most of the known wet-tropical nickel laterites.

From the earliest exploration stages it was recognized that Nkamouna and Mada displayed an unusual laterite stratigraphy, mainly with regard to the presence of abundant ferricrete breccia within the laterite sequence. In most laterites worldwide, ferricrete, if present, occurs at the surface or beneath a surficial ferricrete breccia ("canga"). At Nkamouna and Mada, the ferricrete is well-developed, and is sandwiched between the Upper Laterite and the Ferralite. It is not clear whether the Upper Laterite has formed by dissolution of the Ferricrete, or whether the Ferricrete has formed by continued weathering of the overlying Upper Laterite.

The data in Table 7.1.1.2 indicate that Nkamouna and Mada are highly atypical in terms of their low Ni:Co ratio. Nkamouna and Mada are valued mainly for its cobalt and only the high-cobalt portions of the weathering profile are included in the defined resources. Due to the high Co:Ni ratio in asbolane and the unusually large size of the asbolane wad accretions that occur at Nkamouna and Mada, a significant weight percent of the bulk mineralogy is made up by the asbolane component. At Nkamouna, the Ni:Co ratio averages 3.0 and ranges from 2.0-2.4 at Mada, which is far lower than that of any of the other deposits worldwide. It should be noted, however, that the other deposits are valued for nickel, with or without by-product cobalt, and the resource figures therefore refer to the high-nickel portion of the weathering profile. Because the highest cobalt values and the highest nickel values are often in a separate level of the profile, the Ni:Co ratios are not strictly comparable. For this reason, a Nkamouna Ni:Co ratio was also calculated for some of the Ferralite intervals having at least 0.5% Ni. This ratio (6.3 Ni:Co) is lower than that of any of the cited deposits. Since the upper portion of the Ferralite zone contains nearly 90% of the Nkamouna cobalt resource, it is intuitive that the lower part of the Ferralite must have a somewhat higher ratio than 6.3:1. A separate calculation for 32 intervals at Nkamouna logged as FL/SP (Ferralite-Saprolite transition) yielded a Ni:Co ratio of 8.1:1. This transition carries the highest Ni values in many nickel-laterite deposits. This also indicates that Nkamouna is systematically enriched in Co, compared to other laterite deposits.

Nkamouna is also atypical in the very low MgO content in the Ferralite, averaging approximately 0.5%. Even the highly-leached limonites mined at Moa Bay, Cuba, carried over 1.5% MgO in the early years of that project, where nickel is recovered by high pressure leaching with sulfuric-acid.

As mentioned in the above sections, the Nkamouna and Mada deposits have abundant kaolinite, a mineral not normally indigenous to nickel-cobalt laterites. The clay-mineral issues need further study.

Table 7.1.1.2: Lateritic Nickel-Cobalt Deposits Worldwide

Deposit	Country	Status	Bedrock	Geol. Type +	NI/CO Ratio, Mineral	Process	Metals Recov.	Comments
Nkamouna high-cobalt horizon	***Cameroon***	***Feasibility***	***sheared serpentinite***	***oxide + silicate?***	***3.0***	***sulfurous acid leach***	***Co+Ni (+Mn)***	***subject of this report***
Nkamouna high-nickel horizon (LL)	Cameroon	Feasibility	sheared serpentinite	oxide + silicate?	est. 6 to 8	--	none planned	possible later development
Mada	Cameroon	exploration	serpentinite	oxide+= silicate	*2.0-2.4	sulfurous acid leach proposed	Co+Ni (+Mn) planned	this report
EXMIBAL	Guatemala	former producer	peridotite, serpentinized	oxide + silicate	25	smelting to Ni-S matte	Ni	project idle
Moa Bay	Cuba	producing	peridotite, serpentinized	oxide	10	acid leach	Ni, Co in sulfide	in production
Nicaro	Cuba	producing	sheared serpentinite	oxide + silicate	15	ammonia leach	Ni in oxide	in production
Musongati	Burundi	undevel.	peridotite, serpentinized	oxide + silicate	30	--	Ni, Co	pre-feasibility stage
Nickel Mountain	Oregon, USA	former producer	peridotite, not serpentinized.	silicate	80	smelting to ferronickel	Ni + Fe*	fossil deposit, mined out
Soroako	Indonesia	producing	peridotite, variably serpentinized	silicate	20?	smelting to Ni-S matte	Ni	in production
Bonao	Dominican Rep.	producing	peridotite, sheared serpentinite	silicate	35	smelting to ferronickel	Ni + Fe*	in production
Greenvale	Qld, Australia	former producer	peridotite, serpentinized	clay?	15	ammonia leach	Ni, Co in sulfide	mined out
Murrin-Murrin	Western Australia	producing		clay	13	high-pressure acid leach	Ni, Co	in production
Goro	New Caledonia	construction	peridotite, serpentinized	oxide + silicate	12	high-pressure acid leach	Ni, Co	production scheduled for 2007

NOTE: Manganese is not recovered commercially at any of these operations.
[+] Classification of Gleeson, et al, 2003
* The Fe recovered in ferronickel is not a paying product
Table compiled from numerous sources, including Gleeson, et al. (2003), Evans, et al (1979), Boldt (1968), and others.

8 Exploration (Item 12)

The following section is excerpted from the 2008 PAH Report. Changes to standardizations have been made to suit the format of this report. Edits to text are annotated by the use of square brackets or reference to SRK.

Nickelferous laterite deposits in southeast Cameroon were first discovered and investigated by the UNDP during 1981-1986, in a cooperative project with the Cameroon Ministry of Mines, Water and Energy (UNDP Project CMR/81/005). Following a regional stream sediment geochemical survey which indicated the likely presence of laterite nickel mineralization, the UNDP project drilled eleven core holes in the Nkamouna area, which was the most accessible laterite area at that time.

Several of the UNDP holes intersected laterite and saprolite with interesting nickel and cobalt values. The first hole, KG-S-1, traversed 56 m of lateritic profile and fresh serpentinite, with Ni values up to 1.00% and Co values up to 0.19%. Due to the remote location and the low nickel prices at the time, the discovery did not draw much attention.

The UNDP holes were completed several years prior to Geovic's investigations. The drill apparatus, technical personnel, sampling procedures, and assaying practice were entirely different from those used subsequently by Geovic. Therefore, SRK is of the opinion that the UNDP drillhole data should be excluded for use resource estimation. These 11 holes represent less than 1% of the total sample openings at Nkamouna. In any case, most of the sites of UNDP holes were subsequently offset by gridded Geovic drillholes and pits, and the effective influence of the UNDP holes on resource tonnage and grade estimation is negligible.

In mid-1995, GeoCam received a Prospecting Permit that covered 19,600 km^2. In January 1999, the Prospecting Permit was superseded with an Exploration Permit, PDR 67, which covered 4,876 km^2 and specifically allowed exploration drilling. GeoCam's initial exploration program was based entirely on manually-dug test pits, and subsequently incorporated drilling and limited trenching. The program began at Nkamouna and was later extended to the other laterite plateaus including Mada, which were targeted using satellite images and air photos. Geologists from the Cameroon Ministry of Mines, Water and Energy participated in the work to provide government oversight as well as training. GeoCam's core-drilling program began in 1999, after many hundreds of pits had been completed. A total of 23 holes were drilled (NKM-21 to NKM-43) in the northeast part of West Nkamouna, on an approximate 100 m grid.

In 2002, GeoCam imported an Australian-designed, truck-mounted machine. Holes drilled with this machine are referred to in GeoCam reports as "air core" holes, but intact core was not produced, and these holes are more accurately termed reverse-circulation drill holes. Reverse-circulation holes were drilled between May 2002 and September 2003, when 176 holes (NKM 1010 to 1185, plus NKM-3.3) totaling 3,690 m were completed at Nkamouna. Most of these holes were drilled as infill holes on a series of EW lines which were sampled by pitting, generally at distances greater than 100 m between drillholes. Several of these were twins (within 5 m) of existing pits, and several others were later twinned by pits sunk on the drillhole collar. Twenty-two holes were drilled on a tight grid of approximately 15 x 15 m in West Nkamouna, to test the short-term variability between holes.

A Mining Convention was signed on July 31, 2002 by the Ministry of Mines, Water, and Power of the Republic of Cameroon that defined the general, legal, financial, tax, economic,

administrative, customs, social, land and environmental conditions under which GeoCam shall undertake the mining of cobalt, nickel, and their associated substances within GeoCam's Exploration Permit area. On April 11, 2003, Mining Permit No. 33 which replaced the Exploration Permit was issued by Presidential decree granting an exclusive right to GeoCam to exploit the deposits, and the total area was reduced to 1,250 km^2, which included approximately 337 km^2 of cobalt-nickel mineralized lands.

Geovic's participation in the Mining Permit holder GeoCam is a 60% direct corporate holding by Geovic, Ltd. In addition, another 0.5% is held by Geovic's President William Buckovic. The 39.5% balance is currently held by SNI, a Cameroon government investment corporation.

By 2004, GeoCam had largely completed the reconnaissance sampling and had undertaken pitting and drilling programs of varying densities at Nkamouna, where access was less restricted due to recent logging operations, in order to define deposit parameters for an eventual preliminary feasibility study.

In 2006, Geovic completed a program adding five new test pits and deepening other test pits adding over 730 m of additional sampling in preparation for the final feasibility study.

In 2002 GeoCam contracted with a local survey and civil engineering company in Yaounde (SCET) to provide digital topography for a 12 km^2 area mapped in detail at Nkamouna. Map survey points are accurate to within 1 cm (X, Y, and Z) and are contoured at 1 and 2 m intervals. All pits and drillholes are plotted on this topographic map base. EGIS who later purchased SCET still provides the bulk of surveying required by GeoCam and completed on the surveying in the Mada resource area in 2008 and 2009.

During 2008 and 2009, GeoCam conducted significant infill and step out drilling and pitting in both the Nkamouna and Mada areas, including an additional 975 drill holes at Nkamouna and 1,012 drill holes at Mada. These new data form the basis for the updated Mineral Resource estimates and subsequent Mineral Reserve estimate following the issue of this report.

The geological logging scheme utilized for past and current drill programs is consistent with the stratigraphic units as described in previously. The logging scheme has evolved over the history of GeoCam's work since 1995. All logging was carried out at the pit or drill site by degreed geologists, using standardized logging forms.

8.1 Interpretation

SRK has conducted a review of previous exploration programs and existing sample preparation methodology, and is of the opinion that the current data spacing at both the Nkamouna and Mada deposit areas is sufficient for declaration resources and reserves. The planning and execution of these most recent step-out and infill drilling and pitting programs was conducted in a professional manner, and SRK is of the opinion that the resulting data is adequate for use in resource estimation.

9 Drilling (Item 13)

The following section is excerpted from the 2008 PAH Report. Changes to standardizations have been made to suit the format of this report. Edits to text are annotated by the use of square brackets or reference to SRK.

Because the GeoCam deposits are secondary in nature and represent the decomposition products of bedrock, they present data compilation issues which are unique to laterite deposits. These include sampling of intermixed material, which range from very soft to very hard, and which varies greatly in metal grade from one location to the next, especially in the ferricrete breccia lithologies.

Geovic has described the methodology of collecting and handling samples in written detail, based on existing GeoCam procedures and on systematic analysis of various sampling and sample-processing methods.

The only drilling on any of the properties has been conducted at Nkamouna, Mada and Rapodjombo. The majority of all other samples have been obtained by developing hand-dug test pits or shafts of a nominal 1.2 m diameter to depth of refusal or the water table.

9.1 Sample Data

Most of the sampling at Nkamouna and Mada, and nearly all sampling in the other laterite areas, has been conducted by test pitting, with a lesser amount from drilling. Direct sampling of outcrops and trenches is almost entirely limited to Trench 1 at Nkamouna. A second trench was developed in early 2007. Trench 2 was designed to observe the deposit's continuity, reveal the challenges associated with grade control, and determine the mining characteristics for the expected waste and ore horizons.

The types of sample data as provided to SRK by Geovic are summarized below in Table 9.1.1, as of October 2010**.** Locations of the drill holes and sample openings (pits) for Nkamouna and Mada as provided to SRK by Geovic are shown in Figure 9-1 and 9-2.

Table 9.1.1: Exploration Sample Data

Area	Pits		Diamond Core Holes (UNDP + GeoCam)		Reverse Circulation ('Air Core') Holes		Trenches		Total	
Name	**No.**	**m**	**No.**	**m**	**No.**	**m**	**No.**	**m**	**No.**	**m**
Nkamouna* (all)	1,310	17,466.34	1034	27,215.05	176	3,674.50	2	n/a	2,522	48,355.89
Mada*	593	7,283.20	0	--	928	23,751.10	0	--	1,521	5,683.30
Rapodjombo	**	**	0	--	248	6,472	0	--	**	6,472**
North Mang	15	93.2	0	--	0	--	0	--	15	93.2
South Mang	34	312.7	0	--	0	--	0	--	34	312.7
Messea	25	208.4	0	--	0	--	0	--	25	208.4
Kondong	3	3	0	--	0	--	0	--	3	3
Total (Mining Permit)	**1,980**	**25,366.84**	**1,034**	**27,215.05**	**1,352**	**33,897.60**	**2**	**n/a**	**4,120**	**54,656.49**

Pit Data

A significant proportion of Geovic's assay sampling results (~46%) have been derived from hand-dug pits. Geovic has historically referred to the test pits as both "pits" or "shafts." In this

report, SRK uses the term "pits," which is more customary in laterite exploration, and avoids the impression that they are machine-dug openings of great depth (i.e., "shafts").

In the Nkamouna and Mada areas, GeoCam has excavated 1,898 pits, as shown on Table 9.1.1. The pit sampling program continued during drier weather in East Nkamouna, the area east of the Kongo-Ndu road, until September 2004. Five additional pits were dug in 2006, and an additional 302 pits were dug at Nkamouna (225) and at Mada (77) during the 2008-2009 field season. The density of pitting varies from about 50 x 50 m to 150 x 200 m, but is not uniformly gridded.

Trench Data

Two trenches have been excavated at Nkamouna. Excavated by hand, the first trench (trench 1) is up to 8.5 m deep, includes a 5 m north-to-south cross trench in the middle and extends 20 m east from the site of Pit 923. The trench is located on the western edge of the Nkamouna plateau, west of some surface exposures of ferricrete, at a location where the Upper Limonite appears to have been removed by erosion. Most of the trench exposes only Ferricrete, and does not reach adequate depth to expose the Ferralite or Saprolite. The trench site was selected by Geovic primarily to determine whether blasting would be necessary in the Ferricrete. The trench was thoroughly channel-sampled and assayed by Geovic, but these results were not used in the resource estimation due to change of support issues.

The second trench (trench 2) was excavated between January and May of 2007 in the SE part of Nkamouna near Pits 989, 1251, 1268 and 1269. This trench was excavated with bulldozers and was systematically deepened with hand dug pits. SRK visited this trench during the 2009 site visit, and assay results from this trench have been reviewed. Selected grab samples were also collected from this trench during the site visit, and were subsequently manually upgraded by site personnel as a demonstration of the physical upgrading process. These data from trench 2 have not been utilized in the resource estimation process for the same reason as above.

Drillhole Data

United Nations Drillholes. The first documented samples taken at Nkamouna were the eleven holes drilled by the UNDP in the mid 1980's. The UNDP used a J.K. Smit Model 300 diamond-drill rig. SRK has not examined the original drill core or logs from these initial 11 holes.

The UNDP holes were undertaken several years prior to Geovic's investigations. The drill apparatus, technical personnel, sampling procedures, and assaying practice were different from those used subsequently by Geovic, and the protocols utilized by UNDP with regard to quality assurance/quality control were not necessarily to industry accepted guidelines. Therefore, SRK agrees with both PAH and site geology personnel that these UNDP drillhole data should be excluded from the database utilized for resource estimation. These 11 holes represent less than 1% of the total sample database at Nkamouna. In any case, the sites of most of the UNDP holes were subsequently twinned/offset by gridded Geovic drillholes and test pits, and the effective influence of the UNDP holes on resource tonnage and grade estimation is considered by SRK to be negligible.

Geovic Core Drillholes. Subsequent to the UNDP holes, no further drilling was undertaken at Nkamouna until Geovic's core-drilling program in 1999, after many hundreds of test pits had been completed and an exploration permit was obtained over the mineralized areas. The first rig utilized was a trailer-mounted 20 hp core drill which could be manually maneuvered along forest trails to minimize environmental impacts in prospective areas. A total of 23 holes were drilled

(NKM-21 to NKM-43) in the northeast part of West Nkamouna, on an approximate 100 m spaced grid. The maximum depth drilled was 33 m, with an average hole depth of 26.1 m, for a total of 600 m.

Core recovery was generally good. In the limonite horizons (upper and lower), core recovery was 90% on average, and ranged between 40 and 90% in the breccias. In the saprolite, core recovery was consistently below 70% with values as low as 30% recorded in zones containing serpentinite fragments. As Core recovery in the Lower Ferricrete Breccia and Ferralite zones, where most of the potentially economic mineralized material occurs at Nkamouna and Mada generally exceeds 88%.

Reverse-Circulation Drillholes. In 2002, Geovic imported an Australian-designed, truck-mounted reverse-circulation machine, as shown in Figure 9-3. Holes drilled with this machine are referred to in historic GeoCam reports as "air core" holes, only drill cuttings are produced, and the drilling methods are most accurately described as reverse circulation drilling.

This drilling rig uses three chisel-type or finger-type tungsten carbide bits to cut the laterite, and recovers material by air or water flushing through the inner pipe of a double-walled reverse-circulation recovery system, from the bit to the surface. The outer tube has an external diameter of 74.4 mm, while the inner tube has an internal diameter of 36.6 mm. The drill uses compressed air or water with Baroid drilling mud at 150psi pressure as the drilling fluid. Water was used to flush the drill stem and bit while samples were collected at the cyclone using one-meter sample runs. The drill pipe used is in conventional 3 m lengths.

A two-person drill crew and three labor assistants attend the drill, supervised by a geologist. Set-up time and tear-down time is 5 to 10 minutes. A 30 m hole can typically be drilled in 2 hours, when no drilling difficulty is encountered.

The reverse-circulation drill was used between May 2002 and September 2003, when 176 holes (NKM 1,010 to 1,185, plus NKM-3.3) totaling 3,690.25 m were drilled at Nkamouna. Most of these holes were drilled as fill-in holes on a series of lines which had already been sample by pitting, generally at distances greater than 100 m between drillholes. Several of these were twins (within 5 m) of previous pits, and several others were later twinned by test pits sunk on the drillhole collar. About 20 holes were drilled on a tight grid of approximately 15 x 15 m in West Nkamouna, to test the short-range variability from one hole to the next. The drill rig is currently stored at the Kongo Camp of GeoCam.

Drilling during the period 2004-2009 was conducted using identical drilling equipment to previous programs, targeting infill holes at Nkamouna and infill/step-out holes at Mada. A total of 2,054 holes totaling 54,900 m have been completed at Nkamouna and Mada subsequent to the 2007-2008 resource/reserve statements.

No further geological drilling delineation was conducted at any of the properties subsequent to 2009.

9.2 Interpretation

SRK has conducted a detailed review of all historic and current drilling and pitting program data, and is of the opinion that both the methodology used to collect the samples and the current sample spacing is adequate for use in resource estimation. SRK notes that the historic core sample analyses conducted by GeoCam are adequate, but is of the opinion that the larger sample

sizes afforded by pit and reverse circulation samples constitute a more representative sample, given the coarse grain size and highly variable distribution of asbolane, which is the mineral of economic significance in the Nkamouna and Mada deposits.

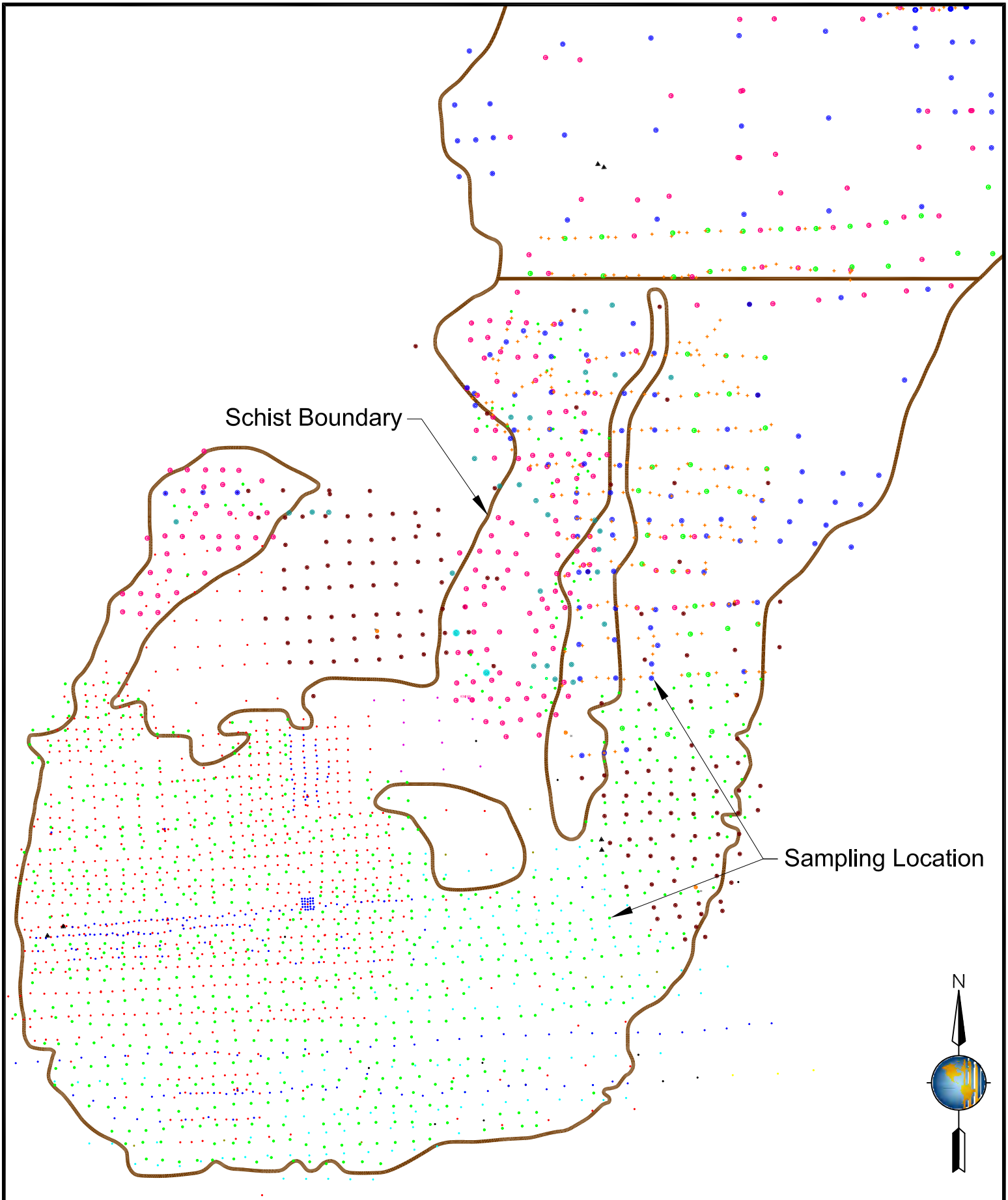

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Figure 9-1
Drillhole and Pit Locations for the Nkamouna Area

Scale:	NTS
Date:	30 Nov. 2009
Dwg:	9-1 Nkm DH's

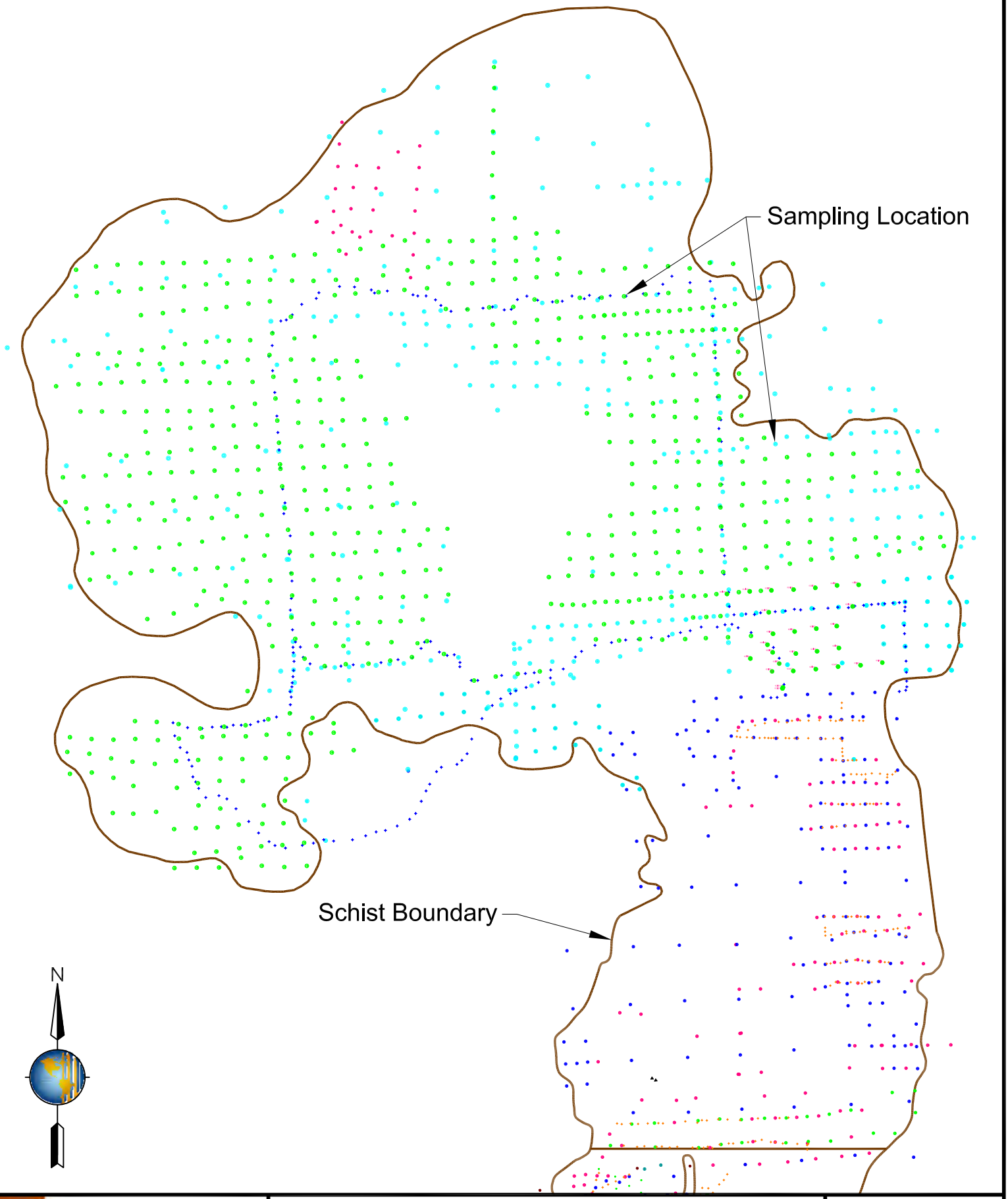

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Figure 9-2
Drillhole and Pit Locations for the Mada Area

Scale:	NTS
Date:	30 Nov. 2009
Dwg:	9-2 Mada DH's

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Figure 9-3
Truck-Mounted Reverse-Circulation Drill Rig at Kongo Camp

Scale:	NTS
Date:	30 Nov. 2009
Dwg:	9-3 Drill Rig

10 Sampling Method and Approach (Item 14)

The following section is excerpted from the 2008 PAH Report. Changes to standardizations have been made to suit the format of this report. Edits to text are annotated by the use of square brackets or reference to SRK.

The sampling techniques utilized by GeoCam are described in considerable detail by Geovic (2003) and Burcham (2003). SRK observed grab sampling and manual processing (physical upgrading) procedures while on-site on November 10, 2009.

The sample bags normally used by GeoCam are white double-thickness polyester, with a drawstring at the neck and a label sewn inside the neck. The geologist records the sample identifier on the inside label and on the outside of the bag, with a permanent marking pen.

10.1 Sampling Methods

10.1.1 Pit Sampling

Most of the sample points were exposed and sampled by test pits, dug using simple hand tools by local labor crews. A standard pitting crew, consisting of two people, is shown in Figure 10-1. A photo of a typical pit is provided in Figure 10-2. The standard equipment for a crew consists of hard hats, a steel pry bar, a bucket, rope, a shovel handle or scoop, and steel-toed rubber boots and a harness. A gasoline-driven air blower and 20 m of vinyl tubing are normally available to provide air when a pit is poorly ventilated, usually at a depth of more than 13 m. Each pit is normally square in section and 1.2 m in diameter, although the diameter may vary slightly. It was reported to SRK that a 20 m deep pit requires 20 days to complete using manual methods.

Spoil not included in the sampling program from the pit is deposited in piles around the pit, but is not rigorously segregated by depth interval. Changes in texture (breccia, limonite) or conspicuous changes in color formed the basis for sampling interval. A sample is collected every meter by cutting a rectangular groove in one wall of the pit, measuring 10 x 5 cm. When more than one sample is collected from an interval concurrently (i.e. adjacent walls), these are oriented following the main cardinal compass points (N, E, W, S). Each pit is inspected one or more times daily by a geologist to log geology, check channel progress, collect the samples from the intervals sampled in his absence, and to decide whether to continue digging.

10.1.2 Trench Sampling

The trench in Nkamouna West was intensively sampled after excavation, using vertical channel samples. Since the trench represents effectively only one sample point in a previously-pitted area, and because the main purpose of the trench was for geotechnical information, the trench samples were not used in the resource estimate.

10.1.3 Drillhole Sampling

Because the data from the UNDP holes were not used by Geovic and its contractors for resource estimation, the UNDP sample quality is not discussed in detail. From SRK's review of the examination conducted by PAH of the original log of hole KG-S-1, it is apparent that the UNDP holes were sampled at irregular intervals, corresponding to both core-barrel lengths and also to geological breaks.

Sample intervals generally varied between 0.5 and 1.65 m. Each interval was logged by color and texture, and by mineralogy where noted. All samples were analyzed for Ni, Co, Mn, Cr, Cu, Zn, Pb, and MgO. Composites representing 5 to 10 m were analyzed for Fe_2O_3.

The GeoCam diamond-drillholes completed during 1999 were sampled at 1.0 or 1.5 m intervals generally, although there were many exceptions due to geological breaks and coring intervals. Core from the laterite zone (soft) was air-dried, crushed, split and halved, with one half sent for assay. The core from the partly weathered, hard serpentinite was cut into two equal parts along the vertical axis of the core, and one part was forwarded for assay while the other was left as backup in the sample store.

Geovic's reverse-circulation holes, drilled during 2002 and 2003, and 2004-2009, were almost invariably sampled at 1 m intervals. The reverse-circulation pathway, including the cyclone and collection buckets, was flushed with water after collection of each one-meter interval, to prevent cross-contamination. The sample expelled by the cyclone, including the water used to flush the sample pathway, was logged by the drill geologist for lithology, and drilling parameters (wet vs. dry, hardness, unusual sample volume, etc.) were recorded.

After the sample-recovery bucket has stood until most fines had settled, the clear water at the top of the bucket was decanted, and the wet sample placed in a previously labeled bag. At the end of the shift, the bags were transported to Kongo Camp.

As is typical of reverse-circulation drilling, precise measurement of the drilling recovery percentage was not possible.

Water-soluble polymer (Baroid EZ Mud) was used to maintain recovery in clay-rich intervals in the lowermost ferralite and upper saprolite horizons. It is recognized that sample recovery in these intervals was unsatisfactory, but these horizons are rarely of economic cobalt grade, and have not been included in the resource estimate

Geovic commissioned a careful study (Burcham, 2003) of potential downhole contamination realized in the previous reverse-circulation drilling programs. The prominent "manganese spike" typically present near the base of the breccia in drillhole assays reveals that vertical cross-contamination is negligible, since the "manganese spike" in drill holes is the same order of magnitude as is observed in channel samples taken from pits.

10.2 Factors Impacting Accuracy of Results

In areas where anomalous concentrations of asbolane mineralization was encountered, care was taken to produce an even split of those intervals. There is also no known geological feature that is preferentially mineralized, or unmineralized, in these deposits. Based on comparison of field duplicate results at the time of the historic drilling, there was no indication of any bias imparted during the core, reverse-circulation or trench sample collection and splitting as part of the sample preparation. Comparisons of the current analytical results obtained by GeoCam were consistent with those produced in previous drilling campaigns, and confirmed the reliability of the original drilling of these deposits based on the opinion of GeoCam personnel in charge of data collection and analysis. Therefore, considering all of the above, the GeoCam samples produced are considered by SRK to be representative of mineralization in the Nkamouna and Mada deposits.

10.3 Sample Quality

Sample preparation was done in a reliable and professional manner, using a riffle splitters on pit, RC and core samples as a basis. The samples were split on an average of 1 m intervals, which is considered an appropriate sample length given the style of mineralization and mining method. Core recovery was reportedly quite good, and field inspection of mineralized material in trenches during the site visits confirms overall envisioned rock quality. Given the generally disseminated nature of mineralization throughout the deposit, and the overall lack of structurally related fracturing and alteration, SRK is of the opinion that the sample preparation practices were done in an acceptable manner and have resulted in reliable and representative samples.

10.4 Sample Parameters

Samples for Nkamouna and Mada were typically collected on an average of 0.91 m, although assay intervals range between 0.0 m and 10.0 m. SRK is of the opinion that the sample intervals as selected by GeoCam are representative of the currently envisioned mining unit and are suitable for their use in resource estimation.

10.5 Relevant Samples

SRK is of the opinion that the current sample database provided by Geovic is adequate for use in resource estimation, and that the resulting analyses are representative of the mineralized tenor and length of mineralized intervals. Drill holes/pits are developed vertically, and no down hole survey was conducted. As mineralization is typically stratiform and oriented horizontally and drill hole lengths are relatively short (19 to 23 m), SRK is of the opinion that the mineralized intercepts as recorded by drill hole assays adequately reflect the true thickness of mineralization.

SRK Consulting Engineers and Scientists	**Nikamouna and Mada Deposits East Province of Cameroon, Africa**	**Two-Man Pitting Crew with Basic Equipment (Pit in Foreground)**		
SRK Job No.: 195100.010				
File Name: Figure10-1	**Source: PAH, 2008**	Date: 11/30/09	Approved: KG	**Figure: 10-1**

SRK Consulting Engineers and Scientists	**Nikamouna and Mada Deposits East Province of Cameroon, Africa**	**Example of Typical Hand Excavated Pit**		
SRK Job No.: 195100.010				
File Name: Figure10-2	**Source: PAH, 2008**	Date: 11/30/09	Approved: KG	**Figure: 10-2**

11 Sample Preparation, Analyses and Security
(Item 15)

The following section is excerpted from the 2008 PAH Report. Changes to standardizations have been made to suit the format of this report. Edits to text are annotated by the use of square brackets or reference to SRK.

11.1 Sample Preparation and Assaying Methods

11.1.1 Sample Preparation

GeoCam maintains a sample-preparation facility at the Kongo Camp, where samples are prepared by GeoCam employees for assay. SRK reviewed the sample preparation procedures in detail, although the sample preparation facility was idle at the time of the site visit.

Upon arrival from the field in polyethylene woven bags, the samples are stored in a sheltered location until processed. As each bag was opened, the sample was placed in a steel tray for drying, and an aluminum tag bearing the sample information on the sample bag placed on the tray. After drying in a wood fired oven for 24 to 28 hours (Figure 11-1), the sample was quartered and placed in a clearly labeled plastic bag, with the sample location and interval number recorded. Another aluminum tag was prepared which accompanied the sample, in transit to the U.S. The aluminum tag placed in the steel tray before oven drying remained with the reject sample on the shelves in the storage facility at the Kongo Camp.

The temperature of the drying oven was not recorded, but was verbally confirmed to be within the vicinity of 100°C, plus or minus 20°. Samples were examined visually from time to time to determine the degree of dryness, and normally after six or seven hours were judged to be sufficiently dry for further sample processing.

Upon removal from the oven and cooling, each sample was visually inspected for the presence of oversize material (coarser than approximately 2 cm). Oversize material was manually crushed in a mortar and pestle and returned to the sample tray (Figure 11-2). At this point, the dried sample was inspected again by a geologist to ensure that the on-site logging did not miss important geological features due to excessive drilling mud or poor lighting. The sample was then split in a Jones-type riffle splitter with openings measuring 10 mm (Figure 11-3). Normally a 200g dried sample was collected for Ferralite and a 500g sample for Breccia and bagged for shipment to the assay lab. The shipment of samples followed industry accepted procedures regarding chain of custody. Samples were shipped by vehicle to GeoCam's office in Yaounde, the capital of Cameroon, where they were delivered to a common carrier for air-freighting to North America.

From 2002 until early 2004, Geovic contracted Mintec of Tucson, Arizona, to oversee the Quality Assurance/Quality Control for Cameroon samples. Mintec provided new 4-digit sample numbers to each sample, before sending the samples to Actlabs in Tucson. Actlabs then pulverized the samples to minus-150 mesh and returned the pulps to Mintec. Mintec then inserted duplicates, standards, and blanks into the sample stream prior to returning the pulps to Actlabs for analysis. Commencing in 2004, Geovic employed the services of Dominic Arrieta of DTA Engineering to oversee assay QA/QC for the GeoCam samples.

SRK has reviewed the sampling procedures that Geovic personnel and Mintec, Inc. implemented during the previous and current drilling and pitting programs and is of the opinion that the collection and handling of samples meets or exceed industry standards.

11.1.2 Assaying

During the period 2003-2009, all samples were shipped to Actlabs (formerly ACTLABS-Skyline), Tucson, Arizona, for analysis.

Sample Preparation for Assaying

The samples received at Bondar-Clegg and Actlabs [formerly Actlabs-Skyline] in Tucson were dried for 24 hours at 150°C before analyses. According to the mineralogical literature on asbolane, there should be no loss of chemically combined water or hydroxyl ions below 150°C. Thus the subsequent assays reflect intact dry asbolane, which is lacking only any loosely-bound water that is not included in the calculated dry tonnes of mineral resource.

11.1.3 Laboratory Qualifications

The Actlabs' Tucson facility (currently Skyline) is accredited to ISO/IEC-17025 and CAN-P-1579 (Canadian) standards, and is thus as fully accredited as a commercial mining assay laboratory.

11.1.4 Laboratory Methods

Following the drying at Actlabs facility, as discussed above, pulps of Geovic samples were digested in a 3-acid solution and 4-acid solutions and analyzed primarily by the ICP-OES (Inductively Coupled Plasma Optical Emission Spectrometry) method for Co, Ni and Mn. The 3-acid digestion is normally sufficient to dissolve all minerals typically present in the Nkamouna samples.

Various other appropriate methods were used for occasional analyses of 34 other elements (Pb, Zn, Cu, Cr, V, Mg, Al, Sc, Zr, MgO, SiO_2, etc.) for bulk samples and other specialty samples.

11.2 Inter-Laboratory Comparisons

Various inter-laboratory checks have been undertaken by Geovic throughout the life of the Nkamouna project.

In 1999, K D Engineering Co Inc. of Tucson, Arizona, visited Nkamouna and undertook to re-sample eight exploration pits. Samples were taken separately from 1 m intervals in channels in the east and west wall of each pit. Splits of each crushed sample were sent for pulverizing and assay to three different laboratories: International Plasma Laboratories (Vancouver); Bondar Clegg Intertek Testing Services (Vancouver); and Genalysis Laboratories (Perth, Australia). The laboratories did not include Actlabs, which subsequently assayed the greater bulk of the Nkamouna samples. Their report (K D Engineering Co. Inc., 2000), indicates that Genalysis and Bondar Clegg agreed closely on Co assays (difference of less than 2% relative, and a Coefficient of Correlation, R^2, of 0.987), whereas the International Plasma results averaged more than 10% low, with R^2, of less than 0.95 when compared to either of the other two labs. No further samples were analyzed by International Plasma.

11.3 Quality Control

The samples assayed by Actlabs were submitted to both Actlabs' and Geovic's independent QA/QC checks. The use of second splits and sample standards are universally recognized methods to provide confidence in the assaying reliability.

11.3.1 Actlabs Quality Control

The Actlabs laboratory runs assay batches of 24 prepared pulp samples, comprising 20 samples plus repeats on the 1st and 20th samples of each batch, in addition to two in-house standards. One sample per client's submitted batch of 20 was reweighed along with both an in-house and a certified reference standard of known Co-Ni-Mn content. Actlabs internal checks allow for a maximum acceptable variance of 2% for duplicates and standards. Given its ISO and CAN-P-1579 certifications, Actlabs is required to have a suitable program in place for periodic round-robin inter-laboratory comparisons.

11.3.2 Geovic Sample Splits

Geovic undertook a comprehensive program of comparing second sample splits from Nkamouna. The pairs of samples extracted from the same sample intervals show a high degree of correlation for Co, Ni, and Mn, providing confidence in the ability of Actlabs to generate reproducible assay results from similar sample material.

The 39 second splits for which assay results have been received, distributed throughout 35 sample submission shipments, were extracted from the same sample rejects stored at the Project Camp (Kongo) as the original samples. Once an original 200g sample was drawn, the reject was remixed (further ensuring complete homogenization) and a second sample was drawn and had a "D" added to the sample number. After sample preparation by Actlabs, all sample pulps were assigned an individual number by Mintec prior to the actual assaying at Actlabs.

Figure 11-4 shows a comparison of splits for cobalt. Similar plots prepared by Geovic for Ni and for Mn indicate that similar correlations occur for those metals.

11.3.3 Geovic Standards

At the request of Geovic in 2003, Mintec fabricated five sample control standards (M5, M6, M7, M8 and M9) of known Co, Ni and Mn value from on-hand Nkamouna material, thereby ensuring that there was no visual difference between the standards and regular samples. The results of 165 analyses of these five standards, distributed throughout 35 sample submission shipments, were received by January 2004.

Perusal of the results strongly suggested that some of the standards had been mislabeled or switched in 32 of the 165 submitted. Mintec personnel therefore examined the anomalous assays of standards, and were able to reassign most of them to the proper standard, according to the Co, Ni, and Mn assays received. Three submitted standard samples did not match any of the five original standards, and it is likely that these three samples were switched with ordinary production samples at the laboratory (three of 168 is about 2%, probably not an atypical error rate for switching of samples in production runs).

The results of assays on the standards are shown in Table 11.3.3.1.

Table 11.3.3.1: Nkamouna and Mada Sample Standards, "Filtered" Results

Standard	N*	Average Co %	std dev Co %	Average Ni %	std dev Ni %	Average Mn %	std dev Mn %
M5	33	0.143	0.0028	0.32	0.0058	0.76	0.013[a]
M6	30	0.275	0.0034	0.62	0.0093	1.52	0.025
M7[b]	31	0.338	0.0038	0.94	0.0129	2.00	0.030
M8	44	0.495	0.0057	0.61	0.0078	2.20	0.031
M9	25	0.287	0.0040	0.58	0.0094	1.70	0.024

*N = number of assays of this standard by Actlabs
a) Excludes one anomalous assay of 2.41% Mn
b) Excludes one anomalous result which is one of the three unresolved standard samples

Table 11.3.3.1 indicates that the precision of the Actlabs assays is very high (i.e. that the Actlab results are highly repeatable). However, the Mintec standards do not appear to have been independently assayed outside Actlabs. Therefore, the Mintec standards program did not elucidate the accuracy (i.e., closeness to absolute truth) of the Actlab assays.

Nevertheless, given that Actlabs are an ISO-certified facility, PAH is prepared to accept the general veracity of the assays on Nkamouna samples.

11.4 SRK Review of 2007 to 2009 QA/QC

Although Geovic has an industry standard QA/QC program in place, they are currently not monitoring results and do not have a compiled QA/QC database. Geovic is using Skyline Assayers and Laboratories (Skyline) in Tucson, Arizona, U.S.A. SRK reviewed QA/QC sample data inserted by Geovic into the sample stream for approximately 5% of the analytical certificates received between October 2007 and September 2009. This is 20 out of 539 certificates. SRK tried to select a representative suite of certificates from each month of the time period, but found some were summary certificates or for other metals. SRK selected 20 certificates at random. From these certificates, SRK found 51 pulp duplicate pairs, 17 blanks and 79 standards. Not all certificates included blanks and duplicates, but for those that included these control samples, duplicates were inserted at intervals on one duplicate per 22 to 31 samples and one blank per shipment. All shipments included standards at one per 20 sample interval and alternated between five separate standards. There is no information that Geovic sends duplicates to a second laboratory as an external check of Skyline.

11.4.1 Standards

Geovic has five different standards used of control samples at the site. SRK noted that in later analytical certificates the standards added an extra number. For instance for M5 the standard was listed as M55, but the expected analytical result remained the same. SRK is unsure whether this is a new standard or simply a new designation. SRK did note that when the standard designation changed there was more variation in the Mn and Ni analytical results for the standards. SRK has not seen certification information for these standards and Geovic did not provide a performance range. However, the standards analyzed during the time period investigated by SRK, show excellent standard performance overall with less then 1sigma (standard deviation) for all standards and for all elements. Standards are listed in Table 11.4.1.1 and standard performance is listed in Table 11.4.1.2. Standard results are shown in the graphs in Figure 11-5.

Table 11.4.1.1: Geovic Standards

Standard	Standard Designation 2009	Co	Ni	Mn
M-5	M55	0.139	0.314	0.702
M-6	M66	0.269	0.605	1.506
M-7	M77	0.323	0.933	1.999
M-8	M88	0.489	0.610	2.210
M-9	M99	0.281	0.576	1.698

Table 11.4.1.2: Standard Performance between October 2007 and September 2009

Standard	N*	Average Co %	std dev Co %	Average Ni %	std dev Ni %	Average Mn %	std dev Mn %
M5/M55	22	0.135	0.0024	0.37	0.0928	0.74	0.0367
M6/M66	8	0.268	0.0042	0.62	0.0141	1.56	0.0716
M7/M77	13	0.318	0.0039	0.94	0.0774	2.04	0.0678
M8/M88	23	0.475	0.0161	0.65	0.0609	2.34	0.1030
M9/M99	13	0.281	0.0062	0.57	0.0243	1.82	0.1138

* N = number of assays of this standard by Skyline

11.4.2 Blanks

SRK found 17 blank samples in the analytical certificates examined. A blank analysis ≥5 times the detection limit is considered a blank failure. There were no blank failures found in this group of analyses. However, in two cases, the analysis for Mn was 0.04%. Although this is not considered a problem, in instances that the analyses for the blank approaches the threshold limits and this information should be monitored on a regular basis.

11.4.3 Duplicates

SRK compared 51 duplicates and 51 original samples submitted to Skyline labs. These are pulp duplicates generated on site and shipped to the laboratory for analysis. Duplicates returned a slightly higher grade analysis than original samples but overall showed very good performance. However, there was one duplicate failure for each element in the analytical range above 0.1%. Duplicate failures are those analysis that do not perform within ±10% of x=y slope on a scatterplot. Failures may be the result of analytical problems or sample variances. All failures must be investigated by reanalysis of the original and the duplicate, to determine the reason for the failure. Scatterplots for the analysis are shown in Figure 11-6 and plots showing percent difference are shown in Figure 11-7.

11.5 Excluded Samples and Reasons

The UNDP drill hole assays were excluded from use in resource estimation due to SRK's inability to independently verify the result, and based on previous work conducted by PAH and recommendations provided by Geovic personnel and confirmed by SRK that the data are not reliable. The assay results obtained from channels 1 and 2 have also been excluded, due to both change in sample support (sample spacing and size) limited vertical extent as compared with the drill hole and pit sample data.

11.6 Interpretation

Although Geovic has not consistently monitored the assay analyses during the 2008 and 2009 sampling programs, the data reviewed by SRK indicates that the analytical results are free of

errors and that the analysis is acceptable for resource estimation. Although there was one failure for each of the elements in the data analyzed by SRK, the standards performed within an acceptable range. This indicates that the duplicate failure could be the result of an improper split during sample preparation. This can only be verified through reanalysis of the duplicate and original samples. All control sample failures must be investigated and reanalyzed if necessary.

SRK also found that the QA/QC samples (blanks, standards and duplicates) are identified on the sample certificates. These are not blind control samples in that the laboratory can identify which samples are QA/QC samples. Based on SRK's review, SRK makes the following recommendations based on industry best practice:

- All sample shipments have had standards, duplicates and blanks inserted into the sample stream. This practice should be continued with all future sampling programs;
- Monitor all QA/QC sample analysis as they are received on a continuous basis and investigate failures by reanalysis;
- Submit QA/QC samples as blind samples assigning a sequential sample number to insure in sequence analysis and that the QA/QC samples are not identified by the receiving laboratory; and
- Submit a percentage of the samples to a second laboratory as an analytical check.

SRK is of the opinion that the number of QA/QC samples inserted into the sample stream as outlined in the Geovic sample handling manual are appropriate for the type of deposit and analytical technique used. SRK recommends the inclusion of control samples by Geovic geologic personnel to monitor the precision and accuracy of analytical results as well as the laboratory's performance and to provide a check for sample numbering problems and sample mix-ups.

SRK Consulting Engineers and Scientists	**Nikamouna and Mada Deposits** **East Province of Cameroon, Africa**	**Photo-Wood Fire Oven**
SRK Job No.: 195100.010		
File Name: Figure11-1.docx		Date: 11/30/2009 | Approved: JAV | **Figure: 11-1**

SRK Consulting Engineers and Scientists	**Nikamouna and Mada Deposits East Province of Cameroon, Africa**	**Photo-Morter and Pestle Crushing**
SRK Job No.: 195100.010		
File Name: Figure11-2.docx		Date: 11/30/2009 | Approved: JAV | **Figure: 11-2**

SRK Consulting Engineers and Scientists	**Nikamouna and Mada Deposits** **East Province of Cameroon, Africa**	**Jones-type Riffle Splitter (10mm)**
SRK Job No.: 195100.010		
File Name: Figure11-3.docx		Date: 11/30/2009 | Approved: JAV | **Figure: 11-3**

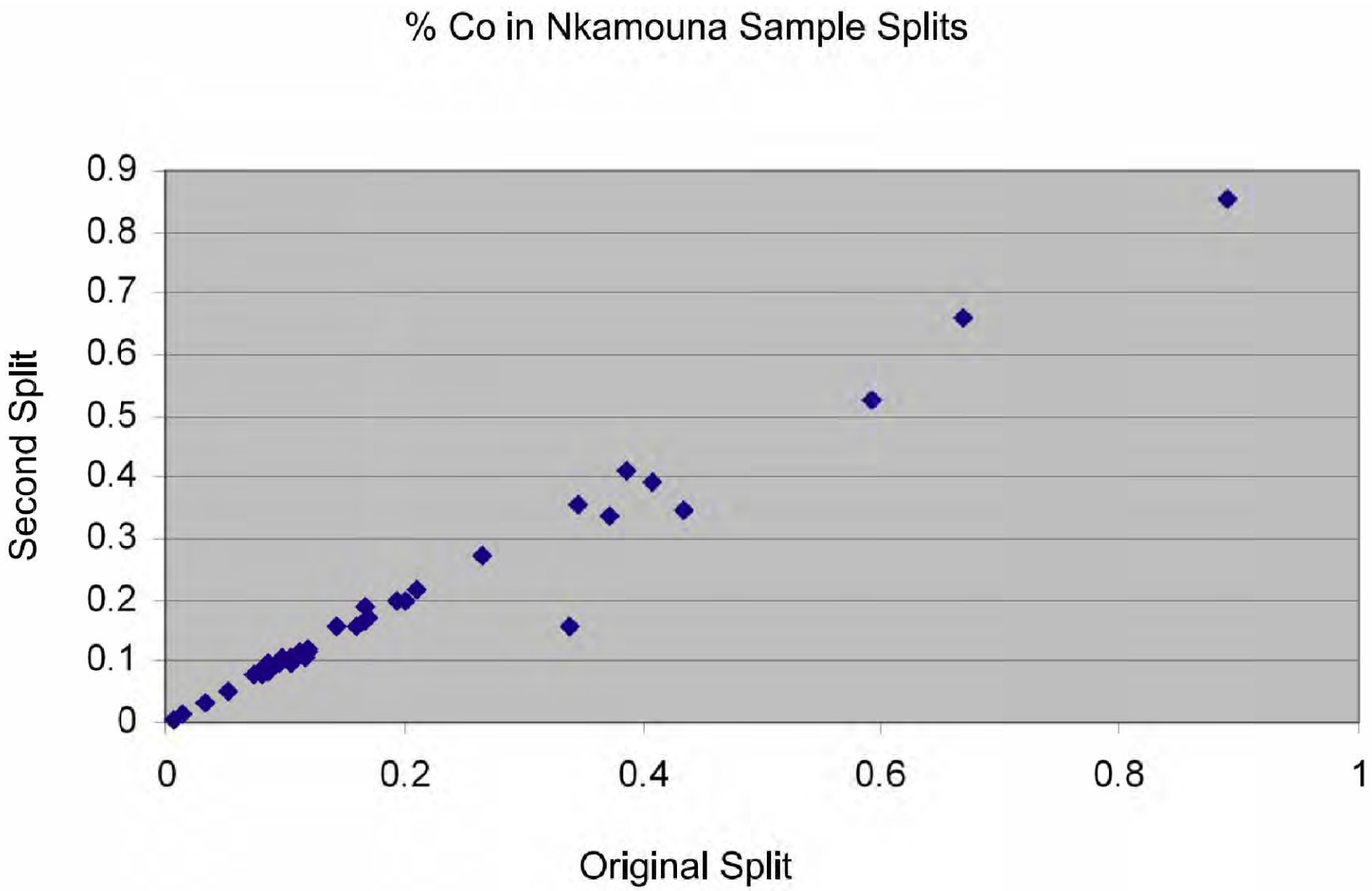

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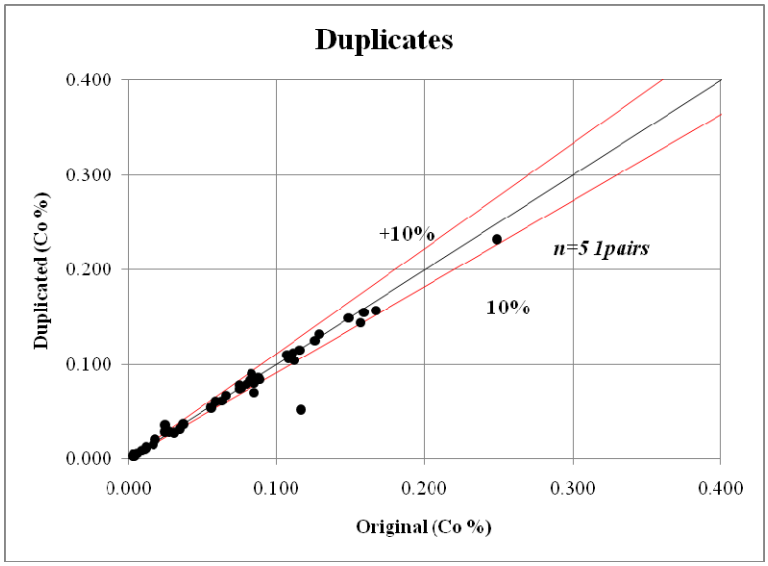
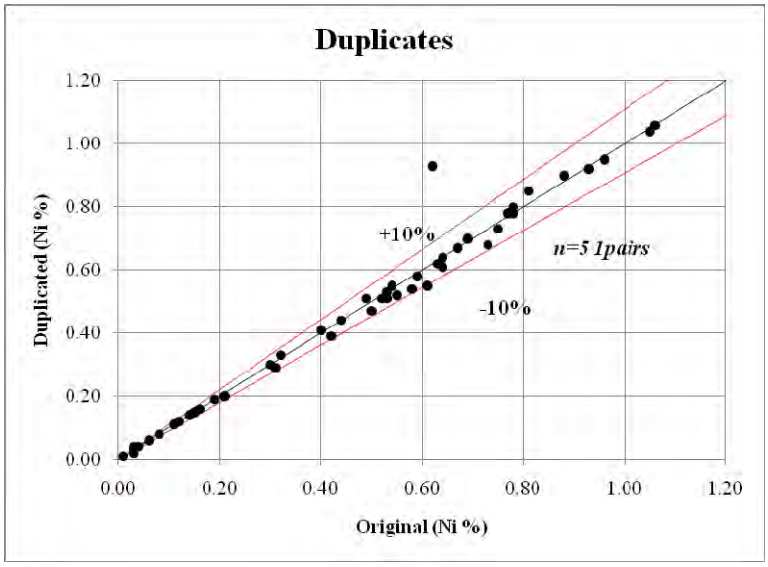
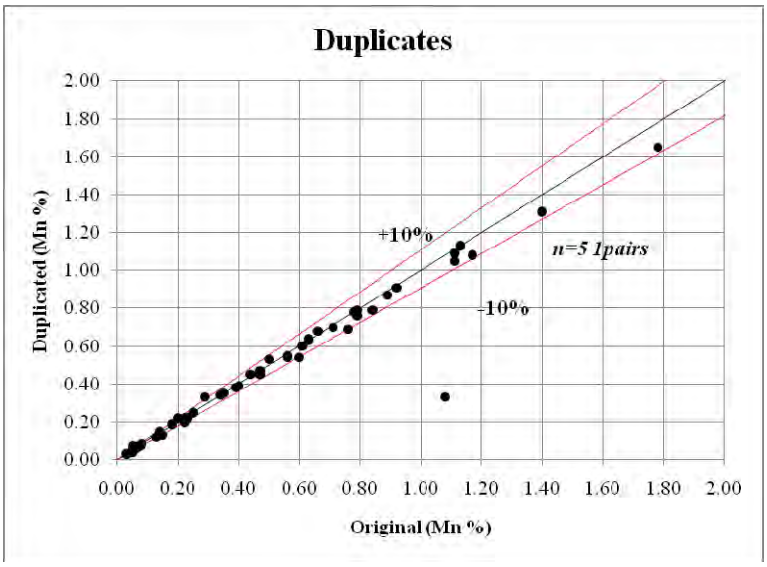
Figure 11-4
Cobalt Assays of Original and Second Splits Nkamouna

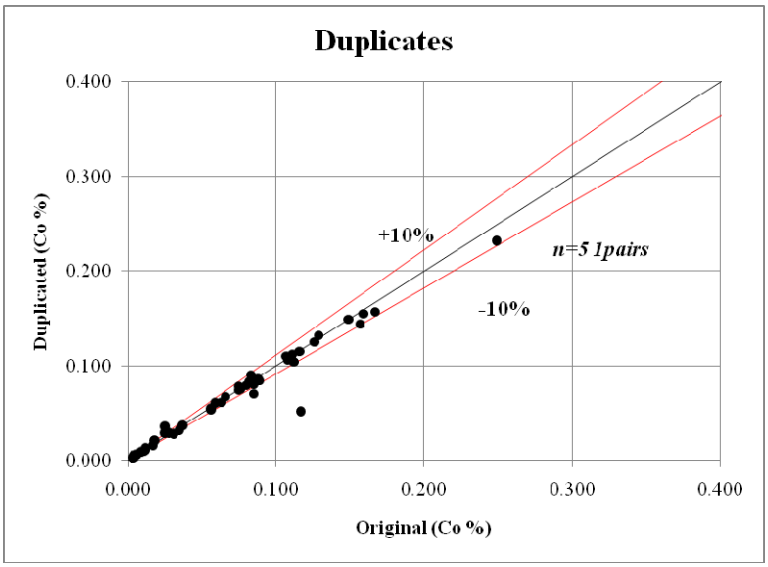
Scale:	NTS
Date:	30 Nov. 2009
Dwg:	11-4 Co Splits

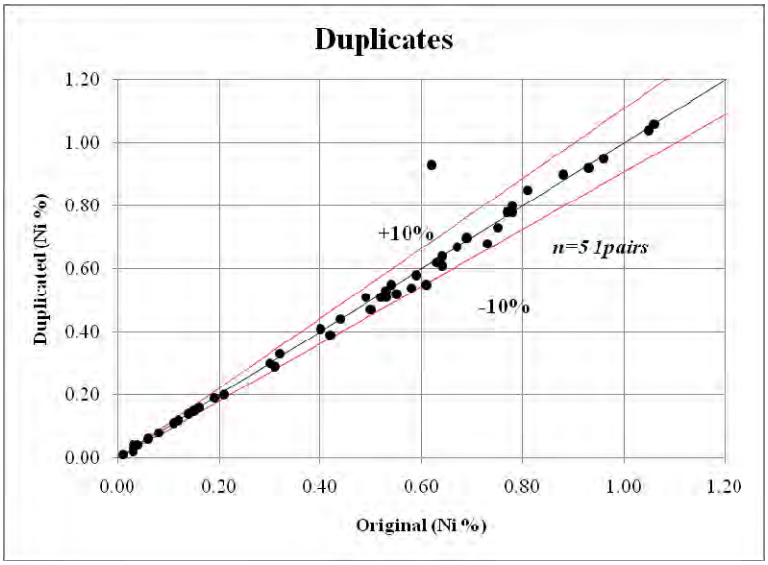


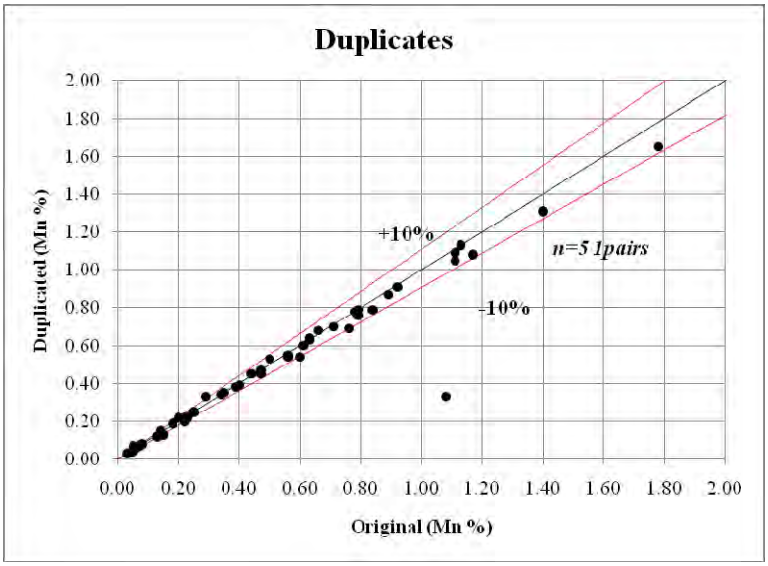




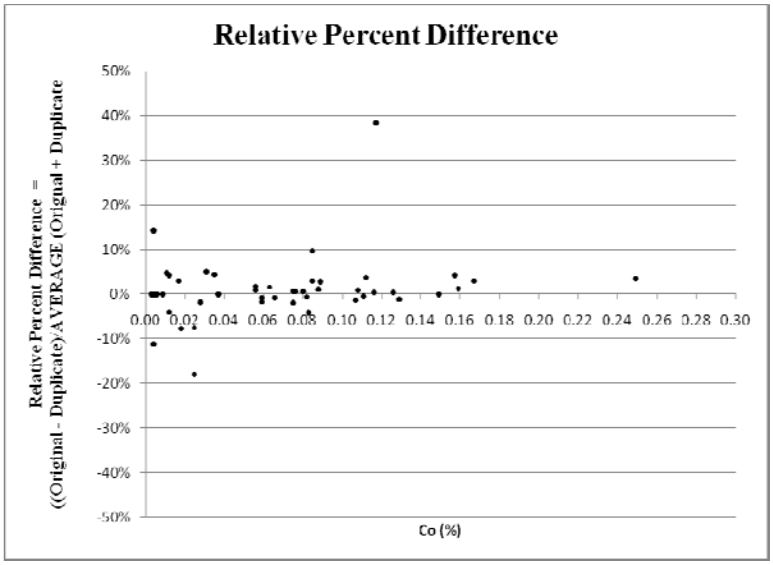
SRK Consulting Engineers and Scientists	Nikamouna and Mada Deposits East Province of Cameroon, Africa	Scatterplots of Duplicate Analysis
SRK Job No.: 195100.010		
File Name: Figure11-5.doc	Date: 11/28/2009 Approved: DKB	Figure: 11-5

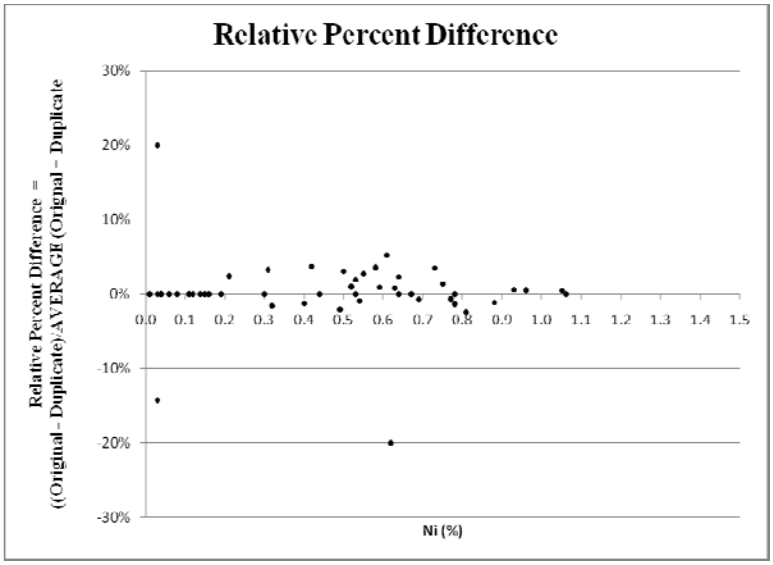


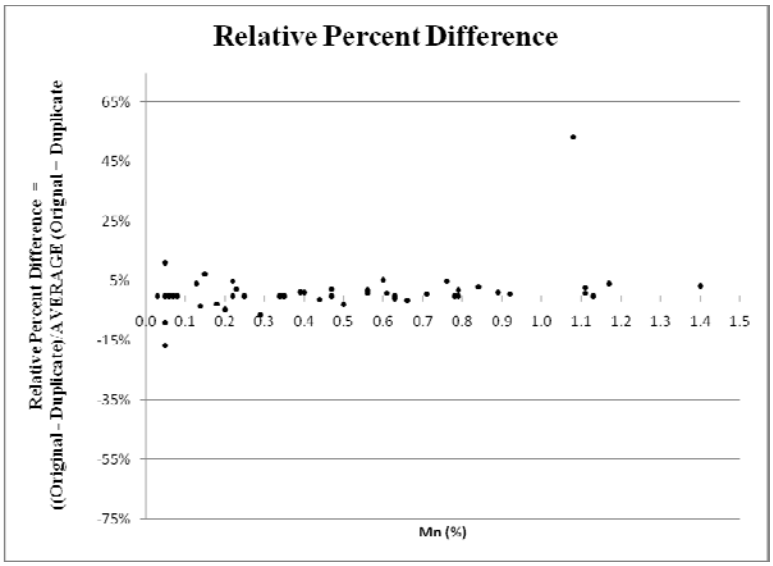




SRK Consulting Engineers and Scientists	Nikamouna and Mada Deposits East Province of Cameroon, Africa	Percent Difference of Duplicate Analysis Scatterplot
SRK Job No.: 195100.010		
File Name: Figure11-6.doc		Date: 11/28/2009 Approved: DKB **Figure: 11-6**







SRK Consulting Engineers and Scientists	**Nikamouna and Mada Deposits East Province of Cameroon, Africa**	**Plots Showing Percent Difference**
SRK Job No.: 195100.010		
File Name: Figure11-7.doc	Date: 11/28/2009 Approved: DKB	**Figure: 11-7**

12 Data Verification (Item 16)

The following section is excerpted from the 2008 PAH Report. Changes to standardizations have been made to suit the format of this report. Edits to text are annotated by the use of square brackets or reference to SRK.

The Nkamouna trench and several pits were inspected during the site visit, although SRK did not verify drillhole locations in the field. Although SRK did not collect additional verification samples during the site visit, SRK conducted a review of the check samples collected by PAH during a previous site visit. A description of these samples and the analytical results are described below.

SRK verified approximately 10% of the 2007 to 2009 Nkamouna and Mada database for cobalt, nickel and manganese by comparison of the original lab certificates from Skyline with the digital database as provided by GeoCam. The results of this analysis show a 1.16% error rate, primarily related to unknown / missing assay certificates (51 errors out of a total of 4,399 random checks). SRK is of the opinion that the error rate is not material with regard to the resource estimate, but recommends that GeoCam correct / address the missing assay certificate issues and correct the database for future resource estimates.

12.1 PAH Samples

PAH collected and sent for analysis several samples, simply to demonstrate that cobalt and nickel mineralization are present on the property. These were all small grab samples of less than 500g each, collected at Nkamouna from pit spoil piles or the large trench (T-1). Samples are described in Table 12.1.1. Because they were from undesignated spoil piles adjacent to each pit mouth, rather than from in-place, the assignments as to depth and laterite unit in Table 12.1.1 are not rigorous. They were analyzed by ALS-Chemex Laboratories in Sparks, Nevada, USA where they were assayed utilizing ICP (Inductively Coupled Plasma) and AA (Atomic Absorption) techniques. ALS-Chemex operates in accordance with ISO/IEC Guide 25.

Samples P-1357 and P-1379 are apparently from profiles developed over or adjacent to schist, as indicated by the high values in Al and Ti, and the low values in Co and Ni in the upper part of the profile. The deeper section of this profile, however, shows high mineralization values indicating the presence of underlying ultramafic rocks.

Table 12.1.1: Analyses of PAH Samples Collected at Nkamouna

Sample Data	Location & Depth	Description	Co ppm	Ni ppm	Mn ppm	Fe %	Al %
P-1241	Pit 1241, unknown depth	siliceous, clayey saprolite, much gibbsite and wad/ (asbolane?)	>10,000 (2.65%)	>10,000 (1.17%)	>10,000	4.77	16.25
P-1357	Pit 1357 < 5 m	Ferricrete Breccia, schist frags, wad (black spots), much gibbsite	73	1,200	362	38.9	7.01
P-1379	Pit 1379, <5 m	Ferricrete breccia, much gibbsite, some wad/(asbolane?)	642	1,500	3270	40	7.07
P-1380	Pit-1380 5.0-6.0 m	porous Ferricrete with gibbsite, wad	6,320	6,330	>10,000	35.4	10
T-1	Trench, approx 3 m	hematitic Upper Ferricrete Breccia	1,515	3,560	7,890	38.7	3.46

SRK notes that although these five samples collected by PAH are not statistically representative given the total number of samples in the Nkamouna-Mada database, SRK is of the opinion that these samples confirm the presence of high grade cobalt and nickel mineralization.

12.2 Quality Control Measures and Procedures

The quality control measure and procedures, as described in Section 12 meet or exceed industry accepted practices.

12.3 Limitations

Some of the historic assay data was excluded from use in resource estimation. These data were not included due to either undocumented collar coordinates or inappropriate assay length and/or undocumented QA/QC protocols. These data were not reviewed by SRK during the course of their investigation. SRK is of the opinion that the data utilized for the resource estimate is reliable, and suitable for use in resource estimation.

13 Adjacent Properties (Item 17)

There are no operating or producing properties adjacent to the Project.

14 Mineral Processing and Metallurgical Testing (Item 18)

14.1 Introduction

The Nkamouna metallurgical processing facility comprises two, essentially independent, processing circuits. Run of Mine (ROM) ore is processed in the physical upgrade (PUG) plant to produce a high grade concentrate product. The concentrate is subsequently processed in the leach and recovery circuit to recover the value metal products.

The major objective of the PUG plant is to separate the coarse, high value material from the fine gangue material in the ROM ore. Separation of the coarse product results in a significant increase in the concentrations of the value metals (cobalt, nickel and manganese) and a significant reduction in the mass of material (concentrate) that requires subsequent processing in the leach and recovery circuit.

The major objective of the leach and recovery circuit is to dissolve essentially all of the value metals from the concentrate and then recover the liberated cobalt and nickel as a mixed sulfide product and the manganese as a manganese carbonate product. Secondary objectives include the rejection of impurity metals, to the maximum extent possible, from the value metal products and the environmental discharge streams.

14.2 Testwork Program History and Summary

GeoCam has conducted a number of testwork programs for the Project to date, with the objective of developing and defining all relevant aspects of the physical upgrading, leaching and metal recovery from the Nkamouna ores. In total, over 530 bench-scale tests and five pilot-scale programs have been performed on 100t of Nkamouna ores. The samples collected over the years have come from throughout the deposit and were selected to cover the variability in grades, spatial distribution and other physical / chemical characteristics present in the Nkamouna cobalt, nickel and manganese deposit.

The testwork programs included a combination of laboratory bench scale, semi-pilot and pilot sized treatments of the ores and concentrates produced through the geological exploration efforts in Cameroon. With the experience gained on-site and at the various testing facilities worldwide, Geovic has developed reproducible methods for the batch production of concentrate samples through controlled attrition scrubbing, washing and screening.

In addition to the metallurgical testwork programs, two mineralogical examinations were also completed in 2002 and 2009 to determine the mineralogical contribution to concentration and leaching response. This information has proven helpful for understanding the physical upgrading and leaching processes.

A list of the testwork programs undertaken to date are summarized below, in chronological order.

- 2010 Ammtec Physical Upgrading: Ammtec Ltd, of Perth, Australia performed a series of batch tests to provide engineering and design data to Lycopodium, on crushing and physical upgrading. This testwork program also provided Lycopodium with the opportunity to directly observe the processing behaviors of this unique ore;

- 2010 Hazen Research PUG pilot program: Hazen Research performed a pilot scale test on a representative Nkamouna ore composite to observe and demonstrate proof of concept of the GeoCam proposed flowsheet, which included a paddle mixer, vibrating screen, one bank of four attrition cells and a screw classifier.
- 2010 Simem vendor testwork program: Simem (Italy) conducted standard paddle mixer suite testwork, to determine required residence time and power input2010 Hazen Research batch concentrate treatment and production of MSP / MnCP products: A bulk sampling and physical upgrading program was completed in Cameroon to produce a selection of concentrates for subsequent processing at Hazen Research. Batch leaching testwork and the subsequent production of representative samples of mixed sulfide and manganese carbonate products for distribution to potential buyers was conducted at Hazen Research. Ore samples representing each of the first five years of Project operation (pre-production and production years one to four) were collected and separately upgraded;
- 2010 Hazen Research continuous leach and recovery pilot program: Subsequent to the developmental batch testwork conducted by Hazen Research (described below), a series of four integrated, continuous pilot plant campaigns were conducted to provide process and engineering data in support of the FS. The testwork campaigns were successfully concluded by the end of August 2010. Pocock Industrial conducted solid / liquid separation testwork for the continuous pilot plant program;
- 2010 Hazen Research batch testwork program: After an unsuccessful pilot plant campaign in February 2010, the GeoCam Technical Advisory Panel (TAP) was requested to assist with the design and provide high level oversight for a laboratory program aimed at developing more suitable testwork conditions for processing the Nkamouna concentrates through the leach and metals recovery process. This program allowed development of improved design and operating conditions as a basis for a second continuous pilot plant program;
- 2009 Hazen Research Mineralogical Analysis: QEMSCAN analysis was conducted on two leach residue samples to assist with the interpretation of the observed leaching responses from selected batch leach tests;
- 2009 Hazen Research Mineralogical Analysis: QEMSCAN Analysis of two PUG concentrate samples with corresponding leach residue samples was conducted to assist with the interpretation of the observed leaching responses from selected batch leach tests;
- 2009 FEI Company Mineralogical Analysis: QEMSCAN mineral abundance and liberation analyses were performed on an asbolane head sample and an Eriez magnetic separation (non-magnetic) product;
- 2009 Hazen Research Concentrate production from Breccia ores: Hazen Research tested six high grade breccia ore samples, to evaluate the potential application of selective mining, as well as magnetic and electrostatic separation processing methods;
- 2008 GeoCam Bulk PUG testwork: GeoCam personnel performed a large scale attrition program primarily aimed at producing approximately 6t of concentrate for a pilot-scale pyrite / sulfuric acid leach test program;

- 2008 and 2009 Concentrate production programs: the purpose of these programs was to produce additional quantities of PUG concentrates, required for flotation, magnetic, gravity and electrostatic separation methods testing;
- 2008 Tests on 2007 Composites (Mountain States, Hazen Research and Jenike & Johanson): In 2008 Mountain States laboratory prepared a representative composite from the high grade ore samples used in the 2007 tests. This composite was split into thirds for detailed PUG testing by Hazen Research and Mountain States and to provide a composite for testing ore flow properties. Jenike & Johanson, a specialized material handling group, were provided with an ore composite of 90% ferralite and 10% breccia to test flow properties in chutes and bins;
- 2007 Tests on samples from 1 m intervals: Mountain States completed 232 PUG tests on samples from intervals of various ore grades and low grade material. The results of these extensive tests on the 2007 samples were used for the statistical analyses that derived the final PUG parameters and performance for use in financial models;
- 2006 Tests on samples from 1 m intervals: Mountain States completed 174 PUG tests on samples from 1 m intervals from Nkamouna and 31 PUG tests on Mada and Rapodjombo samples;
- 2006 Tests on composites: Mountain States prepared a composite of 73 rejects samples from the Nkamouna 1 m samples. A one-third split of this composite was sent to Hazen Research for comparative testing and the last one third split was sent to Metso Minerals for attritioning and comparison of results;
- 2004 Bulk and bench scale tests (Mountain States bulk test and Hazen Research bench tests on bulk sample splits): A 15.6 tonne (dry) bulk sample, deemed to contain 68% ferralite and 32% breccia, was physically upgraded primarily to obtain about 5t of concentrate for leach and solvent extraction pilot tests. The secondary objective of this program was to learn more about physical upgrading while minimizing program cost. Hazen Research performed comparative PUG tests on another split of the samples;
- 2004 Tests on drill cuttings: Seventeen one kilogram samples from seventeen x 1 m intervals were collected from Drill Hole 1017 to test PUG procedures on drill cuttings at depths greater than normally achieved from hand dug test shafts;
- 2003 Mini-bulk test: three composites in aggregate weighing 685 kg wet (550 kg dry) and representing breccia, ferralite and a lower limonitic clay were upgraded by Mountain States, to obtain feed for bench-scale processing tests and to obtain additional information on physical upgrading;
- 2003 Scoping tests: Twenty-two PUG tests were performed on splits of a high grade sample composite that averaged 0.41% Co, 0.57% Ni and 2.43% Mn and contained 58% ferralite and 42% breccia;
- 2003 Mountain States (April 2003 program): testwork was completed on samples from four lithologies of the Nkamouna area. Financial analyses using these test results were used to indicate the best size for upgrading;

- 2003 Metcon Research Inc. (February 2003 program): Metcon Research Inc. of Tucson, Arizona (Metcon) performed extensive column leach tests and preliminary upgrade tests;
- 2002 Pittsburgh Mineral & Environmental Technology Mineralogical Analysis (2008 and 2009 Mineralogy tests and studies): PMET completed a mineralogical laboratory study on 5 samples of ore that included preparation of polished and thin sections, optical microscopy, modal analysis, quantitative XRD analysis, clay analysis and SEM microscopy with EDX elemental analysis;
- 2001 Geovic Ltd. (October 2001 program): In 1997 Bill Buckovic, Founder of Geovic, tested 266 individual samples, from 74 test pits, in the Nkamouna area of the deposit. From those, 217 samples had complete assays and grades above 0.05% Co. A compilation of the 266 tests was prepared by S. Shastry in October 2001;
- 2001 Lakefield Research (March 2001 program): Preliminary PUG results by Lakefield Research of Ontario, Canada; and
- 2003 Oregon State Radiation Laboratories program: OSU developed several ideas and performed many tests that guided and improved the early upgrading by wet screening and testing of wetting agents.

14.2.1 Sample Locations

Figures 14-1 to 14-5 show the location of all samples and their spatial relationship to the Nkamouna deposit. The area most represented by PUG tests lies near the center of the deposit and trends toward the southwest since this area contains most of the higher grade ore that will be sourced during the early mine production years.

Figures 14-2 and 14-3 show the locations of the two bulk samples collected and physically upgraded in late 2008. The PUG concentrates from these two batches were used in the pilot leach tests performed at Hazen Research in 2010. These samples and tests are the basis for the processing design parameters that are used in the feasibility study.

Figure 14-4 shows the location of the 232 samples that were individually tested in 2007 and which served as the basis for the derivation of statistical equations to be applied to the ore production schedule and feed to the crushing plant. Tests were also performed by Hazen Research and Mountain States on 165 of these samples that were blended into composites.

14.2.2 Mineralogy

Pittsburgh Mineral & Environmental Technology (PMET) conducted mineralogical laboratory work on ore samples that included preparation of polished and thin sections, optical microscopy, modal analysis, quantitative XRD analysis, clay analysis and SEM microscopy with EDX elemental analysis.

The overall mineralogy is a complex intergrowth of iron oxides and hydroxides with the Co-Ni-Mn rich mineral asbolane along with minor gibbsite, quartz and clay. The iron oxides include Cr-magnetite inter-grown with chromite and some hematite. About 40% of the material is x-ray amorphous. Data from the PMET report are summarized in Tables 14.2.2.1 to 14.2.2.3.

Table 14.2.2.1: Nkamouna Ore - Bulk Mineralogy

Nkamouna Composites (mineral deportment % w/w)					
	5 Year	**20 year**	**12 Year**	**Mada**	**Rapodjombo**
Asbolane	5.6	5.9	6.0	7.2	3.2
Goethite	23.6	22.7	22.1	32.2	33.0
Amorphous Al-Mn-Fe (O/OH)	43.8	45.6	46.9	43.1	43.2
Hematite	2.9	2.9	3.1	2.9	2.7
Chromite/Cr Magnetite	7.8	8.1	8.5	4.1	4.1
Gibbsite	0.9	0.8	0.7	1.0	1.0
Quartz	6.0	4.3	4.1	3.5	4.2
Kaolinite/Talc	9.4	9.8	8.6	5.9	8.6

In all samples, cobalt, nickel and manganese are concentrated in the mineral asbolane with the generic formula $(Co, Ni)Mn_2O_3(OH)_4{*}H_2O$.

Table 14.2.2.2: SEM-EDX Quantitative Analysis of Asbolane

Nkamouna Composites (elemental deportment % w/w)					
	5 Year	**20 year**	**12 Year**	**Mada**	**Rapodjombo**
Al_2O_3	13.8	14.2	10.8	11.8	7.0
SiO_2	0.1	0.1	0.6	0.0	0.1
MnO	69	59	59	64	62
Fe_2O_3	4	10	11	4	7
CoO	11	15	15	15	9
NiO	5	4	3	5	14

All the cobalt, most of the manganese and up to half the nickel is contained in the mineral asbolane. The oxides of cobalt, nickel and manganese are distributed in the following three major mineral hosts.

Table 14.2.2.3: Distribution of Mn, Co and Ni

Composite	Mineral	CoO (% w/w)	NiO (% w/w)	MnO (% w/w)
5 Year	Asbolane	100.0	42.3	81.8
	Cr-Magnetite	0.0	1.3	2.0
	Goethite	0.0	56.4	16.2
20 Year	Asbolane	100.0	50.5	79.5
	Cr-Magnetite	0.0	7.9	1.4
	Goethite	0.0	41.6	19.1
12 Year	Asbolane	100.0	38.1	85.4
	Cr-Magnetite	0.0	16.2	2.6
	Goethite	0.0	65.5	7.6
Mada	Asbolane	100.0	32.2	90.3
	Cr-Magnetite	0.0	2.3	2.1
	Goethite	0.0	65.5	7.6
Rapodjombo	Asbolane	100.0	52.1	83.0
	Cr-Magnetite	0.0	0.5	3.9
	Goethite	0.0	47.4	13.1

14.3 Metallurgical Testwork – Physical Upgrading

Metallurgical testwork was conducted at a number of laboratories to determine the physical upgrading (PUG) characteristics of the Nkamouna ROM ore. Relevant testwork programs and results are described below.

In general, two different testwork philosophies were adopted for the PUG testwork conducted to date. These are described as follows.

Large scale processing of ROM ore was conducted using expedient techniques and equipment to produce bulk PUG concentrates for subsequent metallurgical leaching testwork. Truck mounted and portable cement mixers and screens were used as the most expedient, effective and economic way to produce suitable quantities of bulk PUG concentrates. These tests were not intended to replicate the equipment and functions proposed in the commercial PUG plant. Metallurgical balances for the bulk programs were generally not closed, as the moisture content of the large sample weights could only be estimated and the number and size of the head, tail and screen fraction samples were generally inadequate to provide the necessary accuracy for the mass balance of the testwork program.

Small scale laboratory attrition tests were conducted to determine the physical upgrade characteristics of a large number of samples representing individual lithologies and selected lithology blends that were obtained from a large cross section of the Nkamouna ore deposit. These tests were conducted in a manner as to achieve a closed mass balance and to provide accurate data on the PUG characteristics (upgrade factors, mass recovery and concentrate grades) on representative samples from the Nkamouna deposit.

The 2007 PUG testwork program completed by Mountain States on 232 individual samples (190 samples of Ferralite and 42 samples of Breccia) forms the basis of the data set used for incorporation of the PUG characteristics into the mine plan as prepared by SRK.

Additional PUG tests, including duplication comparisons between Mountain States and Hazen Research were conducted in 2008 on separate sample splits to those of the 2007 Mountain States

tests above. Similar results were obtained between the laboratories. These results were not included in the statistical analysis data set incorporated into the mine plan as described above.

Both the large scale and small scale testwork results were assessed in order to define the commercial PUG plant design basis.

Table 14.3.1 provides a summary of the results obtained from the major PUG testwork programs, large and small scale, conducted over the duration of the Nkamouna Project to date.

Table 14.3.1: Summary of Results from Major PUG Test Programs

Summary of Major PUG Test Programs
Concentrates sized at + 48 mesh

	No. of Samples	Conc. Wt. %	Recovery, % Co	Ni	Mn	PUG Factors Co	Ni	Mn
2008 Batch #2 on 11.5 tonne Calc 82% Ferralite and 18% Breccia								
Breccia	10	47	80	56	78	1.7	1.2	1.7
Ferralite	68	16	54	23	52	3.3	1.4	3.1
Total	78	22	59	29	56	3.0	1.4	2.9
2008 Attrition Tests on Composite Samples at 90% Ferralite and 10% Breccia (HRI-MSRDI)								
* *Breccia*	4	43	69	32	41	1.6	1.3	1.6
* *Ferralite*	4	15	47	18	47	3.1	1.2	3.0
Total	8	18	51	21	52	2.9	1.2	2.9
Mid-2007	232	One meter samples ~1 kg each						
Samples above cutoff grade. Calculated totals at 90% FL and 10% BR								
* *Breccia*	42	46	84	61	83	1.8	1.3	1.8
* *Ferralite*	171	17	50	21	49	3.0	1.3	3.0
Total	213	20	56	24	56	2.9	1.2	2.9
Mid-2006	174	One meter samples ~ 1 kg each						
Samples above cutoff grade. Calculated totals at 90% FL and 10% BR								
* *Breccia*	23	43	80	60	76	1.9	1.4	1.8
* *Ferralite*	92	12	42	17	46	3.4	1.4	3.8
Total	115	15	50	21	53	3.3	1.4	3.4
Hazen tests on representative composite. Calc. at 90% FL and 10% BR								
Breccia	12	51	89	65	80	1.8	1.2	1.6
Ferralite	61	14	48	17	46	3.4	1.2	3.3
Total	73	18	57	22	53	3.2	1.2	3.0
* BR above 0.15% Co and FL above 0.13% Co								
Dec. 2004		Composite on 15.6 tonnes - 10.9 t from 5 holes						
Sample Composition: 32% breccia & 68% ferralite								
Breccia	109	48	77	46	77	1.3	1.0	1.7
Ferralite	196	20	61	29	61	2.8	1.3	2.6
Total	305	29	66	34	66	2.3	1.2	2.3
Approximated for 90% ferralite and 10% breccia								
Total	305	23	59	29	56	2.6	1.3	2.5
Dec. 2003		13 batch tests on samples 42 kg each						
Sample Composition: 28% breccia & 72% ferralite								
Breccia	9	43	73	54	73	1.7	1.3	1.7
Ferralite	12	16	59	26	54	3.8	1.7	3.5
Total	21	23	63	34	59	3.2	1.6	3.0
Approximated for 90% ferralite and 10% breccia								
Total	21	18	60	29	56	3.3	1.6	3.1
Sept. 2003		20 tests on 1 kg samples of composite and FL and BR						
Composite: 40% breccia & 60% ferralite and Test G-15 results.								
Breccia	4	34	76	45	53	2.2	1.3	1.6
Ferralite	8	24	74	36	81	3.0	1.6	2.8
Total	12	28	75	39	70	2.7	1.4	2.5
Recalculated at 90% ferralite and 10% breccia								
Total	12	25	75	36	79	3.0	1.5	3.2

14.4 Physical Upgrade Circuit – Process Plant Scale-Up Factors and Pilot Plant Data Interpretation

Extensive bench scale PUG testwork has been conducted to date to determine the physical upgrading characteristics of the Nkamouna ROM ores. The bench scale laboratory testwork results were conducted under 'ideal, controlled conditions' and, therefore, depict the optimal PUG response expected from the ore samples. Testwork results obtained from selected bench scale programs are incorporated into the SRK Consultants mine plan to determine the production schedule for the Project. The production data from the mine plan consequently forms the basis for the process plant design.

In order to recover the value rich asbolane minerals from the ROM ore, the fine gangue materials must be rejected. This is achieved by forming a homogenous slurry of the ROM ore in water and separating the particles (at nominally 150 µm), based predominantly on particle size. Beneficial effects on the particle separation process are also expected as a consequence of the higher specific gravity of the asbolane minerals relative to the gangue minerals.

Optimal recovery of the value minerals is achieved by ensuring that the gangue and composite mineral particles are successfully liberated from the asbolane mineral particles prior to size separation and that the size separation is conducted at maximum efficiency.

Particle liberation is achieved by agitating the slurry in high intensity attritioning cells for sufficient duration to produce a slurry with optimal particle liberation and a nominal particle size distribution consistent with that as detailed in the Process Design Criteria document. The intensity and duration of the attritioning process required will vary with respect to ROM feed variation and this will be monitored and adjusted during process operations. Over or under attritioning of the ROM ore may result in sub-optimal particle liberation with a consequent impact on the mass and grade recovery in the concentrate product.

Efficient size separation is achieved by optimal operation of the classification circuit. Presenting a high density, deslimed feed slurry from the dewatering hydrocyclone to the hydrosizer circuit is required to ensure efficient size separation.

14.4.1 Scale-Up Factors

Processing inefficiencies are anticipated in the commercial PUG circuit, relative to the optimal values used in the mine plan, as a consequence of the standard operating characteristics of some of the installed processing equipment. The magnitude of these inefficiencies and the implications on production schedule are discussed in this section.

The primary and secondary crushing circuit, along with the paddle mixer and paddle mixer product screen are not expected to contribute to any process inefficiency issues relative to the mine plan production schedule. This equipment is sized appropriately to meet the necessary mass flow variations expected in day-to-day ore processing.

The attritioning circuit has been designed to provide a nominal energy input of 3.25 kWh/t, based on a retention time of 6 minutes, a design flow rate of 1,198 m^3/h of slurry (542 t/h dry solids) at a slurry density of 35% w/w solids. Variations in attrition energy requirements relative to the nominal design value are anticipated during normal operations in order to produce a discharge slurry with the required particle size distribution. The attrition circuit design provides

the flexibility to significantly vary the attrition energy input (increase or decrease) in order to meet the process requirements.

The retention time in the attritioning circuit can be varied (increased or decreased) by adjustments to the feed slurry density and/or bypassing one or more cells or banks of cells. Additionally, increased production capacity and increased attrition energy may be achieved by operating the PUG circuit for a longer duration than the nominal 13 hours per day as designed. The attrition circuit is sized appropriately to meet the necessary mass flow variations required in day-to-day ore processing and it is not expected to contribute to any process inefficiency issues relative to the mine plan production schedule. No scale-up factors have been applied to the design retention time for the attritioning circuit.

The discharge slurry from the attrition circuit will be dewatered and deslimed in a hydrocyclone bank to prepare a high density, deslimed concentrate slurry for subsequent presentation to a hydrosizer circuit. The normal operation of a hydrocyclone circuit results in process inefficiencies, due to the minor misreporting of material into the overflow and underflow streams.

The process performance of the dewatering cyclones as designed have been modeled by McLanahan Corporation as part of their vendor package for the supply of the classification circuit. Lycopodium have used the McLanahan data to calculate the process inefficiencies associated with the hydrocyclone circuit operation. The data are summarized in Table 14.4.1.1 below.

Table 14.4.1.1: PUG Dewatering Cyclone Performance

Parameter	Units	Feed	Underflow	Overflow
Solids	t/h	640	299.2	340.8
Water	m^3/h	3,383	161	3,221
Slurry	m^3/h	3,620	272	3,348
Slurry % solids	w/w	15.9	65.0	9.6
Slurry % solids	w/v	6.6	40.8	3.8
Slurry density	t/m^3	1.111	1.693	1.064
Particle Size	**Units**	**Cumulative % Passing**		
4,750	µm	100	100.0	100
2,380	µm	98.7	97.5	100
1,700	µm	96.8	94.0	100
1,190	µm	93.9	88.5	100
841	µm	91.1	83.2	100
595	µm	87.2	75.9	100
420	µm	82.8	67.6	100
297	µm	76.7	56.1	100
212	µm	75.0	52.9	100
149	µm	70.7	44.9	99.8
106	µm	62.1	30.1	98.2
75	µm	53.5	18.5	93.0
53	µm	43.7	11.1	80.5
45	µm	40.0	9.10	74.9
38	µm	39.0	8.45	73.5

Hydrosizers will be used to classify the dewatered and deslimed concentrate slurry in accordance with the required separation size (nominally 150 µm for years 1 to 12, and 300 µm for years

13 to 24). The classification step will reject the fine gangue material from the hydrosizer feed stream and produce a concentrate stream comprising the coarser, valuable minerals. The reject stream (dilute overflow slurry from the hydrosizer circuit) is pumped with the hydrocyclone overflow stream to the PUG Tailings Storage Facility (TSF).

The normal operation of a hydrosizer circuit results in process inefficiencies, due to the minor misreporting of material into the overflow and underflow streams.

The process performance of the hydrosizers as designed has been modeled by McLanahan Corporation as part of their vendor package for the supply of the classification circuit. Lycopodium has used the McLanahan data to calculate the process inefficiencies associated with the hydrosizer circuit operation. The data are summarized in Table 14.4.1.2 below.

Table 14.4.1.2: PUG Hydrosizer Performance

Parameter	Units	Feed	Underflow	Overflow
Solids	t/h	288	299.2	340.8
Water	m^3/h	192	161	3,221
Slurry	m^3/h	282	272	3,348
% solids	w/w	60	65.0	9.6
% solids	w/v	31.9	40.8	3.8
Slurry density	t/m^3	1.70	1.693	1.064
Particle Size	**Units**	**Cumulative % Passing**		
4,750	µm	100	100	100
2,380	µm	97.6	95.3	100
1,700	µm	94.0	88.4	100
1,190	µm	88.5	78.0	100
841	µm	83.2	67.8	100
595	µm	75.9	53.8	100
420	µm	67.6	37.9	100
297	µm	56.0	15.9	99.9
212	µm	52.8	9.4	99.5
149	µm	44.9	1.5	90.7
106	µm	30.0	0.1	61.6
75	µm	18.5	0.0	37.9
53	µm	11.0	0.0	22.8
45	µm	9.1	0.0	18.7
38	µm	8.6	0.0	18.1

The combination of product loss to hydrocyclone overflow, hydrosizer oversize and dilution of concentrate as modeled by McLanahan results in a gross loss of approximately 3% of the cobalt in the concentrate relative to the mine plan.

In order to incorporate the effects of the McLanahan modeled losses on the Project economics, SRK Consultants in association with Geovic have incorporated a reduced mass recovery of concentrate into the mine plan relative to the theoretically calculated value as determined from laboratory PUG tests. This is to ensure that the McLanahan modeled PUG circuit operations are consistent with the projections as specified in the mine plan. A reduction of mass recovery into concentrate of 2.5% has been applied in the mine plan, resulting in a 2.5% increase in ROM feed mass required to produce the nominal 656,000 t/y of concentrate.

The estimated benefit of increased cobalt recovery due to differential SG classification as detailed above has not been incorporated into the SRK mine plan. The effect of differential SG classification will offset, to some extent, the requirement to increase the ROM feed mass. The magnitude of the offset will need to be determined by conducting relevant testwork in association with the hydrosizer vendor.

Additionally, it is likely that the classification circuit will be operated, in practice, at a lower nominal separation size ('Operating' cut size) relative to the theoretical separation size of 150 µm. This will result in a reduced loss of concentrate material to the hydrosizer oversize and provide mass and value metal recoveries that are more consistent with theoretically calculated values.

Lycopodium strongly recommends that a PUG pilot plant test be run on a representative sample of Nkamouna ore (for the first nine years of operation) to provide final design and operating data for the dewatering cyclones and hydrosizer equipment, to confirm the required attrition energy input, and to evaluate the beneficial impact of the SG differential on the size separation step. This work, to support detailed engineering design, should be completely scoped and agreed in advance of commencing the work between the client and the selected EPC engineering contractor for the Project. This work should be over-seen by a suitable engineering representative of the client and/or engineering contractor to ensure that it is undertaken to the required standards of quality and completeness.

14.4.2 Pilot Plant Data Interpretation

Two larger scale pilot plant programs were conducted during 2010 in order to demonstrate the integrated operation of the major components of the PUG circuit and to determine the design energy requirements for specific items of process equipment. The programs also provided data with respect to the physical upgrading characteristics of the Nkamouna ROM ores as tested.

A reconciliation of the pilot plant data is provided in Table 14.4.2.1 below and it is compared to the relevant Process Design Criteria (PDC) data.

Table 14.4.2.1: PUG Hydrosizer Performance

Parameter	Units	PDC Yr 1-9	Hazen Pilot Plant 2010	Ammtec Pilot Plant 2010
Attrition Power	kWh/t	3.2	~ 40	1.3
PUG Feed	% - 150 µm	29	19.8	55.8
	% - 300 µm		27.9	61.5
Attrition Feed	% - 150 µm			53.0
	% - 300 µm			61.0
Attrition Discharge	% - 150 µm		70.2	63.2
	% - 300 µm		75.9	68.0
PUG Feed	% Co	0.39	0.32	0.25
Attrition Discharge	% Co		0.34	0.30
PUG Feed	- 150 µm		0.13	0.09
Co grade, %	+ 150 µm		0.36	0.46
	- 300 µm		0.15	0.09
	+ 300 µm		0.38	0.51
Attrition Discharge	- 105 µm		0.13	
Co grade, %	+ 105 µm		0.73	
	- 150 µm		0.14	0.10
	+ 150 µm	0.99	0.82	0.65
	- 300 µm		0.16	0.11
	+ 300 µm		0.93	0.72
	- 600 µm		0.19	0.12
	+ 600 µm		1.11	0.75
PUG Factor	+ 105 µm		2.76	
	+ 150 µm	2.54	3.11	2.15
	+ 300 µm		3.33	2.36
	+ 600 µm			2.48
Cobalt Recovery %	+ 105 µm		73	
	+ 150 µm	72.7	68	80
	+ 300 µm		62	76
	+ 600 µm			71
Concentrate Mass	+ 105 µm		29.0	
(%w/w of feed)	+ 150 µm	28.8	24.0	36.8
	+ 300 µm		20.5	32.0
	+ 600 µm			28.7

Pilot plant attrition data presented in Table 14.4.2.1 demonstrates the effect of over attritioning (Hazen data) and under attritioning (Ammtec data) on the physical upgrading characteristics of the Nkamouna ROM ore. However, notwithstanding the variance in attritioning energies provided, it is noted that similar cobalt and concentrate mass recoveries, as well as PUG factors, are achieved when an appropriate classification cut size is selected. The values of the physical upgrading parameters at the appropriate cut sizes (nominally 105 µm for Hazen data and 600 µm for Ammtec data) are consistent with the PDC values.

During operation of the commercial PUG circuit, the appropriate combination of attrition energy and classification cut size will yield the optimum physical upgrading characteristics for the ROM ore.

14.5 Leaching and Metal Recovery Testing

Acid leaching and metal recovery testwork was conducted in both batch mode and continuous pilot plant mode at Hazen Research. The initial batch testwork was conducted at a laboratory bench scale whereas the later batch testwork was conducted at a larger, pilot scale.

The objective of the initial batch bench scale testwork was to assess the process parameters required for the proposed process flow sheet and to demonstrate acceptable operating conditions for the subsequent pilot plant circuit.

The objectives of the pilot plant testwork were as follows:

- Simulation of the unit process circuits to obtain engineering design data for equipment sizing and selection;
- Corrosion data to assist with material selection;
- Mass balancing data to confirm reagent consumptions and metal recoveries;
- Solid liquid separation testwork for sizing and selection of process equipment;
- Rheology data for pumping and agitator specifications; and
- Particle size data for pumping and agitation specifications.

Optimization of each of the individual circuits was not performed. However, the overall plant was operated in an integrated manner and consequently the leach and recovery testwork was deemed to be definitive from a process design perspective.

The objectives of the pilot scale batch testwork were:

- Production of additional quantities of final products for marketing purposes and to supplement existing intermediate products for refinery testwork;
- Variability testing of selectively prepared PUG concentrates to demonstrate the leaching characteristics of concentrates representing yearly composites for the first five years of Project operation; and
- Supplementary batch bench scale testwork was also conducted to determine the optimal acid addition requirements for the larger scale tests.

14.5.1 Samples and Reagents

A single bulk PUG concentrate composite sample (designated HRI 52103) was blended by Hazen Research to be used for the Technical Advisory Panel (TAP) batch leach tests and the functional component of the pilot plant testwork program. The blended composite was representative of the first 8 years of Project operation. A previously prepared PUG concentrate slurry sample (designated HRI 52066) was used during the commissioning phase of the pilot plant

From the single blended HRI 52103 composite, two major sub-samples were obtained chronologically and separately milled for the relevant batch and continuous testwork. A summary of the head grade analysis of these samples is presented in Table 14.5.1.1.

Table 14.5.1.1: Summary Analysis of HRI 52103 Composite Concentrate Samples

Sample	Co, %	Ni, %	Mn, %	Fe, %	Al, %	Cu, %	Zn, %
52103-2	0.862	0.883	4.41	35.4	8.34	0.035	0.050
52103-3	1.10	0.957	5.78	32.4	9.11	0.049	0.054

Note: 52103-2 was used for TAP batch tests and 52103-3 was used for TAP batch tests and the pilot plant program.

The variability of the head analyses reflects the known 'nugget' effect associated with the relatively coarse and high grade asbolane mineral. Appropriate particle size reduction and blending procedures are required to ensure this 'nugget' effect is eliminated.

The PUG concentrate samples were milled to a nominal size of 80% passing 150 µm in a laboratory ball mill. Bond Ball Mill Work Index testwork, conducted by Hazen Research, indicated a range of 11.2 to 13.3 kWh/t for the PUG concentrate based on a closing screen size of 106 µm.

A single composite sample of pyrite (designated HRI 52324) was used for the testwork. The analysis of this sample is presented in Table 14.5.1.2. The pyrite sample was obtained from Inmet Mining Corporation, a potential supplier of pyrite to the commercial plant, and is produced by flotation.

Table 14.5.1.2: Average Analysis of HRI 52324 Composite Pyrite Sample

Fe %	S %	Al %	Cu %	Zn %	Cr %
45.5	52.0	0.457	0.018	0.008	0.045

14.6 Metallurgical Performance

14.6.1 Concentrate Leaching

The objective of the concentrate leach is to leach cobalt, nickel and manganese from the PUG concentrate. PUG concentrate leaching was carried out at 95 ºC under atmospheric pressure with sulfuric acid. Pyrite was used as the reductant.

The reductive dissolution of cobalt, manganese and nickel from the asbolane mineral in sulfuric acid can be represented by the following individual reaction equations:

$$MnO_2 + 2\ FeSO_4 + 2\ H_2SO_4 \rightarrow MnSO_4 + Fe_2(SO_4)_3 + 2\ H_2O \quad (1)$$

$$2\ CoOOH + 2\ FeSO_4 + 3\ H_2SO_4 \rightarrow 2\ CoSO_4 + Fe_2(SO_4)_3 + 4\ H_2O \quad (2)$$

$$NiO + H_2SO_4 \rightarrow NiSO_4 + H_2O \quad (3)$$

The ferrous ion that acts as the reductant in the above equations is produced from the reduction of the ferric ion and the oxidation of pyrite from the reactions as shown below:

$$FeS_2 + 7\ Fe_2(SO_4)_3 + 8\ H_2O \rightarrow 15\ FeSO_4 + 8\ H_2SO_4 \quad (4)$$

$$FeS_2 + 4\ Fe_2(SO_4)_3 + 4\ H_2O \rightarrow 9\ FeSO_4 + 4\ H_2SO_4 + S \quad (5)$$

The reaction of pyrite within the leaching process is complicated, with the production of elemental sulfur (Reaction 5) a possibility. Mineralogical and stoichiometric analyses indicate that limited production of elemental sulfur is observed, hence Reaction 4 appears to be the dominant reaction.

The ferrous ion is then re-oxidized to ferric ion by the action of the Mn^{4+} and Co^{3+} (as oxidants) in the asbolane mineral as depicted in reactions 1 and 2.

The overall reaction of the MnO_2 and the CoOOH with the pyrite, through the ferrous iron intermediary, can be shown as follows:

$$15\ MnO + 2\ FeS_2 + 14\ H_2SO_4 \rightarrow 15\ MnSO_4 + Fe_2(SO_4)_3 + 14\ H_2O \quad (6)$$

$$30\ CoOOH + 2\ FeS_2 + 29\ H_2SO_4 \rightarrow 30\ CoSO_4 + Fe_2(SO_4)_3 + 44\ H_2O \quad (7)$$

It can be shown that the complete oxidation of the sulfidic sulfur from each mole of pyrite generates sufficient sulfate for its Fe content as ferric sulfate, plus an additional one-half mole of H_2SO_4, hence contributing acid to the overall leach reactions. Additional acid is also produced from the hydrolysis of the ferric sulfate during the leach.

As the leach proceeds and acid is consumed, the iron and aluminium concentrations in solution begin to decrease as these ions are precipitated from solution, mostly as sodium jarosite ($NaFe_3(SO_4)_2(OH)_6$) and sodium alunite ($NaAl_3(SO_4)_2(OH)_6$).

$$3Fe2(SO4)3 + Na2SO4 + 12H2O \rightarrow 2NaFe3(SO4)2(OH)6 + 6H2SO4 \quad (8)$$

$$3Al2(SO4)3 + Na2SO4 + 12H2O \rightarrow 2NaAl3(SO4)2(OH)6 + 6H2SO4 \quad (9)$$

The major parameters in the reductive acid dissolution of the Nkamouna PUG concentrates with pyrite as the reductant are as follows:

Leach Temperature

Adequate leach temperature is necessary to effect the following:

- Reductive acid leaching of the asbolane mineral by the pyrite;
- Hydrolysis and precipitation of excess ferric iron; and
- Hydrolysis and precipitation of the dissolved aluminium.

Sulfuric Acid

Adequate acid needs to be added to solubilize the cobalt, nickel and manganese and the unavoidable dissolution of a portion of Al and Fe. A slight excess is required to maintain an adequate pH for effecting the leach.

Pyrite

Adequate pyrite (mass and surface area) is required for the reductive leach to be successful. An excess of finely ground pyrite to stoichiometric requirements has been shown to be required.

Sodium Sulfate

As a consequence of the use of sulfuric acid in the leach circuit and soda ash as a neutralization and precipitation agent in downstream processes, extensive sodium sulfate production occurs. Sodium sulfate will build-up in the process water circuit as a consequence of the requirement to have a closed circuit water balance. The sodium sulfate concentration in the process water is reduced to acceptable levels by removing sodium sulfate in the Glauber's salt plant.

The complete removal of the sodium sulfate from the process water was deemed impractical and prohibitively expensive. Consequently, a decision was taken to develop the leach parameters and metal recoveries in a sodium sulfate matrix. Unless otherwise stated, only the results

pertaining to leaching in sodium sulfate solutions are presented in this section as the proposed commercial process will be carried out in sodium sulfate solutions.

Where references to leach tests are made throughout this section, the actual leach test number will be stated. Essentially all of the tests referenced in this section are from the Technical Advisory Panel (TAP) testwork program; hence, tests will generally be prefixed with the 'TAP' acronym.

14.6.2 Primary Purification

The objective of the primary purification circuit is to precipitate a majority of the aluminium and iron from the leach solution. Primary purification is conducted on the slurry directly after leach discharge.

The chemistry of aluminium and iron precipitation in primary purification is essentially equivalent to that occurring in the leaching circuit. Aluminium and iron are predominantly precipitated as alunite and jarosite respectively by adjusting the pH with soda ash to a target pH value of 3. The precipitation chemistry is presented in equations 8 and 9 as previously indicated. Equation 10 represents the neutralization of the acid produced by the precipitation reactions with sodium carbonate.

$$3Fe_2(SO_4)_3 + Na_2SO_4 + 12H_2O \rightarrow 2NaFe_3(SO_4)_2(OH)_6 + 6H_2SO_4 \quad (8)$$

$$3Al_2(SO_4)_3 + Na_2SO_4 + 12H_2O \rightarrow 2NaAl_3(SO_4)_2(OH)_6 + 6H_2SO_4 \quad (9)$$

$$H_2SO_4 + Na_2CO_3 \rightarrow Na_2SO_4 + CO_2 + H_2O \quad (10)$$

In addition to the precipitation equations as represented above, aluminium and iron may also precipitate as hydroxides, gibbsite ($Al(OH)_3$) and goethite (FeOOH) respectively according to equations 11 and 12.

$$Al_2(SO_4)_3 + 3Na_2CO_3 + 3H_2O \rightarrow 2Al(OH)_3 + 3Na_2SO_4 + 3CO_2 \quad (11)$$

$$Fe_2(SO_4)_3 + 3Na_2CO_3 + H_2O \rightarrow 2FeOOH + 3Na_2SO_4 + 3CO_2 \quad (12)$$

14.6.3 Secondary Purification

The objective of secondary purification is to remove the majority of the remaining aluminium and iron from the decanted primary purification pregnant leach solution (PLS). Secondary purification follows counter current decantation and is performed on the thickener overflow solution. The aluminium and iron are precipitated as aluminium hydroxide and iron hydroxide respectively by adjusting the pH with soda ash to a target pH value of 4.3. Air is also sparged into selected secondary purification tanks to effect the oxidation of ferrous to ferric to facilitate the near complete precipitation of iron from solution.

Two aluminium precipitation reactions are considered to occur during these steps. At high sodium concentrations, lower pH, and high temperatures in the leach liquors, aluminium precipitation as sodium or hydronium alunite is favored according to Reaction 11. At higher pH values, gibbsite ($Al(OH)_3$) precipitates according to Reaction 12.

$$3Al_2(SO_4)_3 + 6Na_2CO_3 + 6H_2O \rightarrow 2NaAl_3(SO_4)_2(OH)_6 + 5Na_2SO_4 + 6CO_2 \quad (11)$$

$$Al_2(SO_4)_3 + 3Na_2CO_3 + 3H_2O \rightarrow 2Al(OH)_3 + 3Na_2SO_4 + 3CO_2 \quad (12)$$

Similar iron precipitation reactions take place, with sodium or hydronium jarosite predominating at lower pH values and goethite (FeOOH) precipitating at higher pH values.

$$3Fe_2(SO_4)_3 + 6Na_2CO_3 + 6H_2O \rightarrow 2NaFe_3(SO_4)_2(OH)_6 + 5Na_2SO_4 + 6CO_2 \quad (13)$$

$$Fe_2(SO_4)_3 + 3Na_2CO_3 + H_2O \rightarrow 2FeOOH + 3Na_2SO_4 + 3CO_2 \quad (14)$$

In the secondary purification step, air was bubbled through the solution in the pilot plant, oxidizing ferrous iron to ferric iron and subsequently precipitating ferric iron hydroxide ($Fe(OH)_3$).

$$4FeSO_4 + O_2 + 4Na_2CO_3 + 6H_2O \rightarrow 4Fe(OH)_3 + 4Na_2SO_4 + 4CO_2 \quad (15)$$

Precipitated solids are proposed to be recycled back to the leach to recover any co-precipitated value metals.

Preliminary redissolution testwork on the solids produced during secondary purification indicated that the majority of the value metals along with a majority of the aluminium are dissolved from the precipitate under acidified water conditions. Testwork has not been conducted under the process leaching conditions.

14.6.4 Mixed Sulfide Precipitation

The objective of the mixed sulfide precipitation circuit is to precipitate cobalt and nickel as a high grade and high purity mixed sulfide product from the pregnant leach solution. Mixed sulfide precipitation will be carried out on the secondary purification clarifier overflow solution.

Sodium hydrosulfide is used to recover cobalt and nickel from solution by precipitating them as a mixed sulfide product. Base, either Na_2CO_3 or NaOH, is used to maintain the pH, converting the NaHS to Na_2S as follows:

$$2NaHS + Na_2CO_3 \rightarrow 2Na_2S + CO_2 + H_2O \quad (16)$$

$$NaHS + NaOH \rightarrow Na_2S + H_2O \quad (17)$$

Sodium sulfide precipitates cobalt and nickel as sulfides. The chemistry may be represented simplistically as follows:

$$CoSO_4 + Na_2S \rightarrow CoS + Na_2SO_4 \quad (18)$$

$$NiSO_4 + Na_2S \rightarrow NiS + Na_2SO_4 \quad (19)$$

X-ray diffraction analysis of pilot plant MSP product has shown that complex cobalt and nickel sulfides species, such as cobalt pentlandite (Co_9S_8 and $(Co,Ni,Fe)_9S_8$) are dominant in the MSP product.

Other divalent metal cations, including copper and zinc, co-precipitate as sulfides (CuS and ZnS). Depending on the pH, varying amounts of ferrous iron and manganese will also precipitate as FeS and MnS. The reactions of these other metals are similar to Reactions 18 and 19.

Batch bench scale testwork was conducted in a sealed resin kettle under anaerobic conditions. The extent of cobalt and nickel precipitation at pH values greater than 3 was in excess of 98%. Higher pH values lead to higher amounts of manganese in the mixed sulfide products, resulting in contamination of the mixed sulfide product. Aluminium and iron showed similar trends with regards to solution pH.

14.6.5 Tertiary Purification

The objective of tertiary purification is to remove the remaining cobalt, nickel, aluminium and iron from the mixed sulfide discharge solution prior to manganese carbonate recovery. Tertiary

purification follows mixed sulfide precipitation and is performed on the thickener overflow solution.

The residual cobalt and nickel in solution are precipitated as sulfides with a small addition of NaHS in accordance with equations 18 and 19. The aluminium and iron are precipitated as aluminium hydroxide and iron hydroxide respectively by adjusting the pH with soda ash to a target pH value of 5.2. The basic chemistry of tertiary purification is the same as that of the secondary purification chemistry, in accordance with equations 12 and 15.

14.6.6 Manganese Carbonate Precipitation

The objectives of the manganese carbonate precipitation circuits are to produce a relatively pure primary manganese carbonate as the major product ($MnCO_3$ Precipitation 1) and a small quantity of lower quality secondary manganese carbonate product ($MnCO_3$ Precipitation 2) containing most of the residual impurities. Manganese carbonate precipitation follows tertiary purification and is performed on clarifier overflow solution.

It is evident that a relatively small excess to stoichiometric addition of carbonate is required to quantitatively precipitate the manganese.

An increase in precipitation pH results in higher concentration of magnesium, sodium and calcium in the manganese carbonate product. This response can be used to control the purity of the primary manganese carbonate product by implementing two sequential precipitation circuits

14.6.7 Variability Testwork

Five PUG concentrates, produced in Cameroon from Nkamouna ores representative of feed to a commercial leach and recovery plant and totaling 4.7t, were processed at large scale, primarily in batch mode, through the Geovic Process flowsheet at Hazen Research Inc. during September and October 2010. The primary objective had been the production of additional high purity mixed Co-Ni sulfide (MSP), as well as manganese carbonate as a by-product, for evaluation by prospective purchasers. In the case of MSP, some of the MSP was also intended for refinery process development testwork.

14.7 Elemental Deportment

An essential part in the assessment of a complex hydrometallurgical flowsheet is to determine the extent of any potential impurity accumulation within the circuit. The behavior of 17 elements was followed throughout the pilot testwork campaigns conducted on the leach, purification, and precipitation unit processes. More complete analysis, including 63 elements, was conducted on two sets of circuit profile samples from each of the four campaigns, collected during periods of relatively steady state operation. The suite of analyses was performed on both solid and solution samples taken from a number of locations within the leach, purification, and precipitation processes of the pilot plant campaigns (Figure 14-6).

14.7.1 Methodology

In determining the deportment of elements within the Nkamouna flowsheet, an assessment was first made of the extent to which the concentrate dissolves and the subsequent solubilization of elements during the atmospheric acid leaching process (Tables 14.7.1.1 and 14.7.1.2).

Table 14.7.1.1: Leach Feed Concentrate-Pyrite Blend - Multi-element Solids Analyses

		Campaign 3				Campaign 3				Campaign 3	
		Profile 1	Profile 2			Profile 1	Profile 2			Profile 1	Profile 2
Ag	mg/kg	0.3	0.3	Hf	mg/kg	0.9	0.9	Sb	mg/kg	2	2
Al	%	7.62	7.64	Hg	mg/kg	<0.2	<0.2	Sc	mg/kg	55	51
As	mg/kg	23	23	Ho	mg/kg	0.6	0.5	Se	mg/kg	<5	<5
B	mg/kg	613	590	In	mg/kg	<1	<1	Si	%	3.7	3.71
Ba	mg/kg	700	656	K	mg/kg	52	62	Sm	mg/kg	4.2	3.5
Be	mg/kg	<1	1	La	mg/kg	17	16	Sn	mg/kg	<1	<1
Bi	mg/kg	0.2	<0.2	Li	mg/kg	114	108	Sr	mg/kg	6	6
Ca	mg/kg	<10	<10	Lu	mg/kg	<0.5	<0.5	Ta	mg/kg	<2	<2
Cd	mg/kg	<0.5	<0.5	Mg	mg/kg	3,290	3,220	Tb	mg/kg	0.6	0.5
Ce	mg/kg	672	607	Mn	%	5.31	5.3	Te	mg/kg	<0.5	<0.5
Co	%	1.04	1.03	Mo	mg/kg	7	7	Th	mg/kg	3	3
Cr	%	1.88	1.84	Na	mg/kg	682	652	Ti	mg/kg	1,190	1,190
Cs	mg/kg	<0.5	<0.5	Nb	mg/kg	<100	<100	Tl	mg/kg	2	2
Cu	mg/kg	413	411	Nd	mg/kg	18	16	Tm	mg/kg	<0.5	<0.5
Dy	mg/kg	1.7	2.8	Ni	%	1.01	0.997	U	mg/kg	2	1
Er	mg/kg	1	1	P	mg/kg	1260	1330	V	mg/kg	260	230
Eu	mg/kg	0.9	0.8	Pb	mg/kg	125	120	W	mg/kg	7	8
Fe	%	37.6	37.61	Pr	mg/kg	5	4.5	Y	mg/kg	8	7.2
Ga	mg/kg	15	13	Rb	mg/kg	0.2	0.2	Yb	mg/kg	1	1
Gd	mg/kg	3.3	3.3	Re	mg/kg	<0.2	<0.2	Zn	mg/kg	600	582
Ge	mg/kg	1.1	1.2	S	%	1.89	1.89	Zr	mg/kg	40	35

Table 14.7.1.2: Leach Feed and Discharge - Multi-element Solution Analyses

		Campaign 3				Campaign 3				Campaign 3	
Element	Solution Sample	Profile 1, mg/L	Profile 2, mg/L	Element	Solution Sample	Profile 1, mg/L	Profile 2, mg/L	Element	Solution Sample	Profile 1, mg/L	Profile 2, mg/L
Ag	LPF	<0.1	<0.1	Hf	LPF	<0.1	<0.1	Sb	LPF	<0.1	<0.1
	LPT7	<0.1	<0.1		LPT7	<0.1	<0.1		LPT7	<0.1	<0.1
Al	LPF	1.5	0.2	Hg	LPF	<0.1	<0.1	Sc	LPF	<0.1	<0.1
	LPT7	6400	4470		LPT7	<0.1	<0.1		LPT7	2.6	2.5
As	LPF	0.1	<0.1	Ho	LPF	<0.1	<0.1	Se	LPF	<0.1	<0.1
	LPT7	0.1	0.1		LPT7	<0.1	<0.1		LPT7	<0.1	<0.1
B	LPF	<5	<5	In	LPF	<0.1	<0.1	Si	LPF	7.7	4
	LPT7	<5	<5		LPT7	<0.1	<0.1		LPT7	237	240
Ba	LPF	<0.1	<0.1	K	LPF	<10	<10	Sm	LPF	<0.1	<0.1
	LPT7	<0.1	<0.1		LPT7	<10	<10		LPT7	0.1	0.1
Be	LPF	<0.1	<0.1	La	LPF	<0.1	<0.1	Sn	LPF	<0.1	<0.1
	LPT7	<0.1	<0.1		LPT7	0.4	0.4		LPT7	<0.1	<0.1
Bi	LPF	<0.1	<0.1	Li	LPF	<2	<2	Sr	LPF	<0.5	<0.5
	LPT7	<0.1	<0.1		LPT7	116	129		LPT7	<0.5	<0.5
Ca	LPF	53	55	Lu	LPF	<0.1	<0.1	Ta	LPF	<0.1	<0.1
	LPT7	63	63		LPT7	<0.1	<0.1		LPT7	<0.1	<0.1
Cd	LPF	<0.1	<0.1	Mg	LPF	45	46	Tb	LPF	<0.1	<0.1
	LPT7	<0.1	<0.1		LPT7	166	181		LPT7	<0.1	<0.1
Ce	LPF	<0.1	<0.1	Mn	LPF	160	145	Te	LPF	<0.1	<0.1
	LPT7	75	75		LPT7	57900	65100		LPT7	<0.1	<0.1
Co	LPF	6.2	8	Mo	LPF	<0.1	<0.1	Th	LPF	<0.1	<0.1
	LPT7	11000	12100		LPT7	<0.1	<0.1		LPT7	0.2	0.3
Cr	LPF	<0.1	<0.1	Na	LPF	18200	20300	Ti	LPF	<0.1	<0.1
	LPT7	6.2	4.7		LPT7	12700	15500		LPT7	0.6	0.6
Cs	LPF	<0.1	<0.1	Nb	LPF	<0.1	<0.1	Tl	LPF	<0.1	<0.1
	LPT7	<0.1	<0.1		LPT7	<0.1	<0.1		LPT7	<0.1	<0.1
Cu	LPF	<0.1	<0.1	Nd	LPF	<0.1	<0.1	Tm	LPF	<0.1	<0.1
	LPT7	217	186		LPT7	0.4	0.5		LPT7	<0.1	<0.1
Dy	LPF	<0.1	<0.1	Ni	LPF	1.5	3.0	U	LPF	<0.1	<0.1
	LPT7	0.1	0.3		LPT7	6840	7470		LPT7	0.4	0.4
Er	LPF	<0.1	<0.1	P	LPF	<10	<10	V	LPF	<0.1	<0.1
	LPT7	0.1	0.1		LPT7	<10	<10		LPT7	8.5	7.3
Eu	LPF	<0.1	<0.1	Pb	LPF	<0.1	<0.1	W	LPF	<0.1	<0.1
	LPT7	<0.1	<0.1		LPT7	<0.1	<0.1		LPT7	<0.1	<0.1
Fe	LPF	2	<1	Pr	LPF	<0.1	<0.1	Y	LPF	<0.1	<0.1
	LPT7	379	267		LPT7	0.1	0.1		LPT7	1	1.1
Ga	LPF	<0.1	<0.1	Rb	LPF	<0.1	<0.1	Yb	LPF	<0.1	0.1
	LPT7	<0.1	<0.1		LPT7	<0.1	<0.1		LPT7	0.1	0.1
Gd	LPF	<0.1	<0.1	Re	LPF	<0.1	<0.1	Zn	LPF	<0.1	<0.1
	LPT7	0.1	0.1		LPT7	<0.1	<0.1		LPT7	242	258
Ge	LPF	<0.1	<0.1	S	LPF	11400	13200	Zr	LPF	<0.1	<0.1
	LPT7	<0.1	<0.1		LPT7	60100	63500		LPT7	<0.1	<0.1

From these solution assay data, the following elements were identified as having been solubilized during the leaching process: aluminium (Al), arsenic (As), calcium (Ca), cerium (Ce), cobalt (Co), chromium (Cr), copper (Cu), dysprosium (Dy), iron (Fe), lanthanum (La), lithium (Li), magnesium (Mg), manganese (Mn), nickel (Ni), scandium (Sc), silicon (Si), thorium (Th), titanium (Ti), uranium (U), vanadium (V), yttrium (Y), and zinc (Zn). Additional elements, specifically erbium (Er), gadolinium (Gd), neodymium (Nd), praseodymium (Pr), samarium (Sm), and ytterbium (Yb), were also found in the leach solutions at trace concentrations, close to their detection limits.

Of the 28 elements listed above, Ce, Dy, Er, Gd, La, Nd, Pr, Sc, Sm, Y, and Yb make up 11 of the 17 rare earth elements (REEs), of which Ce exhibited the highest soluble concentration increase during the leaching process. Due to the relatively low concentrations of Dy, Er, Gd, La, Nd, Pr, Sc, Sm, Y, and Yb, it is assumed that these all follow the same deportment route as Ce, as any attempt to distinguish more precise deportment patterns based on their low concentrations would be highly questionable. This assumption is also based on the fact that, as they all belong to the same rare earth group of elements, their chemical properties will be similar.

In addition, sulfur (S) and sodium (Na) are present in high concentration in both the leach feed and discharge liquors. While neither are present in significant quantities in the PUG concentrate itself (0.005%$^{w}/_{w}$ and 0.05%$^{w}/_{w}$ respectively), their introduction, predominantly through reagent addition (including pyrite in the leach feed solids blend), and the subsequent generation and

recirculation of high concentrations of sodium sulfate solution within the circuit cannot be ignored.

This leaves 18 elements (excluding S and Na) for evaluation, and the next step is to determine at what stage in the processing plant each is likely to be removed and to which residue and / or product stream(s) it is likely to deport. Thus the remaining 43 of the original 63 elements are excluded from any subsequent elemental deportment commentary.

14.7.2 Discussion

The following provides a deportment summary for each of the elements dissolved during the leaching of Nkamouna concentrate. Additional commentary is provided for sodium, sulfur, and cadmium.

Aluminium

The concentrate contained 7.48% Al (and 1.03% Co). In the leach, which under the design conditions results in extraction of up to 95% of the Co, it is estimated that about 35% of the Al undergoes dissolution, but the major portion of this Al is subsequently precipitated under the prevailing conditions, primarily as a sodium alunite. The leach product liquor, at ~ 12 to 15 g/L Co, contains 4 to 8 g/L Al, corresponding to a net dissolution of 5 to 6% of the Al.

The barren liquor from the manganese carbonate precipitation circuit, therefore, contained <0.5 mg/L Al. Thus, with the MSP and the MnCP products containing only about 0.05% of the Al in concentrate, it is evident that >99.9% of the Al reports with the solids contained in the CCD tailings storage facility. The recycled barren solution contains only traces of Al.

Arsenic

Arsenic is present in the leach feed in low, but detectable, quantities (25 mg/kg). It is estimated that 1% of the As undergoes dissolution in the leach circuit to yield a 0.1 mg/L As concentration in the leach discharge liquor.

Concentration from the MSP barren liquor was below detectable limits (0.1 mg/L), indicating that the majority of As which reports to the sulfide precipitation circuit will be precipitated with the mixed sulfide product.

Cadmium

Solids assays of the leach feed concentrate used during the pilot testwork returned Cd values below detectable limits (<0.5 mg/kg). Although leach discharge solution assays also returned Cd concentrations below detectable limits (<0.1 mg/L), large scale batch leach tests performed on year 1 Nkamouna concentrate returned a Cd concentration in the leach solution of 0.7 mg/L Cd.

As no detectable concentrations of Cd were recorded in the leach discharge solution from the pilot testwork campaigns, predictions regarding the deportment of trace quantities of Cd throughout the circuit are speculative.

The concentration of Cd in the mother liquor for environmental discharge is expected to be below analytical detection limits (<0.1 mg/L).

Calcium

The concentrate used in the pilot testwork contained approximately 0.007% Ca. Despite the absence of Ca in any of the reagents (neutralizing reagents such as lime and limestone have been

specifically excluded from the flowsheet to mitigate scale accumulation and to prevent potential product contamination), concentrations in the leach discharge solution of 60 to 110 mg/L Ca were recorded during the pilot testwork campaigns, with an average value of 96 mg/L. This represents a net Ca dissolution 96% (this excludes any contribution made to the Ca concentration following the use of Denver tap water during the pilot testwork).

Subsequent feed of the MnCP2 barren solution to the Glauber's Salt crystallizer in the commercial facility is unlikely to result in Ca contamination of the Glauber's Salt, other than physical entrainment. Thus, as the water associated with the Glauber's Salt product accounts for less than 5% of the total flow to the Glauber's Salt crystallizer, the Ca concentration in the mother liquor for environmental discharge is expected to be similar to the crystallizer feed solution concentration (5 to 10 mg/L Ca).

Cerium

Cerium is one of 17 metals comprising the rare earth elements (REEs), and exhibited significant dissolution during the leaching process. Elemental analyses were recorded for a further 15 REEs during the pilot testwork program, of which ten (dysprosium, erbium, gadolinium, lanthanum, neodymium, praseodymium, samarium, scandium, yttrium, and ytterbium) yielded varying concentrations within the leach discharge solution. Henceforth it is assumed, for the purposes of further discussion, that the deportments of these ten elements throughout the Nkamouna circuit are sufficiently similar to that of cerium to be represented by cerium in the discussions.

No deportment of Ce to the Glauber's Salt product is anticipated, and the concentrations of REEs reporting to the environmental discharge stream are expected to be less than detectable limits (0.1 mg/L).

Cobalt

Cobalt is the principal pay element in the Nkamouna flowsheet and comprises approximately 1% of the leach feed concentrate mass. Cobalt dissolution within the leach circuit, under design conditions, is expected to be approximately 95%, yielding a leach discharge solution concentration of 12 to 15 g/L Co.

Partial neutralization of the leach discharge slurry in the subsequent primary purification circuit will precipitate a small quantity of Co (anticipated to be basic cobalt sulfate) which will report as a loss (estimated at 0.4%) to the CCD tailings storage facility. Correct pH control within the primary purification circuit is, therefore, vital to ensure minimization of Co loss to tailings. The commercial facility will incorporate a six stage CCD circuit to maximize the recovery of soluble Co (>98.5% Co recovery). The combined solids and soluble loss of cobalt to tailings may, therefore, be expected to be between 6 and 7%.

In the subsequent secondary purification stage minor quantities of Co (estimated at between 1 and 1.5%) are precipitated (anticipated to be as a basic cobalt sulfate) together with a number of impurity and pay elements, notably Fe, Al, and Mn. The precipitated solids are returned to the leaching circuit where it is expected that the associated Co will re-dissolve. The impact on acid consumption due to the re-dissolution of this small percentage of recycled Co is expected to be minor.

In the sulfide precipitation circuit the precipitation of Co is maximized by the addition of NaHS, yielding a barren solution typically <5 mg/L Co. Thus, based on an average feed solution assay from the pilot plant campaigns of ~5 g/L and allowing for dilution, the sulfide precipitation stage

results in the precipitation of approximately 99.9% of the Co from solution. This generated a mixed sulfide product assaying 40% Co, equivalent to >93% of the total Co in the concentrate.

The object of the Tertiary Purification stage is to effect precipitation of base and other metals to prevent subsequent contamination of the manganese carbonate product. Despite the intention of this circuit, the testwork campaigns demonstrated only a partial removal of Co from the feed solution. Profile data from campaign 4 indicated an average feed solution assay of 9 mg/L Co compared to 8 mg/L Co in the discharge solution from tertiary purification. Individual solution sample concentrations, however, varied from 1 to 24 mg/L Co; the high residual concentrations are attributable to insufficient residual NaHS. It is expected in a commercial facility that proper residual concentrations of NaHS in the barren solution from the sulfide precipitation circuit, together with the elevated pH (target pH 4.8) will result in the precipitation of Co as a sulfide to trace levels in the tertiary purification circuit. Return of the tertiary purification precipitate to the leach, and subsequent dissolution of the associated Co, is expected to have negligible impact on reagent consumption due to the small quantities involved.

The elevated pH within the first stage manganese carbonate precipitation circuit (target pH 8.5) is expected to precipitate most of any residual Co (as cobalt carbonate) with the MnCP1 product. The solids assays from the campaign 4 pilot testwork data varied between 10 and 410 mg/kg Co in the MnCP1 product, commensurate with the soluble Co concentrations in the feed solutions. The higher values may be attributed to less than optimal pH control and insufficient NaHS in the upstream tertiary purification circuit. It is expected that commercial plant operations will receive feed solution assays of 0.2 mg/L Co, resulting in a MnCP1 product containing <20 mg/kg Co. This represents <0.02% of the Co in the concentrate.

The barren liquor from MnCP1 generally contains less than detectable limits (0.1 mg/L) of Co. Thus, while there may indeed be further precipitation of trace amounts of Co in the second stage manganese carbonate precipitation circuit, this is not quantified.

Chromium

The barren liquor from the MSP stage contained less than detection concentrations (0.1 mg/L) of Cr as most of the Cr in the feed solution is readily precipitated as a sulfide and thus reports with the mixed sulfide product. One of the benefits of the Nkamouna reducing leach circuit is the deportment of Cr in the trivalent form which is readily precipitated in the impurity removal circuits.

In a commercial facility, therefore, it is expected that the manganese carbonate products will contain no discernable Cr and less than detectable concentrations (<0.1 mg/L) of Cr are expected to report to the environmental discharge stream.

Copper

The concentrate used in the pilot testwork campaigns contained 0.044% Cu (and 1.03% Co) and yielded a leach discharge liquor of ~250 mg/L Cu; this equates to a net Cu dissolution of approximately 40%.

Most of the Cu remaining in the purified solution is precipitated as a sulfide in the sulfide precipitation circuit, yielding a mixed sulfide product containing approximately 0.2% Cu; this represents about 10% of the Cu in the concentrate. The resulting MSP barren solution Cu concentration is below analytical detection limits (0.1 mg/L).

In summary, the MSP contained approximately 10% of the Cu in the concentrate, while no detectable Cu is anticipated in the MnCP1 product. As the Cu concentration in the barren liquor is expected to be at or below detection limits (0.1 mg/L), it is evident that approximately 90% of the Cu reports to the tailings storage facility.

Iron

Iron is introduced to the leaching circuit from two sources; concentrate feed and pyrite reductant. Iron in the concentrate feed to the leaching circuit comprises approximately 38% of the total concentrate mass which, together with the pyrite addition, yields a leach discharge solution of approximately 0.4 g/L Fe. Although some of the Fe of pyrite is initially dissolved, and a portion of the Fe of the concentrate is also extracted, the majority of this iron is precipitated during the course of leaching. Based on a combined concentrate and pyrite feed to leach of 40% Fe, the net dissolution of Fe is estimated as less than 0.1%.

The final barren solution typically contains less than detectable concentrations of Fe (1 mg/L). As the MSP and MnCP products contain only about 0.02% of the Fe in concentrate, it is evident that >99.9% of the Fe reports to the CCD tailings storage facility.

Lithium

The concentrate used for the continuous testwork campaigns had a composition containing approximately 0.011% Li. This material partially leached to yield an average leach discharge solids composition of 0.0011% Li, indicating approximately 90% dissolution.

The subsequent primary purification circuit yielded no detectable precipitation of Li and thus only about 10% of the Li in concentrate is expected to report to the tailings storage facility.

In many respects Li behaves similarly to Na, i.e. it dissolves to form highly soluble lithium sulfate which would partially explain why there is only limited precipitation of Li within the various impurity removal and product precipitation stages. Unlike Na, however, no Li is introduced into the circuit via reagents and thus the lithium sulfate concentrations throughout the circuit are significantly lower. As these concentrations are low, it is anticipated that none of the residual soluble Li will report with the Glauber's salt crystals.

Magnesium

The average Mg content of the concentrate used for the pilot testwork campaigns was 0.313% of the total concentrate mass. Partial dissolution of the Mg, assumed to be present predominantly as silicates, during the leaching process yielded a leach discharge solution containing 0.21 g/L Mg, representing a net dissolution of approximately 5%.

Further precipitation of Mg in the second manganese carbonate precipitation stage recovered magnesium carbonate to small amounts of MnCP2 product, resulting in a barren liquor of ~60 mg/L Mg. The multi-element scan analyses of two MnCP2 solids samples indicated an average Mg content of ~ 0.4%; this is equivalent to between 0.5 and 1% of the Mg in the concentrate.

Manganese

The concentrate used for the continuous testwork campaigns had a composition containing approximately 5.2% Mn. The Mn, primarily associated with the asbolane, leached to yield a leach discharge solution concentration of ~ 70 g/L Mn, equivalent to a net Mn dissolution of about 94%.

Unlike each of the preceding impurity removal and precipitation stages, the second stage manganese carbonate precipitation circuit has no downstream process for the removal of Mn from solution. Consequently, failure to operate this circuit within the correct design parameters may result in mother liquor concentrations discharged to the environment exceeding the target value of <5 mg/L Mn.

Sodium

While sodium makes up less than 0.05% in the concentrate feed to leach, the main source of Na within the Nkamouna circuit originates from reagents, most notably sodium carbonate and, to a lesser extent, sodium hydrosulfide.

Deportment of the barren liquor from manganese carbonate precipitation to the Glauber's salt crystallizer is expected to yield a Glauber's salt product with minimal impurities. Generation of Glauber's salt provides the main mechanism for the removal of Na from the Nkamouna flowsheet.

Environmental discharge of the excess Glauber's salt mother liquor will be approximately 14 g/L Na (44 g/L sodium sulfate).

Nickel

The concentrate used in the pilot testwork campaigns contained 1.00% Ni, and yielded an average leach discharge solution of ~9 g/L Ni; this equates to a net Ni dissolution of 61%.

Although subsequent partial neutralization (target pH 3.0) of the pregnant leach solution in the primary purification stage resulted in a slight reduction in the Ni concentration in the discharge solution, this may be attributed to dilution effects and there was little, if any, Ni precipitation within this circuit.

It is not expected that any Ni will report with the Glauber's salt product, and a final mother liquor concentration for environmental discharge will typically contain Ni at below analytical detection limits (0.1 mg/L). Therefore, as all of the Ni in the recycled secondary and tertiary purification precipitates is expected to re-dissolve in the leach, and the combined MSP and MnCP products contain approximately 60% of the Ni in the concentrate, it is evident that ~40% of the Ni in the concentrate reports to the CCD tailings storage facility.

Sulfur

Sulfur makes up less than 0.005% of the leach feed concentrate, and the vast majority is introduced into the Nkamouna flowsheet via reagent addition; pyrite (~40 kg/t of concentrate) and sulfuric acid (~180 kg/t of concentrate) addition to the leaching circuit; and sodium hydrosulfide (~17 kg/t concentrate) to the sulfide precipitation circuit. This results in elevated sodium sulfate concentrations throughout the leaching, purification, and precipitation circuits.

Generation of Glauber's salt from the manganese carbonate precipitation circuit barren liquor provides an additional mechanism for the removal of S from the Nkamouna flowsheet.

Environmental discharge of the excess Glauber's salt mother liquor is expected to be 10 g/L S (as a 44 g/L sodium sulfate solution).

Silicon

The concentrate used in the pilot testwork campaigns comprised 3.71% Si, primarily as quartz, while kaolinite and silicates of magnesium, sodium and aluminium make up the balance. Partial Si dissolution in the leach circuit produced a leach discharge solution containing an average 238 mg/L Si; this equates to a net Si dissolution of about 0.5%. The nominal solubility of Si or, more commonly silica in solution, under these conditions is about 200 mg/L SiO_2, i.e. about 90 mg/L Si. These solutions are, therefore, supersaturated with respect to Si.

The barren liquor from the MnCP circuit contained ~5 mg/L Si, equivalent to 0.03% of total Si in the concentrate feed. No additional removal of Si to the Glauber's salt product is anticipated. Therefore, as the MSP and MnCP products contain a combined 0.05% of the Si in concentrate, it is evident that >99.5% of the Si in concentrate reports to the CCD tailings storage facility.

Thorium

Partial dissolution of the trace quantities of Th (3 mg/kg), reported in the leach concentrate used for the pilot testwork campaigns, resulted in an average leach discharge solution of 0.25 mg/L Th; this represents a net Th dissolution of approximately 6%.

Subsequent primary purification resulted in the precipitation of an estimated 50% of the soluble Th, to 0.1 mg/L which, together with the un-leached Th, reports to the CCD tailings storage facility.

The residual Th concentration in the mother liquor for environmental discharge is expected to be less than detectable limits (<0.1 mg/L). Thus, as there is no detectable deportment of Th to any of the MSP, MnCP1, MnCP2, or Glauber's salt products, it is estimated that >99% of the Th in the concentrate will report to the CCD tailings storage facility.

Titanium

Significant quantities of titanium (c. 0.12%) were reported in the elemental analysis of the concentrate used as feed to the leach circuit for the pilot testwork program. Partial dissolution of this material yielded a leach discharge solution of 0.6 mg/L Ti. This is equivalent to a net Ti dissolution of <0.1%.

Concentration of Ti in the mother liquor for environmental discharge is expected to be less than detectable limits (<0.1 mg/L). As the MSP, MnCP1, MnCP2, and Glauber's salt products contain only about 0.1% of the Ti in concentrate, it is evident that c.99.9% of the Ti reports to the CCD tailings storage facility.

Uranium

Trace quantities of uranium within the concentrate used for the pilot testwork campaigns accounts for less than 2 mg/kg of the total concentrate mass. Partial dissolution of this material in the leaching circuit yielded a leach discharge solution containing 0.4 mg/L U; this is equivalent to a net U dissolution of approximately 20%.

The barren liquor from the second stage manganese carbonate precipitation circuit is expected to contain less than detectable concentrations of U (<0.1 mg/L), and there is expected to be no deportment of trace quantities of U from the barren solution to the Glauber's salt product. It is expected that most of the soluble U will report to the MnCP1 product while the majority of the U in concentrate reports to the CCD tailings storage facility.

Vanadium

Vanadium occurs in the Nkamouna ore in small quantities and comprised approximately 0.025% of the concentrate used as feed to the leach circuit for the pilot testwork campaign. Partial dissolution of this material yielded a leach discharge solution assaying approximately 8 mg/L V. This corresponds to a net V dissolution of approximately 2.3%.

Test results indicate that 40% of the leached V is precipitated in the primary purification circuit which, together with the un-leached V, reports to the CCD tailings storage facility. Thus, it is estimated that 98.5% of the total V in the concentrate reports to the CCD tailings storage facility.

Zinc

Zinc accounted for 0.053% of the mass of the concentrate used as feed to the leach circuit for the pilot testwork campaign. Partial dissolution of this material yielded an average leach discharge solution concentration of ~0.3 g/L Zn. This corresponds to a net Zn dissolution of approximately 40%.

The final barren liquor is expected to contain concentrations of Zn below analytical detection limits (<0.1 mg/L). The combined leach and primary purification residue accounts for approximately 60% of the Zn in the concentrate and reports to the CCD tailings storage facility, while the balance is predominantly recovered to the mixed sulfide product.

14.7.3 Summary

Based on an interpretation of the data from the continuous pilot plant campaigns conducted on the Nkamouna concentrate, an assessment has been made of the likely deportment of each of the elements solubilized during the leaching process.

A summary of the solution compositions that may be expected in the Nkamouna circuit is provided in Table 14.7.3.1. The data for the major elements has been taken from multiple profile data sets from the results of the campaign 3 and campaign 4 pilot testwork program, as supplied by Hazen Research. The minor elements are based on the multi-element analyses supplied by Huffman Laboratories. Note, therefore, that these are absolute values and they may vary considerably during commercial plant operation due, for example to variations in concentrate feed grade.

Expected solids compositions for the various precipitate and product streams are summarized in Table 14.7.3.2. Again these are calculated absolute values and will vary with changes in concentrate feed grade and composition. Nevertheless, they provide a good indication of the expected quality of the mixed sulfide and manganese carbonate products. (Note that deportment of impurity elements to the Glauber's salt product has been assumed to be below detection limits in all cases).

Finally, Table 14.7.3.3 provides a deportment summary of each element to the MSP, MnCP1, MnCP2, and Glauber's salt products. The table also provides information on the elemental deportment to the environmental discharge stream and, by difference, the elemental deportment to the CCD tailings storage facility.

Using Co as an example, the data shows that 94.8% of the Co in the feed solids of 1.03% dissolves in the leach, which results in the production of a subsequent mixed sulfide product containing 40% Co, representing 93.5% of the total Co in the concentrate feed to leach. Residual trace quantities of Co in solution after the sulfide precipitation circuit are precipitated with the

MnCP1 and MnCP2 products, representing respectively 0.002% and 0.0005% of the total Co in the feed concentrate to leach. There is no deportment of Co to the Glauber's salt product, and the estimated 0.0001 g/L Co in the environmental discharge liquor represents approximately 0.001% of the total Co in the concentrate feed to leach. Therefore, an estimated 6.48% of the total Co in the concentrate feed to leach reports to the CCD tailings storage facility – the CCD tailings solids are expected to contain roughly 0.056% Co.

14.7.4 Conclusion

Despite the limitations of this elemental deportment review, interpretation of the analytical data from the pilot testwork campaigns has enabled an estimate to be made of the likely composition of the various products (MSP, MnCP1, MnCP2, and Glauber's salt) and of the tailings residue. In addition the composition of the final effluent stream for environmental discharge has also been estimated.

Based on interpretation of the available data, obtained from leaching and impurity removal testwork of a representative concentrate sample, there appears to be no significant elemental accumulation within the Nkamouna circuit that could adversely affect product quality, and the quality of the final effluent stream.

The SysCAD mass and energy balance, as used for the design of the Nkamouna processing facility, has been adapted to assist with the elemental deportment evaluation and supports the conclusion that there appears to be insignificant build-up of elements within the circuit.

Table 14.7.3.1: Expected Solution Composition

	Leach Feed	Leach Discharge	Primary Purification Discharge	Secondary Purification Discharge	Sulphide Precipitation Discharge	Tertiary Purification Discharge	Manganese Carbonate Precipitation 1 Discharge	Manganese Carbonate Precipitation 2 Discharge
	Solids, % of concentrate	Solution, mg/L	Solution, mg/L	Solution, mg/L	Solution, mg/L	Solution, mg/L	Solution, mg/L	Solution, mg/L
Al	7.480	5,730	386	55	35	<20	<2	<0.5
As	0.0023	0.45	0.43	0.2	<0.1	<0.1	<0.1	<0.1
Ca	0.007	96	81	41	42	42	17	8
Ce	0.062	212	115	19	19	10	<0.1	<0.1
Co	1.030	13,900	11,600	5,800	<5	<2	<0.1	<0.1
Cr	1.250	23	<0.5	<0.1	<0.1	<0.1	<0.1	<0.1
Cu	0.044	243	84	44	<0.1	<0.1	<0.1	<0.1
Fe	36.50	350	43	13	7	7	<0.1	<0.1
Li	0.011	123	111	58	59	58	27	23
Mg	0.313	209	172	102	100	105	82	63
Mn	5.210	69,800	56,900	29,700	28,000	28,600	336	<10
Na	0.072	16,400	22,600	25,200	31,600	32,800	43,200	45,500
Ni	1.00	8,690	7,020	3,660	9	5	2	<0.1
S	1.960	79,000	60,100	40,800	40,700	41,300	31,600	35,200
Si	3.71	238	155	24	24	18	10	6
Th	0.0003	0.25	0.1	<0.1	<0.1	<0.1	<0.1	<0.1
Ti	0.119	0.60	0.4	0.1	<0.1	<0.1	<0.1	<0.1
U	0.0002	0.4	0.4	0.2	0.2*	0.2*	0.1	<0.1
V	0.025	7.9	4.3	2.2	2.0	2.0	<0.1	<0.1
Zn	0.053	293	216	117	<0.2	<0.1	<0.1	<0.1

Table 14.7.3.2: Expected Solids Composition

		Leach Feed Solids	Primary Purification Precipitate	Secondary Purification Precipitate	Mixed Sulphide Precipitate	Tertiary Purification Precipitate	Manganese Carbonate Precipitate Stage 1	Manganese Carbonate Precipitate Stage 2	Glauber's Salt	CCD Tailings Solids
Al	**%**	7.48	7.65	5.58	0.032	2.87	0.02	0.025	<0.0001	7.35
As	**%**	0.002	0.002	<0.001	0.0007	<0.0001	<0.0001	<0.0001	<0.0001	0.002
Ca	**%**	0.007	0.006	<0.001	0.002	<0.0001	0.077	0.256	<0.0001	0.0002
Ce	**%**	0.062	0.016	<0.001	<0.0005	<0.0001	0.025	<0.0005	<0.0001	0.052
Co	**%**	1.03	0.07	7.89	40.0	4.98	<0.002	<0.0005	<0.0001	0.056
Cr	**%**	1.25	1.33	0.06	0.003	<0.001	<0.0005	<0.0005	<0.0001	1.23
Cu	**%**	0.04	0.04	<0.001	0.20	0.003	<0.0005	<0.0005	<0.0001	0.043
Fe	**%**	36.5	38.0	0.24	<0.05	0.73	<0.02	0.025	<0.0001	37.6
Li	**%**	0.011	0.002	0.25	<0.001	<0.0001	0.020	<0.0001	<0.0001	0.002
Mg	**%**	0.31	0.31	<0.0005	<0.001	<0.001	0.040	0.40	<0.0001	0.30
Mn	**%**	5.21	0.32	28.2	0.12	47.1	46.0	42.6	<0.0001	0.44
Na	**%**	0.072	0.803	<0.001	0.044	<0.001	1.27	2.19	14.29	0.94
Ni	**%**	1.00	0.41	3.62	23.4	2.91	<0.01	<0.01	<0.0001	0.38
S	**%**	1.96	3.71	2.09	36.4	31.9	0.66	1.12	9.94	3.88
Si	**%**	3.71	3.74	6.78	0.03	1.52	0.01	0.10	<0.0001	3.72
Th	**%**	0.0003	0.0003	0.006	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	0.0003
Ti	**%**	0.119	0.118	0.008	0.001	<0.0001	< 0.001	<0.0017	<0.0001	0.117
U	**%**	0.0002	0.0001	0.000	<0.0001	<0.0001	<0.0005	<0.005	<0.0001	0.0001
V	**%**	0.025	0.024	0.017	0.001	0.003	0.003	0.003	<0.0001	0.025
Zn	**%**	0.053	0.035	0.000	0.841	0.071	<0.0005	<0.0008	<0.0001	0.035

Table 14.7.3.3: Elemental Deportment Summary

	Leach		MSP		MnCP 1		MnCP 2		Glauber's Salt		Environmental Discharge		CCD Tailings Storage	
	Feed Solids % w/w	Net Dissolution, %	Solids Composition, % w/w	% of Element in Concentrate	Solids Composition, % w/w	% of Element in Concentrate	Solids Composition, % w/w	% of Element in Concentrate	Solids Composition, % w/w	% of Element in Concentrate	Solution, g/L	% of Element in Concentrate	% of Element in Concentrate	Tailings Solids % w/w
Al	7.480	5.4%	0.032	0.01%	0.020	0.03%	0.025	0.00%	0.000	0.0%	0.0005	0.00%	99.96%	7.35
As	0.0023	1.4%	0.0007	0.78%	0.0001	0.43%	0.0001	0.02%	0.000	0.0%	0.0001	0.27%	98.49%	0.0015
Ca	0.007	96.3%	0.002	0.73%	0.077	70.00%	0.256	15.00%	0.000	0.0%	0.0075	6.70%	7.57%	0.0002
Ce	0.062	24.0%	0.0005	0.02%	0.025	4.01%	0.0005	0.00%	0.000	0.0%	0.0001	0.01%	95.95%	0.052
Co	1.030	94.8%	40.0	93.50%	0.002	0.02%	0.0005	0.00%	0.000	0.0%	0.0001	0.001%	6.48%	0.056
Cr	1.250	0.1%	0.003	0.01%	0.001	0.00%	0.0005	0.00%	0.000	0.0%	0.0001	0.001%	99.99%	1.229
Cu	0.044	38.8%	0.20	11.66%	0.001	0.11%	0.0005	0.01%	0.000	0.0%	0.0001	0.01%	88.21%	0.043
Fe	36.50	0.1%	0.05	0.00%	0.020	0.01%	0.025	0.00%	0.000	0.0%	0.0001	0.00002%	99.99%	37.59
Li	0.011	90.3%	0.001	0.23%	0.020	17.94%	0.011	0.51%	0.000	0.0%	0.084	47.32%	33.99%	0.002
Mg	0.313	4.7%	0.001	0.01%	0.040	1.27%	0.40	0.68%	0.000	0.0%	0.0630	1.26%	96.78%	0.30
Mn	5.210	94.1%	0.12	0.06%	46.0	87.90%	42.6	4.38%	0.000	0.0%	0.0010	0.001%	7.67%	0.44
Na	0.072	N/A	0.044	N/A	1.27	N/A	2.19	N/A	14.29	N/A	14.32	N/A	N/A	0.94
Ni	1.00	61.0%	23.4	60.02%	0.01	0.10%	0.010	0.01%	0.000	0.0%	0.0001	0.00%	39.87%	0.38
S	1.960	N/A	36.4	N/A	0.66	N/A	1.12	N/A	9.94	N/A	10.09	N/A	N/A	3.88
Si	3.71	0.4%	0.03	0.02%	0.01	0.03%	0.1000	0.01%	0.000	0.0%	0.0130	0.02%	99.92%	3.72
Th	0.0003	5.9%	0.00001	0.09%	0.00001	0.33%	0.0001	0.18%	0.000	0.0%	0.0000	0.21%	99.20%	0.0003
Ti	0.1190	0.04%	0.0011	0.02%	0.001	0.08%	0.0017	0.01%	0.000	0.0%	0.0001	0.01%	99.88%	0.117
U	0.0002	18.7%	0.00001	0.17%	0.0002	13.27%	0.0001	0.36%	0.000	0.0%	0.0000	0.42%	85.78%	0.0001
V	0.025	2.3%	0.001	0.10%	0.003	1.22%	0.0028	0.06%	0.000	0.0%	0.0001	0.03%	98.59%	0.025
Zn	0.053	40.3%	0.841	38.21%	0.001	0.09%	0.0008	0.01%	0.000	0.0%	0.0001	0.01%	61.68%	0.035

14.8 Metallurgical Design Parameters

14.8.1 Treatment Flowsheet

The metallurgical flowsheet selections are based on the results from the testwork and more specifically from the pilot plant campaigns. The flowsheet configuration is summarized as follows:

- Primary crushing;
- Paddle mixer for slurrying;
- Attritioning;
- Screw classification;
- Pug concentrate milling;
- Leaching;
- Primary purification;
- Counter current decantation;
- Secondary purification;
- Mixed sulfide precipitation;
- Mixed sulfide precipitation solid liquid separation;
- Mixed sulfide product bagging;
- Manganese carbonate precipitation 1;
- Manganese carbonate precipitation 1 solid liquid separation;
- Manganese carbonate precipitation 1 product bagging;
- Manganese carbonate precipitation 2;
- Manganese carbonate precipitation 2 solid liquid separation; and
- Glauber's salt crystallization.

14.8.2 Key Process Design Parameters

The key process design parameters are based on the results from the testwork and more specifically from the pilot plant campaigns. The key process design parameters that were derived from the testwork are summarized in Table 14.8.2.1.

Table 14.8.2.1: Key Process Design Parameters as Derived From Testwork

Parameter	Value	Unit
PUG Circuit		
Paddle mixer discharge density	50	% solids w/w
Attritioning residence time	6	minutes
Attritioning feed density	35	% solids w/w
Screw classification separation size	300	µm
Leaching		
Sulfuric acid addition	170	kg/t concentrate
Pyrite addition	40	kg/t concentrate
Pyrite size – P_{80}	<15	µm
Temperature	95	ºC
Concentrate size – P_{80}	106	µm
Leach density	50	% solids
Leach residence time	14	h
Primary Purification		
Residence time	6	h
Discharge pH	3	pH
pH modifier	Soda ash	
Temperature	95	ºC
Slurry density	50	% solids
Secondary Purification		
Residence time	2	h
Discharge pH	4.3	pH
pH modifier	Soda ash	
Temperature	60	ºC
Mixed Sulfide Precipitation		
Residence time	1	h
Discharge pH	3	pH
pH modifier	Soda ash	
Temperature	80	ºC
Precipitation agent	Sodium hydrosulfide	
Excess precipitation agent	15	% of stoichiometric
Seed recycle	300	%
Tertiary Purification		
Residence time	1.5	h
Discharge pH	5.5	pH
pH modifier	Soda ash	
Temperature	60	ºC
Manganese Carbonate Precipitation 1		
Residence time	1.5	h
pH modifier	Soda ash	
Temperature	80	ºC
Precipitation agent	Soda ash	
Precipitation agent addition	95	% of stoichiometric
Seed recycle	300	%
Manganese Carbonate Precipitation 2		
Residence time	1.5	h
Discharge pH	8.7	pH
pH modifier	Soda ash	
Temperature	80	ºC
Precipitation agent	Soda ash	
Seed recycle	300	%

14.8.3 Reagent Consumption

Reagents consumptions for the PUG concentrate grade tested are summarized in Table 14.8.3.1. It is noted that the reagent consumption is mostly underpinned by the gangue or impurity elements in the PUG concentrate and not by the cobalt.

Reagent consumptions are based on the average plant feed rate for the first 5 years of operation. The reagent consumptions are based on a PUG circuit feed rate of 8,500 t/d resulting in 2,000 t/d of concentrate. The leach and recovery circuit reagent consumptions are based on 2,000 t/d at the PUG concentrate grades represented in Table 14.8.3.2. Deviations in concentrate grade will lead to changes in reagent consumptions; for example reagent consumptions will increase with an increase in iron and aluminium.

Table 14.8.3.1: Reagent Consumptions as Derived from the Testwork

Reagent	Area of use	Unit	Consumption
Sulfuric acid	Leach	kg/t concentrate	170
Pyrite	Leach	kg/t concentrate	40
Sodium Hydroxide	Sulfide area scrubber	kg/t concentrate	1
Dense Soda Ash		kg/t concentrate	125
	Primary Purification	kg/t concentrate	22
	Secondary Purification	kg/t concentrate	5
	Sulfide pH control	kg/t concentrate	12
	Tertiary Purification	kg/t concentrate	2
	Manganese Carbonate Precipitation 1	kg/t concentrate	80
	Manganese Carbonate Precipitation 2	kg/t concentrate	5
Sodium Hydrosulfide (70%)		kg/t concentrate	20
Flocculant	Counter Current Decantation	g/t concentrate	505
	Secondary Purification	g/t concentrate	2
	Sulfide Precipitation	g/t concentrate	1
	Tertiary Purification	g/t concentrate	3
	Manganese Carbonate Precipitation 1	g/t concentrate	6
	Manganese Carbonate Precipitation 2	g/t concentrate	3

Table 14.8.3.2: Concentrate Composition

Constituent	Concentration, % w/w
Cobalt - Co	0.9
Nickel - Ni	0.8
Manganese - Mn	4.8
Iron - Fe	35.6
Aluminium - Al	8.7
Copper - Cu	0.1
Zinc - Zn	0.1

14.9 Overall Metallurgical Recoveries

The overall recoveries of the valuable metals in the circuit for the average PUG concentrate with an average cobalt grade of 0.85%, nickel grade of 0.80 and a manganese grade of 4.7% are summarized in Table 14.9.1. This recovery is incorporating the PUG circuit recovery as well as the leach and recovery circuit recovery. It should be noted that variations in PUG factor vary the

overall recovery according to mine head grade. Recoveries stated in the mineral reserve statement are based on life of mine recoveries and not those of steady state analysis presented here.

Table 14.9.1: Overall Recovery of Valuable Metals at Stated Grade Input

Metal	Recovery, %
Cobalt	50.8
Nickel	15.3
Manganese	46.0

14.9.1 PUG Recovery

The PUG circuit recoveries for the valuable metals over the Life of Mine are summarized in Table 14.9.1.1.

Table 14.9.1.1: PUG Circuit Life Of Mine Weighted Average Recovery

Metal	Weighted Average Recovery, %
Cobalt	54.4
Nickel	25.3
Manganese	53.5

14.9.2 Leach Cobalt Recovery

The cobalt recovery through the leach and recovery circuit, based on an average cobalt grade of 0.85%, is summarized in Table 14.9.2.1 and Figure 14-7.

Table 14.9.2.1: Cobalt Recovery in the Leach and Recovery Circuit at Stated Grade

Section	Recovery, %
Leach	94.8
Primary Purification	99.7
Counter Current Decantation	98.7

From this analysis it can be seen that the primary loss of leached cobalt occurs via the counter current decantation circuit.

14.9.3 Leach Nickel Recovery

The nickel recovery through the leach and recovery circuit, based on an average nickel grade of 0.80%, is summarized in Table 14.9.3.1 and Figure 14-8.

Table 14.9.3.1: Nickel Recovery in the Leach and Recovery Circuit at Stated Grade

Section	Recovery, %
Leach	61.5
Primary Purification	99.7
Counter current decantation	98.7

14.9.4 Leach Manganese Recovery

The manganese recovery through the leach and recovery circuit, based on an average manganese grade of 4.7%, is summarized in Table 14.9.4.1 and Figure 14-9.

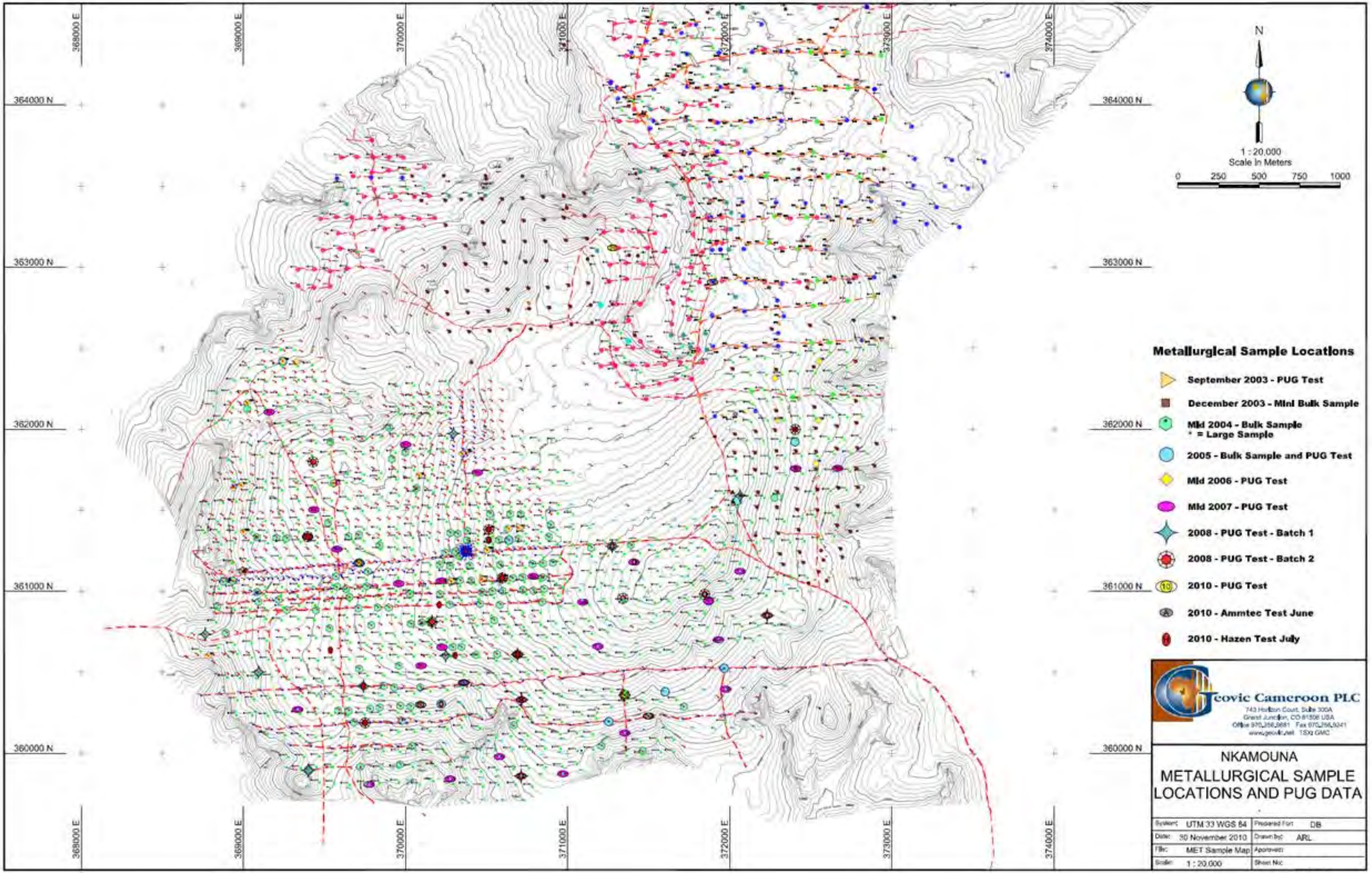
Table 14.9.4.1: Manganese Recovery in the Leach and Recovery Circuit at Stated Grade

Section	Recovery, %
Leach	92.1
Primary Purification	99.7
Counter current decantation	98.7
Manganese carbonate precipitation 1	95
Manganese carbonate precipitation 2	91

14.10 Test Work Conclusions

Extensive small and large scale testwork has been conducted at reputable testwork laboratories over the duration of the project to date, culminating in the completion of multiple pilot plant testwork programs to demonstrate the operation of the selected process flowsheet and provide reagent consumption data for process design requirements. The results obtained from the pilot plant testwork programs demonstrate the robustness of the process flowsheet selected.

The pilot plant testwork programs have been conducted by competent and internationally renown laboratories to a standard suitable for incorporation into the level of Feasibility Study conducted to determine the process design, as well as capital and operating costs presented in this report.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N
1 : 20,000
Scale In Meters
0 250 500 750 1000
Metallurgical Sample Locations
September 2003 - PUG Test
December 2003 - Mini Bulk Sample
Mid 2004 - Bulk Sample
* = Large Sample
2005 - Bulk Sample and PUG Test
Mid 2006 - PUG Test
Mid 2007 - PUG Test
2008 - PUG Test - Batch 1
2008 - PUG Test - Batch 2
2010 - PUG Test
2010 - Ammtec Test June
2010 - Hazen Test July
Geovic Cameroon PLC
743 Horizon Court, Suite 300A
Grand Junction, CO 81506 USA
Office 970.256.9661 Fax 970.256.9241
www.geovic.net (Sq GMC
NKAMOUNA
METALLURGICAL SAMPLE LOCATIONS AND PUG DATA
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Prepared For: DB
Date: 30 November 2010
Drawn by: ARL
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Approved:
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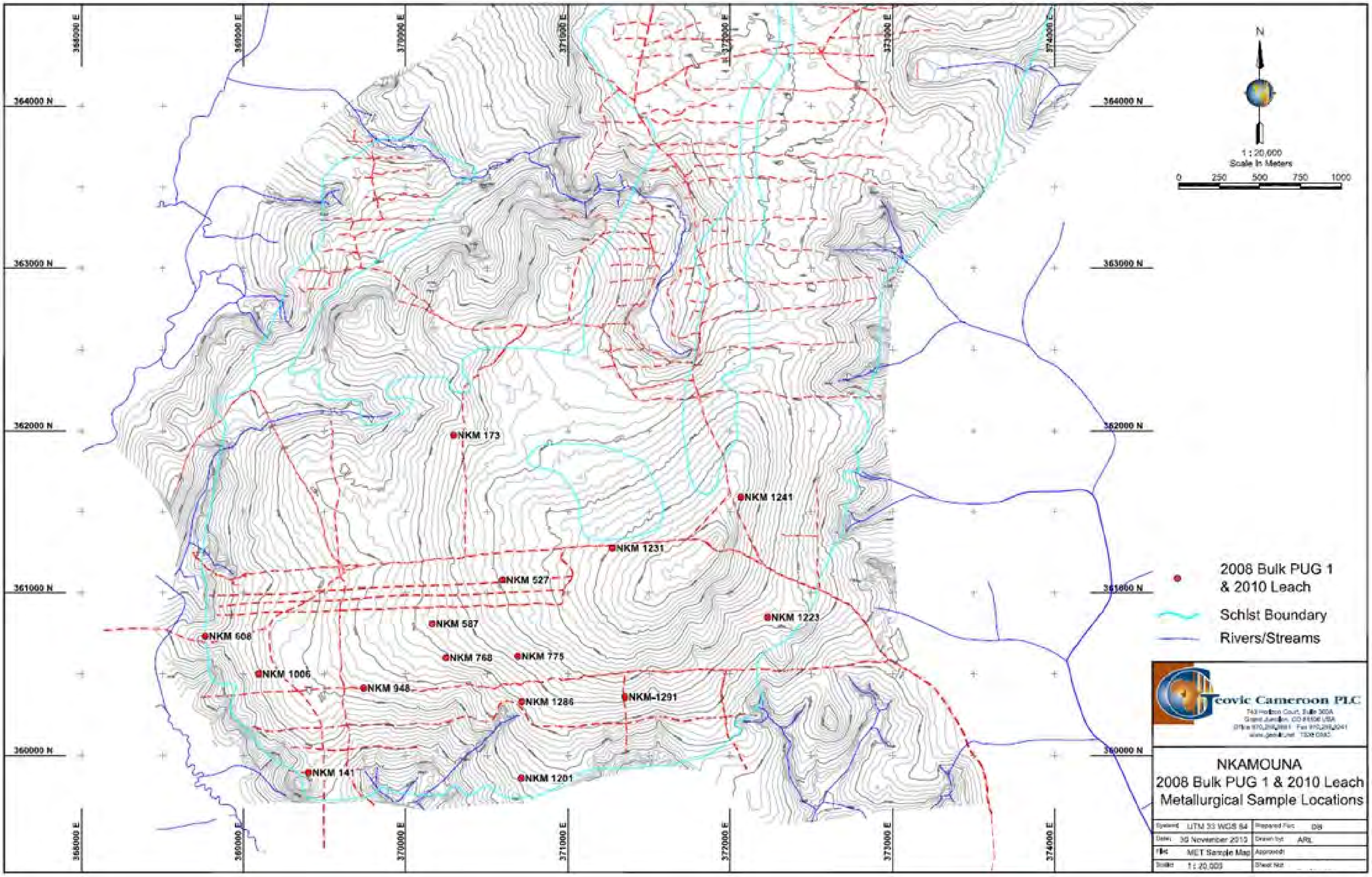
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Scale In Meters
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NKM 1241
NKM 1231
NKM 527
NKM 587
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NKM 775
NKM 1006
NKM 948
NKM 1286
NKM-1291
NKM 141
NKM 1201
2008 Bulk PUG 1 & 2010 Leach
Schist Boundary
Rivers/Streams
Geovic Cameroon PLC
NKAMOUNA
2008 Bulk PUG 1 & 2010 Leach
Metallurgical Sample Locations
Datum: UTM 33 WGS 84
Prepared For: DB
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1 : 20,000
Scale In Meters
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NKM 1373
NKM 1397

2008 Bulk PUG 2 & 2010 Leach
Schist Boundary
Rivers/Streams

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www.geovic.net TSXV GMC

NKAMOUNA
2008 Bulk PUG 2 & 2010 Leach Metallurgical Sample Locations

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Date:	30 November 2010	Drawn by:	ARL
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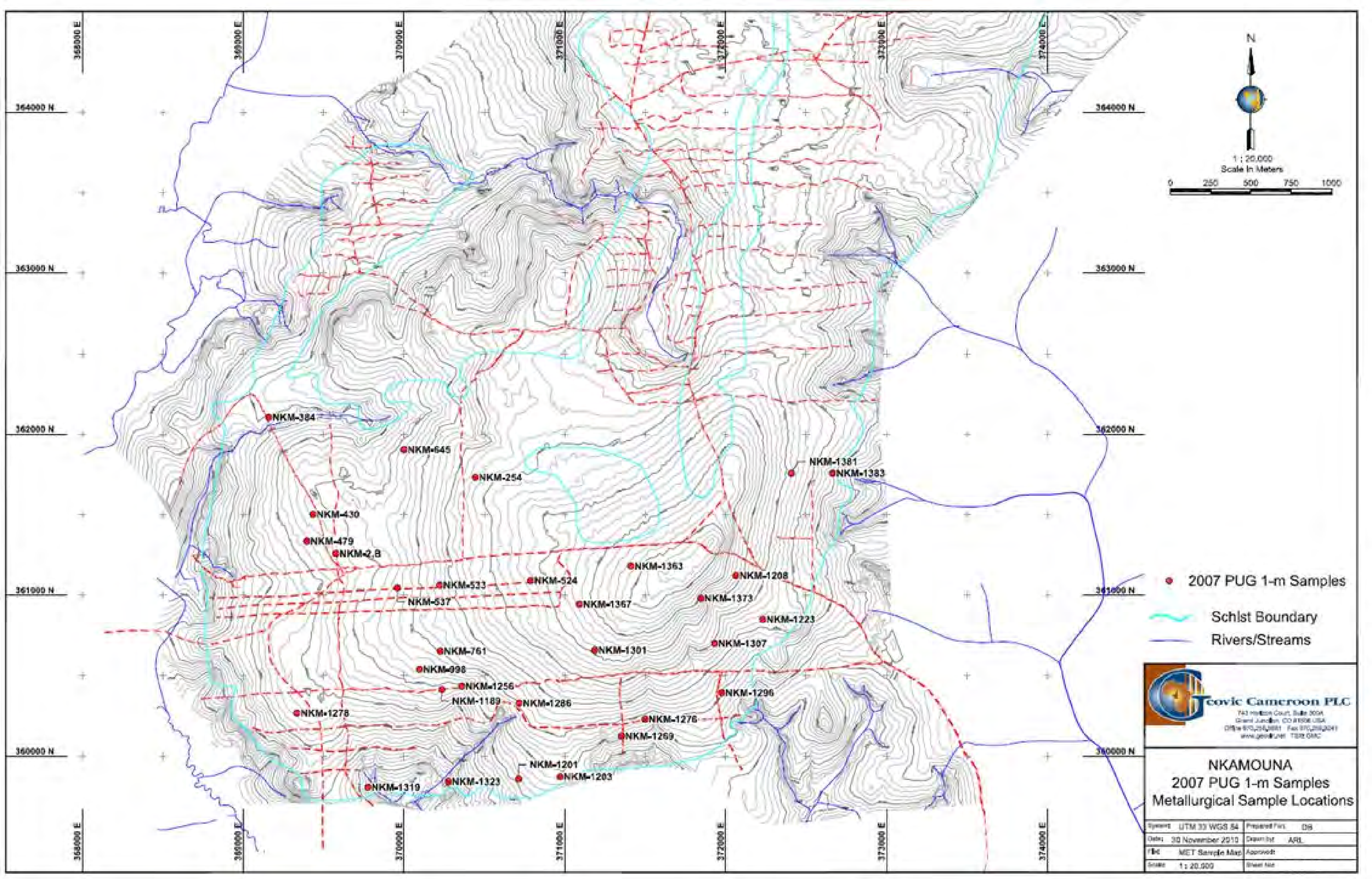
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2007 PUG 1-m Samples
Schist Boundary
Rivers/Streams
Geovic Cameroon PLC
NKAMOUNA
2007 PUG 1-m Samples
Metallurgical Sample Locations
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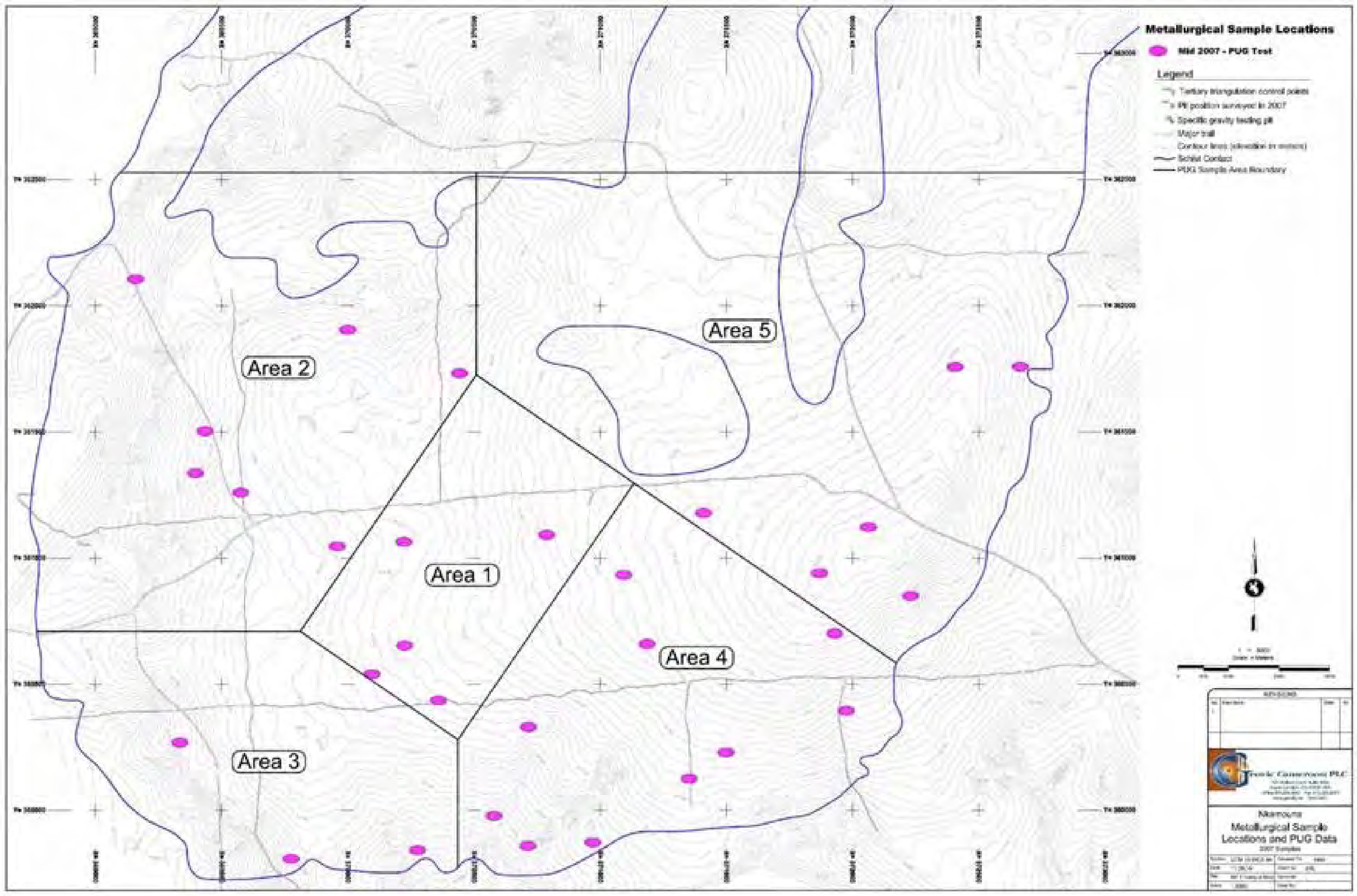
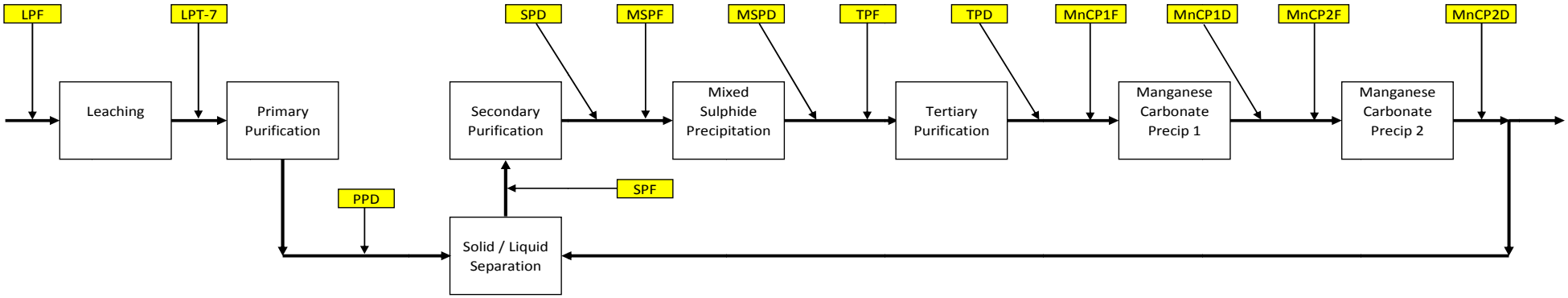

Area 1
Area 2
Area 3
Area 4
Area 5
Metallurgical Sample Locations
Mbl 2007 - PUG Test
Legend
Tertiary triangulation control points
Pit position surveyed in 2007
Specific gravity testing pit
Major trail
Contour lines (elevation in metres)
Schist Contact
PUG Sample Area Boundary
Namoura
Metallurgical Sample Locations and PUG Data
2007 Samples



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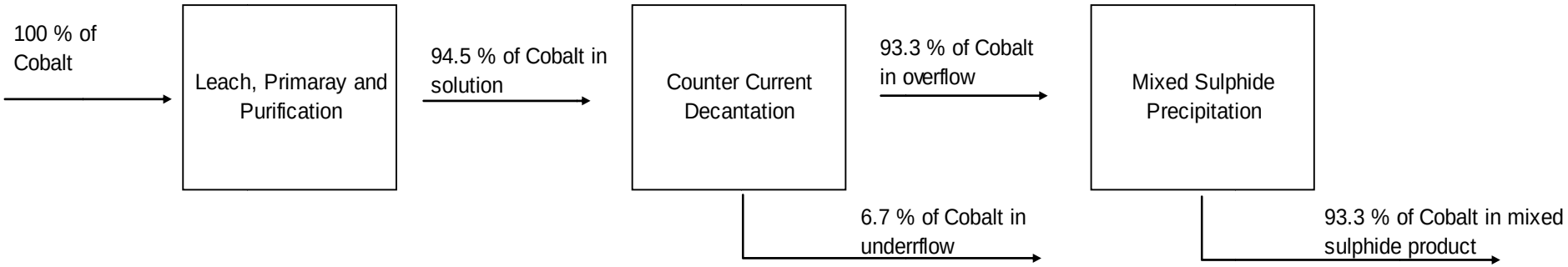
SRK Job No.: 195100.020

File Name: Figure 14-6.docx

Nkamouna and Mada Deposits
East Province of Cameroon, Africa

Sampling Block Flow Diagram

Date: 04/01/2011 | Approved: BC | **Figure: 14-6**



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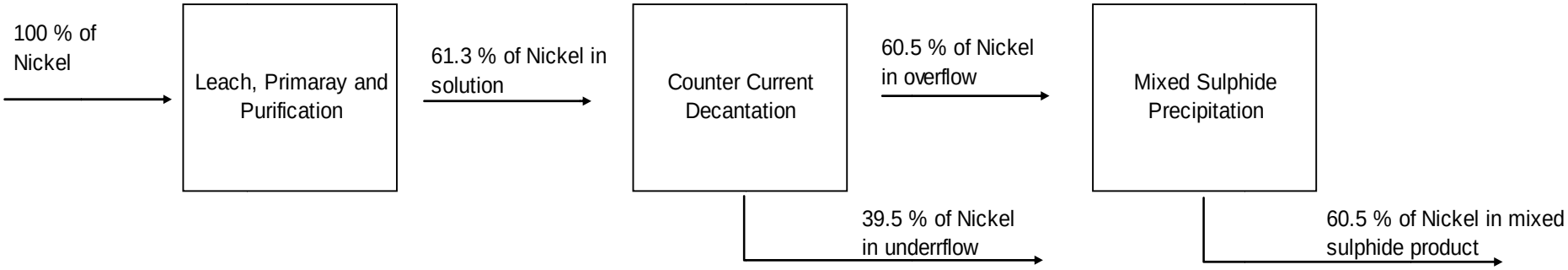
SRK Job No.: 195100.020		
File Name: Figure 14-7.docx		Date: 04/01/2011 Approved: BC **Figure: 14-7**



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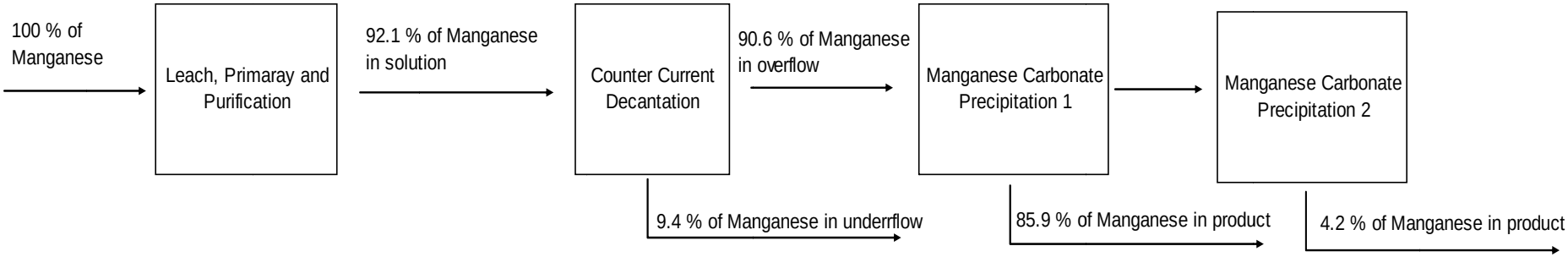
SRK Job No.: 195100.020

File Name: Figure 14-8.docx

Nkamouna and Mada Deposits
East Province of Cameroon, Africa

Nickel Recovery in the Leach and Recovery Circuit

Date: 04/01/2011 | Approved: BC | **Figure: 14-8**



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15 Mineral Resources and Mineral Reserve Estimates (Item 19)

15.1 Resource Estimation

15.1.1 Mineral Resource Model

The mineral resource estimate was prepared by Alan Noble, P.E., independent consultant, under the direction of Brian Briggs, P.E., Geovic's Manager of Technical Services. The mineral resource estimate was prepared using three-dimensional block models to estimate cobalt, nickel and manganese grades for individual 10 x 10 m horizontal by 1 m vertical blocks for Nkamouna and 20 x 20 m horizontal by 1 m vertical blocks at Mada. In addition, lithology and resource classification codes were assigned to each block. This estimate is an update to the feasibility study Nkamouna resource model and the 2007 Mada resource model using additional data from the 2007 to 2009 drill programs conducted at Nkamouna and Mada. The updated resource estimation was conducted using Datamine Studio 3.0 geologic modeling software and the methodology remains essentially the same to that utilized in prior resource estimates. No resource estimate was conducted at Rapodjombo, as drilling is considered insufficient for classification of resources.

Because the Nkamouna and Mada deposits are thin (averaging 13.5 m including mineralized material and overburden) compared to the horizontal extent (over 5,000 m at Nkamouna and over 10,000 m at Mada), a 10X to 30X vertical exaggeration is required to view the entire deposit in cross-section. With greater than 10X vertical exaggeration, however, cross-sections become unacceptably tall, so basic interpretation and modeling were conducted using a flattened coordinate system. In this flattened coordinate system, an elevation of zero is the topographic surface and an elevation of -10 is 10 m below existing topography.

The general procedure used for resource estimation was as follows:

- Sampling data were processed to average multiple assays that were conducted for some intervals. Multiple samples on different sides of pits were averaged to create a single averaged assay for the pit. Data from 2005, 2007 and 2009 database were merged into a single file;
- The raw data were filtered to remove redundant points and control lines were added to improve the topographic model for Mada. A triangulated digital terrain model (DTM) was created using the thinned topographic data and control lines. Triangulated DTM models were created to represent the depth from surface to the bottom of the granular (GR), breccia (BX) and ferralite (FL) zones. For example, the bottom of FL / top of rock was defined by locating all but the uppermost saprolite / serpentinite / schist sample in each pit / hole. A DTM was then formed using the elevations of the top of those samples. The DTM was limited to the area inside the schist boundary, which was assigned an elevation of zero. Since many of the pits / holes are not deep enough to intersect the bottom of FL / top of rock, some FL samples were below the DTM;
- The initial DTM was updated by locating the lowest point in each pit / hole that contained GR, BX, or FL. Those data were compared to the DTM surface and GR-BX-FL bottom points below the DTM surface were added to the top-of-rock point data. The DTM

surface was re-triangulated with the updated data. The search for GR+BX+FL points below the DTM surface was repeated twice until no GR+BX+FL points were below the bottom of FL / top of rock. The same procedure was used to define the top of FL / bottom of BX and the top of BX / bottom of GR;

- Based on cross-section plots of cobalt grade in the flattened model, it was observed that there was generally a sharp contact between low-grade and high-grade cobalt grade at the top of mineralization. The top 2 m of mineralization generally contain the highest cobalt grades in the deposit;
- While the top of mineralization appears to be continuous, it is slightly irregular and locally correlates sub-parallel to the topographic surface, the breccia / ferralite contact and/or the bottom of the ferralite. A top-of-mineralization, or 'TOPMIN' model was created to remove the irregularities in the top-of-mineralization by moving the collar elevations of pits / holes up until the top of mineralization was at an elevation of zero. The advantage of this model is that the optimum correlation between the metal grades is horizontal and the shape and continuity of the mineralization can be viewed directly on plan maps. This model also went through several iterations of editing and remodeling to remove inconsistencies in the data from shallow holes that did not penetrate the top of mineralization and from multiple pits / holes within a few meters of each other;
- Basic statistics, using the TOPMIN coordinate system, showed that there are three cobalt grade populations, including low-grade (poorly mineralized), mid-grade (mineralized) and high-grade (strongly mineralized). Manganese was found to have grade distributions similar in shape, but higher grade than cobalt, consistent with the strong correlation between cobalt, manganese and abundance of asbolane. Nickel appears to be much more evenly distributed than cobalt and manganese and was found to only have two grade zones, low-grade, and high-grade;
- Pits / holes were composited to 1 m down-hole intervals and grade zones were assigned to composites using cut-off grades. The composite grade zones were refined using a jack-knifing procedure to adjust grade zones relative to the grades of nearby pits / holes. Grade zones were assigned to the model using nearest-neighbor assignment from the composite grade zones;
- Basic statistics were conducted within the grade zones to confirm the grade distributions and variograms were run to confirm continuity of grades within the zones;
- Block grades were estimated for cobalt, nickel, and manganese using inverse-distance-power (IDP) estimation with grade-zoning controls. IDP estimation parameters were adjusted so the estimated block distributions adequately reflected mining selectivity. Nearest neighbor grades were also estimated, in order to provide a comparative model used to validate the IDP grades; and
- A sample spacing model was prepared in TOPMIN model coordinates that measured the spacing of samples around each block. This model was used to classify the resources into measured, indicated, and inferred resource classes based on pit / hole spacing.

15.1.2 Modeling Coordinate System

The models were constructed using UTM Zone 33/WGS84 coordinates. The Datamine project used double precision data files. Extended precision was used to provide better computing precision and to eliminate problems from the limited range of the Datamine IJK parameter.

15.1.3 Block-Model Location and Size Parameters

A single block model was used for resource estimation for each model area, but the models contain Z-coordinates for viewing and manipulation of the model in the flat coordinate system, the TOPMIN coordinate system, and true UTM coordinates. It is relatively simple to convert the model from one type to the other by replacing the elevation of the block center (ZC) with the Z-coordinate from the desired coordinate system. Each XY location in the model was defined as a single parent block where the thickness of the block is equal to the thickness of the model. Individual blocks were created as 1 m high Datamine sub-blocks. This implementation of parent blocks / sub-blocks is required because the 1 m high sub-blocks do not coincide with regular boundaries in the UTM coordinates.

The true-elevation coordinate space is simply the true elevation of the block centroid. The flat-model coordinate space defines elevation zero (0) as the current topographic surface. Thus, an elevation of minus 12 m (-12 m) in the flat-model system is equivalent to 12 m below surface. The TOPMIN model coordinate space defines zero elevation as the top of mineralization. Thus, the top of mineralization, which is an irregular, undulating surface in flat and true elevation coordinates, is a flat, horizontal plane in the TOPMIN coordinates. The TOPMIN model may be visualized as a seam model with sub-blocks running parallel to the top of mineralization. The size and location parameters for the resource model are shown in Table 15.1.3.1.

Table 15.1.3.1: Block Model Size and Location Parameters

	Number Blocks	Block Size (m)	Minimum Value (m)	Maximum Value (m)	Length (m)
Nkamouna					
Easting (Columns)	550	10	368,600	374,100	5,500
Northing (Rows)	490	10	359,600	364,500	4,900
UTM Elevation(Levels)	200 (Sub-blocks)	200	580	780	200
Mada					
Easting (Columns)	440	20	366,100	374,900	8,800
Northing (Rows)	540	20	364,500	375,300	10,800
UTM Elevation(Levels)	300 (Sub-blocks)	300	500	800	300

15.1.4 Pit and Drill Hole Data

Drill hole and data were provided by Geovic as EXCEL files containing collar coordinates, lithologic codes, and assay data. Data for Nkamouna for the 2005 and 2007 periods was used from the ORE project archives. New Nkamouna data files for the 2009 data were provided by Geovic. A new EXCEL file was provided for the Mada area that was complete through 2009 and the previous data from 2005 was not used for Mada. Each of these files was edited slightly for data checking and/or to facilitate later use of the data. In addition, some pits / holes were

renamed by Geovic, and the 2005 - 2007 data were edited for those changes. The resulting data were imported into Datamine using ODBC v2 data import drivers.

Collar Data

Collar data included a unique database ID number, an area code (NKM, KON), sample type (Pit, UN Core Hole, Core Hole, Reverse Circulation, SG sample, bulk sample) and collar coordinates (Easting, Northing, and Elevation). The 2008-2009 Nkamouna collar coordinates, which have been recently resurveyed, were used except for a few 2005 holes, which were missing collar coordinates in the 2009 data. A minor difference between the 2005 and 2009 collar coordinates is that the 2008 - 2009 coordinates were measured at the center of pits, while the 2005 coordinates were measured at the location of the sample point on the pit. 2009 collar coordinate data were used for Mada.

Some pits / holes were not used in previous estimates including extremely shallow pits, pits / holes twinned with other data, and samples on multiple sides of pits. The only data not used for this estimate were the UN core holes, because they are considered unreliable and holes with unsurveyed locations.

Summaries of the number of pits / holes and meters used for this estimate shown in Table 15.1.4.1.

Table 15.1.4.1: Summary of Samples Used for Resource Estimation

Area	Sample Type	Number of Samples	Meters Sampled	Average Length (meters)
Nkamouna	Core	502	591.40	1.18
	Pit	12,603	10,894.12	0.86
	Reverse Circulation	2,983	2,727.50	0.91
	Drill Hole	20,906	20,664.80	0.99
	Test Pit	327	292.70	0.90
	Total Meters Sampled	**37,321**	**35,170.52**	**0.94**
Mada	Pit	4,082	3,716.76	0.91
	Reverse Circulation	16,356	16,178.60	0.99
	Total Meters Sampled	**20,438**	**19,895.36**	**0.97**

Lithologic Data

The lithologic data for the 2005 data is in a separate EXCEL worksheet that includes drill hole name (BHID), interval depths (FROM, TO) and alphanumeric lithologic code. Lithologic coding for the 2007 and 2009 data is included in the assay data and each assay interval is assigned an alphanumeric lithologic code. A summary of lithologic codes is shown in Table 15.1.4.2.

Table 15.1.4.2: Summary of Lithologic Codes Used For Estimation

			Nkamouna			Mada		
Lith Group	Lith Code	Lith Description	Total Meters	Number Assayed Intervals	Average %Co	Total Meters	Number Assayed Intervals	Average %Co
1 Granular)	GR	Granular	8,057.5	934.0	0.031	4,879.5	488.0	0.009
2 (Breccia)	UB	Upper Breccia	3,357.2	1,981.0	0.044	2,674.5	1,968.0	0.019
	FB	Ferricrete Breccia	5,192.1	4,249.0	0.070	5,323.0	4,468.0	0.026
	LB	Lower Breccia	6,644.7	6,408.0	0.129	4,610.6	4,590.0	0.074
3 (Ferralite)	FL	Ferralite	19,870.7	20,034.0	0.138	10,354.8	10,262.0	0.107
	LFL	Lower Ferralite	75.0	73.0	0.132			
	CB	Carbonate	5.8	3.0	0.018	2.0		
	QS	Quartz Sand	83.9	36.0	0.132			
	SI	Silica	491.6	399.0	0.092	171.4	126.0	0.072
4 (Bedrock)	SE	Serpentinite	272.2	258.0	0.051	206.3	167.0	0.048
	SH	Shist	1,328.7	677.0	0.041	771.4	506.0	0.036
	SP	Saprolite	2,953.1	2,250.0	0.067	2,033.9	1,759.0	0.063
	Missing		16.8	22.0	0.067	3.6	2.0	0.066

When multiple samples on different sides of pits were combined into a single pit, the from-to intervals and lithology codes on different sides were not always the same. This problem was resolved by splitting the from-to intervals into smaller intervals so that all sides had common from-to intervals. If more than one lithology code interval was present in an interval, the majority code was used.

The lithologic units were consolidated into four lithology groups for resource estimation, as shown by the 'Lith Group' designation in Table 15.1.4.2.

Assay Data

Assay data included the original assay data file from 2005, the assay update data from 2007 and the new data from 2008 to 2009. All of these data files had been edited so that the area code and database-identifier-number was combined into a single, alphanumeric drill hole name (BHID). In addition, values below detection limit were set to half the detection limit provided in the EXCEL spreadsheet.

Each record in the assay data includes the BHID, sample interval (from, to), plus assays for cobalt, nickel, and manganese. Some intervals were re-assayed up to four times for quality control purposes, so some intervals have more than one data record. The 2005 data included a number of additional fields such as the Assay Date, Report Number, and Issue Codes that were not in the 2007 - 2009 data. These additional fields were not used for resource estimation purposes.

Averaging of Assay Data for Resource Estimation

The assay data were processed before resource estimation to average duplicate assays, to average multiple samples on different sides of pits and to merge data from the 2005, 2007 and 2009 database files into a single file.

- The first part of the process was to average the duplicate assays. Duplicate assays were averaged for 972 intervals of the 2005 Nkamouna data and 1,114 intervals of the 2009 data. Duplicate assays were averaged for 1,679 intervals at Mada;
- Assays on multiple sides of pits in the 2005 data were averaged into single, averaged intervals. In some cases, particularly at the bottom of pits, the assay interval (FROM and

TO) were not the same on all sides of the pit, in which case intervals were split to align the intervals before averaging. A total of 1,149 samples from 120 pits were averaged in this process and 368 averaged records were created for 38 averaged pits;

- Assays from the 2007 data, which generally just extended existing pits, were merged with the averaged 2005 data. Where intervals from 2005 overlapped with intervals from 2007, they were averaged. Intervals with mismatching FROM and TO were split into smaller intervals before averaging to align the intervals;
- Assays from the 2009 data (after averaging duplicate assays) were merged with the combined 2005 - 2007 assay data;
- Drill hole files were created for actual elevation (UTM coordinates) and flat-model coordinates. The collar elevation was set to zero for all pits / holes to create the flat-model drill hole files; and
- Assays below detection limit were set to one-half the detection limit in the original EXCEL data file prior to importing into Datamine.

15.1.5 Topographic Model

Topographic data was provided by Geovic as a single AutoCAD drawing file that covers all of Nkamouna and Mada. This drawing, named 'general_topographic_map_of_geovic_2009.dwg', is at an original scale of 1:10,000.

Topographic mapping covers the entire area inside the schist / serpentinite boundary for Nkamouna, but small portions of Mada were outside the boundary.

15.1.6 Compositing

Drill holes were composited in the TOPMIN coordinate system using simple, length-weighted compositing to combine the original sample intervals into equal 1 m lengths starting from the top of the pit / hole. Missing assays were treated as unknown values and they were not included in the weighted average. At least 0.5 m of assayed length was required before a composite value was stored. Lithologic codes were assigned to composites based on the assay lithology that covered the majority of the composite interval.

Since over 85% of the samples were collected using regular 1 m lengths starting from the top of the pit / drill hole, the effect of compositing on the data is minimal.

After the TOPMIN composites were computed, the Z-coordinate was rounded to the nearest mid-bench elevation, i.e. -1.5, -0.5, 0.5, etc., then 0.001 m was added to the Z-coordinates. This manipulation was done so that the octant search used for resource estimation would effectively select from only the upper four octants.

Missing cobalt and nickel grades in the granular and breccia zones were set to 0.0249% Co and 0.1599% Ni, respectively, prior to compositing.

Estimation of Missing Manganese Assays from Cobalt

Approximately 10% (2,197) of the intervals in the 2005 assay data were not assayed for manganese.

Since there is a good general correlation between manganese and cobalt grade, with a regression R2 = 0.9, it is reasonable to estimate the missing manganese values from the cobalt grade. In

addition, manganese has little economic value and most of the unassayed manganese values have a relatively low cobalt grade (averaging 0.1% Co). A regression study was done to determine the best method for estimating the missing manganese assays, as follows:

- A data set was assembled that contained all the samples with assays for both cobalt and manganese.
- Preliminary statistics were done which indicated that manganese and cobalt were best correlated on log-log plots, which implies a power curve in the form

  $Y = A \times B$.
- Based on the log-log correlation and assumption that relative sampling errors are the same for cobalt and manganese, an intermediate variable, MnCo, was created that was equal to the square root of manganese times cobalt, i.e. the geometric mean of cobalt and manganese.
- Average manganese and cobalt grades were computed for each of the cobalt-bearing lithologic units, using small grade ranges of the MnCo variable with about the same number of samples in each cell. A total of 64 outlier pairs were identified at this stage. The outliers were removed from the data before computation of the regression equations and 14,147 assay pairs were used for regression analysis.
- The average manganese and cobalt grades were plotted on log-log graphs. Since the averages were computed using the geometric mean of the two assays, the resulting curves are equivalent to doing a lognormal major-axis regression.
- The resulting graphs were very linear, but appeared to have different slopes for low and high grade values. Lithology was confirmed to be a significant variable, and power curves were derived for low and high cobalt values as shown in Table 15.1.6.1.
- The regression equations were tested against the original data, as shown in Table 15.1.6.2 and were found to be globally unbiased. Estimation error varied between 15 and 36% RSD (relative standard deviation) in the cobalt-bearing zones. Outside the cobalt-bearing zones, the correlation between Co and Mn is less reliable and the relative standard error is only 44% RSD.

Table 15.1.6.1: Regression Coefficients for Estimating Manganese from Cobalt

Lithology	Low Power	Low Constant	Low-High Crossover	High Power	High Constant
Granular (GR)	0.8830	5.3799	0.6579	1.0000	5.6500
Upper Breccia (UB)	0.8873	4.8428	0.1408	0.9898	5.9212
Ferricrete Breccia (FB)	0.9536	5.6337	0.3753	1.1002	6.5042
Lower Breccia (LB)	0.8733	4.6533	0.3736	1.0606	5.5956
Ferralite (FL)	0.7040	3.2144	0.1873	1.0401	5.6438
Other	0.8491	4.4092	0.1965	1.0020	5.6552

Table 15.1.6.2: Results from Applying Regression Equations to the Test Data Set

Lith Code	Count	Average Co	Average Mn	Average Mn Regress	Average Regress Error	Average log(e) (RegErr)	StdDev log(e) (RegErr)	R2
GR	750	0.0332	0.2591	0.2584	-0.0007	0.0027	0.1496	0.9319
UB	630	0.0698	0.4460	0.4460	0.0001	-0.0026	0.2758	0.9090
FB	2,151	0.1238	0.7801	0.7886	0.0085	-0.0077	0.3528	0.9266
LB	3,273	0.1459	0.8419	0.8457	0.0039	-0.0024	0.3084	0.9295
FL	6,664	0.1593	0.9197	0.9207	0.0010	-0.0060	0.2756	0.8128
Other	679	0.0934	0.5985	0.5954	-0.0031	-0.0865	0.4171	0.8444

15.1.7 Bulk Specific Gravity

Specific gravity was assigned to the block model based on lithology. Average dry specific gravities were determined based on testwork results on 163 samples taken from the variety of laterite lithology's observed at Nkamouna and Mada. These samples were collected and analyzed in 2008.

The following specific gravities were assigned to the model blocks, based on the lithologic coding assigned by the modeled lithologic surfaces:

- Granular (GR) material, dry, in place = 1.31 t/m^3;
- Breccia (UB, FB, LB) material, dry, in place = 1.7 t/m^3; and
- Ferralite (FL) material, dry, in place = 1.4 t/m^3.

15.1.8 Lithologic Surface Models

Triangulated DTM models were created to represent the depth from surface to the bottom of the granular (GR), breccia (BX), and ferralite (FL) zones. For example, the bottom of FL / top of rock was defined by locating all but the uppermost saprolite / serpentinite / schist sample in each pit / hole. A DTM was then formed using the elevations of the top of those samples. The DTM was limited to the area inside the schist boundary, which was assigned an elevation of zero.

Since many of the pits / holes were not deep enough to intersect the contact between the bottom of the FL/top of saprolite, but were deeper than the initial DTM model for the bottom of FL based on surrounding pits / holes, this caused some FL samples to erroneously project below the initial DTM. The initial DTM was therefore updated by locating the lowest point in each pit / hole that contained GR, BX, or FL. Those data were then compared to the DTM surface and GR-BX-FL bottom points below the DTM surface were added to the top-of-rock point data. The DTM surface was re-triangulated with the updated data. The search for GR+BX+FL points below the DTM surface was repeated twice until no GR+BX+FL points were below the bottom of FL / top of rock. The same procedure was used to define the top of FL / bottom of BX and the top of BX / bottom of GR.

15.1.9 Flat Model and Top-of-Mineralization Model

Based on cross-section plots of cobalt grade in the flattened model, it was observed that there was generally a sharp contact between low-grade and high-grade cobalt grade at the top of mineralization. The top 2 m of mineralization generally contain the highest cobalt grades in the deposit.

While the top of mineralization appears to be continuous, it is slightly irregular, and locally correlates sub-parallel to the topographic surface, the breccia / ferralite contact, and/or the bottom of ferralite. A top-of-mineralization, or 'TOPMIN' model was created to remove the irregularities in the top-of-mineralization, by moving the collar elevations of pits / holes up until the top of mineralization was at an elevation of zero. The advantage of this model is that the optimum correlation between the metal grades is horizontal and the shape and continuity of the mineralization can be viewed directly on plan maps.

The TOPMIN model was created iteratively, as follows:

- Initial values for the depth to mineralization at each pit/hole were estimated by computing optimized seam composites, in flat model coordinates. A cut-off grade of 0.10% Co, a minimum mineralized thickness of 2 m, and a minimum of 4 m thickness for inter-burden waste were used for computing the composites. The initial value for the TOPMIN depth was set to the depth of the top of the highest composite with an average grade above cut-off. (Note a TOPMIN depth of 5.0 is the same as a flat-model Z-coordinate of -5.0.) If no composite was above cut-off grade, TOPMIN was set to the depth of the top of the FL unit. If no FL was present, TOPMIN was set to the depth of top of BX. Otherwise, the TOPMIN depth was set to zero.
- A DTM of TOPMIN-shifted surface topography was constructed using the TOPMIN depths with the XY locations of the corresponding pits / holes. The schist / serpentinite contact, was used as a DTM limit with a depth of zero. The TOPMIN depth was added to the formation boundary points (Bottom FL, Top FL, and Top BX) and TOPMIN-shifted DTM models were created using the point elevations. A TOPMIN-shifted drill hole file was created by setting the collar elevation of each pit / hole to the TOPMIN depth before de-surveying.
- A flat model DTM of the TOPMIN surface was created by setting the TOPMIN depths to negative values, i.e. flat model elevation and triangulating the point elevations.
- The TOPMIN DTM and TOPMIN points were visually inspected in the Datamine design window as cross-section plots with pits / holes. Anomalous TOPMIN depths were adjusted interactively and the TOPMIN DTM surfaces were recreated starting with the second step of this procedure.

This model went through several iterations of editing and remodeling to remove inconsistencies and to improve the consistency of the interpretation of the top of mineralization.

The cross-section plots in Figure 15-1 shows pits and drill holes plotted in untransformed coordinates, flat model coordinates and coordinates indexed parallel to the top of mineralization. Cobalt grades are shown as color-coded histograms and the interpreted lithology contacts are shown as lines where the lithology contacts intersect with the cross section.

The section with untransformed coordinates, at the top of Figure 15-1, demonstrates the difficulty of visualizing the deposit without a vertical exaggeration. Even though the maximum topographic relief is only about 200 m, any significant vertical exaggeration rapidly becomes unwieldy and correlations between pits / drill holes are distorted. (Vertical exaggerations between 10x and 30x were used for interpretation and review of the model in the Datamine design window).

Because the deposit is essentially a thin skin just below the surface and the surface topography is relatively flat, the deposit is more easily viewed as though the surface topography is flat, as shown in the middle section of Figure 15-1. This plot also demonstrates the variability in the thickness of the various lithologic units and that the deposit is composed of a variable-thickness barren zone consisting of the Granular Zone and most of the breccia.

It is also observed that the contact between the barren overburden and the mineralized horizon is generally very sharp and is sub-parallel to the top of the Lower Breccia, the top of the Ferralite, and/or the bottom of the Ferralite. In addition, higher grade mineralization tends to be associated with the upper parts of the mineralized zone, so resource modeling was done in the TOPMIN coordinate system in which the top of mineralization was defined as Z equal to zero (0) elevation. The bottom cross-section of Figure 15-1 shows the pits and drill holes plotted in the indexed (TOPMIN) coordinate system.

SRK has reviewed the procedures and resultant DTM surfaces produced by the processes as described above and is of the opinion that the transformed coordinate systems (TOPMIN and flat model) are both justified and required for the grade estimation and model visualization / validation processes.

15.1.10 Basic Statistics by Lithologic Unit

Basic statistics were performed on each lithologic unit to evaluate the correlation between cobalt, nickel, and manganese as a function of lithology. These statistics show that cobalt grade is strongly correlated with lithology with the lowest cobalt grade in the uppermost unit, the Granular (Unit 1) and highest grade in the bottom unit, the ferralite zone (Unit 5). Tables 15.1.10.1 shows statistics for cobalt, nickel and manganese, respectively. Figure 15-2 demonstrates that there is a very strong linear correlation between manganese grade and cobalt grade. This is consistent with the strong association between cobalt and the manganese mineral asbolane. Nickel is well correlated with cobalt except, within the breccia units at Nkamouna, which are lower-grade nickel compared to the overall correlation (Figure 15-3). The reason for the lower nickel grades in the Nkamouna breccias is likely related to the development of the high-grade-cobalt zone at the bottom of the Nkamouna breccias.

Table 15.1.10.1: Basic Statistics for 1 m Composite Grades

Assay	Area	Lith Group	Number Assays	Minimum	Maximum	Average	Std. Dev	Coef of Variation
Cobalt	Mada	Granular	488	0.002	0.117	0.009	0.011	1.19
		Breccia	11,026	0.000	2.546	0.045	0.100	2.21
		Ferralite	10,388	0.000	2.531	0.107	0.103	0.97
	Nkamouna	Granular	934	0.002	0.644	0.031	0.037	1.18
		Breccia	12,638	0.001	2.806	0.096	0.200	2.09
		Ferralite	20,545	0.001	2.265	0.137	0.140	1.02
Manganese	Mada	Granular	488	0.020	6.630	0.115	0.308	2.69
		Breccia	11,023	0.005	24.190	0.344	0.784	2.28
		Ferralite	10,384	0.004	19.020	0.737	0.685	0.93
	Nkamouna	Granular	529	0.030	3.400	0.228	0.229	1.01
		Breccia	11,716	0.002	22.880	0.599	1.197	2.00
		Ferralite	19,976	0.002	16.700	0.858	0.798	0.93
Nickel	Mada	Granular	488	0.002	0.540	0.080	0.057	0.71
		Breccia	11,024	0.003	1.960	0.213	0.151	0.71
		Ferralite	10,388	0.005	2.150	0.536	0.234	0.44
	Nkamouna	Granular	931	0.008	0.680	0.181	0.084	0.46
		Breccia	12,633	0.003	9.810	0.248	0.234	0.94
		Ferralite	20,540	0.005	5.060	0.620	0.249	0.40

SRK has conducted an independent statistical analysis using the data provided by Geovic, and has produced similar results and conclusions to those determined by Geovic.

Grade Distributions and Grade Zoning

Previous work on the Nkamouna deposit has indicated that cobalt, manganese, and nickel are distributed as distinct populations that may be modeled as spatially continuous grade zones. Cobalt and manganese were partitioned into low-grade, mid-grade, and hi-grade zones and nickel was partitioned into low-grade and a high-grade zones.

The same general concept of grade zones is used for this resource estimate, but the grade-zones were defined using an automated procedure, rather than manually drawn outlines. While the automated procedure does not capture the level of detail that is possible with the manually drawn outlines, the results are generally similar. Furthermore, because the grade zones are used in the resource as overlapping, gradational (soft) boundaries, exact definition of the zones is not required.

The algorithm for assignment of grade zones is as follows:

- An indicator for high-grade cobalt (HG_Co) is set to 1 if cobalt grade is greater than 0.275%.
- An indicator for mineralized, mid-grade cobalt (MIN_Co) is set to 1 if cobalt grade is greater than 0.05%.
- Jack-knifing is used to estimate cobalt grade, nickel grade, and the two indicators for each composite. Inverse-distance-power weighting is used for estimation. A power of 4.0 is used for grade and a power of 1.0 for the indicators.
- The geometric means of the actual composite grade and the jack-knifed grade is computed for both nickel and cobalt.

- Grade zones are assigned using the actual composite grades, the geometric mean grade, and the indicators as shown in Table 15.1.10.2 and Table 15.1.10.3.

Table 15.1.10.2: Logic Table for Cobalt Grade Zoning

Composite Co Grade	Geomean (Co,CoEst)	Hi-Grade Indicator	Mid-Grade Indicator	Grade Zone
>0.4	Any	HG_Co>0.01		3 (Hi-Grade)
>0.2	>0.275	HG_Co>0.30		3 (Hi-Grade)
>0.1	>0.275	HG_Co>0.50		3 (Hi-Grade)
>0.15				2 (Mid-Grade)
<0.15	>0.05		>0.25	2 (Mid-Grade)
>0.10			>0	2 (Mid-Grade)
Does not match any of the above				1 (Low Grade)
Non-grade-related conditions				
LITH is GR (granular)				1 (Low Grade)
More than 2 meters above top of mineralization				1 (Low Grade)

Table 15.1.10.3: Logic Table for Nickel Grade Zoning

Composite Ni Grade	Geomean (Ni,NiEst)	Grade Zone
>0.6		3 (Hi-Grade)
>0.5	>0.25	3 (Hi-Grade)
>0.4	>0.30	3 (Hi-Grade)
>0.3	>0.35	3 (Hi-Grade)
>0.2	>0.40	3 (Hi-Grade)
Does not match any above		1 (Low Grade)

Basic statistics for the composite grades by grade zone are summarized in Table 15.1.10.4. Composite histograms and lognormal cumulative frequency plots, as shown in Figure 15-4 through Figure 15-9, indicate that the grade-zoning method partitions cobalt grade into generally lognormal populations. A significant outlier population is observed at Nkamouna in the mid-grade distribution. The high-grade outliers are high-grade samples that are spatially surrounded by low-grade samples. These samples appear to be discontinuous, higher-grade zones with little continuity that should be capped for resource estimation. A similar group of outliers is observed in the low-grade cobalt zone at Nkamouna. These are also believed to be higher-grade material with poor continuity.

Because cobalt and manganese grades are highly correlated, the cobalt grade zones are used for manganese zoning. The histograms and cumulative frequency plots for manganese, as shown in Figure 15-5 and Figure 15-8, indicate that it is reasonable to use the cobalt grade zones as a proxy for manganese grade zoning.

Only two zones were used for nickel zoning based on work conducted during previous resource estimates. The histograms and cumulative frequency plots for nickel, as shown in Figure 15-6 and Figure 15-9, indicate the nickel could be better modeled using three zones than two. The high-grade zone is nearly a perfect lognormal distribution, as shown by the symmetrical histograms and the nearly straight lines in the cumulative frequency plots. The histograms for the low-grade zone are skewed to the left and the cumulative frequency plots curve down at the bottom, indicating multiple populations. Use of three nickel populations is recommended for future estimates, which may provide a slight improvement in the accuracy of nickel estimates.

Table 15.1.10.4: Basic Statistics for 1 m Composited Grades by Grade Zone

Assay	Area	Grade Zone	Number Composites	Minimum	Maximum	Average	Std. Dev.	Coef. of Var.
Cobalt	Mada	Low-Grade	17,434	0.0005	0.1920	0.0207	0.0159	0.7665
		Mid-Grade	8,942	0.0040	1.4620	0.1291	0.0766	0.5935
		High-Grade	358	0.1160	2.5460	0.5248	0.3453	0.6579
	Nkamouna	Low-Grade	21,926	0.0010	0.5020	0.0262	0.0193	0.7383
		Mid-Grade	17,952	0.0070	2.4000	0.1363	0.0886	0.6501
		High-Grade	1,987	0.0770	2.8060	0.5127	0.3614	0.7050
Manganese (Missing Values Estimated from Regression)	Mada	Low-Grade	17,434	0.0049	8.9800	0.1971	0.2423	1.2295
		Mid-Grade	8,942	0.0500	11.9000	0.8581	0.6223	0.7252
		High-Grade	358	0.4200	24.1900	3.1861	2.8484	0.8940
	Nkamouna	Low-Grade	21,926	0.0020	4.6000	0.2309	0.1785	0.7728
		Mid-Grade	17,952	0.0600	17.6000	0.8307	0.5315	0.6399
		High-Grade	1,987	0.1400	19.5757	2.8842	2.2707	0.7873
Nickel	Mada	Low-Grade	17,062	0.0025	0.6000	0.1724	0.0867	0.5030
		High-Grade	9,672	0.2100	2.1500	0.5899	0.1900	0.3221
	Nkamouna	Low-Grade	21,782	0.0026	0.6000	0.1696	0.0794	0.4684
		High-Grade	20,078	0.2100	9.8100	0.6568	0.2332	0.3550

SRK has reviewed the data and procedures utilized to formulate the grade zone boundaries, and is of the opinion that this process is suitable, given the variable distributions of metal typically observed in lateritic deposits.

15.1.11 Grade Zone Models

Grade zone codes were assigned to the block model using nearest-neighbor assignment from composite grade zones in the TOPMIN coordinate system.

15.1.12 Variogram Analysis and Modeling

Variograms were run for the Nkamouna data to evaluate the continuity of cobalt, nickel, and manganese mineralization. 1 m composite assays with the log-transformed grades were used for variogram computation in the TOPMIN coordinate space. The log-transformed variograms were converted to relative variograms using the standard covariance transformation method. Directional variograms were computed parallel to the top of mineralization at azimuths of 0°, 45°, 90° and 135° to evaluate directional anisotropies. In addition, the average variogram parallel to the top of mineralization and the vertical variogram were run to assess the average continuity parallel and perpendicular to mineralization. Variograms were not computed for Mada, because of the wider sample spacing.

The variograms, which are shown graphically in Appendix C, have the characteristics described in the following paragraphs.

Cobalt Low-Grade Zone Variograms

The directional variograms for low-grade cobalt indicate a very slight directional trend at an azimuth of approximately 45 degrees, but the horizontal variograms are essentially isotropic. The sill of the vertical variogram is 1.53, with a nugget effect of 0.10 and a range of about 15 m. The average horizontal variogram rises sharply from the nugget effect for 22 m, then increases gradually for the next 92 m, and finishes with a long-range component with a range of 2,000 m.

These variograms indicate a continuous long-range process combined with erratic short-range continuity and a very strong vertical anisotropy.

Cobalt Mid-Grade Zone Variograms

The horizontal variograms for mid-grade cobalt are nearly isotropic, although the long-range component shows a weak anisotropy to the northeast. The sill of the vertical variogram is 0.332, with a nugget effect of 0.062 and a range of 21 m. The average horizontal variogram rises sharply from the nugget effect within the first 50 m, then increases gradually over the next 800 to 1,000 m.

Again these variograms indicate a very continuous long-range process combined with erratic short-range continuity and a very strong vertical anisotropy.

A reasonable explanation for this variogram is that cobalt grade was initially very continuous over a range of 800 to 1,000 m. As the deposit weathered, the cobalt distribution became more erratic as cobalt concentrated preferentially in the mineral asbolane, and repeated collapse and brecciation in small pockets created short-range discontinuities.

Cobalt High-Grade Zone Variograms

The horizontal variograms in the cobalt high-grade zone are essentially pure nugget effect, with a range less than 50 m. A possible 'hole effect' component may be present, which is suggestive of small pods of mineralization with sizes smaller than 25 m. The vertical variogram has a nugget effect of 0.060, a sill of 0.407 and a range of 6 meters.

Nickel Low-Grade Zone Variograms

Variograms in the nickel low-grade zone indicate very continuous mineralization with a nugget effect that is essentially zero. The horizontal variogram rises rapidly for the first 90 m to about 60% of the total variability, and then levels off to a long-range structure with a range of 2,500 m. The vertical anisotropy is extreme, however and the vertical range is only 10 m.

Nickel High-Grade Zone Variograms

Variograms in the nickel high-grade zone indicate a weak anisotropy with better continuity to the in a northwest-southeast direction. The overall horizontal variogram indicates a short-range continuity of less than 25 m followed by a long-range continuity of 700 m. The vertical anisotropy is extreme, with a range less than 14 m.

Manganese Variograms

Manganese variograms are similar to cobalt variograms.

15.1.13 Grade Estimation

Block grades were estimated for cobalt, nickel and manganese using inverse-distance-power (IDP) estimation with grade-zoning controls. The primary function of the grade zones was to control selection of composites so that the composites used to inform block grades were statistically representative. Since there is considerable overlap between the grade-zone populations, the grade estimation composite selection procedure was designed to treat the grade-zone boundaries as soft boundaries rather than hard boundaries that divide the populations on exact lines. This was accomplished as follows:

- All composites from the low-grade zone plus the lower-grade composites from mid-grade zones were used to estimate the low-grade zone. Only a few low-grade outliers from the high-grade zone were used to estimate the low-grade zone except for nickel, which did not have a mid-grade zone.

- The mid-grade zone was estimated using all mid-grade zone composites plus the higher-grade composites from the low-grade zone and the lower-grade samples from the high-grade zone.
- The high-grade zone was estimated using all high-grade zone composites plus the higher-grade composites from the mid-grade zone. None of the composites from the low-grade zone were used to estimate the high-grade zone except for nickel, which did not have a mid-grade zone.

In addition to the grade-range selection parameters for each grade zone, capping grades were established for each grade zone based on the composite grade distributions. These parameters are summarized in Tables 15.1.13.1 through 15.1.13.3.

SRK has reviewed the data provided to assess for the presence of high grade outliers that could potentially adversely impact grade estimation. Although SRK is of the opinion that the raw data should be capped prior to compositing, the average sample length approximates the composite length utilized for the resource estimate. Based on this observation, SRK concludes that the composite capping strategy utilized by Geovic is appropriate.

Table 15.1.13.1: Grade Range Parameters and Capping Grades for Cobalt

	Composite Zone Grade Ranges (%Co)						Capping Grade
	Low-Grade		Mid-Grade		High-Grade		
Estimated Zone	Min	Max	Min	Max	Min	Max	(% Co)
Low-Grade	0.00	100	0.00	0.10	0.00	0.10	0.10
Mid-Grade	0.08	100	0.00	100	0.00	0.40	0.60
High-Grade	0.25	100	0.25	100	0.00	100	2.00

Table 15.1.13.2: Grade Range Parameters and Capping Grades for Nickel

	Composite Zone Grade Ranges (%Ni)						Capping Grade
	Low-Grade		Mid-Grade		High-Grade		
Estimated Zone	Min	Max	Min	Max	Min	Max	(% Ni)
Low-Grade	0.00	100	NA	NA	0.00	0.40	1.00
High-Grade	0.30	100	NA	NA	0.00	100	2.00

Table 15.1.13.3: Grade Range Parameters and Capping Grades for Manganese

	Composite Zone Grade Ranges (%Mn)						Capping Grade
	Low-Grade		Mid-Grade		High-Grade		
Estimated Zone	Min	Max	Min	Max	Min	Max	(% Mn)
Low-Grade	Cobalt composite selection was used						0.70
Mid-Grade	Cobalt composite selection was used						2.50
High-Grade	Cobalt composite selection was used						12.0

Because of the high vertical-to-horizontal anisotropies, a five-pass search was implemented so that the horizontal search radius could be expanded independently of the vertical search radius. All composite selection was conducted using no more than one composite from any individual pit / hole. At least one composite was required for block grade estimation. The octant search option was used to provide de-clustering to ensure that data were evenly selected on all sides of the block. Because the TOPMIN block elevations were rounded to even mid-bench units and the

TOPMIN composites were rounded to mid-bench units plus a 0.001 m shift, the octant search is effectively a quadrant search for the first three search passes.

The IDP power and number of total composites used for grade estimation were set for each zone to provide the appropriate level of variance reduction, or smoothing, as estimated from the variogram. The final search and IDP parameters are listed in Table 15.1.13.4 and Table 15.1.13.5.

Table 15.1.13.4: Composite Selection Parameters

	Search Ellipse					Octant Search	
Search Pass	Primary Radius	Secondary Radius	Vertical Radius	Minimum Compos.	Maximum Compos.	Max Comp. per Octant	Minimum Octants
Search Pass 1	100	100	0.5	5	9	3	4
Search Pass 2	200	200	0.5	5	9	3	4
Search Pass 3	300	300	0.5	5	9	3	4
Search Pass 4	400	400	1.5	3	9	3	3
Search Pass 5	800	800	5.0	1	9	3	1

Table 15.1.13.5: Inverse Distance Weighting Parameters

	IDP Weighting Anisotropies			
Zone	Primary Anisotropy	Secondary Anisotropy	Vertical Anisotropy	Exponent Power
Cobalt - Low Grade	10	10	1.0	4.5
Cobalt - Mid Grade	10	10	1.0	4.5
Cobalt - High Grade	10	10	1.0	2.8
Nickel - Low Grade	20	20	1.0	4.5
Nickel - High Grade	20	20	1.0	3.6
Manganese - Low Grade	10	10	1.0	4.5
Manganese - Mid Grade	10	10	1.0	4.2
Manganese - High Grade	10	10	1.0	4.0

Inverse distance modeling statistics and smoothing factors are shown in Table 15.1.13.6.

Table 15.1.13.6: Inverse Distance Modeling Statistics and Smoothing Factors

Deposit-Metal	Grade Zone	Num. IDP Blocks	Average IDP	Rel. Var. IDP	Num. NN Blocks	Average NN	Rel. Var. NN	Ratio IDP/NN	Smoothing Factor
NKM-Co	Low-Grade	1,031,554	0.0250	0.5587	1,031,554	0.0251	0.6523	0.994	0.665
NKM-Co	Mid-Grade	971,968	0.1356	0.2199	971,968	0.1365	0.3856	0.994	0.561
NKM-Co	High-Grade	86,221	0.5035	0.1886	86,221	0.5028	0.4492	1.001	0.541
NKM-Mn	Low-Grade	1,031,554	0.2314	0.3953	724,557	0.2664	0.9400	0.869	0.400
NKM-Mn	Mid-Grade	971,968	0.8220	0.1653	962,512	0.8400	0.3662	0.979	0.461
NKM-Mn	High-Grade	86,221	2.8180	0.2997	85,492	2.8305	0.5470	0.996	0.662
NKM-Ni	Low-Grade	1,012,984	0.1692	0.2623	1,012,984	0.1687	0.2679	1.003	0.718
NKM-Ni	High-Grade	1,077,748	0.6515	0.0724	1,077,748	0.6509	0.1117	1.001	0.636
Mada-Co	Low-Grade	443,741	0.0203	0.6162	443,741	0.0203	0.6921	0.999	0.716
Mada-Co	Mid-Grade	151,728	0.1383	0.2225	151,728	0.1364	0.3511	1.014	0.661
Mada-Co	High-Grade	9,038	0.5161	0.1301	9,038	0.5175	0.3618	0.997	0.508
Mada-Mn	Low-Grade	443,741	0.1831	0.7164	390,103	0.1896	2.2399	0.965	0.275
Mada-Mn	Mid-Grade	151,728	0.9101	0.2225	151,502	0.9437	0.6070	0.964	0.419
Mada-Mn	High-Grade	9,038	2.9652	0.2327	9,038	3.0168	0.6380	0.983	0.535
Mada-Ni	Low-Grade	424,864	0.1875	0.3201	424,864	0.1866	0.2535	1.005	0.759
Mada-Ni	High-Grade	179,820	0.5594	0.0528	179,820	0.5549	0.0904	1.008	0.654

15.1.14 Sample Grid-Spacing Resource Classification Model

Sample grid spacing was measured using the estimation variance from point kriging with a zero-nugget, linear variogram that had a slope of 0.5. This particular linear variogram and point kriging is used because it provides a simple, direct index to the pit / hole spacing, as follows:

- With these parameters, the kriging variance for a block that is estimated from a single, isolated composite is equal to the distance from the composite to the block center.
- The kriging variance for blocks around the margins of sets of composites is slightly smaller than the distance to the nearest point.
- The kriging variance for a block in the center of a square grid of composite points is equal to approximately 28% of the size of the grid. The kriging variance becomes smaller as blocks get closer to the grid points.
- Thus, selecting blocks with a kriging variance less than 28 will select blocks inside a 100 m grid, plus an extrapolation of 28 m outside the 100 m grid.

A FLAG variable was defined with a value of 1.0 for all composites with cobalt values. The FLAG variable was point-kriged to block centers using the same 5-pass search ellipse procedure as was used for estimating metal values. A minimum of eight and a maximum of 15 composites were used for this model with no more than one composite from any pit / hole, except search Pass 5, for which a minimum of one composite was allowed. Grade zone limits were not used for this process. Composites with absent cobalt values were not used.

15.2 Resource Classification

The mineral resources at the Nkamouna and Mada deposits have been classified in accordance with the CIM definition standards for mineral resources and mineral reserves (December 2005). The classification parameters are defined as a function of distance to sample data and number of samples utilized to inform block grades and are intended to encompass zones of reasonably continuous mineralization.

Resource classification was done for each block based on the sample grid spacing model. Determination of the appropriate grid size for each resource class was conducted based on the continuity of grade above a cut-off grade of 0.10% cobalt. The sample grid spacing and extrapolation limits for each resource category are as follows:

- Measured Mineral Resources - maximum 100 m grid spacing or 28 m extrapolation and search pass less than or equal to 2;
- Indicated Mineral Resources - maximum 200 m grid spacing or 56 m extrapolation and search pass less than or equal to 3; and
- Inferred Mineral Resources – All blocks estimated that are not classified as Measured and Indicated Resources.

15.3 Block Model Validation

15.3.1 Visual Inspection

The block model was inspected visually in plan and section by comparison to the underlying composite data. Overall, there is close agreement between block model grades and composite grades.

15.3.2 Comparison of Inverse Distance (IDP) and Nearest Neighbor (NN) Models

The effect of IDP smoothing on the resource model is evaluated by comparing the IDP model to the NN model, which was created without caps or grade zoning. This comparison shows the typical smoothing effects that are expected for an IDP model compared to an NN model. In general, tonnage is higher and grade is lower in the IDP model than in the NN model, which reflects dilution and ore losses that are introduced by the smoothing from the IDP estimation. Comparisons of the IDP and NN models are shown for a range of cut-off grades in Table 15.3.2.1 for Nkamouna and Table 7.3.2 for Mada. Overall, the IDP and NN model tonnage and grade compare well above the cut-off grade ranges analyzed, and generally confirms the IDP grade estimate.

Table 15.3.2.1: Comparison of Nkamouna IDP Model and Nearest Neighbor Models

Nkamouna Deposit (Measured and Indicated Only)												
	IDP Model				NN Model				Ratio IDP to NN			
Cut-off (%Co)	Tonnes	% Co	% Ni	% Mn	Tonnes	% Co	% Ni	% Mn	Tonnes	% Co	% Ni	% Mn
0.06	151,250	0.169	0.625	1.005	147,703	0.169	0.617	1.015	1.024	1.000	1.014	0.990
0.10	111,634	0.201	0.657	1.159	106,533	0.200	0.635	1.160	1.048	1.004	1.035	0.999
0.11	99,414	0.212	0.660	1.215	95,057	0.212	0.639	1.221	1.046	1.000	1.031	0.995
0.12	87,336	0.226	0.657	1.279	84,068	0.227	0.642	1.294	1.039	0.993	1.024	0.989
0.13	76,434	0.240	0.654	1.349	73,954	0.243	0.643	1.374	1.034	0.987	1.018	0.981
0.14	66,581	0.256	0.651	1.425	65,113	0.261	0.649	1.466	1.023	0.979	1.004	0.972
0.15	58,194	0.272	0.648	1.505	57,151	0.279	0.651	1.557	1.018	0.973	0.996	0.967
0.16	50,756	0.289	0.647	1.593	50,418	0.300	0.659	1.664	1.007	0.964	0.982	0.958
0.17	44,188	0.307	0.646	1.690	44,646	0.323	0.671	1.787	0.990	0.952	0.963	0.946
0.18	38,654	0.326	0.645	1.791	39,717	0.347	0.684	1.917	0.973	0.941	0.944	0.934
0.19	34,035	0.346	0.644	1.896	35,583	0.372	0.695	2.053	0.957	0.930	0.927	0.923
0.20	30,112	0.365	0.644	2.003	31,791	0.396	0.703	2.183	0.947	0.923	0.916	0.917
0.25	18,976	0.450	0.649	2.475	19,930	0.490	0.719	2.707	0.952	0.919	0.902	0.914
0.30	14,380	0.507	0.656	2.804	14,195	0.538	0.696	2.967	1.013	0.943	0.942	0.945
0.35	11,251	0.558	0.670	3.111	10,760	0.589	0.702	3.250	1.046	0.948	0.954	0.957
0.40	8,735	0.611	0.688	3.437	8,333	0.654	0.727	3.613	1.048	0.934	0.947	0.951
0.45	6,649	0.669	0.710	3.801	6,550	0.746	0.776	4.128	1.015	0.897	0.914	0.921
0.50	5,131	0.727	0.731	4.145	5,230	0.845	0.841	4.651	0.981	0.860	0.870	0.891

Table 15.3.2.2: Comparison of Mada IDP Model and Nearest Neighbor Models

Mada Deposit (Measured and Indicated Only)												
Cutoff	IDP Model				NN Model				Ratio IDP to NN			
(%Co)	Tonnes	% Co	% Ni	% Mn	Tonnes	% Co	% Ni	% Mn	Tonnes	% Co	% Ni	% Mn
0.06	98,063	0.160	0.512	1.029	96,668	0.159	0.507	1.078	1.014	1.006	1.011	0.955
0.10	69,771	0.192	0.533	1.205	63,746	0.187	0.499	1.223	1.095	1.031	1.067	0.985
0.11	62,165	0.203	0.535	1.265	57,015	0.198	0.503	1.294	1.090	1.025	1.062	0.978
0.12	54,604	0.215	0.538	1.334	50,027	0.211	0.507	1.369	1.092	1.021	1.061	0.974
0.13	47,834	0.228	0.542	1.404	44,348	0.226	0.518	1.460	1.079	1.009	1.045	0.962
0.14	41,554	0.242	0.545	1.480	39,475	0.244	0.537	1.567	1.053	0.991	1.015	0.944
0.15	35,862	0.257	0.549	1.563	34,884	0.264	0.554	1.694	1.028	0.973	0.992	0.922
0.16	30,959	0.274	0.554	1.652	31,135	0.288	0.578	1.838	0.994	0.951	0.960	0.899
0.17	27,066	0.289	0.557	1.738	28,134	0.311	0.600	1.988	0.962	0.931	0.930	0.874
0.18	23,635	0.306	0.562	1.827	25,153	0.334	0.620	2.117	0.940	0.916	0.906	0.863
0.19	20,575	0.324	0.568	1.922	22,318	0.358	0.638	2.248	0.922	0.904	0.890	0.855
0.20	18,260	0.340	0.573	2.007	19,975	0.379	0.650	2.376	0.914	0.898	0.881	0.845
0.25	11,065	0.417	0.599	2.404	12,904	0.483	0.726	2.980	0.857	0.864	0.825	0.807
0.30	7,739	0.480	0.623	2.724	8,372	0.531	0.711	3.241	0.924	0.904	0.877	0.841
0.35	5,858	0.530	0.646	3.014	6,004	0.570	0.714	3.469	0.976	0.930	0.904	0.869
0.40	4,559	0.574	0.659	3.281	4,432	0.604	0.706	3.671	1.029	0.951	0.934	0.894
0.45	3,303	0.632	0.685	3.607	3,255	0.683	0.745	4.089	1.015	0.925	0.920	0.882
0.50	2,401	0.691	0.721	3.978	2,596	0.810	0.862	4.763	0.925	0.853	0.837	0.835

15.3.3 Mineral Resource Statement

The Mineral resources for the Nkamouna and Mada cobalt-nickel deposits have been audited by SRK at 59.8 Mt grading an average of 0.24% cobalt, 0.68% nickel and 1.37% manganese classified as Measured Mineral Resources with an additional 60.8 Mt grading an average of 0.22% cobalt, 0.62% nickel and 1.32% manganese classified as Indicated Mineral resources. An additional 202.5 Mt grading an average of 0.20% cobalt, 0.59% nickel and 1.20% manganese is classified as Inferred Mineral resources. The resource is stated above a 0.12% cobalt cut-off for ferralite and a 0.23% cobalt cut-off for breccias, constrained above the bedrock surface.

The mineral resources are reported in accordance with CSA NI 43-101 and have been estimated in conformity with generally accepted CIM 'Estimation of Mineral Resource and Mineral Reserves Best Practices' guidelines. Mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resource will be converted into mineral reserves. The audit of this resource estimate was completed by Jeffrey Volk, P.Geo, an independent Qualified Person, as this term is defined in NI 43-101. The effective date of this resource estimate is October 14, 2009. The mineral resources statement for the Nkamouna and Mada cobalt-nickel deposits is presented in Table 15.3.3.1.

The mineral resource is summarized by resource category and lithologic unit in Table 15.3.3.1. The cut-off grades in this table are different for each lithology and are approximate economic cut-offs based on the different processing characteristics of each lithology.

Table 15.3.3.1: SRK Mineral Resource Statement for the Nkamouna and Mada Cobalt-Nickel-Manganese Deposits, October 14, 2009

Lithology	Resource Category	Cut-off (%Co)	Tonnes (kt)	Average Grade		
				Co (%)	Ni (%)	Mn (%)
Measured						
Nkamouna						
Breccia	Measured	0.23	6,527	0.47	0.55	2.60
Ferralite	Measured	0.12	53,277	0.22	0.69	1.22
Subtotal	Measured		59,805	0.24	0.68	1.37
Mada						
Breccia	Measured	0.23	-	-	-	-
Ferralite	Measured	0.12	-	-	-	-
Subtotal	Measured		-	-	-	-
Total	Measured		59,805	0.24	0.68	1.37
Indicated						
Nkamouna						
Breccia	Indicated	0.23	672	0.43	0.49	2.53
Ferralite	Indicated	0.12	20,247	0.19	0.68	1.07
Subtotal	Indicated		20,918	0.19	0.67	1.12
Mada						
Breccia	Indicated	0.23	6,625	0.38	0.53	2.21
Ferralite	Indicated	0.12	33,251	0.21	0.60	1.28
Subtotal	Indicated		39,876	0.23	0.59	1.43
Total	**Indicated**		**60,794**	**0.22**	**0.62**	**1.32**
Total	**M+I**		**120,599**	**0.23**	**0.65**	**1.35**
Inferred						
Nkamouna						
Breccia	Inferred	0.23	766	0.39	0.49	2.19
Ferralite	Inferred	0.12	19,163	0.18	0.66	1.05
Subtotal	**Inferred**		**19,929**	**0.19**	**0.65**	**1.09**
Mada						
Breccia	Inferred	0.23	14,790	0.40	0.53	2.47
Ferralite	Inferred	0.12	167,831	0.18	0.59	1.10
Subtotal	**Inferred**		**182,621**	**0.20**	**0.58**	**1.21**
Total	**Inferred**		**202,551**	**0.20**	**0.59**	**1.20**

Note: Mineral resources are not mineral reserves and do not have demonstrated economic viability.
All figures have been rounded to reflect the relative accuracy of the estimates.
Reported at cut-off grades of 0.12 and 0.23% % cobalt contained within Ferralite and Breccia, respectively.

15.4 Results of SRK Audit

SRK has verified and validated this model using a number of different methodologies, and find the model acceptable as a basis for resource reporting under CSA NI 43-101 guidelines. This verification included:

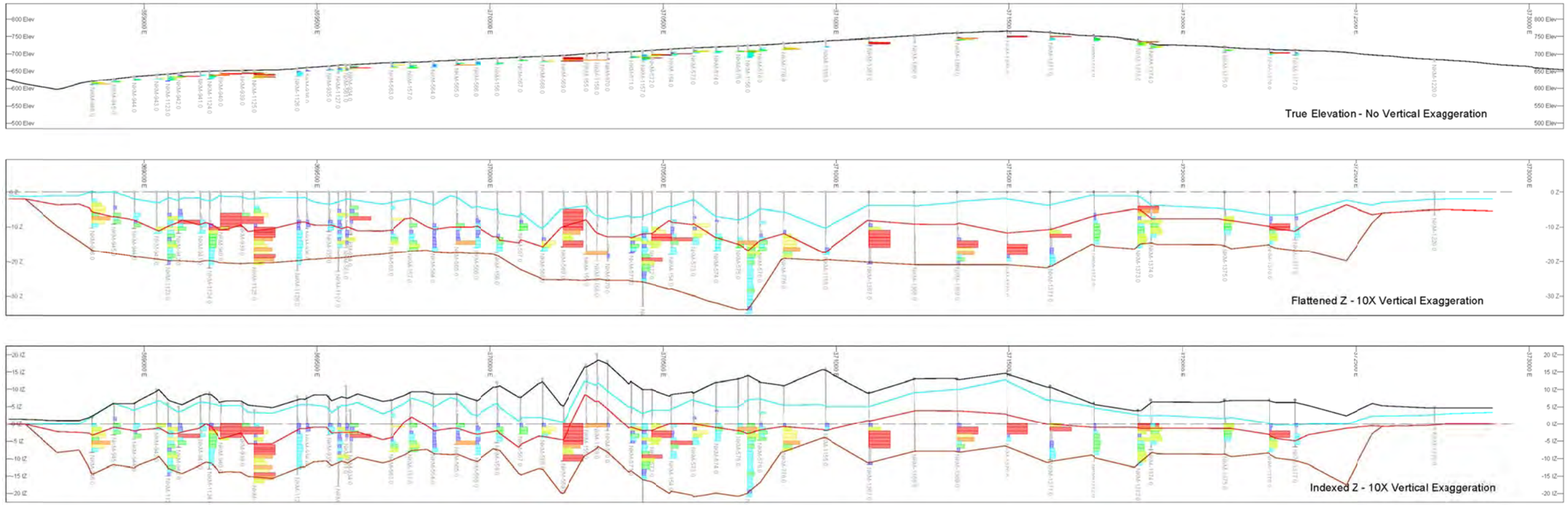
- Visual comparison of block grades and composite grades in plan and section
- Statistical comparisons between block and composite grade distributions
- Review of variography studies and resource classification parameters
- Independent grade estimation (nearest neighbor model).

All of the above independent checks show good agreement between the Geovic model and all other ancillary checks.

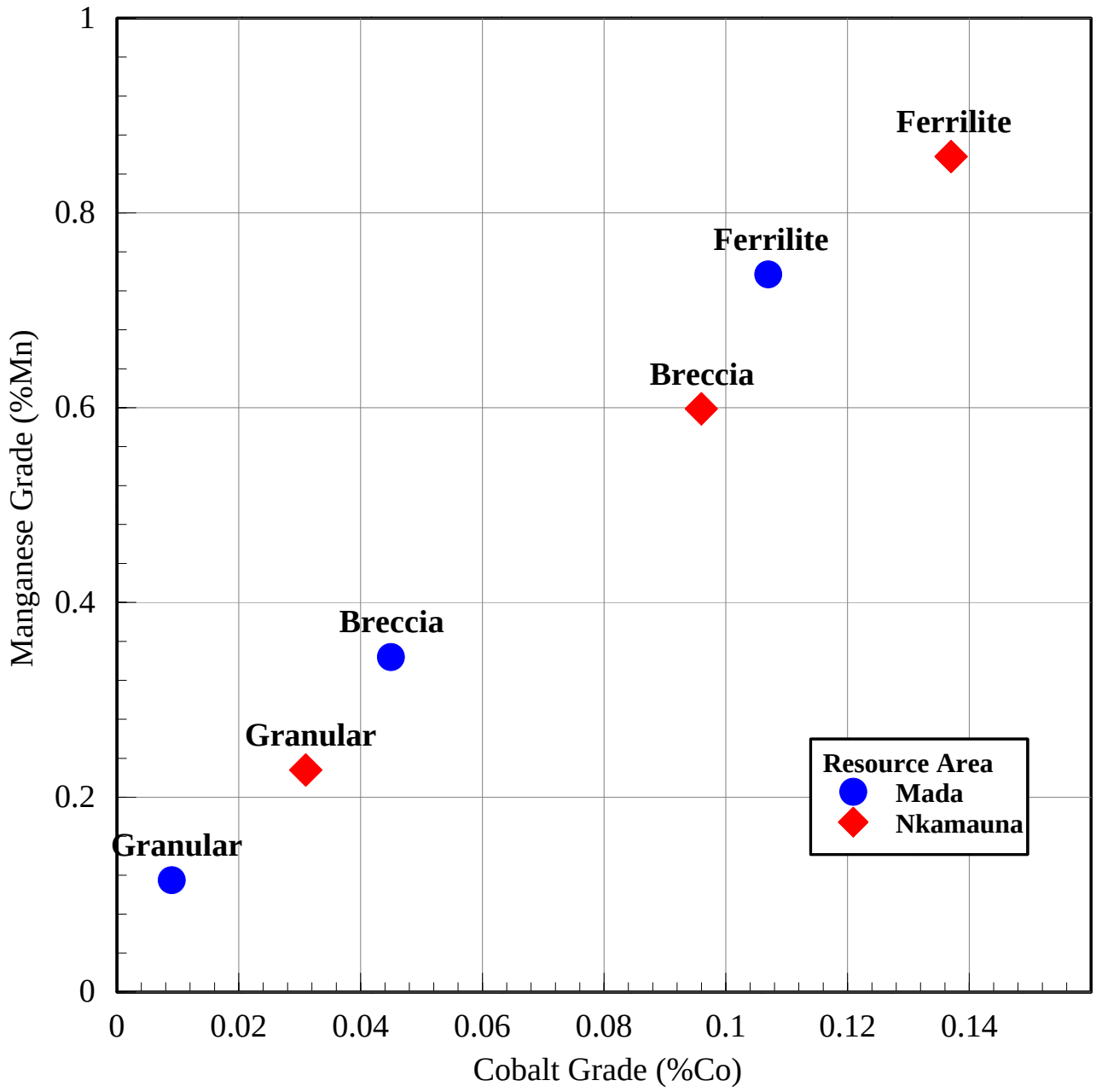
15.5 Recommendations

Based on the SRK audit of the resource estimation process and resource classification procedures, the following activities are recommended:

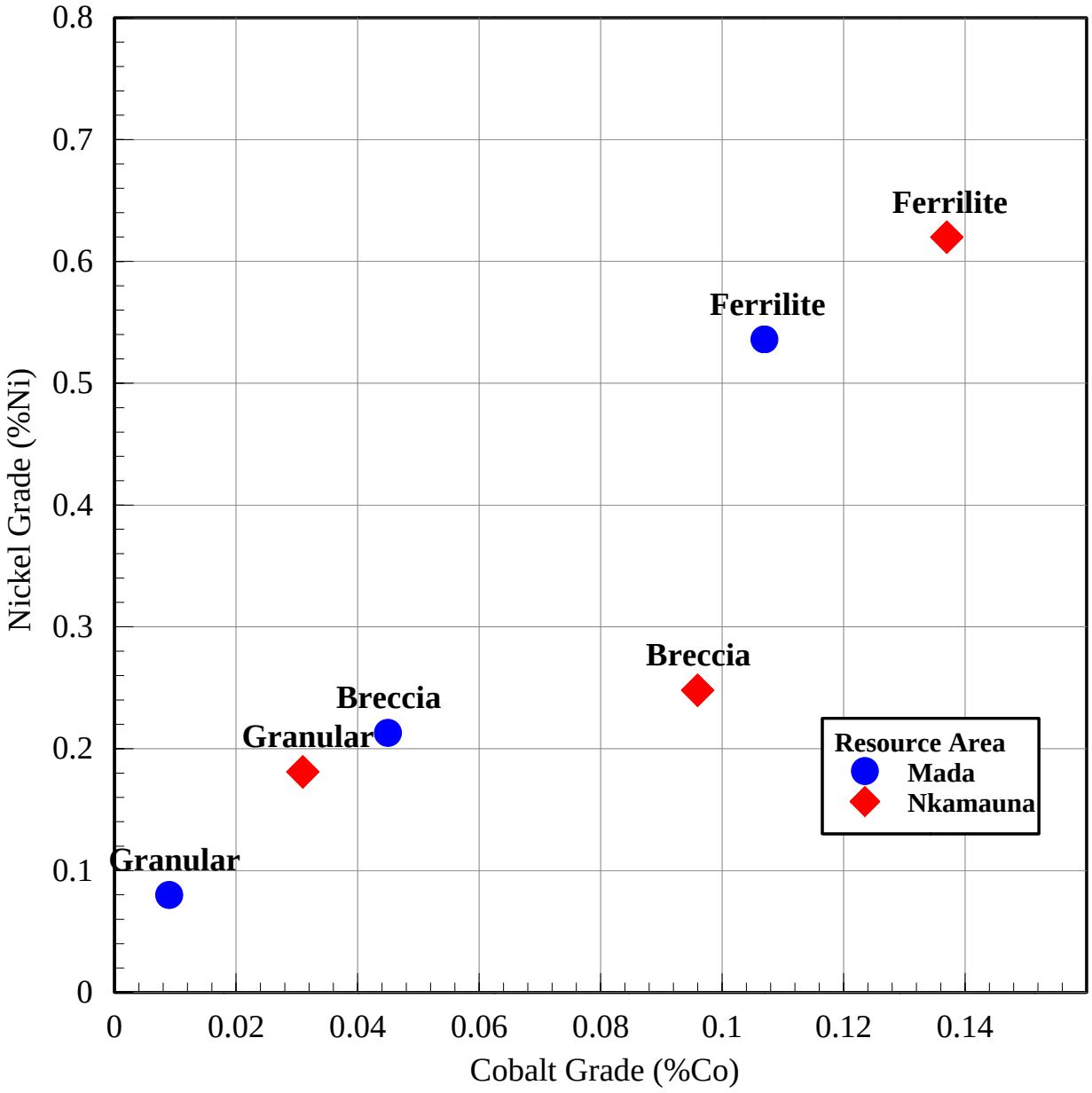
- Given the current resource classification parameters and the highly nuggetty distribution of grade in the breccia units, SRK recommends that the next model iteration attempt to subdivide the various breccia units that were combined for the 2009 resource update;
- In order to convert the current resource to higher confidence levels, SRK recommends infill drilling to a spacing of 25 m x 25 m for the following year mining and drilling the subsequent next three years planned production at a nominal spacing of 50 m x 50 m. This will allow adequate and timely definition of overburden thickness and depth to bedrock, as well as provide additional assay data for use in short range model construction to allow for better a better estimate of tonnage and grade in the short-medium term production schedule.
- The current grade estimation methodology, while scientifically defendable, is a reasonably complex and time consuming process that is likely not suitable once the project is in production. SRK is of the opinion that the process can be simplified, as well as designed to cater more directly to the needs of mine production, including detailed mine design and mine reconciliation.



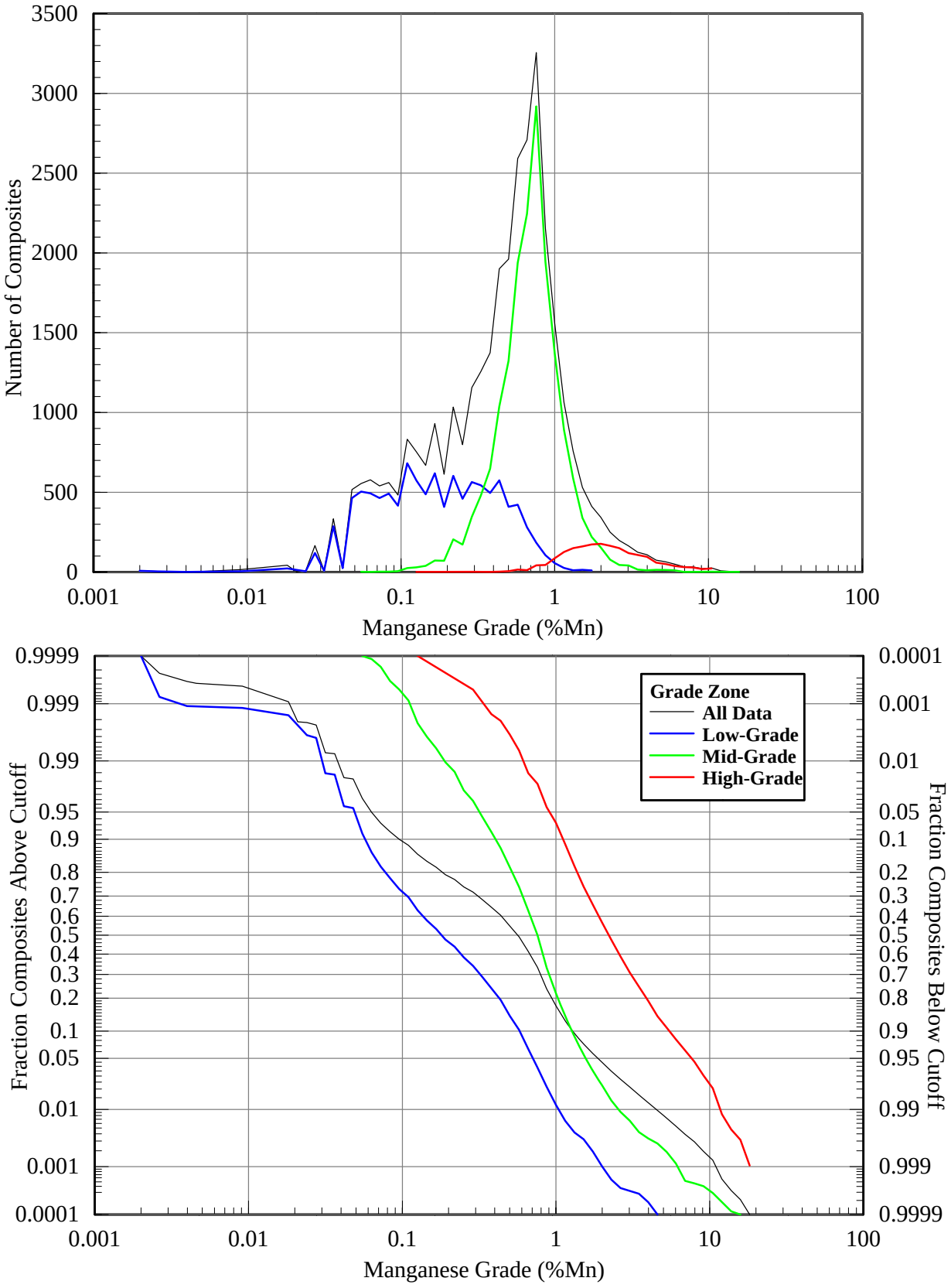
SRK Consulting Engineers and Scientists	Nikamouna and Mada Deposits East Province of Cameroon, Africa	Cross-section Plot
SRK Job No.: 195100.020		
File Name: Figure 15-1.docx	Date: 11/30/2009 Approved: JAV	Figure: 15-1



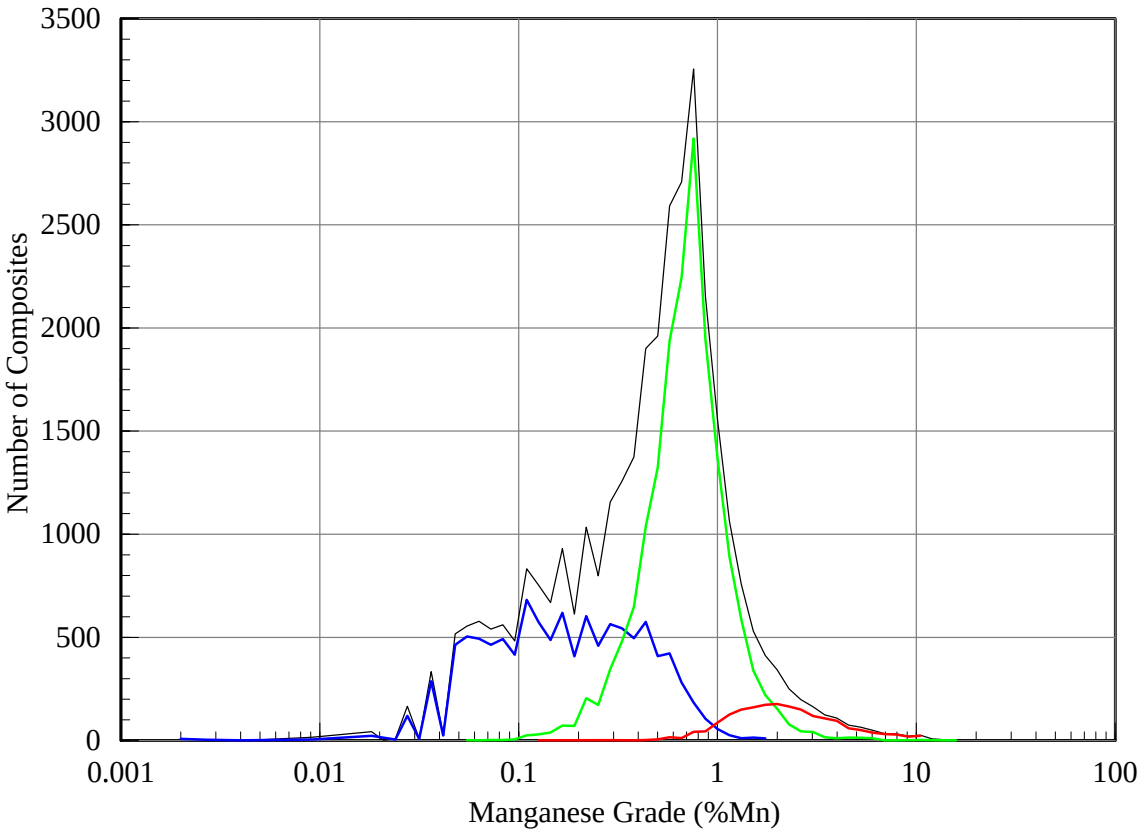
SRK Consulting Engineers and Scientists	Nikamouna and Mada Deposits East Province of Cameroon, Africa	Manganese Grade Versus Cobalt Grade for the Major Lithologic Groups		
SRK Job No.: 195100.020				
File Name: Figure15-2.docx		Date: 11/28/2009	Approved: JAV	Figure: 15-2

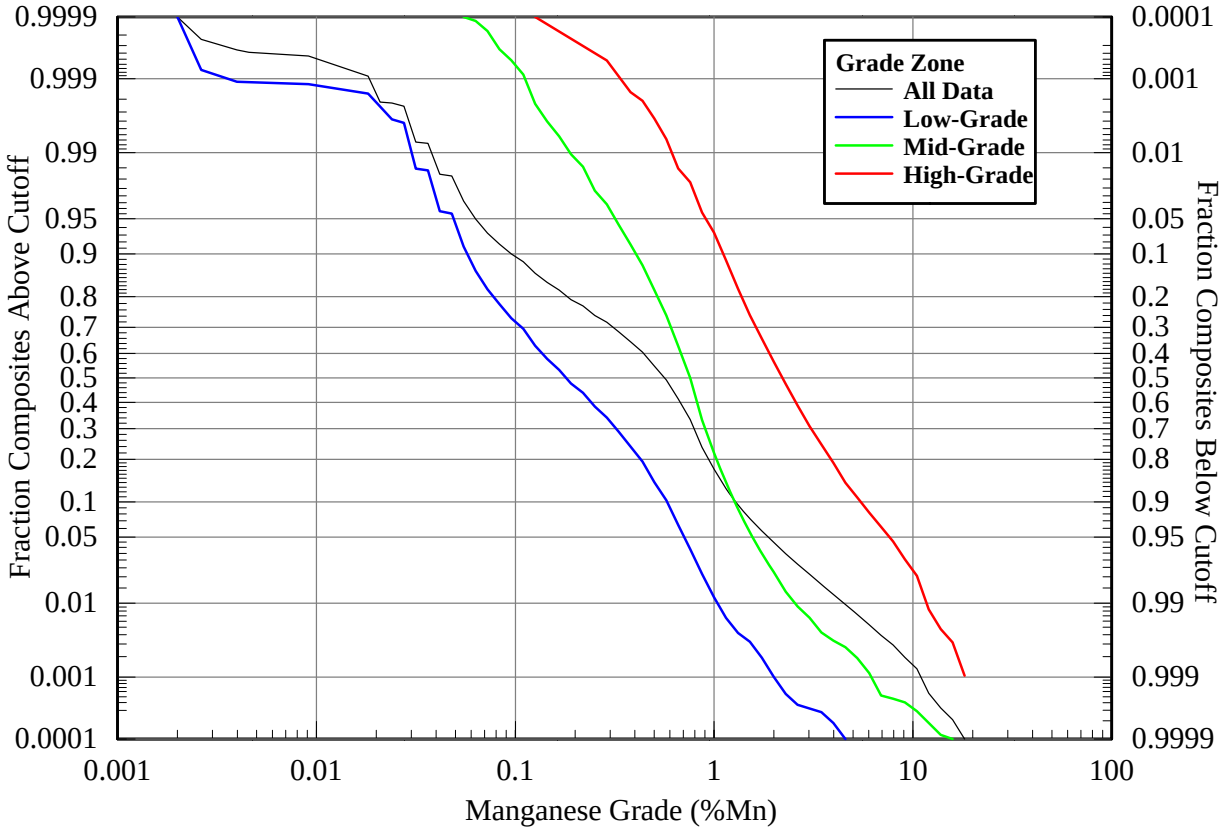


SRK Consulting Engineers and Scientists	Nikamouna and Mada Deposits East Province of Cameroon, Africa	Nickel Grade Versus Cobalt Grade for the Major Lithologic Groups
SRK Job No.: 195100.020		
File Name: Figure15-3.docx		Date: 11/28/2009 Approved: JAV Figure: 15-3

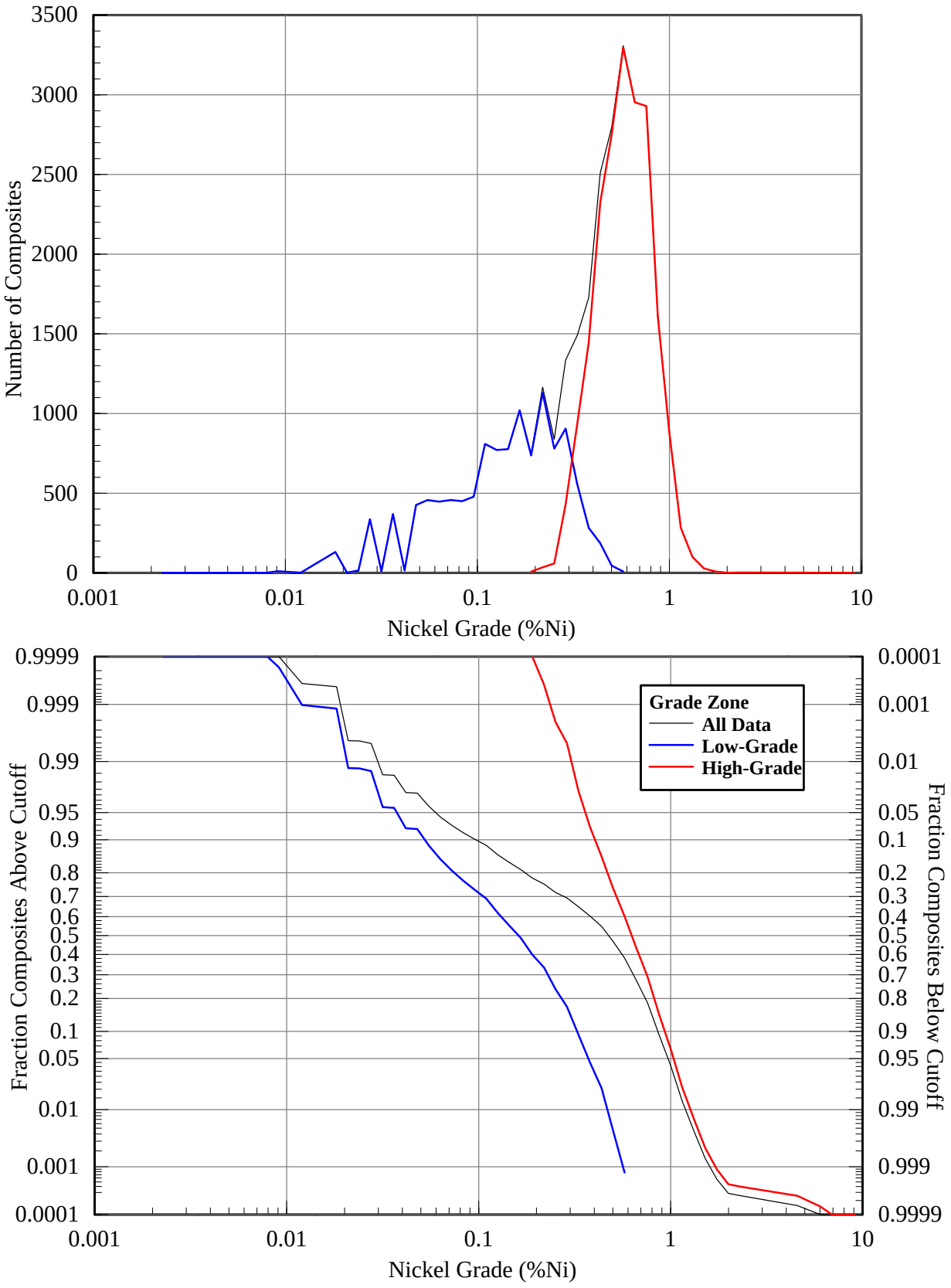


SRK Consulting Engineers and Scientists	**Nikamouna and Mada Deposits East Province of Cameroon, Africa**	**Nkamouna Deposit – Cobalt Grade Histograms and Cumulative Frequency Plots by Grade Zone**
SRK Job No.: 195100.020		
File Name: Figure15-4.docx		Date: 11/28/2009 Approved: JAV **Figure: 15-4**

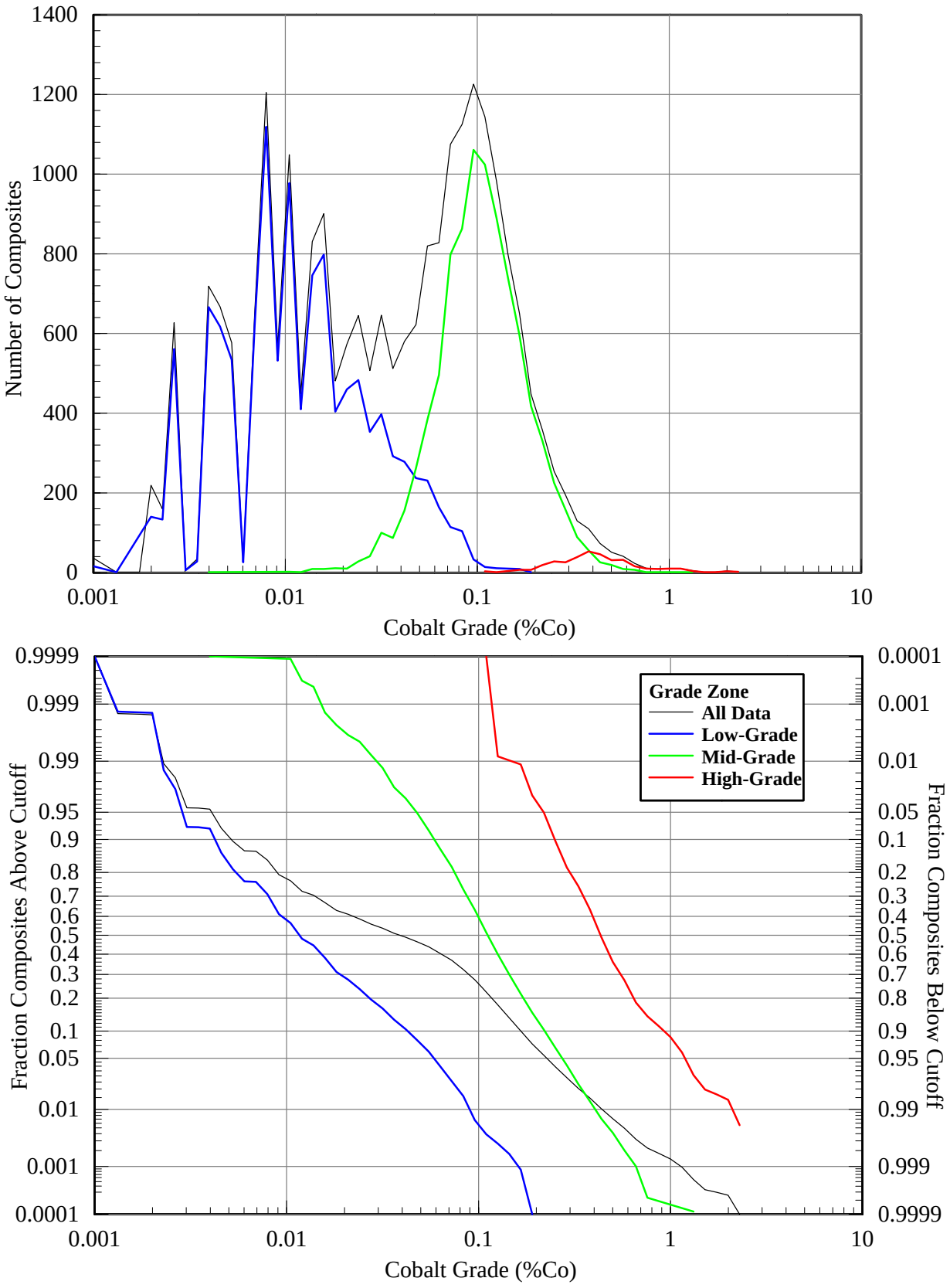




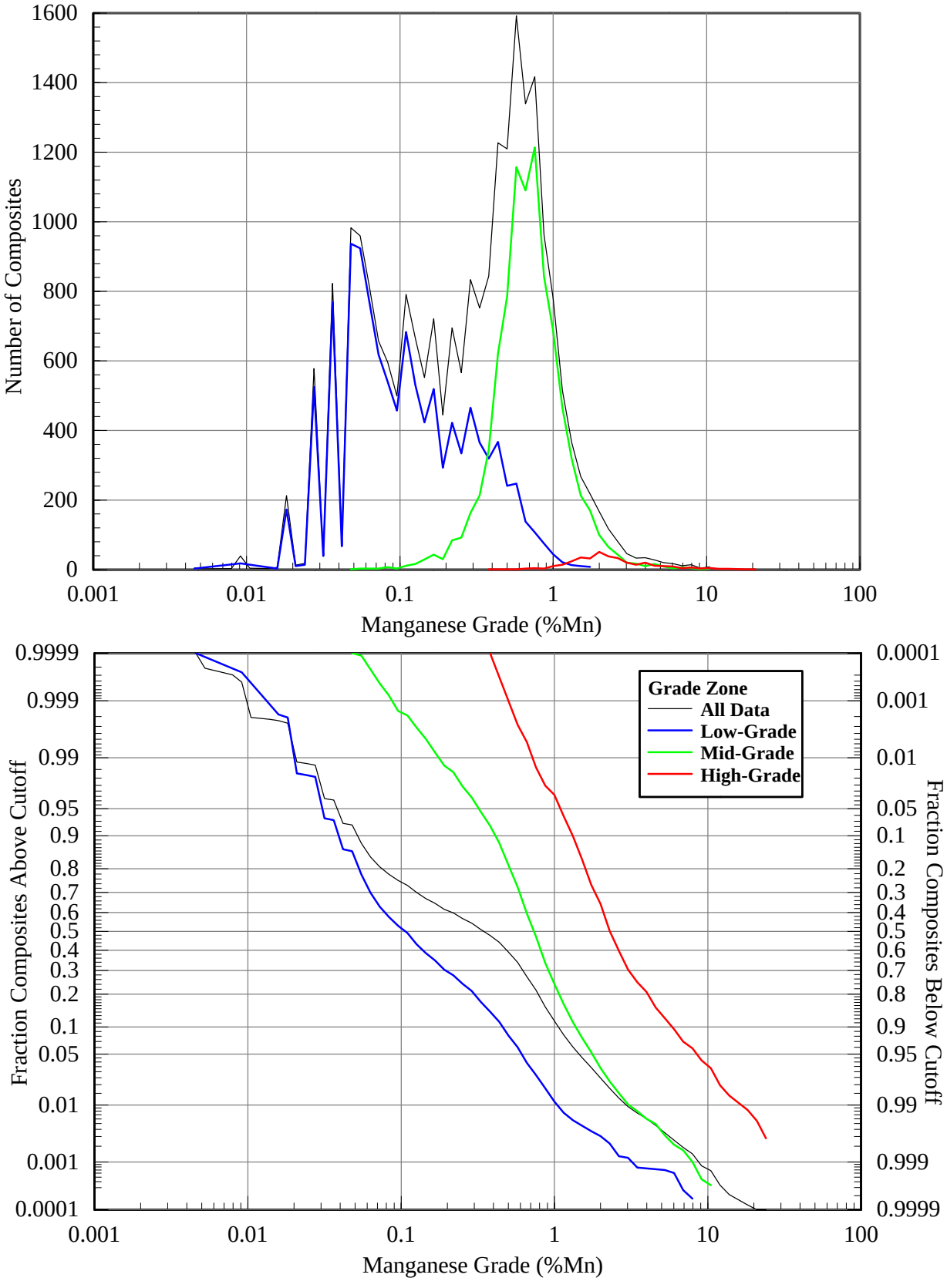
SRK Consulting Engineers and Scientists	**Nikamouna and Mada Deposits East Province of Cameroon, Africa**	**Nkamouna Deposit – Manganese Grade Histograms and Cumulative Frequency Plots by Grade Zone**
SRK Job No.: 195100.020		
File Name: Figure15-5.docx		Date: 11/28/2009 Approved: JAV **Figure: 15-5**



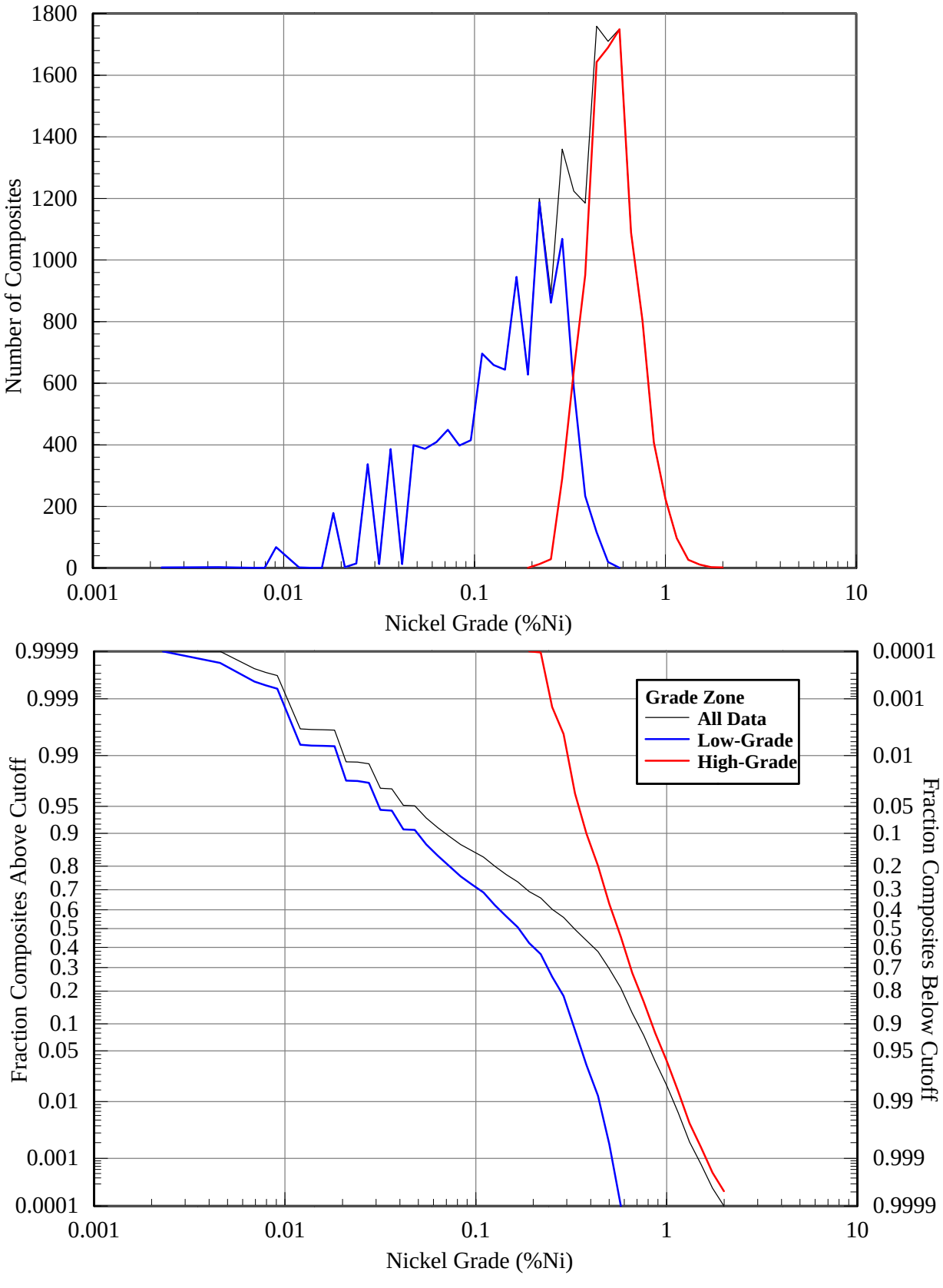
SRK Consulting Engineers and Scientists	**Nikamouna and Mada Deposits East Province of Cameroon, Africa**	**Nkamouna Deposit – Nickel Grade Histograms and Cumulative Frequency Plots by Grade Zone**
SRK Job No.: 195100.020		
File Name: Figure15-6.docx	Date: 11/28/2009 Approved: JAV	**Figure: 15-6**



SRK Consulting Engineers and Scientists	**Nikamouna and Mada Deposits East Province of Cameroon, Africa**	**Mada Deposit – Cobalt Grade Histograms and Cumulative Frequency Plots by Grade Zone**
SRK Job No.: 195100.020		
File Name: Figure15-7.docx	Date: 11/28/2009 Approved: JAV	**Figure: 15-7**



SRK Consulting Engineers and Scientists	**Nikamouna and Mada Deposits East Province of Cameroon, Africa**	**Mada Deposit – Manganese Grade Histograms and Cumulative Frequency Plots by Grade Zone**
SRK Job No.: 195100.020		
File Name: Figure15-8.docx	Date: 11/28/2009 Approved: JAV	**Figure: 15-8**



SRK Consulting
Engineers and Scientists

SRK Job No.: 195100.020

File Name: Figure15-9.docx

Nikamouna and Mada Deposits
East Province of Cameroon, Africa

Mada Deposit – Nickel Grade Histograms and Cumulative Frequency Plots by Grade Zone

Date: 11/28/2009 | Approved: JAV | **Figure: 15-9**

16 Mineral Reserves (Item 19)

LOM plans and resulting reserves are determined based on a cobalt price of US$57,761/t (US$26.20/lb), a nickel price of US$19,208/t (US$8.71/lb) and a manganese price of US$1,360/t (US$0.544/lb) for the Nkamouna and Mada deposits. These prices are based on the three year moving average for cobalt, nickel and manganese calculated in December 2010. As such, the reserves are valid for December 31, 2010.

Reserve Classification: Ore tonnes which lie within the final pit design shape are classified as proven or probable reserves based on the geological classification for measured and indicated resources. Proven reserves are measured resources within the design pit shape and probable reserves are indicated resources within the design pit shape. Inferred material which lies within the pit design is not included in the reserve statement and is treated as waste.

16.1 Conversion of Mineral Resources to Mineral Reserves

16.1.1 Modifying Factors

Ore reserves are based on the economic balance between the value per tonne of rock against the costs to mine and process each tonne of rock. The value is based on estimated metal concentration, estimated metal value and process recovery. The costs include mining, processing, overhead and rehabilitation.

For the Nkamouna and Mada deposits, a variable cut-off grade strategy was designed to high grade the deposit early in the mine life but still provide adequate reserves for a twenty year plus mine life. As such, the true definition of economic value for a block model block was not used as the differentiation of ore and waste. This is possible given the large quantum of potential resource, i.e. not reserve limited, within the mining region and strategic planning decisions governing mine cash flow.

As a reserve check, the lowest cut-off grade bins created in the production schedule and used in the economic model contained a breccia cobalt cut-off of 0.20% and ferralite cobalt cut-off of 0.12%. When only the low grade bins were used as a source of ore, the resultant cash flow was cash flow positive in the SRK economic model.

In addition to cut-off calculations, open pit modifying factors include the following:

Block Model Compositing and Analysis

This is the primary tool to determine areas of good grade continuity and mining width. Small mining areas were excluded from possible reserve selection.

Pit Design

The conversion of mineable areas to mine blocks of specific width, length and depth determine reserve quantum.

Indicated and Inferred Classification

Inferred material is excluded from optimization calculation and reserve calculation and treated as waste. Thus the classification determined by the geologist directly affects the mineable reserve.

16.2 Mineral Reserve Statement

Table 16.2.1: Nkamouna and Mada Reserve Statement by Ore Stream and Rock Type (as of December 31, 2010)

Ore Stream, Type and Class	Ore Tonnes (000's)	Co Grade (%)	Mn Grade (%)	Ni Grade (%)
Total Proven and Probable Ore	68,132	0.26	1.48	0.66

Notes:

- Reserves are based on a Co price of US$57,761/t (US$26.20/lb), a Ni price of US$19,208/t (US$8.713/lb) and a manganese price of US$1,360/t (US$0.544/lb);
- Full mining recovery is assumed;
- Mine reserves are not diluted;
- Cut-off grades are not representative of internal or break-even calculations but rather stockpile grade bin classification above 0.12% Co for ferralite and 0.2% Co for breccia; and
- In-situ Co, Mn and Ni grade does not include average metallurgical recovery of 58.66% Co, 16.43% Ni and 53.06% Mn as calculated in economic model.

Table 16.2.2: Nkamouna and Mada Reserve Statement by Rock Type (as of December 30, 2010)

Ore Type	Ore Tonnes (000's)	Co Grade (%)	Mn Grade (%)	Ni Grade (%)
Ferralite Ore	57,097	0.23	1.30	0.69
Breccia Ore	11,035	0.42	2.37	0.54
Total Proven and Probable	68,132	0.26	1.48	0.66

Notes:

- Reserves are based on a Co price of US$57,761/t (US$26.20/lb) Ni price of US$19,208/t (US$8.713 /lb) and a manganese price of US$1,360/t (US$0.544/lb);
- Full mining recovery is assumed;
- Mine reserves are not diluted;
- Cut-off grades are not representative of internal or break-even calculations but rather stockpile grade bin classification above 0.12% Co for ferralite and 0.2% Co for breccia; and
- In-situ Co, Mn and Ni grade does not include average metallurgical recovery of 58.66% Co, 16.43% Ni and 53.06% Mn as calculated in economic model.

17 Other Relevant Data and Information (Item 20)

There is no other relevant data and information for the Project.

18 Additional Requirements for Development Properties and Production (Item 25)

18.1 Mining

The Nkamouna Deposit occurs in a heavily wooded tropical environment located in eastern Cameroon. The Deposit is located in moderately undulating topography covering an area approximately 4.5 km east to west and 2 km north to south and covers an area of approximately 9 km^2. The orebody is semi-circular in shape ranging in depth from approximately 5 m to 30 m and bounded by a shiest contact defining the limit of mineralization. The deposit contains numerous small drainages which feed rivers systems down-stream of the deposit.

The two ore types breccia and ferralite are targeted for physical upgrading and mined at approximately 10,000 to 20,000 t/d using small excavators and articulated dump trucks (ADTs). Both breccia and ferralite will be transported from the pit face to run-of-mine (ROM) stockpiles located near the plant site in Nkamouna, the southern extent of Mada and an emergency stockpile between the two. From the stockpile fingers holding desired grade, the ore types are transported to the physical upgrade (PUG) plant using front end loaders and haul trucks. Waste is removed through the use of bulldozers and side casting of rehandle with long boom excavators.

The mine production schedule is based on consistent excavation of in-situ ROM material where a unit variable of cycle time multiplied by tonnage is targeted. This variable accounted for haul distance from different parts of the mine to achieve a consistent mining fleet as defined by pre-production requirements. The mine production rate allowed for sufficient high grade breccia and ferralite to be stored in stockpiles for utilization early in the mine life by the PUG plant. By doing so, the maximum high grade stream during the payback period was exploited from the deposits thus providing the leach and recovery plant with the highest ferralite and breccia concentrate grade possible. Given the accelerated mining during the ramp up period for the leach and recovery plant, ROM material is to be placed in individual grade bin stockpiles for breccia and ferralite. After 10 years of in-situ mining, the lower grade bins that are stockpiled become the major source of feed for the plant. The production schedule estimates a total stockpile storage capacity of 44.6 Mm^3 be built in three locations for ferralite above 0.12% Co and 0.20% Co for breccia. The stockpile is required from a reserve allocation and economic perspective, but is unlikely to be built if additional exploration drilling identifies other high grade low strip ratio resources.

Average annual mining rates over the first 10 years are 6.55 Mt/y of ore stockpiled and 21 Mt/y of waste relocated (not including waste rehandle) with a strip ratio of 3.22:1.

Since the mining and the pre-production earthworks for the plant, tailings dam and infrastructure will be performed by the subsequent mine equipment fleet, relatively large (construction-sized) but small mining-sized equipment has been selected. Initial major equipment will include tracked hydraulic excavators (2.5 m3 to 6.0 m3), wheel loaders (6 m3), dozers (D9 class) and 40 t articulated trucks to cope with tropical conditions in a lateritic environment. This pre-production fleet forms the basis of mine production and when the mine is operational fleet replacements and rebuilds have been costed and scheduled.

Given that Nkamouna and Mada deposits have a relatively shallow ore depth spread over a large aerial extent, strip mining was immediately identified as the lowest cost mining method versus

traditional open pit bench advance. A modified open cast procedure was then designed to take advantage of these natural conditions, allowing flexibility in strip advance, ore blending and grade control. The mine method is developed by creating cuts or panels 30 m wide and generally from 200 m to 1,000 m long within the ore reserve outlines. As dozer operation relocates waste, ore is exposed for detailed grade control analysis and liberation through excavators and articulated dump truck transport to designated stockpiles. Waste internal to the mine block and associated re-handle at the mine face is side cast into old workings. Progressive rehabilitation of mine blocks is expected to follow main mine operations.

18.2 Pit Design and Schedule

Rather than using a toe and crest pit design, the flat nature of the deposit lends itself to a stratigraphic pit layout using strips of mineable areas within an ore polygon.

Using the floor points determined during the pit identification process, a grid mesh of the pit floor is modeled and converted into a surface triangulation. Manual definition of mining areas results in a series of polygons that form the basis for strip layout. 30 m strips are orientated perpendicular to the average surface dip within the polygon and are named sequentially up dip by pit and strip number. At the end of the strip layout, small or odd shaped strips are coalesced into reasonable mining shapes.

With strips identified and labeled, polygons are projected vertically through the pit floor and topographical surface. The result of the process is a polygon projected onto the pit floor and topographical surface which can be converted into a solid triangulation. Solid triangulations form the basis of a mining block and are used in production scheduling and extraction of reserve numbers from the block model.

Areas which show marginal economic viability are also identified, but instead of applying a 30 m strip, a 100 m strip (or single polygon) is used. This allows potential mining blocks to be scheduled later in the mine life if need be. Low grade, marginal ore regions in Mada were treated as mining blocks rather than mine strips.

Figure 18-1 illustrates the results of the three block compositing runs with polygon and strip layout. Figure 18-2 gives a cross sectional view of the mine blocks as they are projected through the mine stratigraphic sequence.

18.2.1 Production Schedule

The production schedule is broken into pre-production and full-production. Preproduction includes mining of ore within the tailings impoundment footprint, plant site foundation excavation, tailings dam construction and quarry excavation. The production schedule is modeled in accordance with the economic model with pre-production beginning in October 2011 and running through to December of 2013. From January 2014, full mine production is implemented. The production schedule is modeled monthly through 2015, quarterly through 2021 and annually thereafter. At the end of 2024, no further mining of in-situ blocks is considered and production feed comes entirely from stockpiled material from the previous ten years of mining.

18.2.2 Physical Upgrading and Concentrate Grade Prediction

PUG or physical upgrading is a process in which raw run of mine ore is sized and screened to improve the quality of ore sent to the process plant. Due to the nuggetty nature of the ferralite

and breccia ore, the ability to predict concentrate grade, mass yield and recovery has been made through the use of statistical regressions (or averages) based on ROM cobalt grade. These regressions are based on raw sample data of attritioning breccia and ferralite for 6 minutes using 100 mesh screens (150 µm aperture size) and 48 mesh (300 µm aperture size) screens. Other mesh sizes have been analyzed but are not used in this study.

The ore types are particularly amenable to physical upgrading because nominally 100% of the cobalt, 82% of the manganese and 44% of the nickel are contained in asbolane accretions. Separation of the asbolane allows PUG performance to be predicted.

The steps for calculating PUG performance from cobalt, nickel and manganese head grades of breccia or ferralite ore types are:

PUG Factor	=	Conc. Grade % ÷ Head Grade % or
		Recovery % ÷ Conc. Weight %
Recovery	=	PUG Factor x Conc. Wt Ret %
Concentrate Wt.	=	Recovery ÷ PUG factor

The three equations are applied to the respective head grades of cobalt, nickel and manganese to determine respective concentrate grades for breccia and for ferralite.

The equation to determine concentrate weight is applied only to cobalt head grade concentrate grades are divided by respective head grades to determine PUG factors.

PUG factors are multiplied by the concentrate weight to determine respective recoveries.

Table 18.2.2.1 summarizes the equations for calculating PUG parameters for concentrates sized at +300 µm (100 mesh). These equations were utilized in the Chronos production schedule produced by SRK.

Table 18.2.2.1: +100 Mesh Equations to Calculate Concentrate Grades from PUG Grades

Item	Element	Equation (Concentrates Sized at +100 Mesh & Attritioned Six Minutes)
Ferralite		
Conc. Grades	Cobalt	-0.9152 x (Co Head %)^2+ 3.4721 x (Co Head %)
	Nickel	-0.5632 x (Ni Head %)^2+ 1.6436 x (Ni Head %)
	Manganese	-0.1166 x (Mn Head %)^2 + 2.7731 x (Mn Head %)
Conc. Wt, %		(8.7693 x LN(Co Head %) + 35.7357) x 97.5%
PUG Factors	Cobalt	Co Conc. % ÷ Co Head %
(P.F.)	Nickel	Ni Conc. % ÷ Ni Head %
	Manganese	Mn Conc. % ÷ Mn Head %
Recoveries, %	Cobalt	Co P.F. x Concentrate Wt %
	Nickel	Ni P.F. x Concentrate Wt %
	Manganese	Mn P.F. x Concentrate Wt %
Breccia		
Conc. Grades	Cobalt	-0.1912 x (Co Head %)^2 + 1.8536 x (Co Head %)
	Nickel	0.1647 x (Ni Head %)^2 + 1.1831 x (Ni Head %)
	Manganese	-0.0504 x (Mn Head %)^2 + 1.9501 x (Mn Head %)
Conc. Wt., %	*	(6.895 x LN(Co Head %) + 55.0873) x 97.5%
PUG Factors	Cobalt	Co Conc. % ÷ Co Head %
	Nickel	Ni Conc. % ÷ Ni Head %
	Manganese	Mn Conc. % ÷ Mn Head %
Recoveries, %	Cobalt	Co P.F. x Concentrate Wt %
	Nickel	Ni P.F. x Concentrate Wt %
	Manganese	Mn P.F. x Concentrate Wt %

*Last constant of these equations were slightly adjusted to better match the data base of derivation.

Table 18.2.2.2: +48 Mesh Equations to Calculate Concentrate grades from PUG Grades

Item	Element	Equation (Concentrates Sized at +48 Mesh & Attritioned Six Minutes)
Ferralite		
Conc. Grades	Cobalt	-1.1171 x (Co Head %)2 + 3.9296 x (Co Head %)
	Nickel	-0.878 x (Ni Head %)2 + 1.8973 x (Ni Head %)
	Manganese	-0.1428 x (Mn Head %)2 + 3.0928 x (Mn Head %)
Conc. Wt, %		(9.3311 x LN(Co Head %) + 31.0918) x 97.5%
Breccia		
Conc. Grades	Cobalt	-0.2428 x (Co Head %)2 + 1.9563 x (Co Head %)
	Nickel	0.1513 x (Ni Head %)2 + 1.2227 x (Ni Head %)
	Manganese	-0.0591 x (Mn Head %)2 + 2.0536 x (Mn Head %)
Conc. Wt., %	*	(7.4726 x LN(Co Head %) + 52.2074) x 97.5%

*Last constant of these equations were slightly adjusted to better match the data base of derivation.

Figure 18-3 illustrates the combined regression curves for expected cobalt grade, mass yield and recovery for ferralite using 100 mesh (150 µm aperture size). Inputs are referenced to ROM cobalt grades.

Figure 18-4 illustrates the combined regression curves for expected cobalt grade, mass yield and recovery for breccia using 100 mesh (150 µm aperture size). Inputs are referenced to ROM cobalt grades.

Figure 18-5 illustrates the combined regression curves for expected cobalt grade, mass yield and recovery for ferralite using 48 mesh (300 µm aperture size). Inputs are referenced to ROM cobalt grades.

Figure 18-6 illustrates the combined regression curves for expected cobalt grade, mass yield and recovery for breccia using 48 mesh (300 µm aperture size). Inputs are referenced to ROM cobalt grades.

18.2.3 Schedule Inventory Creation

Reserves which form the basis of the production schedule are calculated in two passes. The first pass exports both Nkamouna and Mada tonnages with no application of cut-off grades. The second pass includes the 'cut classification' of different ore bins for ferralite and breccia. Within Chronos, the PUG equations are applied to each rock type to determine concentrate values for breccia and ferralite using 100 mesh regressions. Table 18.2.3.1 illustrates the number of reserve runs performed and application of classification fields required to create the schedule inventory.

Table 18.2.3.1: Reserve Calculations for Schedule Inventory – Nkamouna and Mada Area

Manipulation	Variables	Cut-Off, Bins and Regression	Period Application	Comment
1	None	None	all	Base tonnage check
2	Ferralite Grade Bins	F12, F14, F16, F18, F20, F22, F24	all	F24 represents co grade greater than 0.24%
3	Breccia Grade Bins	B20, B24, B28, B32	all	B32 represents co grade greater than 0.34%
4	Target Ore Grade	Greater than F20, Greater than B28	all	Needed to ensure 0.97%>Co Cnct<1.03% in stockpiles
5	Target Concentrate Grade and Tonnage	100 Mesh Regressions	2014-2024	48 mesh implemented in 2022 of economic model
6	Cycle Time and Distance for Breccia and Ferralite	Cycle Time x Tonnage	all	Target variable for mine fleet production limit

18.2.4 Production Schedule Results

The Nkamouna and Mada production schedule is illustrated in Table 18.2.4.1.

The mine production schedule does not match the reported grade and tonnage lines in the economic model. This is because:

- 5,172 kt of F12 stockpile bin was not transported to the PUG facility at the end of the mine life as the tailings capacity had been filled. This material was left in the stockpile and not used.
- The pre-production blocks were not reported in the mine production schedule. Comprising 1,288 kt of ore, this material and grade was added to the grade bins of F12, F22, B20 and B28.

Table 18.2.4.1: In-Situ Ore Production Schedule Results

Values	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	Grand Total
Target Ore Bin Tonnage	2,408,207	2,225,546	2,142,118	2,647,511	2,532,171	2,955,259	3,407,378	2,921,680	2,196,898	2,274,204	2,135,743	27,846,716
Cycle Tonnage Production Limit	38,000,000	46,500,000	48,000,000	48,000,000	74,000,000	110,000,000	130,000,000	140,000,000	140,000,000	140,000,000	140,000,000	1,054,500,000
Target Ferralite Tonnes	1,463,253	1,590,182	1,206,356	2,208,626	1,803,606	2,281,971	2,690,671	2,251,450	1,646,191	1,760,532	1,526,500	20,429,339
Target Ferralite Co (%)	0.39	0.39	0.40	0.37	0.37	0.37	0.35	0.37	0.31	0.30	0.30	0.36
Target Ferralite Ni (%)	0.70	0.72	0.66	0.64	0.69	0.68	0.70	0.75	0.63	0.68	0.67	0.69
Target Ferralite Mn (%)	2.06	2.11	2.11	1.93	1.99	2.14	1.93	2.01	1.72	1.66	1.64	1.94
Target Breccia Tonnes	944,954	635,364	935,762	438,886	728,565	673,288	716,707	670,230	550,706	513,672	609,242	7,417,377
Target Breccia Co (%)	0.56	0.53	0.56	0.52	0.50	0.47	0.54	0.49	0.47	0.42	0.42	0.51
Target Breccia Ni (%)	0.60	0.60	0.56	0.55	0.57	0.57	0.59	0.60	0.52	0.49	0.62	0.57
Target Breccia Mn (%)	3.17	2.89	3.03	2.86	2.83	2.69	2.96	2.78	2.55	2.58	2.49	2.84
Ferralite Stockpile	2,017,486	2,559,018	1,862,397	3,009,134	3,334,078	3,603,976	5,515,010	4,533,294	4,673,897	4,997,704	4,668,451	40,774,445
Breccia Stockpile	250,258	142,609	265,908	119,271	411,180	429,473	309,796	301,996	410,059	400,148	359,014	3,399,712
Combined Stockpile Tonnes	2,267,744	2,701,628	2,128,306	3,128,405	3,745,257	4,033,449	5,824,806	4,835,289	5,083,956	5,397,852	5,027,465	44,174,157
Total Ore Mined	4,675,951	4,927,174	4,270,424	5,775,917	6,277,428	6,988,708	9,232,184	7,756,969	7,280,853	7,672,056	7,163,208	72,020,873
Waste Tonnes	9,690,164	7,143,864	10,579,685	12,174,024	12,800,096	18,826,489	19,846,852	32,346,673	28,887,205	33,083,636	46,353,451	231,732,139
Total Tonnes	**14,366,115**	**12,071,039**	**14,850,109**	**17,949,941**	**19,077,524**	**25,815,197**	**29,079,036**	**40,103,642**	**36,168,059**	**40,755,692**	**53,516,659**	**303,753,012**
Stripping Ratio	2.07	1.45	2.48	2.11	2.04	2.69	2.15	4.17	3.97	4.31	6.47	3.22

18.2.5 Fleet Estimation

Cycle time and haul distances were flagged into the Nkamouna and Mada block models and acted as reserve variables when exported to the production schedule. A 3-D haul string was drawn from the pit polygon to the process plant location so accurate distances and grade variations along the haul roads are captured. In order to estimate cycle times, articulated dump truck (ADT) rimpull statistics are emulated in Vulcan and capped at a maximum speed of 45 km/h. Table 18.2.5.1 illustrates the speed versus grade relationship that is applied to each line segment during cycle time calculation. Speeds are estimated for loaded, unloaded, uphill and downhill haul segments.

Table 18.2.5.1: Haul Cycle Time Rim Pull Parameters

Truck State	% Grade Cut-Off	Uphill km/h	Downhill km/h
Loaded	2	45	45
	5	20	40
	7	13	35
	10	8	21
	12	7	21
	15	6	14
	20	4	14
	40	2	6
Empty	2	45	45
	5	45	45
	7	35	34
	10	25	34
	12	22	21
	15	19	21
	20	13	21
	40	8	14

Table 18.2.5.2 illustrates the resultant average ore cycle times for each production period modeled. Additional delays for loading, spot time, dump time and general delays are added in the economic model. The average haul speed has been estimated based on cycle time and distance. It should be noted that the average speeds are closer to the averaged capped speed of 45 km given the flat topography with ore coming from Mada. An additional three minutes of unburdened cycle time was used for all stockpile rehandle.

Table 18.2.5.2: Haul Cycle Time and Distance (unburdened with additional delays)

Year	Total Ore Mined	Average Ferralite Cycle Time (mins)	Average Ferralite Haul Distance (km)	Average Ferralite Haul Speed (km/h)	Average Breccia Cycle Time (min)	Average Breccia Haul Distance	Average Breccia Haul Distance (km)
2014	4,675,951	8	2,240	33	8	2,166	34
2015	4,927,174	9	2,478	33	10	3,118	36
2016	4,270,424	11	3,119	35	15	4,872	39
2017	5,775,917	8	2,308	34	10	3,080	37
2018	6,277,428	10	2,728	34	16	5,146	38
2019	6,988,708	12	3,389	34	22	6,958	39
2020	9,232,184	17	5,471	39	16	5,274	39
2023	7,672,056	24	8,194	41	30	10,344	42
2021	7,756,969	23	7,778	41	25	8,720	42
2022	7,280,853	21	6,972	40	26	8,756	41
2024	7,163,208	22	7,378	40	28	9,741	41
Grand Total	**72,020,873**	**17**	**5,453**	**39**	**23**	**7,646**	**41**

18.2.6 Disturbance Schedule

Disturbance areas have been calculated based on the proposed mine sequence for Nkamouna and Mada mining areas. Areas deemed as pre-production have been grouped together and will vary according to project execution. The disturbance has been broken into areas that are currently

(2011) inside the lease boundary and those outside. There is a reasonable expectation that additional lease area will cover the Mada areas as development commences. Table 18.2.6.1 illustrates the annual disturbance within and outside the current lease boundary.

Table 18.2.6.1: Annual Disturbance Schedule (Update pre-prod when detail arrives)

Period	In Lease m^2	Out of Lease m^2	Total m^2
PreProduction	5,364,858	1,938,821	7,303,679
Year 1	494,338		494,338
Year 2	409,008	35,200	444,208
Year 3	387,465	231,061	618,526
Year 4	509,024	145,234	654,258
Year 5	552,051	234,136	786,187
Year 6	420,422	451,569	871,991
Year 7	865,612	151,544	1,017,156
Year 8	482,767	772,651	1,255,418
Year 9	533,476	723,329	1,256,805
Year 10	972,569	269,348	1,241,917
Year 11	581,880	1,911,259	2,493,139
Total	**11,573,469**	**6,864,152**	**18,437,621**

A two year lag between mining and final rehabilitated areas has been assumed, but it is likely that, during operations the time difference between excavation and rehabilitation will be shortened, thus reducing the rehabilitation balance to less than 60 Ha as required in the mine permit.

18.3 Pre-Production Development

18.3.1 Clearing and Grubbing

Before mining operation commence, it will be necessary to remove saleable hardwood from strip locations. It has been assumed that saleable hardwood will be tendered to a local timber contractor for extraction and processing of available hardwood. Areas to be cleared also include the tailings dam footprint, mine services area, mine access roads, ROM stockpile, soil stockpile, storm water management diversions, plant site and soil stockpile areas and pre-production mine development. The primary clear and grubbing equipment will be the D8 and D9 bulldozers.

18.3.2 Diversion and Sediment

Catchment areas have been defined in Figure 18-7. They are based on Nkamouna and Mada contour maps. As mining progresses, every attempt will be made by production staff to classify catchments as either dirty or clean depending on possible sediment transportation to collection areas. Where necessary, open diversion drains will be constructed using excavators and ADTs to divert upstream water from mining areas to clean catchments. The contour diversion drains will be graded at 1% to minimize water flow (scouring) and allow heavy sediments to drop out of suspension.

Water downstream of mining areas will report to sedimentation control structures. These structures will be designed to reduce turbidity levels before discharge into current river tributaries. This concept will keep 'Clean' water clean and manage 'Dirty' water. If heavy element or external contaminates are discovered as part of the monitoring policy, remediation

measures will be conducted at each sedimentation pond. It is expected that water management in the mine areas will not require excessive pumping given the up-dip nature of mine progression.

18.3.3 Serpentinite Quarry

As part of preproduction activities, it will be necessary to generate crushed and screened hard serpentinite for tailings dam construction, cement aggregate and road course. Three serpentinite locations have been sourced with one area containing geotechnical logs and testing results. The approximate volumes (as of October 2010) are listed in Table 18.3.3.1.

Table 18.3.3.1: Primary Serpentinite Volume Estimates

Outcrop	Thickness Serpentinite m	Area m^2	Volume m^3	Haul Time (avg CCD and PUG dam) mins	Comments
1	20	8,522	170,447	20.5	Contains Geotechnical Logs
2	15	45,147	677,205	19	No Drill Holes - Estimated
3	n/a	n/a	n/a	12	No Survey - identified in field

Table 18.3.3.2 illustrates the October 2010 estimates for serpentinite usage. Although the second serpentinite source has not been tested for quality, it is expected additional geological information will become available before construction begins in 2011. The combined sources are currently estimated to contain sufficient quarry resources to meet the demands of site construction.

Table 18.3.3.2: Estimated Serpentinite Usage

Usage	Type Aggregate	Quantity m^3
Plant Site	Concrete	14,000
	Rip-Rap	25,000
	General Ground Cover	15,000
	Road Base	60,000
Tailings Pond	Chimney Drain Filter Zone	47,224
	Filter Blanket	45,141
CCD Pond	Under Drain Aggregate	103,000

Due to the specialized nature of quarry operations, it is expected that the drill, blast and crushing operations will be handled by third party contractors. It will be the contractor's responsibility to manage explosive permits and magazines, mining, crushing and loading of serpentinite. There is also a mobile rock breaker sourced by the process plant that may be available for small quarry jobs. GeoCam will be responsible for clear and grubbing as well as transportation of crushed rock to designated construction areas. Without drill, blast and haul cost, an international mining contractor has provided an initial quotation of crushing services at US$5.20/t.

As mining operations progress, it is expected alternate sources of road wearing course and construction materials may be discovered. Potential products include ferricrete which is high iron content laterite formed above the saprolite orebody, if deemed acceptable for road course, the ferricrete will be screened and stockpiled for later usage. Other road course and construction products include gravels from local flood plains and aggregates from local suppliers in Abong Mbang. It is expected exploration and sourcing of these materials will continue through 2011.

18.4 Mining Method

The Nkamouna deposit has a relatively shallow ore depth spread over a large aerial extent. With this in mind, strip mining was immediately identified as the lowest cost mining method versus traditional open pit bench advance. A modified open cast procedure was then designed to take advantage of these natural conditions allowing flexibility in strip advance, ore blending and grade control.

As noted in previous sections, the ore reserve is initially identified using block and stratigraphic modeling of the drilling data. Once identified it is then separated into distinct logical mining units (LMUs) which were further cut into 30 m strips parallel to the slope. An overall view of the Nkamouna project logical mining units by year along with the first five years of proposed access roads are given in Figure 18-8.

18.4.1 Mine Operations Pit Development

In Figure 18-9 one can also see that each LMU is split by the main haulage ramp into two distinct sides allowing each side to progress at different mining rates as grade control and blending programs dictate. Current estimates are that 2 to 3 years are required to mine a complete LMU. Estimated blending requirements indicate that three LMUs are required to be in production which will allow up to six separate mining faces. An example of this mine production scenario is given in Figure 18-10.

18.4.2 Box Cut and Overburden Relocation

Mining of an LMU begins with the development of the boxcut at the lowest elevation of the LMU. Overburden from the boxcut and half of the next cut is hauled to stockpiles placed on the edge of the LMU near the last cut of the LMU as shown in Figure 18-11. As the last cut of the LMU is mined, the boxcut overburden is then spoiled into the final cut allowing regrading and reclamation to proceed.

After overburden and ore are removed from the initial boxcut, a stripping sequence then is implemented where overburden is stripped ½ cut in advance and dozed into the previous cut. Once overburden is stripped down to ½ to 1 m above the ore level for the entire cut, drilling will commence for grade control. Grade control drilling will be implemented on 5 m spacing as determined in a preliminary sample spacing study conducted by SRK. Contiguous with grade control drilling, excavators with a nominal 2.5 m^3 bucket capacity will be used to rehandle spoil or wasted overburden along the ore/waste interface as seen in the cross-section view in Figure 18-11.

18.4.3 Ore Mining

Following grade control drilling, excavators with a 6 m^3 nominal bucket capacity will begin mining ore along with wasting of any interburdens. The excavators will sit on the waste or the very top of the ore grade breccias for ore mining.

Typically both breccia and ferralite ore zones will be mined in separate sequences by either separating the breccias and ferralite in a single pass lift or by mining the entire breccia sequence down to ferralite and returning for a second pass along the cut to remove the ferralite ore. A plan view of a typical sequence can be seen in 18-9.

Ore liberated above the economic cut-off will be transported directly to the PUG plant or ROM stockpile. Marginal material, which is still classified as ore but destined for later processing, will be stockpiled near or over existing mined areas.

18.4.4 Reclamation

The overburden relocated by bulldozers or through rehandle, will be re-graded in preparation for reclamation. It has been estimated that the delay between ore removal and rehabilitation will be approximately four strips of development. As mining operations are optimized during production, it will be necessary to control the reclamation to ensure disturbed land areas are kept below annual permitted disturbance levels. Soil from clear and grubbing operations in new pit strips will be directly transported to contoured spoil piles to finish reclamation.

18.4.5 ROM Stockpile Management

The production schedule is aimed at full utilization of the mining fleet to uncover high grade material early in the mine life. To do so, higher grade stockpile material is fed to the PUG plant first and lower grade material stockpiled for later use when mining is finished. Although it is unlikely that the stockpiles will ever get as big as designed given additional sources of ore to be delineated once exploration drilling recommences, for the purpose of this study they must be defined and sized.

Three stockpile locations have been identified and sized. The main ROM stockpile is located to the west of the process plant as shown in Figures 18-8 and 18-10 and has been sized for 25 Mm^3 of ore. This area will store the majority of Nkamouna ore as well as the majority of high grade feedstock. A second ROM stockpile is located in the Mada area and represents an additional 15 Mm^3 of stockpile volume. A third 'emergency stockpile' has been located in between Mada and Nkamouna and represents an additional 10 Mm^3 of storage if needed.

The stockpile quantities have been estimated in Table 18.4.5.1**.**

Table 18.4.5.1: Maximum Run of Mine Stockpile Volume and Capacity (2024)

Max Stockpile Bins - 2024	Required Tonnage (kt)	Required Volume (000's m^3)	Nkamouna stockpile (m^3)	Mada Stockpile (m^3)	Emergency Stockpile (m^3)
Ferralite Ore Bins					
F12	8,763	8,426	9,821.1	5,157	3,429
F14	11,975	11,514			
F16	8,640	8,307	7,518.3	4,131	2,766
F18	6,549	6,297	5,670.0	3,483	2,307
F20	2,177	2,093			
F22	-	0			
F24	2,937	2,824			
Total Ferralite	***41,040***	***39,461***	***23,009.4***	***12,771***	***8,502***
Breccia Ore Bins					
B20	2,084	2,004	1,579	958.5	664.5
B24	1,347	1,295	973	729	473.25
B28	1,070	1,029			
B32	893	859			
Total Breccia	***5,187***	***5,187***	***2,552***	***1,687.5***	***1,137.75***
Total Stockpile	**46,434**	**44,648**	**25,561.4**	**14,458.5**	**9,639.75**

Front end loaders will be used to transfer material from stockpile fingers to the PUG plant and when possible, mining trucks will directly dump ore from one of the three tip heads. The PUG surge bin has the capacity to store 150 t of ore while continuously feeding the apron feeder. Since there are three tip heads, the PUG plant will accommodate one truck load from each tip head every 10 minutes or one truck load every three minutes. The PUG throughput production (623 wt/h) would equate to a truck clearance time of four minutes. It has been estimated a haul truck will deliver an ore load every six minutes during full production on day shift. Unless there is a blockage or downtime in the crusher, there should not be any limitation with regards to the truck clearance rate on mine production.

18.4.6 Grade Control

In order to assess grade control sample spacing requirements, SRK conducted a preliminary sample spacing analysis using the existing exploration data set. SRK understands that GeoCam are planning to conduct grade control using air coring drilling techniques and assumes that the resultant sample data will be statistically similar to the exploration data utilized in this analysis. SRK also assumes that the sample spacing statistics of the cobalt analyses will dictate the sample spacing requirements for the grade control process. The optimal grade control sample spacing is 5 m x 5 m in ore. Assay results will be stored in a grade control database and subsequently analyzed by grade control engineers. Engineers will then tape out stockpile and grade bin designation on the pit face for the excavators to dig to.

18.4.7 Primary Production Fleet

The production equipment fleet is based on the life-of-mine (LOM) production schedule developed by SRK. The equipment requirements are estimated based on the mining of 236.9 Mt of prime waste and 68.1 Mt of ore as detailed in the economic model (not in-situ schedule). In addition, there will be complete rehandle of ore and an additional 35.5 Mt of waste rehandle within the pit limits. The primary equipment fleet consists of hydraulic excavators, articulated dump trucks and dozers. In addition to this primary equipment there will also be front end loaders (FELs), dozers, rubber tire dozers (RTDs), graders, water trucks, backhoe, and compactors.

As the pre-production fleet will be utilized during full mine production, the production schedule and hence fleet estimation was designed to maintain a balanced haul fleet through 2018. After that time, the stripping ratio begins to increase and required haul units increase proportionally along with longer hauls from Mada. The maximum fleet has been costed in 2021-2024 when the combination of stripping ratio and haul distance are at the mines maximum. Table 18.4.7.1 illustrates the mine production fleet available when mining commences in 2014 and the maximum number of equipment required during the mine life.

Table 18.4.7.1: Production Equipment Fleet

Model	Initial	Replacements	Additions
CAT 374D	3	9	6
CAT 345C L	0	0	6
CAT D9T	2	11	18
CAT 740	12	53	19
CAT CS64	1	0	0
CAT CP64	2	0	0
CAT CP76	2	0	0
CAT 988	1	1	2
CAT 980	1	1	0
CAT D8T	3	0	0
CAT 834H	1	3	2
CAT 14M	4	0	0
Water Truck (27 kL)	1	2	1
CAT 444E	1	1	0
Cubex QXR1120	3	0	3

The maximum number of units that are estimated after 2021 - 2024 have been costed for reserve expansion rather than optimization of mine fleet utilization or cash-flow. The fleet requirements do suggest alternate and/or improved confidence of low stripping reserves will be required after 2020 when the majority of the low stripping ratio Nkamouna reserves have been depleted.

The primary loading and hauling equipment has been selected to provide good synergy between mine selectivity of ore grade and ability to operate in wet conditions. As part of the unit price estimation the relationship of loader size and haul truck parameters must be calculated.

Table 18.4.7.2 illustrates the unit cost breakdown for the each major earthwork process estimated for the mine production schedule. On a cost per tonne basis, the load and haul of ore material contributes the majority of mining costs. Conversely, the overburden relocation using dozers (As opposed to load and haul) provides the most significant cost savings. As with the sustaining capital schedule, mining cost will benefit with the optimization of the production schedule after 2022.

Table 18.4.7.2: Unit Operating Costs – Reference Mining Cost

	2014-2024		2025-2037	2014-2037		
Operation	In-situ Mining	Stockpile Rehandle	Stockpile Rehandle	Total LOM		
Units	$/ -ROM	$/t-Stockpile	$/t-Stockpile	$/t-ROM moved*	$/t-Moved	LOM ($000's)
Loading	$0.80	$0.42	$0.42	$0.61	$0.20	$82,633
Dozing	$0.75			$0.37	$0.12	$50,353
Hauling	$0.85	$0.32	$0.32	$0.58	$0.19	$79,025
Roads, Dumps & Stockpiles	$0.42		$0.25	$0.30	$0.10	$40,760
Sample Drilling	$0.27			$0.13	$0.04	$17,941
Mining Support Equipment	$0.09		$0.04	$0.05	$0.02	$ 7,418
Mine G&A	$0.57		$0.28	$0.37	$0.12	$50,938
Total Mining	$3.76	$0.74	$1.30	$2.41	$0.81	$329,067

* $/t ROM moved – ROM moved includes re-handle tonnage in addition to in-situ ore tonnage.

18.5 Process Description

The process plant consists of two process circuits, namely the physical upgrade plant (PUG) and the leach and recovery circuit.

The objective of the PUG plant is to concentrate the cobalt, nickel and manganese by rejecting the fine gangue material.

The objective of the leach and recovery circuit is to dissolve the value metals from the PUG concentrate and then recover cobalt and nickel as a mixed sulfide product and manganese as a manganese carbonate product from the PUG concentrate.

Mined ore will be deposited in a ROM bin from where it will report to the primary crushing circuit. Crushed material will be slurried in a paddle mixer. The slurry will report to a screen where oversize material will be removed and presented to a secondary crushing circuit for further size reduction. Secondary crushed material will be re-introduced to the paddle mixer.

Slurried ore will report to an attritioning circuit where fine gangue material will be scrubbed from the coarse asbolane mineral surfaces. Dewatering cyclones and hydrosizers will be used to separate the coarse and fine material from the attritioned product to yield a PUG concentrate representing approximately one quarter of the original PUG circuit feed tonnage. This classification step will result in recovery of the coarse material to a higher grade PUG concentrate.

The PUG concentrate will be stockpiled in a combination of live and standby stockpiles. The PUG concentrate will be reclaimed and milled in an open circuit ball mill to produce a concentrate slurry with a nominal particle size P_{80} of 106 µm.

The milled concentrate will be leached with finely ground pyrite and sulfuric acid at atmospheric pressure and at 95°C. The slurry will then be passed through a primary purification circuit where the pH will be adjusted using soda ash to precipitate and remove aluminum and iron impurities. Co-precipitation of some additional minor impurities also occurs in this stage. The leach liquor will be recovered through a counter current decantation (CCD) circuit.

The washed slurry from the CCD circuit is sent to the CCD tailings storage facility. The recovered leach solution will be sent to a secondary purification circuit where additional soda ash will be added to further remove aluminum and iron impurities. The barren solids will be recycled to the leach circuit to recover any valuable metals that co-precipitate.

The recovered leach solution will be sent via a heat exchanger to the sulfide precipitation circuit. A mixed cobalt and nickel sulfide product will be precipitated by the addition of sodium hydrosulfide and soda ash.

The mixed sulfide slurry will be thickened, filtered, and packaged as a wet filter cake into sealed bulk bags. The cobalt and nickel barren solution will be pumped to the third purification stage for further impurity removal ahead of the manganese precipitation circuit.

Purified solution will be transferred to the first of two stages of manganese carbonate precipitation. Manganese carbonate will be precipitated by the addition of soda ash. Thickening and belt filtration will be used to recover the solid manganese carbonate products.

Figure 18-12 is a summary process flowsheet for the PUG circuit. Figure 18-13 is a summary process flowsheet for the leach and recovery circuit. Figure 18-14 provides a site general

arrangement drawing of the process plant, TSF and accommodation village site and Figure 18-15 provides a general arrangement drawing of the process plant.

18.5.1 Process Design

Process design for the two circuits is based on parameters that were generated during the testwork phases of this study. The general design basis for years 1 to 12 of the Project are presented in Table 18.5.1.1.

Table 18.5.1.1: General Design Basis (Years 1 -12)

General Information	Units	Data	Source
Average PUG plant feed tonnage, Years 1 to 12	t/y (dry)	2,047,000	SRK
Average PUG plant feed tonnage, Years 13 to 24	t/y (dry)	3,631,000	SRK
Operating days per year	days	365	GeoCam
Utilization	%	90	GeoCam
PUG plant operating hours - nominal	h/d	16	GeoCam
Ore moisture	%	25	Testwork
Design head grade – cobalt	%	0.41	GeoCam
Design head grade – nickel	%	0.66	GeoCam
Design head grade – manganese	%	2.24	GeoCam
Design PUG recovery mass	t/y (dry)	656,000	Testwork
Design concentrate grade – cobalt	%	1.00	GeoCam
Design concentrate grade – nickel	%	0.85	GeoCam
Design concentrate grade – manganese	%	5.34	GeoCam
Cobalt recovered in mixed sulfide	t/y (dry)	6,158	Testwork
Nickel recovered in mixed sulfide	t/y (dry)	3,389	Testwork
Manganese recovered as $MnCO_3$	t/y (dry)	65,308	Testwork

18.5.2 Engineering Design Philosophy

The processing facility comprises a series of in line unit operations and activities. Adequate surge capacity will be installed between each of these unit processes to ensure that short term interruptions to any individual unit process will not halt production.

The processing plant design is such that short duration routine planned maintenance activities will be undertaken while the plant is operational, with longer duration maintenance activities being undertaken during scheduled total plant shut downs. Design of the process plant will reflect:

- A robust process flowsheet;
- Sturdy, well proven and easy to maintain equipment;
- Provision of installed spares on equipment subject to high wear or susceptible to failure;
- Provision of bypass pipework and launders to enable individual tanks to be bypassed in the event of tank or agitator failure;
- Sufficient instrumentation and automation to achieve the design production rates, to enable stable process operation and to facilitate safe operation;
- Control of the majority of equipment in the plant will be via the Supervisory Control and Data Acquisition (SCADA system) with only complex equipment being fitted with on board PLCs; and

- The SCADA system will be used to monitor and control process equipment in the plant. The process control system will have process data trending and alarming capabilities to facilitate troubleshooting.

Underlying the engineering design philosophy is the requirement for a robust process plant design to offset potential lost availability due to unscheduled maintenance requirements and less than optimal operator skill levels during the early years of operation.

18.5.3 Plant Operating Schedule

The PUG circuit is designed to treat 2.9 Mt/y (8,695 t/d) of run-of-mine (ROM) ore to produce 0.656 Mt/y (2,000 t/d) PUG concentrate for subsequent processing through the leach and recovery circuit. Process plant operating design criteria are summarized in Table 18.5.3.1.

Table 18.5.3.1: Process Plant Operating Schedule

Description	Unit	Value	Source
PUG			
Nominal Operating Time	hours / day	16	Geovic
	days / year	365	Geovic
	hours / year	5,248	Calculated
Utilization	%	60	Geovic
Grinding / Leach & Recovery			
Operating Time	hours / day	24	Geovic
	days / year	365	Geovic
	hours / year	7,872	Calculated
Utilization	%	90	Geovic

18.5.4 ROM Ore Crushing and Repulping

Design criteria for ore crushing and repulping are shown in Table 18.5.4.1. ROM ore will be delivered from the open pit mining areas by truck and directly tipped into the ROM bin or stockpiled on or adjacent to the ROM pad.

Crushed ore will be conveyed to a paddle mixer, which will actively blend the ore with water to a pulp density of approximately 50% $^{w}/_{w}$ solids. The paddle mixer product slurry will discharge directly onto a double deck vibrating screen. The screen oversize material will be recycled to a secondary crusher, while the fines will be pumped to a distributor, splitting the pulp between the four banks of attritioning cells.

Table 18.5.4.1: Design Criteria for Ore Crushing, Handling and Storage

Description	Unit	Value	Source
Ore dump truck capacity	tonnes	40	Geovic
ROM Bin Capacity	tonnes	150	Lyco
	m^3	94	Lyco
Primary Crusher			
- Type		Mineral Sizer	Geovic
- Size (roll diameter)	mm	500	Lyco
- Configuration		4 tooth 9 ring	Lyco
Primary Crusher Feeder			
- Type		Apron Feeder	Lyco
- Capacity	t/h	866	Lyco
Secondary Crusher			
- Type		Impact Crusher	Lyco
- Size		Terex Canica or similar	Vendor
Paddle Mixer			
- Type		Twin shaft	Lyco
- Maximum feed size	mm	150	Vendor
- Discharge density	%solids	50	Testwork
Paddle Mixer Product Screen			
- Type		Vibrating, double deck	Lyco
- Size	mm	1,800 x 6,100	Lyco
- Top deck aperture	mm	35	Lyco
- Bottom deck aperture	mm	6	Lyco

18.5.5 Attritioning and Classification

The attritioning and classification circuit will consist of four banks of attritioning cells, each bank having four cells in series. Slurry discharge from each bank will be pumped to a dedicated dewatering cyclone, where approximately 50% of the solids will report to the overflow (fine fraction) and 50% to the underflow (coarse fraction).

The dewatering cyclone underflow will gravitate to a pair of hydrosizers. Teeter water will be introduced to the distribution manifold of each hydrosizer to effect separation of the coarse, dense particles from the fine, less dense gangue material. The fine particle stream will overflow a peripheral launder where it will be combined with the dewatering cyclone overflow slurry and subsequently discharged to the PUG tailings facility. The coarse 'heavy' underflow slurry from each hydrosizer will discharge to a dedicated dewatering screen. Fine material and water passing through the dewatering screen will be pumped back to the hydrosizer inlet, while the dewatering screen oversize (PUG concentrate) will be conveyed to a 'live' stockpile with a 2,000 tonne capacity. Alternatively, concentrate may be diverted to an 'intermediate' stockpile and subsequently transferred by front end loader (FEL) to a number of 'dead' stockpiles. These will provide approximately 90,000t of buffer concentrate storage capacity as a contingency against reduced or curtailed mining activity during heavy rainfall periods.

Table 18.5.5.1: Design Criteria for Attritioning and Classification

Description	Unit	Value	Source
Attritioning Cells			
- Number of parallel banks	#	4	Vendor
- Cells in series per bank	#	4	Vendor
- Type		Denver 80” x 80” or similar	Vendor
- Installed power per cell	kW	110	Vendor
- Residence time	min	6	Geovic
Dewatering Cyclones			
- Number of parallel units	#	4	Lyco
- Type		HS2420 Hydrocyclone	Vendor
- Feed density	% solids	24	Vendor
- Feed F_{80}	µm	367	Calc
- Cut size D_{50}	µm	150	Calc
Hydrosizers			
- Number of parallel units	#	2	Lyco
- Type		DMS105 Hydrosizer	Vendor
- Separation size	µm	150 (year 1 to 9)	Geovic
	µm	300 (year 10 to 20)	Geovic
- Feed density	% solids	65	Vendor
- Underflow density	% solids	68	Vendor
- Overflow density	% solids	24	Vendor
- Underflow solids per unit	t/h	62.5	Mass Balance
Dewatering Screens			
- Number of parallel units	#	2	Lyco
- Type		VD-15 Vibrating, single deck	Vendor
- Size	mm	1,500 x 3,600	Vendor
- Aperture	mm	0.250	Lyco

18.5.6 Concentrate Grinding

The PUG concentrate grinding circuit will consist of a single ball mill that will be operated in an open circuit configuration. Ore will be withdrawn from the live stockpile by belt feeders and discharged onto the mill feed conveyor. Recycled process water will be added to the mill feed chute to control the pulp density into the ball mill to 70% $^{w}/_{w}$ solids.

The ball mill product will discharge into the ball mill discharge hopper and be pumped to the trash screen to remove oversize material. Screen undersize slurry will report to the leach feed tank from where it will be pumped to the leach circuit.

Table 18.5.6.1: Design Criteria for PUG Concentrate Grinding

Description	Unit	Value	Source
Ball Mill			
- Number of mills	#	1	OMC
- Circuit configuration		Open circuit	Geovic
- Design mill feed size F_{80}	mm	4.5	OMC
- Design mill product size P_{80}	µm	106	Testwork
- Installed power	kW	1,375	OMC
- Mill diameter (inside shell)	mm	3.80	OMC
- Mill length (EGL)	mm	6.50	OMC
- Volume steel charge (operating)	%	31	OMC
- Mill discharge slurry	% solids	70	OMC
Trash Screen			
- Type		Vibrating, Horizontal	Lyco
- Size	mm	900 x 2,400	Vendor
- Aperture	mm	0.630	Lyco

18.5.7 Leaching

The objective of the leaching circuit will be to dissolve cobalt, nickel and manganese from the concentrate. Continuous atmospheric leaching of the concentrate will be accomplished at 95°C in five closed mechanically agitated tanks arranged in a series cascade overflow configuration. Concentrated sulfuric acid will be added to the leaching circuit to effect mineral dissolution, while the addition of finely milled pyrite slurry will maintain reducing conditions. The general leaching reactions are exothermic, and thus minimal heating will be anticipated. Slurry heating, if and when required, will be by direct steam injection into the leach tanks. Primary purification will follow the leaching process.

Table 18.5.7.1: Design Criteria for Leaching

Description	Unit	Value	Source
Leach Tanks			
- Number of tanks	#	5	Lyco
- Configuration		In series	Lyco
- Temperature	°C	95	Testwork
- Total residence time	h	14	Testwork
- Live volume (each)	m^3	250	Lyco
Sulfuric Acid Addition			
- Acid concentration	% H_2SO_4	98.5	Vendor
- Addition rate	kg/t concentrate	182	Mass Balance
Pyrite Addition			
- Particle size - P_{80}	µm	10	Testwork
- Addition rate	kg/t concentrate	40	Testwork

18.5.8 Primary Purification

The objective of primary purification is to remove the soluble aluminum and iron from the leach liquor. This will be achieved by increasing the pH of the leach discharge solution with soda ash to facilitate alunite and jarosite precipitation. Primary purification will be performed at atmospheric pressure and 95°C in three closed mechanically agitated tanks, arranged in a series

cascade overflow configuration. Heating will be achieved by direct steam injection. Counter current decantation will follow primary purification.

Table 18.5.8.1: Design Criteria for Primary Purification

Description	Unit	Value	Source
Primary Purification Tanks			
- Number of tanks	#	3	Lyco
- Configuration		In series	Lyco
- Temperature	°C	95	Testwork
- Total residence time	h	6	Testwork
- Target pH	pH	3.0	Testwork
- Live volume (each)	m^3	200	Lyco
Soda Ash Addition			
- Concentration	g/L	175	Lyco
- Target pH	pH	3.0	Testwork
- Addition rate	m^3/h	9.8	Mass Balance

18.5.9 Counter Current Decantation

The objective of counter current decantation (CCD) is to separate and recover the leach liquor from the leached solids residue. This will be accomplished in six CCD thickeners using partially acidified barren solution from the manganese carbonate precipitation circuit.

Table 18.5.9.1: Design Criteria for Counter Current Decantation

Description	Unit	Value	Source
Thickeners			
- Number of thickeners	#	6	Testwork
- Configuration		Counter Current	Lyco/Geovic
- Type		High density thickeners	Testwork
- Settling flux	$t/m^2.h$	6.9	Testwork
- Maximum allowable rise rate	m/h	1.8	Testwork
- Wash recovery	% Co	98.5	Geovic
- Wash ratio	t/t U/F liquid	1.70	Calculated
- Wash solution pH	pH	3.0	Testwork
- Mixing efficiency per stage	%	95	Lyco
- Thickener diameter	m	22	Calculated
- Underflow density	% solids	55	Testwork
- Overflow clarity	ppm	< 500	Testwork

18.5.10 Tailings Processing

The objective of the tailings processing section is to collect all tailings process streams and pump them to the CCD tailings storage facility.

Table 18.5.10.1: Design Criteria for Tailings Processing

Description	Unit	Value	Source
Tailings Processing Tanks			
- Number of tanks	#	2	Lyco
- Configuration		In series	Lyco
- Temperature	°C	60 - 80	Mass Balance
- Total residence time	h	2	Lyco
- Live volume (each)	m^3	100	Lyco

18.5.11 Leach Area Scrubber

The objective of the leach area scrubber is to collect and scrub all acidic aerosols in the vent gas streams from the leaching and primary purification tanks before discharging to atmosphere. Process water will be used as the scrubbing medium in a packed column, and small amounts of soda ash will be added to maintain pH >2.5.

Table 18.5.11.1: Design Criteria for the Leach Area Scrubber

Description	Value	Source
Type	Packed tower	Vendor
Scrubbing medium	Process water / Soda ash	Lyco

18.5.12 Secondary Purification

The objective of the secondary purification circuit is to remove residual aluminum and iron from the leach liquor and ensure a clean liquor stream advancing to the sulfide precipitation circuit. This will be achieved by adding soda ash to the solution to increase pH and effect the precipitation of the majority of residual aluminum and iron from solution in a series of three mechanically agitated cascade overflow tanks. Subsequent liquid-solid separation will be achieved by conventional clarification. Three quarters of the clarifier underflow stream will be recycled as seed material to the secondary purification tanks, while the remainder will be returned to the leaching circuit. The overflow stream from the clarifier will proceed to sulfide precipitation.

Table 18.5.12.1: Design Criteria for Secondary Purification

Description	Unit	Value	Source
Secondary Purification Tanks			
- Number of tanks	#	3	Lyco
- Configuration		In series	Lyco
- Temperature	°C	60 - 80	Testwork
- Total residence time	h	2	Testwork
- Target pH	pH	4.3	Testwork
- Live volume (each)	m^3	100	Lyco
Soda Ash Addition			
- Concentration	g/L	175	Lyco
- Addition rate	m^3/h	0.8	Mass Balance
Clarifier			
- Settling flux	$t/m^2.day$	0.14	Testwork
- Maximum allowable rise rate	m/h	1.05	Testwork
- Clarifier diameter	m	18	Lyco

18.5.13 Sulfide Precipitation

The objective of the sulfide precipitation circuit is to recover cobalt and nickel from the purified leach liquor by precipitating a mixed sulfide product (MSP). This will be achieved by adding sodium hydrosulfide (NaHS) to the purified leach liquor in four pressure controlled mechanically agitated tanks, arranged in a series cascade overflow configuration. The mixed sulfide will be recovered by thickening and washed in the sulfide filter to remove soluble impurities. The washed sulfide product will be packaged into sealed bulk bags and dispatched for export.

Table 18.5.13.1: Design Criteria for Sulfide Precipitation

Description	Unit	Value	Source
Oxygen Stripping Tower			
- Type		Packed tower	Vendor
- Oxygen removal efficiency	%	> 90	Lyco
- Vacuum required	kPa(abs)	65	Vendor
Sulfide Conditioning Tank			
- Number of tanks	#	1	Lyco
- Operating temperature	°C	80	Testwork
- Type		Insulated pressure vessel	Lyco
- Total residence time	h	0.25	Testwork
- Live volume	m^3	45	Lyco
Sulfide Reactors			
- Number of tanks	#	4	Lyco
- Configuration		In series	Lyco
- Temperature	°C	80	Testwork
- Type		Insulated pressure vessel	Lyco
- Total residence time	h	1	Testwork
- Live volume (each)	m^3	45	Calculated
Sulfide Thickener			
- Type		High rate thickener	Testwork
- Settling flux	$t/m^2.day$	2.45	Testwork
- Maximum allowable rise rate	m/h	1.51	Testwork
- Thickener diameter	m	13	Lyco
Sulfide Filter			
- Type		Tower filter press	Testwork
- Filtration flux	$kg/m^2.h$	179	Testwork
- Filter cake moisture content	$\%\ ^w/_w$	15	Testwork
- Filter area	m^2	12.5	Vendor

18.5.14 Tertiary Purification

The objective of the tertiary purification process is to remove trace metal impurities from the sulfide precipitation circuit barren liquor, thereby minimizing the potential for contamination of manganese carbonate product from the subsequent precipitation stage. This will be accomplished in two mechanically agitated tanks arranged in a series cascade overflow configuration. Soda ash will be added to increase pH to facilitate impurity precipitation. Liquid solid separation will be achieved by conventional clarification. Three quarters of the clarifier underflow stream will be recycled as seed material to the tertiary purification tanks, while the remainder will be returned to the leaching circuit. The overflow stream from the clarifier will report to the manganese carbonate Stage 1 precipitation circuit.

Table 18.5.14.1: Design Criteria for Tertiary Purification

Description	Unit	Value	Source
Tertiary Purification Tanks			
- Number of tanks	#	2	Lyco
- Configuration		In series	Lyco
- Temperature	°C	60 - 80	Testwork
- Total residence time	h	1.5	Testwork
- Target pH	pH	4.8	Testwork
- Live volume (each)	m^3	150	Lyco
Tertiary Purification Clarifier			
- Type		Conventional clarifier	Testwork
- Settling flux	$t/m^2.h$	0.10	Testwork
- Maximum allowable rise rate	m/h	1.27	Testwork
- Clarifier diameter	m	23	Vendor

18.5.15 Sulfide Area Scrubber

The objective of the sulfide area scrubber is to remove toxic hydrogen sulfide from the vent gases produced in equipment in the sulfide areas. A dilute solution of sodium hydroxide will be used as the scrubbing medium in a packed column to remove the majority of the hydrogen sulfide from the gas feed stream.

Table 18.5.15.1: Design Criteria for the Sulfide Area Scrubber

Description	Unit	Value	Source
Sulfide Area Scrubber			
- Type		Packed column	Vendor
- Scrubbing liquor		20% sodium hydroxide	Lyco
- Scrubbing efficiency (H_2S removal)	%	99.0	Vendor

18.5.16 Manganese Carbonate Precipitation 1

The objective of the manganese carbonate precipitation 1 circuit is to recover approximately 95% of the manganese from the cobalt and nickel depleted solution as a high quality manganese carbonate precipitate. This will be achieved by addition of soda ash in three mechanically agitated tanks in series. Liquid solid separation will be achieved by a high rate thickener followed by a vacuum belt filter. A portion of the thickener underflow will be recycled as seed material to the precipitation tanks. The overflow from the thickener will proceed to the manganese carbonate precipitation 2 circuit. The manganese carbonate product will be packaged into sealed bulk bags and dispatched for export.

Table 18.5.16.1: Design Criteria for Manganese Carbonate Precipitation 1

Description	Unit	Value	Source
Manganese Carbonate Precipitation 1 Tanks			
- Number of tanks	#	3	Lyco
- Configuration		In series	Lyco
- Temperature	°C	80	Testwork
- Total residence time	h	1.5	Testwork
- Target pH	pH	8.5	Testwork
- Live volume (each)	m^3	100	Lyco
Manganese Carbonate Precipitation 1 Thickener			
- Type		High rate thickener	Testwork
- Settling flux	$t/m^2.day$	12.0	Testwork
- Maximum allowable rise rate	m/h	4.4	Testwork
- Thickener diameter	m	8.0	Calculated
Manganese Carbonate Precipitation 1 Filter			
- Type		Vacuum belt filter	Testwork
- Filtration flux	$kg/m^2.h$	376	Testwork
- Filter cake moisture content	$\%\ ^w/_w$	14	Testwork
- Filter area	m^2	25	Calculated

18.5.17 Manganese Carbonate Precipitation 2

The objective of the manganese carbonate precipitation 2 circuit is to remove the remainder of the manganese and other impurity elements from the cobalt and nickel depleted solution. This will be achieved by addition of soda ash in three agitated tanks in series. Liquid solid separation will be achieved by conventional clarification followed by a belt filter. A portion of the clarifier underflow will be recycled as seed material to the precipitation tanks. Clarifier overflow will be partially acidified and used as wash solution in the CCD circuit, while excess barren solution will report to the sodium sulfate rich water pond for subsequent feed to the Glauber's salt crystallizer.

The manganese carbonate precipitate may either be packaged into bulk bags and exported or mixed with the PUG circuit tailings.

Table 18.5.17.1: Design Criteria for Manganese Carbonate Precipitation 2

Description	Unit	Value	Source
Manganese Carbonate Precipitation 2 Tanks			
- Number of tanks	#	3	Lyco
- Configuration		In series	Lyco
- Temperature	°C	80	Testwork
- Type		Insulated flat bottom with closed top	Lyco
- Total residence time	h	1.5	Testwork
- Target pH	pH	9.5	Testwork
- Live volume (each)	m^3	100	Lyco
Manganese Carbonate Precipitation 2 Clarifier			
- Type		Conventional clarifier	Testwork
- Settling flux	$t/m^2.day$	0.05	Testwork
- Maximum allowable rise rate	m/h	2.0	Testwork
- Clarifier diameter	m	17	Calculated
Manganese Carbonate Precipitation 2 Filter			
- Type		Vacuum belt filter	Testwork
- Filtration flux	$kg/m^2.h$	238	Testwork
- Filter cake moisture content	$\%\ ^w/_w$	19	Testwork
- Filter area	m^2	5	Calculated

18.5.18 Glauber's Salt Recovery

The objective of the Glauber's salt recovery circuit is to treat solution from the sodium sulfate rich water pond to remove sodium sulfate as Glauber's salt (sodium sulfate decahydrate), thereby producing a mother liquor that will be used for soda ash make-up, and excess solution will be pumped to a discharge buffer pond prior to a controlled discharge into the Edjé River. The controlled discharge has been modeled to meet water quality discharge standards and includes conceptual planning of a mixing zone for discharged and Edjé River waters, while meeting appropriate water discharge guidelines. The Glauber's salt will be packaged into bulk bags and transferred to a dedicated storage area.

Table 18.5.18.1: Design Criteria for Glauber's Salt Recovery

Description	Unit	Value	Source
Glauber's Salt Crystallizer			
- Type		Double effect crystallizer	Vendor
- Required discharge concentration	Na_2SO_4 g/L	45	Testwork
- Operating temperature	°C	0 - 3	Vendor
- Cooling water requirement	m^3/h	TBA	Vendor

18.6 Support Facilities and Services

18.6.1 Fresh Water Supply

The Edjé River is the only supply of fresh water makeup to the Project. The water abstraction point currently planned is from a location approximately 3 km west North West from the plant. There is an elevation rise of approximately 135 m from the river to the plant site and 2 stage pumping via an above ground pipeline into the fresh water dam located at the plant site is

planned. This water will be pumped from the dam, filtered and treated in a number of stages to produce filtered, demineralized and potable water supplies for the plant.

The water management system is based on maximizing the use of recycled water from the tailings storage facilities (TSFs) and the process plant area thereby reducing the reliance on and consumption of fresh water make-up from the Edjé River.

Stormwater run-off captured within the process areas in the plant site will be returned to the process circuit. General un-contaminated runoff from non process areas will be re-diverted into natural drainage routes surrounding the site.

Potable water for the accommodation village and the construction camp will be pumped from the main process plant water treatment facility.

18.6.2 Process Water

Two independent process water circuits will exist in the plant, the PUG circuit water and the leach and recovery plant process water circuit. The PUG circuit water will essentially be chemically uncontaminated and will be used to wash and upgrade the ROM ore to produce a PUG concentrate for subsequent chemical processing in the leach and recovery plant. The bulk of the water flow in the PUG circuit will be used as a dilution / transport media only. The leach and recovery process water will contain dissolved salts and will be isolated from the environment. This water will be recovered into a storage pond from a lined CCD tailings storage facility and subsequently distributed throughout the leach and recovery circuit.

The PUG tailings storage facility is designed to be the prime source of process water and to store process waters required during prolonged dry periods. During such times and when water cannot be taken from the Edjé River for the process, process waters for the PUG and CCD circuits, fresh water to the Plant, and cooling requirements will be provided from reclaim water from the PUG tailings storage facility. This operational provision is intended to eliminate dependency on the Edjé River flow during prolonged dry periods as currently modeled

CCD tailings decant water will be returned from the CCD tailings storage facility via an above ground polyethylene pipeline into a lined process water pond. This pond will also receive fresh make-up water.

18.6.3 Water Treatment Plant

The water treatment plant will receive water from the fresh water pond which will be used to store approximately 36 hours of unfiltered river water. Debris from the river will, however, be prevented from entering the pumps, pipeline and storage pond using coarse screens at the river harvesting point. Suspended solids will be removed by the filtration plant.

The water treatment plant will consist of multimedia sand filters and carbon filters and will generate filtered water that is the feed to the potable water and demineralized water treatment processes.

Filtered water will be used for product filter washing cycles, and for gland seal water.

Potable water will be pumped to various areas of the process plant, site buildings, mine services area and the accommodation village. Potable water will also be distributed to various safety shower and eyewash stations located throughout the process plant and mine services areas.

Demineralized water will be distributed to the sulfuric acid plant and steam generating equipment.

18.6.4 Cooling Water

Cooling water circuits will be required in the sulfuric acid and the Glauber's Salt crystallizer plants.

The Glauber's Salt plant, which has relatively low temperature cooling requirements, will require the use of refrigerative cooling systems, while the sulfuric acid plant will generally deploy evaporative cooling methods.

18.6.5 Sodium Sulfate Rich Water

Excess barren solution from the second stage manganese carbonate precipitation circuit will report to the sodium sulfate rich water pond. The sodium sulfate rich water will be pumped to the Glauber's Salt crystallizer. The objective is to control the sodium sulfate concentrations in the process plant solutions by crystallizing Glauber's Salt and removing it from the process.

18.6.6 Fire Water

The Project will require the construction of three independent fire water systems located at the main plant, the accommodation village and the construction camp.

There will be a dedicated fire water system for the village comprising a main hydrant and hose reel system. The fire water reserve will comprise the lower volume within the potable water storage tank, i.e. the fire system will have a lower suction nozzle than potable water takeoff.

The fire hydrants will be served by a dedicated pumping facility comprising an electric pump, a back-up diesel pump, and a jockey pump (to maintain system pressure). The system will service the accommodation village and will also be suitable for defense against bush or scrub fires which may threaten the village.

The fire water systems will be supported by fire extinguishers located strategically around and within buildings in the village including high risk areas such as the kitchen and main electrical systems.

The main process plant system will be similar to that used in the village but will include a dedicated fire water tank. Provision will also be made to enable the pumps to draw directly from the fresh water dam or other water sources.

18.6.7 Power Supply and Electrical

Maximum Demand and Steady State Load

The maximum demand for the site has been calculated to be in the order of 18.5 MVA (16 MW), including processing plant, village, construction camp and other facilities.

The electrical design has been based on the following inputs:

- Mechanical equipment list (process plant).
- Projected infrastructure loads.

In addition to these loads, the design capacity includes building services and area lighting, including perimeter lighting at the process plant and at the accommodation village.

Power Generation

Given the remote site, a connection to the utility has not been considered. Electricity will be generated using high speed diesel generator sets. The configuration will be 10 x 2 MVA diesel units.

A back-up generator at the accommodation village and construction camp will mitigate the threat of power outage due to a fault on the 11 kV distribution networks.

The 11 kV supply is distributed in two networks, each of which includes approximately 12 km of overhead powerline.

All cabling within the process plant will be run above ground in cable trays except road crossings, which will be run underground. There will be no overhead powerlines within the process plant boundary.

High pressure steam produced in the sulfuric acid plant will be used to drive a steam turbine / alternator to generate electrical power, thereby augmenting diesel power generation.

18.6.8 Communications

Communications Infrastructure

Arterial communications will use Optical Pilot Ground Wire (OPGW) - a fiber optic cable embedded within the overhead ground conductor on the overhead powerline. Buried fiber optic cables will also be used either at lead-ins or at the process plant in order to create a ring.

The communications head-end will be located at the existing compound which, given the OPGW approach, will require the powerline to service this location.

Network Cabling Infrastructure

Within the process plant, a fiber optic network will link several network switches. This will provide the backbone for VoIP, data, security and Supervisory Control and Data Acquisition (SCADA) networks.

At the village, data and satellite television will be serviced by a combination of dedicated satellite and OPGW channels. It is envisaged that, given an expanded functional GSM mobile telephone network, voice communication will be limited to a small number of VoIP lines for use by facility management.

Radio System

The existing 70 m communications tower will be used to support an expanded Radio Base Station. This will provide for mine operational fleet and process control operators' radio, emergency channel and provide repeater functionality to extend the area of radio coverage.

Security CCTV

Security CCTV will monitor fence lines and access points at the process plant and accommodation facilities. These will be monitored at the process plant gatehouse and accommodation village gatehouse. CCTV data will be sent over the fiber optic network to monitoring stations.

CATV Network

Cable Television (CATV) will be provided by a separate subscription, and will use a dedicated receiver.

18.6.9 Fuel and Oil

Diesel fuel for the power station and mobile equipment will be delivered via road tankers, filtered and then pumped into four 1,000 m^3 storage tanks located within a concrete bund in the processing plant area. Fuel bowsers will be provided for heavy mine equipment and light vehicles, and day tanks will be provided for the acid plant and fire water pump.

Lubricant oil products will be delivered in drums and stored at the mine workshop area. Waste oil and coolant will be collected in a waste oil tank and waste coolant tank and periodically removed from site.

18.6.10 Sewage

Sewage and grey water from the mine services area, control rooms and plant site buildings will be collected in various sumps and forwarded to the site sewage treatment plant.

Sewage from the accommodation village and the construction camp will be collected in various sumps and forwarded to the sewage treatment plant located a distance from the senior staff village.

18.6.11 Refuse Handling

A waste incinerator and compactor will be provided within the process plant secure area. A waste storage dump for site waste will also be constructed.

18.6.12 Security

The process plant will be secured by a 2,400 mm high galvanized chain link security fence including loops of razor wire at the base and top. Other features of process plant security are CCTV surveillance of the plant perimeter and areas within the plant, single pedestrian access for authorized persons using a proximity card, and a single vehicle access system.

The workshops, warehouses and main administration facilities will be secured within a 1,800 mm high single fence. The warehouse will be located within the process plant secure area.

The entire accommodation village site will be secured by a 2,400 mm high galvanized chain link security fence including loops of razor wire at the base and top. There will be a single point of entry to the village. This will be controlled and accessible by authorized personnel only.

18.7 Infrastructure

18.7.1 Site Development

The Project will require development at the following major centers:

- Main access road upgrade.
- Mine site roads and tracks.
- Construction camp located near the accommodation village.
- Accommodation village.

- Process plant including associated workshops, warehouses, various site buildings and fuel storage.
- Mine service area including heavy vehicle workshop, mine change house, core shed and emergency response center.
- Edjé River water intake system and pipeline to plant site.
- PUG tailings storage facility.
- CCD tailings storage facility.
- Glauber's salt pads (GS Pads).
- 11 kV powerlines.
- Communications system upgrade.

Figure 18-16 shows the process plant in perspective.

18.8 Reagent and Product Transportation

The importation of reagents into the Port of Doula and road transport to site and the export of products to overseas markets from the Port of Doula represent a significant portion of the operating costs of the Project.

18.8.1 Reagent and Product Quantities

Table 18.8.1.1 provides the quantity of bulk reagents, containerized reagents and products to be transported.

Table 18.8.1.1: Transportation Quantity of Reagents and Products

Imports	**Bulk Reagents**	**Tonnes/Year**
	Sulfur	39,080
	Pyrite	27,621
	Dense Soda Ash	86,830
	Subtotal	**153,531**
	Containerized Reagents	**Tonnes/Year**
	Flocculant	435
	Sodium Hydrosulfide	15,864
	Sodium Hydroxide	814
	Corrosion inhibitor	3
	Antiscalent	3
	Biocide	31
	Subtotal	**17,150**
Exports	**Products**	**Tonnes/Year**
	Mixed Sulfide	18,042
	Manganese Carbonate	72,373
	Subtotal	**90,415**
Total		**261,096**

18.8.2 Port Warehousing Facilities

There are currently no warehousing facilities in and around the Douala Port Precinct suitable for storage of bulk reagents or products for the Project. However, there is sufficient land close to the bulk materials section of the port that is available to construct purpose built warehouses.

It is estimated that 25,000 m2 of yard will be required and that a purpose built facility will be constructed to store and handle the reagents and products. The facilities will entail a 15,000 m2 warehouse for the storage of soda ash, sulfur, pyrite and other reagents when needed, a 5,000 m2 hardstand of which 2,500 m2 will be covered for the storage of manganese carbonate.

18.8.3 Route Survey

All supplies and exported products are required to be transported via overland trucks from the port facilities to the mine site. Given the length and variation in road quality, a road survey was undertaken to determine the expense of upgrading the transportation route before commercial production.

Douala Port to Yaounde (235 km)

Once exiting the Douala Port precinct, the route follows the N3 towards Edea. Along this part of the route there are a number of overpasses which have no impediments for the transportation of normal trucks.

On route to Yaoundé there are two weighbridges at which it is compulsory for all trucks to be weighed and three pay tolls. Both the pay tolls and weighbridges are limited with access for out of gauge cargo although they have no limitations.

The road between Douala and Yaounde is in excellent. There are a number of overtaking lanes and continued work on road widening.

Yaounde to Abong Mbang (225 km)

From Yaounde there is a by-pass of the city and the route continues along the N10 from Yaoundé to Abong Mbang. The city limits are congested with low power lines in areas although once the city limits were cleared the congestion then subsides

Once the congestion of the city limits has been passed, the road becomes clear with little traffic. The road between Yaoundé and Abong Mbang is newly laid. Currently there are restrictive truck movements on this road due to government regulation. It is envisaged that this regulation will change in the coming months and that pay tolls will be placed on the road, allowing trucks to use the road.

There are no impediments for regular transport to travel between Yaoundé and Abong Mbang.

Abong Mbang to Nkamouna Site (134 km)

This part of the route is by far the most difficult to traverse. The road is unsealed and in most parts a single lane road with few passing points for vehicles. Once the village of Abong Mbang is cleared the first part of the route is low lying and with numerous culverts to disburse water. This area is subject to flooding during the wet season.

There are numerous bridges on this route with uncertain weight capacity. In their current condition, it is a requirement that all bridges will be replaced in accordance to the projects.

18.8.4 Product Storage - Site and Port

All Mixed Sulfide Product (MSP) will be loaded and transported in containers due to its higher value and more stringent handling requirements. Where feasible, double handling will be avoided. Approximately 19% of the manganese carbonate product will be backloaded to Douala in containerized form with the balance being loaded into bulk transport trailers for unloading and containerization at the port. A dedicated product storage shed will be constructed to store 21 days of product under cover, should extended road closures occur then product storage under tarpaulins or in other storage areas would be required.

At the port, product transported in bulk bags in bulk product trucks will be stored in rented warehouses and transferred into 6 m sea containers for final dispatch to the customer.

18.8.5 Site Storage and Handling Equipment

Storage sheds have been allowed at the plant site for reagents based on 42 days consumption. The shed will be steel framed and metal clad and have appropriate ventilation depending on the type of reagent. All storage sheds will have reinforced concrete slab floors capable of handling front end loader and truck wheel loads. The storage sheds for sulfur, pyrite and soda ash will have 4 m high side walls to enable material to be stacked to a greater height while reducing the floor area.

Table 18.8.5.1 provides the area of buildings allowed for each reagent and product.

Table 18.8.5.1: Reagent and Product Storage Sheds

Buildings	Area (m^2)
NaSH Storage Shed	1,800
Pyrite Storage Shed	1,200
Sulfur Storage Cover	2,200
Soda Ash Shed	4,800
General Reagents Shed	1,200
Product Warehouse	4,140

18.9 Project Implementation

18.9.1 Project Implementation Strategy

The Project implementation strategy is based on an EPCM delivery approach with the Owner executing certain works.

GeoCam will engage an EPCM Engineer to provide EPCM services for the plant and infrastructure. The Engineer will undertake design and procurement from its home office and the Owner will have representatives in the Engineer's office to approve the design and procurement.

The Owner will manage the main road upgrade via a contractor with the main earthworks for the plant site bulk earthworks, site roads and tailings storage facilities being Owner performed. Initially contractor's equipment will be used until the mining fleet arrives and the contractor equipment will be progressively demobilized.

It is proposed to award work construction contracts on a horizontal basis by discipline of work i.e. earthworks, concrete, building works, tankage, etc.

Fabrication contracts will be awarded separately from the main construction contracts for the supply of structural steel and platework items.

All purchase orders and contracts will be directly with the Owner and managed by the EPCM Engineer.

A specialist transport and logistics group will be engaged by the EPCM Engineer and report to the Engineer's Materials Manager.

18.10 Tailings Storage

18.10.1 PUG and CCD TSF Structures

The PUG and CCD tailing facilities will be built within the Napene Creek drainage basin and immediately to its north. Stored tailing will essentially fill the upper end of the Napene Creek basin. The location reduces offsite run-on to the facilities thus limiting the need to handle excess waters over the Project life. The tailing facilities will be formed by the construction of embankment dams built in four stages (Stages 1 – 4) over the life of the Project to spread capital costs over the mine life while meeting project needs. At build-out in Stage 4, the tailing facilities will extend about 2,500 m in the north-south direction and 2,200 m in the east-west direction, and will cover about 4 km^2, about the entire Napene Creek drainage area.

The PUG TSF will comprise four embankment sections built during Stages 1 – 4. The West Dam, built to a height of about 85 m in Stage 4, will be built across Napene Creek where the creek currently exits the site flowing westerly to the Edjé River. As the facility is raised, embankments between 85 and 10 m high will bound the PUG TSF on the north, east, and south sides. At the end of Stage 4, the embankments will form one embankment enclosing the upper Napene Creek basin. The PUG tailing dams are designed as water retention structures considering (1) the very soft and wet tailing to be stored, (2) the size of the operational water pond and impoundment of water against the dams, particularly against West and South Dams at least during a portion of the mine life, and (3) because provisions are included in the water balance to store additional water in the PUG TSF to supply water to the Plant during dry periods. Dam crests are based on the tailing and water storage requirement in the facility, freeboard requirements, and accounting for a sloped tailing beach.

The CCD TSF will be formed by embankments built during Stages 1 – 4 to form two, four-sided cells to the north of the PUG TSF. These two cells and the PUG TSF share embankments between them. Cell 1 and 2 embankments range from about 15 to 36 m high, with final crest elevations of 735 and 748 m, respectively. The CCD cells will have geocomposite liners along the inside embankment slopes and basin bottoms. The basin bottoms will also be covered with an underdrainage collection system and protective layer (retardant layer).

The design of the PUG TSF embankments provides for zoned earthfill structures, and the design of the CCD embankments provides for homogeneous earthfill structures. Dam raises will be accomplished using the downstream raise method. Overburden materials will be removed from the dam “footprints” and replaced with the appropriate embankment fill zones. Underlying materials appear adequate for founding the tailing dams. Dam foundation excavation work is estimated in this report but investigations for further design phases will add to knowledge of the site and work requirements for dam construction.

Key zones of the PUG TSF dams include a relatively impermeable sloping core zone with a cut-off bottoming in the relatively impervious dam base materials. They include a filter zone that serves as both a filter and drain providing for the removal of water seeping through the core zone. At the base of the West Dam, the filter zone will extend partially below the downstream shell forming a filter blanket adjacent to the core zone. This filter zone will be extended to the downstream toe in the valley bottom. Internal drainage will be assisted with the installation of perforated seepage collection pipes in the filter blanket, connecting to a seepage collection sump at the downstream toe of the Stage 4 West Dam. The filter zone in the remaining PUG TSF embankments will be extended to the downstream toes at select locations via finger drains. Perforated pipe in the finger drains will connect to seepage collection sumps.

Construction of the embankments is based on the use of on-site materials. Materials for the core zone and structural fill will be soils from the laterite profile at the site that can be sourced from the tailing basin or mine pits. Some stockpiling, re-handling, and moisture conditioning should be expected. Filter zone and pipe surround materials for internal drainage will be processed from competent bedrock on-site. This report assumes that the proposed tailing facility embankments are built by GeoCam, who will also develop embankment fill materials.

18.10.2 PUG and CCD Tailing Management, Water Management, and Water Balance

The tailing storage facilities have been designed to contain the PUG and CCD tailing, and associated waters, produced over the approximate 23-yr mine life. This corresponds to about 53 million dry metric tonnes of PUG tailing and 15.4 million dry tonnes of CCD tailing.

Results of the operational water balance have also been used to size the PUG and CCD TSFs and provide parameters for operating the tailing facilities. This includes parameters related to providing fresh water and return waters to the process facilities, and discharging waters from the facilities to the Edjé River. Discharges to the Edjé from the PUG TSF are modeled; controls on the amount of total suspended solids in the discharges to meet water quality guidelines will be implemented. Water will be discharged from the CCD TSF via the Glauber's Salt Plant and buffer pond. This has been modeled to meet water quality discharge standards and includes conceptual planning of a mixing zone for discharged and Edjé River waters, while meeting appropriate water discharge guidelines.

The current PUG TSF tailing deposition concept provides for deposition of the tailing from a few individual drop bars (open pipe discharges), strategically located around the PUG TSF perimeter. The intent is to produce a systematic filling, forming a tailing slope of about 0.25 percent from north to south in the facility, and to control the size of the decant pond, maintaining it in the southern portion of the facility. The tailing will be flocculated before deposition to improve sedimentation and decant development rates of this clayey tailing. Design of the tailing delivery and deposition system is by others.

If experience during operations using the individual drop bars indicates that a significant tailing beach could be established using groups of drop bars on a rotational basis around the facility, then GeoCam can consider the option of converting to this thin-layer rotational deposition system. This would entail operating a single group of drop bars at any time. Placement of tailing in thin layers around the PUG TSF would potentially allow for some air drying of fresh-placed tailing leading to increased tailing density in the beached materials, and some increase in storage capacity. As an added potential benefit, improved tailing densities may accelerate

closure and reclamation activities. The amount of increased tailing densities that could be achieved is dependent on site conditions and operational controls, but provides the impetus to control tailing deposition and the decant water pond and to work toward developing a tailing beach.

The current CCD TSF tailing deposition concept provides for deposition from individual drop bars, strategically located around the CCD TSF cell perimeters. With progressive tailing deposition from the northern and eastern limits of the two cells, the goal is to form tailing slope/beach slopes of about 0.5 percent to the southwest corners of the cells where the decant structures and underdrain collection sumps are located. Conversion to a thin layer, rotational deposition system, as in the PUG TSF, is not anticipated, because the decant ponds are expected to cover significant portions of each CCD cell. Design of the tailing delivery and deposition system is by others.

18.10.3 Glauber's Salt Pad and Alternative GSF

The concept-level Glauber's Salt (GS) Pad will temporarily store up to four years of Glauber's Salt production (0.83 million metric dry tones), until the point that GeoCam begins its removal from the facility and project site.

The GS Pad will be built to the north of the CCD TSF on relatively flat ground. It will be stage-built in four separate cells, each with the ability to store one year of Glauber's Salt production. At build-out, it will cover an area of about 215,000 m^2 (about 550 by 400 m). Each yearly cell will have a geocomposite liner along its bottom that will be covered with a secondary protective fill layer containing an underdrain system. The liner, protective layer, and underdrains will extend under the entire GS Pad. Low perimeter berms will preclude run-on onto the stage-built cells or the built-out GS Pad. Perimeter channels within each cell will collect precipitation and convey it to a stormwater collection pond outside of the GS Pad.

The Glauber's Salt will be placed into lined bulk bags at the process plant. The bags will be transported to the GS Pad, where they will be placed, stacked, and periodically covered with plastic sheeting. Design of the bags and design of the bagging, stacking, and cover procedures is by Lycopodium. Geovic and Lycopodium currently anticipate that damage to bags in the facility will be sufficiently limited such that precipitation onto the GS Pad that contacts spilled Glauber's Salt will not require water treatment before release to the environment. Thus these waters are not included in the current project water balance. Lycopodium is designing the facilities to release these waters to the environment.

If modifications to the current Project prevent removal of the Glauber's Salt from the Project site, the GS Pad will be converted into the alternative, permanent GSF, storing the total anticipated Glauber's Salt production over the mine life of about 4.8 million metric dry tonnes. The alternative GSF would consist of stage-built earth embankments built around the GS Pad limits and the GS Pad liner system would be extended to cover the GSF basin and interior embankment side slopes. GSF embankments would range in height between 37 and 44 m, with a final, sloped, crest elevation ranging from 760 to 767 m.

Under the current alternative GSF concept, Glauber's Salt would be blended with clayey breccia overburden at a 2:1 ratio, by weight, that would be placed and compacted into the GSF. The overburden soils would likely require some moisture conditioning prior to blending to reduce the

dissolution potential of the salts once mixed into the soil. Blending is conceptually considered to be conducted in a small PUG mill or other similar process facility.

The soil-salt mixture lifts would need to be planned to promote encapsulation of prior lifts and to promote precipitation runoff toward a sump in the GSF and to control against erosion. Precipitation and runoff would be removed from the sump such that no pond develops against the placed salt-soil mixture. This operational procedure would be intended to reduce the dissolution potential of salts from the mixture, and to reduce development of hydraulic head on the liner, reducing the potential for seepage through the liner system and release of saturated sodium sulfate solution to the environment.

If further work on the project identifies that the alternative GSF is required, water removal, treatment, and controlled discharge to the Edjé River meeting water quality discharge standards would be implemented.

18.11 Environmental Mitigation and Management

Based on the ESIA findings, the ESAP focuses on what GeoCam believes are the mitigation measures requiring specific action items for implementation, as summarized below.

18.11.1 Land Use Restoration

GeoCam will implement effective reclamation strategies aimed at maintaining forest productivity on reclaimed land disturbances. The Mine Reclamation and Closure Plan (MRCP) reflects the Project layout and mine plan presented in this study. The Plan outlines the general program for stabilizing and revegetating the Nkamouna Mine. GeoCam plans to concurrently reclaim disturbed areas to the maximum practical extent and will reclaim Project disturbances to a forest / wildlife habitat post-mining land use. Specific procedures have been established in the MRCP including surface preparation, soil placement, seeding and planting, soil amendments and monitoring of revegetation progress. Reclamation strategies for specific types of disturbances have also been included, as well as an estimate of the overall cost of reclamation activities.

18.11.2 Ecosystem Conservation and Biodiversity Improvement Areas

GeoCam has a strong commitment to support the conservation and sustainable use of the biodiversity resources in and around the Project area. As part of this commitment, GeoCam has developed a Biodiversity Management Plan. In support of this plan a wildlife survey was conducted on the Nkamouna site in April 2008 (Ngandjui and Linus, 2008). The process for development of the Plan has been coordinated with the Ministry of Forests and Fauna (MINFOF), a team of national consultants and several conservation and non-governmental organizations.

The main objectives of the Biodiversity Management Plan are to:

- Contribute to the preservation of riparian protected areas in the Project area;
- Contribute to the sustainable management of natural resources in the Project area;
- Rehabilitate land degraded by mining and contribute to carbon sequestration; and
- Develop environmental education programs.

Several programs / projects / activities have been developed with estimated annual budgets to carry out and verify the outcomes of the Plan objectives, including the following:

- Monitoring program for fishing biodiversity and water quality in the vicinity of the mine site;
- Poaching and bush meat alternatives development program;
- Mining affected parcel reforestation program;
- Support to community forestry in the area of the Project; and
- Local communities' environmental education programs.

The Biodiversity Management Plan emphasizes progressive rehabilitation of Project impacts and the involvement of local communities and NGOs, as these groups are interested stakeholders for the environmental issues associated with the Project. Another aspect of the Plan is the creation of a monitoring program to assess the potential impacts of the Project. GeoCam may also support actions initiated by governmental and non-governmental parties in the planning and implementation of programs related to protecting species and habitats.

18.11.3 Sodium Sulfate and Glauber's Salt Management

The beneficiation process will create elevated sodium sulfate concentrations in process waters exceeding 100,000 mg/L. GeoCam plans to install a Glauber's Salt water treatment plant to reduce sodium sulfate concentrations in process waters prior to recycle and/or controlled release to the environment. Process streams with elevated sodium sulfate concentrations and Glauber's Salt produced from treating process water will require mitigation and management to reduce impacts to the environment.

Sodium sulfate will report to the CCD TSF in the CCD tailings. Mitigation measures include lining the facility with an engineered, geomembrane liner system. The facility will contain a seepage collection system and sump to recycle water with elevated sodium sulfate concentrations to the process. Glauber's Salt will be placed in bags that are temporarily stored on the GS Pad (or will report to the alternative GSF if the Glauber's Salt is not sold and removed from the project) The GS Pad (and the GSF, if constructed) will also be lined with an engineered, geomembrane liner system and stormwater collection and recycle system. The Glauber's Salt plant will reduce sodium sulfate concentrations to approximately 40,000 mg/L in water to be discharged. This concentration exceeds applicable guidelines for water quality and therefore a mixing zone is proposed within which in-stream sulfate concentrations will reduce to meet water quality guideline concentrations. Treated water from the Glauber's Salt plant will be stored in a lined buffer pond and discharged to the Edjé River based on river flow rate.

In addition to the mitigation and management measures described above, a surface and groundwater monitoring program will be implemented during operations and post-closure to evaluate the effectiveness of such measures. If necessary, additional mitigation measures will be designed and implemented.

18.11.4 Sediment Control and Water Discharges

An Erosion and Sediment Control Plan (ESCP) has been developed setting forth the Company's plans for controlling erosion and sedimentation associated with mining operations. The ESCP details the structures and procedures that will be implemented to reduce the production and transport of sediment from land disturbance related to mining activities. The ESCP provides

specific controls and design for the plant site, ore stockpiles, haul and access roads, mine box cut panels, and waste fill stockpiles. Diversion and collection channels, rock check dam sediment traps, culverts, and sediment pond structure designs are presented in the ESCP. Scheduling and sequencing of construction activities, concurrent reclamation, revegetation, and site grading to control runoff are important elements of the overall ESCP.

18.11.5 Water Supplies and Public Drinking Water Systems

Process water and CCD tailing decant water is to be recycled or treated in the Glauber's salt plant and the leach tailing is contained in a geosynthetically lined containment facility. In addition, monitoring wells will be installed to monitor the quality of groundwater proximate to the tailings facility. In addition the Emergency Response and Contingency Plan will address procedures for mitigating the unanticipated release of adverse-quality water. It is noted, however, that no public drinking water supplies are located near the tailings facility or within the Project area boundary. Consequently, it is not anticipated that there will be adverse impacts to groundwater quality.

Nevertheless, GeoCam will monitor local drinking water supplies according to the commitments presented in its Water Monitoring Plan.

During the dry season, the average monthly low flow in the Edjé River is approximately 45 times the water requirements of the Project. However, in order to ensure that the Project does not adversely affect water supplies, the Project has been designed to minimize water use in the dry season. In addition, excess capacity for water storage has been incorporated into the design of the PUG TSF, reducing the potential need for extraction of water from the Edjé River during times of low flow. Therefore, impacts to surface water flows will be insignificant.

Regarding surface water, during the dry season, base flow will diminish and may cease in some smaller drainages. However, this will be offset by the fact that use of water from the Napene Creek tailings facility will obviate the need for pumping of water from the Edjé River floodplain. A surface water monitoring program will also be implemented.

18.11.6 Waste Management

The Waste Management Plan details the handling, storage and disposal of wastes generated by Project activities, including mining and processing wastes, as well as the management of wastes requiring special handling including hazardous, medical and sanitary waste. The Company has committed to updating the Plan periodically to ensure that the Plan content reflects actual site conditions and waste management procedures.

18.11.7 Fuel and Chemical Transport, Storage, and Containment

Process reagents, laboratory chemicals, fuels, flammable materials, bottled gases, oils, lubricants, solvents, degreasing agents, and waste oils and solvents will be used onsite. In addition to an Emergency Response and Contingency Plan, the ESAP presents appropriate approaches for transporting, storing and handling the various chemicals, oils, and fuels.

18.11.8 Emergency Response

The Emergency Response and Contingency Plan outlines the process for responding to on-site emergencies, including accidental spills and releases, fires, explosions and medical emergencies. It is designed to minimize employee exposure to risk and injury and limit potential impacts to the environment in case of emergency. It also addresses prevention, containment and cleanup

methodologies. Adequate training and equipment for responding to emergencies will be provided and maintained. The Company has committed to updating the Plan to keep it current with site conditions and activities.

18.11.9 Weeds and Disease Vectors

GeoCam will take precautions to limit the establishment and/or spread of invasive species. Revegetation of disturbed lands will take place as part of concurrent reclamation efforts and will include measures to promote native species. Employee training programs on invasive weed species and mosquito / malaria habitat will also be conducted.

18.11.10 Air Quality and Noise Control

Project-associated air quality and noise issues are not anticipated to be a concern for public health given the distance to the nearest building. However, mitigation measures such as road watering, speed and off road restrictions, equipment maintenance and hearing protection devices will be employed to prevent impacts to workers and the public.

18.11.11 Embankment Safety

GeoCam has retained qualified engineers to design the Napene Creek tailings facility embankments according to current international safety guidelines and standards. Rigorous quality control / assurance procedures will be implemented during construction, thereby reducing the potential for embankment failure and consequent downstream impacts. The Company will provide care and maintenance on all of the embankments during the operating phase to mitigate against structural deterioration and unstable conditions.

18.11.12 Management and Corporate Commitment

Effective implementation of the environmental management system framework established in the ESAP begins with sound Company environmental policies. It also requires commitment of the senior and executive management of the Company to ensure that the programs are adequately implemented and financed.

GeoCam is committed to long-term involvement in protecting the environmental quality and human interests in Cameroon. The environmental and social programs aim to improve environmental and safety performance continually in the workplace, maintain multi-directional communication among the Company, local communities, and interested stakeholders and limit local community dependence on the Project. The Company has adopted two policies that define its corporate commitment, policy and specific goals:

- Environmental, Social, Health and Safety Policy.
- Community Development Policy.

The Environmental, Social, Health and Safety Policy commits to long-term protection of environmental quality, human health and safety by providing sufficient financial support for environmental and social programs, educating and training employees in these disciplines and multi-directional communication with the surrounding communities and other interested stakeholders.

The Community Development Policy is designed to improve local community conditions and infrastructure through financial support, soliciting input from local communities and other stakeholders and the establishment of partnerships with government and non-government

agencies. GeoCam will continue to support socio-humanitarian programs in its effort to promote sustainable local community development initiatives.

Environmental and social management will be administered through an organization that includes corporate oversight, site management, local employees, contractors and subcontractors. GeoCam will appoint experienced managers for key positions who will be responsible for:

- Complying with local, national, international, and financial lending institution laws, regulations, policies and guidelines.
- Ensuring that the required environmental and social management activities are implemented and maintained.
- Reporting on the effectiveness of such activities to executive management and the Board of Directors for review and corrective action, if necessary.

The Health, Safety, Social and Environmental Manager will maintain records and report on any significant environmental matters, including monitoring data, accidents and occupational illnesses related to waste management. Training programs for workers will reflect the level and type of expertise necessary for a given position. Safety precautions will include protective clothing pertinent to the work activity, area and schedule. Security of all facilities will be further evaluated for adequacy on an ongoing basis.

18.11.13 Contractors

Site contractors will be held responsible for ensuring that their employees have an appropriate level of health, safety, environmental, community relations and emergency training. All contractors will be required to adhere to GeoCam's commitments to environment, health, safety and community issues.

18.12 Mine Reclamation and Closure Plan

The objective of the mine reclamation and closure plan is to return land disturbed by the mine to forest/wildlife habitat following the cessation of mining. Facilities that are not needed to support post-closure will be reclaimed. Mine reclamation and closure details are provided in the MRCP (Volume 2). The mine panels, roads, waste facilities, stockpiles quarries, tailings facilities (PUG TSF and CCD TSF), Glauber’s Salt storage pad, process, and storm water retention ponds, and structures are all considered in the Project closure plan.

GeoCam plans to reclaim disturbed areas concurrently with Project development to the maximum practical extent and will reclaim Project disturbances to a forest / wildlife habitat post-mining land use. Guidance and specific procedures have been established in the MRCP. Reclamation strategies for specific types of disturbances have been included and a reclamation budget estimated based on these procedures for closure and reclamation.

The MRCP is a discipline-specific plan of the Project environmental management system and should be viewed as a 'living document' that will be updated and continuously improved upon so that it is current with the actual site conditions and reclamation performance throughout the life of the Project.

18.12.1 Tailings Reclamation

The design and operating criteria for the tailing impoundments considers closure and reclamation of the facilities. Physical stabilization of the PUG and CCD tailing surfaces will be required for revegetation of the PUG TSF, and to allow cover placement over the CCD TSF followed by revegetation.

Once the Glauber's Salt bags are removed from the GS Pad, this facility and the associated stormwater pond can be closed, and the area reclaimed. This is expected to entail removal of associated berms and channels, scarifying the ground, removal of underdrainage collection pipes in the GS Pads, removal of the stormwater pond geomembrane liner, backfilling the pond and regrading the area as needed for drainage, and revegetation. If the GS Pad is converted to the alternative GSF, the 'dry' placement of the soil-salt mixture into the GSF would provide for ease of cover placement over the GSSF and subsequent revegetation

18.13 Markets

GeoCam anticipates producing two saleable products: a high quality mixed cobalt-nickel sulfide (MSP) and manganese carbonate. The metallurgical processes required to generate these products have been fully tested and demonstrated by pilot testing in the laboratory. Over the life of the project, some 105,000t of cobalt, 75,000t of nickel, and 535,000t of manganese will be produced. Based on the mining, processing and related Project requirements described, GeoCam envisages production rates during the mining years (years 1 to 11) ramping up to about 15,000 t/y of mixed cobalt-nickel sulfide product (MSP) and 30,000 t/y of manganese carbonate. Cobalt and nickel derived from the MSP will be about 6,100 t/y and 3,200 t/y, respectively. These production rates will gradually drop off during the processing-only phase from years 12 to 24 after the mining phase is completed if additional resources are not identified during that time to supplement the existing resource base.

18.13.1 Potential Consumers

Potential consumers for the MSP have been identified as parties either operating existing processing facilities or as those considering developing processing facilities, and having the technical and commercial competence to recover, account, and commercially pay for the cobalt and nickel contents at conditions acceptable to the Project. While all sectors of the cobalt market are considered potential target consumers, the projected growth of battery demand for cobalt places greater emphasis toward consumers supplying or anticipating their participation in the battery sector. All of the products - cobalt, nickel, and manganese - are inherent elements to the manufacture of many types of batteries.

The cobalt value within the MSP is the major economic influence for the Project. Most of the interest in the Project to date has been expressed by Asian consumers. These potential consumers are typically looking for a reliable long-term supply of cobalt to support their manufacturing operations.

There are three general types of cobalt consumers:

- Existing Facilities: including consumers of direct shipping ores, primarily from the Democratic Republic of Congo (DRC), refiners of cobalt and nickel matte, scrap, and precipitate, and processors of MSP having excess capacity within their circuits.

- New Facilities: including those parties who are planning to participate in the battery sector and are considering development of new intermediate product refineries as a means of securing long-term supplies.
- Traders: including merchants who have commercial and financial relationships with the two groups identified above and wish to secure long-term sources of supply that enhances their market presence. This group has been evaluated with reference to their ability to organize acceptable financial considerations for Project development.

18.13.2 Nickel

The Nkamouna Project will produce nickel as an important by-product of its primary cash commodity cobalt. The financial model indicates that nickel sales will comprise about 15.30% of the total life of mine project revenue, compared to 67.03% cobalt revenue; and the remaining 17.67% being derived from the manganese carbonate product.

18.13.3 Value Determination of Mixed Sulfide Product

The value of MSP is established by the actual recovery during processing of the cobalt and nickel contents, adjusted for the negotiated accountability for each, and factored by the established and agreed market quotations for finished products. Many factors feed into these valuations, including: profit margin, processing charges, delivery conditions, quality discounts and premiums, penalties, payment conditions, and determined out-turned qualities and quantities to be delivered to consumers. Further consideration includes the MSP consumers' cost relative to the liquidation of their final products. These include delivery, payment conditions, pricing basis, quotation periods, premiums and discounts, and their ultimate consumer risks.

Methods of value determination vary by region and by factors that influence the aggressiveness of the consumers. Recovery of the cobalt and nickel contents has been determined to be feasible (in the high ninety percentages) while the operating costs and desired profit margins of the consumers are variable. Assuming that all variables are static and combined within an accountability function, discussions with serious consumers have been established at or greater than the mid-eighty percent accountability level. In fact, all current discussions with major potential consumers are based upon their ability to account for 85% minimum of both nickel and cobalt. The accountability assumptions used in the financial model are believed to be conservative and set at or below the minimum considered in the current discussions, 85% for cobalt and 82% for nickel. GeoCam has consumer correspondence validating these assumptions.

18.13.4 Manganese Carbonate Product

The manganese carbonate product was produced as an output of the same continuous pilot testing of representative Nkamouna ore samples as was accomplished for the MSP. The mine production schedule, ore grade, physical upgrading and leach and recovery plant throughput combine to determine the volume and quality of cobalt, nickel and manganese that will be produced.

The annual quantity of manganese carbonate is expected to be approximately 65,000 t/y, subject to variations in the mine production, quality of ore processed, and the continuity of the anticipated quality of the manganese carbonate.

Table 18.13.4.1: Manganese Carbonate Analysis

Element	Percent

Manganese (Mn)	44.4
Carbonate	47.2
Sodium (Na)	0.62
Iron (Fe)	0.016
Aluminum (Al)	ND
Calcium (Ca)	0.067
Magnesium (Mg)	0.041
Sulfur (S)	0.76
Cerium (Ce)	0.056
Chromium (Cr)	0.008
Cobalt (Co)	0.001
Copper (Cu)	ND
Nickel (Ni)	0.006
Scandium (Sc)	0.002
Zinc (Zn)	ND

ND – not detected

18.13.5 Value Determination for Manganese Carbonate Product

The quality of the manganese carbonate lends itself to consumption at the final stage of production within EMD and EMM facilities. This circumvents the need for the costly leaching and precipitation stages required of lesser quality raw materials. As a result, consumers have expressed willingness to secure the Nkamouna manganese carbonate based upon a percentage of the London Metal Bulletin manganese quotation for 99.9% metal and/or flake. The price history as quoted in the LMB shows a US$3,000/t manganese metal price used in the financial model against the recent price history. Since GeoCam will produce manganese carbonate containing about 47% manganese metal content, the price assumption for the sale of manganese carbonate used in the model is 40% of the manganese metal price, or US$0.54/lb.

18.14 Contracts

As of December 2010, there were no contracts of significance that may cause material liability to Geovic/Geocam. Contracts were generally limited to consulting services for the development of the feasibility study.

18.15 Taxes and Royalties

SRK is not aware of any royalty agreements pertaining to the project other than an unsigned "rental" agreement between the Cameroonian government and GeoCam for land disturbance within the mine permit (2,490 Ha). This rate was estimated at $US400,000/yr and SRK does not consider this potential agreement material to the project.

Income tax is estimated at 38.5% of net income from the 3rd Quarter of 2017.

If unskilled Cameroonian nationals are hired, a 25% "Cameroonian Labor Reduction" is credited to net income before income tax is calculated. An additional credit of 0.5% on gross revenue is added to net income for a "Product Export Reduction".

An Ad Valorem Tax of 2% on gross revenue is applied based on negotiations with the Cameroonian government. Ad valorem tax is considered an operating expense.

18.16 Capital Costs

Table 18.16.1 summarizes the capital cost estimate for the Project including contingency. The treatment plant, support services and infrastructure serve as the basis for the capital cost estimate. Table 18.16.2 provides a breakdown of cost by discipline.

Table 18.16.1: Capital Cost Estimate Summary by Main Area (4Q10, ±15%)

Main Area Code	Subtotal US$ 000's	% of Capital
Construction Indirects	44,370	8.0%
Treatment Plant Costs	167,470	30.1%
Reagents & Plant Services	40,003	7.2%
Infrastructure	117,911	21.2%
Mining	36,554	6.6%
Management Costs	73,429	13.2%
Owner's Project Costs	76,476	13.7%
Owner's Operations Costs	0	0%
Contingency	60,951	11%
Project Total	**617,163**	**100.0%**

Table 18.16.2: Capital Cost Estimate Summary by Discipline (4Q10, ±15%)

Primary Discipline	Total US$ 000's	% of Capital
A - General	28,865	4.7%
B - Earthworks	61,297	9.9%
C - Concrete	35,528	5.8%
D - Steelwork	8,218	1.3%
E - Platework (including tankage)	17,997	2.9%
F - Mechanical	125,706	20.4%
G - Piping	30,617	5.0%
H - Electrical	59,609	9.7%
M - Buildings & Architectural	26,686	4.3%
N - Freight	20,880	3.4%
O - Owner's Costs	84,464	13.7%
P - EPCM	80,889	13.1%
Q - Mining	36,407	5.9%
Project Total	**617,163**	**100.0%**

18.16.1 Estimate Basis

The estimate is expressed in United States dollars (US$) based on prices and market conditions as at fourth quarter 2010 (4Q10).

Preliminary general arrangement drawings were produced with sufficient detail to permit the assessment of the engineering quantities for concrete, steelwork and mechanical for the process plant and overland piping. Unit rates that reflect the current market conditions were established for bulk materials, capital equipment and labor. The rates have been provided by contractors with experience in the area derived from recent market enquiries to local Cameroonian and overseas contractors and benchmarked against rates received for other projects.

Budget pricing for equipment and facilities has been obtained from suitably experienced suppliers. The supply rates used in the estimate have been reviewed and deemed to reflect the current market conditions.

The following exchange rates have been used to convert input pricing sourced in currencies other than US dollars for the capital cost estimates.

Table 18.16.1.1: Foreign Exchange Rates

Currency	Exchange to	Rate
AUD	US$	0.98844
US$	US$	1.00000
EUR	US$	1.33450
ZAR	US$	0.14527
XAF	US$	0.001953

18.16.2 Treatment Plant Capital

Table 18.16.2.1: Summary Costs by Main Area for Treatment Plant

Main Area Code	Cost excluding contingency (US$ 000's)
Treatment Plant - General	4,848
Physical Upgrade Plant (PUG)	26,164
Grinding	5,108
Leaching and Counter Current Decantation	29,803
Sulfide Precipitation	15,204
Manganese Carbonate Recovery	24,020
Acid Plant	62,323
Total	**167,470**

18.16.3 Reagents and Services Capital Costs

Table 18.16.3.1: Summary Costs by Main Area for Reagents and Services

Main Area Code	Cost excluding contingency (US$ 000's)
Reagents & Plant Services - General	402
Reagents / Process Consumables - Plant Facility	14,078
Water Services	6,355
Plant Services	10,181
Air Services	1,107
Fuels	2,532
Electrical Services	5,350
Total	**40,003**

18.16.4 Infrastructure Capital

Table 18.16.4.1: Summary Costs by Main Area for Infrastructure

Main Area Code	Cost excluding contingency (US$ 000's)
Infrastructure - General	20,087
Environmental	2,168
Utilities & Services	1,922
Power Supply	28,261
Tailings Storage Facility	33,822
Site Buildings	11,076
Permanent Village	11,647
Mobile Equipment	8,927
Total	117,911

18.16.5 Mine Capital

Mine capital costs of the study and have been estimated by SRK.

The unit cost for transportation and compaction of prime material types has been detailed in Table18.16.5.1.

Table 18.16.5.1: Pre-Production Earthworks that are Capitalized

Model	$/m^3	Total Preproduction Cost (US$ 000's)
Quarry 1	3.64	655
Quarry 2	3.67	440
TSF to Waste	1.81	1,379
Plant Borrow to TSF	2.02	7,152
Plant Breccia to Plant	1.93	966
TSF to Fill	1.93	977
Mine Borrow to TSF	1.93	2,334
Mine Breccia to Plant	1.97	2,757
Total	**2.03**	**16,663**

18.16.6 EPCM

A summary of EPCM management costs are presented in Table 18.16.6.1.

Table 18.16.6.1: Summary of EPCM Management Costs

Category	Estimated Cost (US$ 000's)
501 EPCM - Home Office	37,112
520 EPCM - Site	33,255
540 Specialist Consultants	2,485
560 Vendor Representatives	577
Total	**73,429**

18.16.7 Owner's Costs

A summary of Owner's costs are presented in Table 18.16.7.1.

Table 18.16.7.1: Summary of Owner's Costs

Category	Estimated Cost (US$ 000's)
Salaries (Excluding Mining)	11,923
Douala Office	197
Site Office	497
Insurances	1,114
Financial	301
Fees	146
Consultants	533
Personnel	2,622
Contracts	1,767
General	2,960
First Fills	10,482
Total	**32,542**

18.16.8 Light Vehicles and Ancillary Equipment Costs

A summary of light vehicles and ancillary equipment costs are presented in Table 18.16.8.1.

Table 18.16.8.1: Summary of Light Vehicles and Ancillary Equipment Costs

Description	Quantity	Unit Cost (US$ 000's)	Total Cost (US$ 000's)
Light Vehicles	55	37.5	2,060
Ambulance	1	224	224
Cranes	5	699	3,494
Utility Vehicles	10	14	140
Buses	10	84	840
Ancillary Equipment	Lot	1,884	1,884
Total		**2,942.5**	**8,642**

18.16.9 First Fill

A summary of first fill costs are presented in Table 18.16.9.1.

Table 18.16.9.1: Summary of First Fill Costs

Category	Quantity	Unit Cost (US$/T)	Cost (US$ 000'S)
Sulfur	4,692	408	1,912
Pyrite	3,537	295	1,043
Sodium Hydroxide	87	1,018	89
Dense Soda Ash	10,618	431	4,571
Sodium Hydrosulfide	1,696	1,150	1,950
Flocculant	44	5,547	242
Bulk Bags	4,600	18	81
Grinding Media			594
Total			**10,482**

18.16.10 Deferred Capital

Deferred capital cost items are included in the financial model but not in the capital cost estimate for the continued construction of the TSFs and replacement of mining fleet and light vehicles. The operating costs include a calculation for maintenance and replacement cost that will adequately provide for ongoing replacement capital for the Process Plant.

A deferred capital cost estimate is shown in Table 18.16.10.1.

Table 18.16.10.1: Deferred Capital Cost Estimate

	Cost US$ 000's										
Category	**2014**	**2015**	**2016**	**2017**	**2018**	**2019**	**2020**	**2021**	**2022**	**2023+**	**Total**
Mine Equipment	13,275	483		11,278	3,068	13,427	13,485	21,399	1,603	8,452	86,471
Process Plant			278		1,388		278		2,777	14,164	18,885
TSFs	338	980	974	6,374	122	203	203	203	7,414	48,142	64,954
Mine Closure										51,252	51,252
Total	**13,613**	**1,463**	**1,252**	**17,652**	**4,578**	**13,630**	**13,965**	**21,602**	**11,795**	**122,009**	**221,561**

After commissioning of the Project (Stage 1), the TSFs will operate for a number of years before they need to be raised. Over the life of the mine the height of the TSFs will be built/raised in a total of four stages, each stage being approximately nine to four years apart in the case of the PUG TSF, and approximately three to ten years in the case of the CCD TSF. The TSF staging will be optimized during the final design phase.

18.16.11 Contingency

The purpose of contingency is to make specific provision for uncertain elements of cost within the Project scope. Contingency makes no allowance for scope changes, escalation or exchange rate fluctuations.

The contingency by main area code is presented in Table 18.16.11.1.

Table 18.16.11.1: Contingency by Main Area Code

Main Area Code	**Subtotal Costs US$ 000's**	**Contingency US$ 000's**	**% of Capital**
Construction Indirects	44,370	5,735	12.9%
Treatment Plant Costs	167,470	18,229	10.9%
Reagents & Plant Services	40,003	5,097	12.7%
Infrastructure	117,911	14,567	12.4%
Mining	36,554	2,215	6.1%
Management Costs	73,429	7,460	10.2%
Owner's Project Costs	76,476	7,648	10.0%
Owner's Operations Costs	0	-	
Future Capital	0	-	
Total	**556,213**	**60,951**	**11.0%**

The contingency by discipline are provided in Table 18.16.11.2.

Table 18.16.11.2: Contingency by Discipline

Primary Discipline	Subtotal Costs US$ 000's	Contingency US$ 000's	% of Capital
A - General	25,953	3,893	15.0%
B - Earthworks	53,420	8,013	15.0%
C - Concrete	32,298	3,230	10.0%
D - Steelwork	8,308	831	10.0%
E - Platework	17,224	1,802	10.5%
F - Mechanical	124,023	12,402	10.0%
G - Piping	27,580	5,273	19.1%
H - Electrical	54,696	5,470	10.0%
M - Buildings & Architectural	25,093	2,509	10.0%
O - Owner's Costs	78,656	7,961	10.1%
P - EPCM	73,429	7,460	10.2%
Q - Mining	35,533	2,108	5.9%
Total	**556,213**	**60,951**	**11.0%**

18.17 Steady State Operating Costs

Costs are presented in United States dollars (US$) and are based on prices obtained during the fourth quarter of 2010. Operating costs presented in this section are representative of steady state operations. Please refer to the life of mine operating costs section for total operating cost disclosure.

The summary operating cost is presented in Table 18.17.1 and Figure 18-17 and 18-18.

Table 18.17.1: Steady State Summary of Operating Cost

Cost Center	Total Cost		Fixed Cost		Variable Cost	
	US$/annum (Millions)	US$/t Ore	%	US$/annum (Millions)	US$/annum (Millions)	US$/t Ore
Mining	24	10.54	0	-	24	10.54
Labor	12	5.45	100	12	-	0.00
Operating Consumables	52	22.77	0	-	52	22.77
Transportation for Consumables	35	15.41	0	-	35	15.41
Product Transportation and Sales	15	6.76	0	-	15	6.76
Power	26	11.42	90	23	3	1.18
Maintenance	11	4.94	33	4	7	3.29
General and Administration	16	7.20	100	16	-	0.00
Total	**192**	**84.50**	**29**	**56**	**136**	**59.96**

Note: Mining costs assume fixed operating cost per year divided by Pug feed tonnes. This does not represent stockpile material mined (i.e.: total ore tonnes mined in a given year)

The operating cost of cobalt produced is US$31,231/t before credits for nickel and manganese are taken into account. The operating cost expressed in terms of US$/t cobalt produced is fixed for a given head grade (including cobalt, nickel, manganese, aluminum and iron). This value

will change with variations in head grade. The value needs to be assessed for each period of the Life of Mine Operations.

18.18 Life Of Mine Operating Costs

Life of mine operating costs vary from those reported in the annualized operating cost estimate as the results have been reported from the economic model which is not a steady state analysis.

18.18.1 Mine Operating Costs

Open-cut mining operating costs were determined using first principle engineering, consumption rates from cost guides and vendor supply bids. US$317 million will be spent over the life of mine which equates to approximately US$4.56/ore.t (US$1.05/t).

Table 18.18.1.1: Mine Operating Costs

Description	US$/ore.t	LOM ($000's)
Loading	$1.21	$82,633
Dozing	$0.74	$50,353
Hauling	$1.16	$79,025
Roads, Dumps and Stockpiles	$0.60	$40,760
Sample Drilling	$0.26	$17,941
Mining Support Equipment	$0.11	$7,418
Mine G&A	$0.75	$50,938
Total Mining	$4.83	$329,067

18.18.2 PUG Plant

The PUG Plant operating costs total US$288 million (US$4.23/ore.t) over the LOM (Table 18.18.2.1).

Table 18.18.2.1: PUG Plant Operating Costs

Description	US$/ore.t	LOM (US$ 000's)
Power	$3.12	$212,425
Consumables	$0.40	$27,049
Maintenance	$0.54	$36,655
Labor	$0.18	$12,144
Total PUG Plant	**$4.23**	**$288,272**

18.18.3 Leach and Recovery Plant

The Leach and Recovery Plant operating costs total US$1,921 million (US$28.19/ore.t) over the LOM (Table 18.18.3.1).

Table 18.18.3.1: Leach and Recovery Plant Operating Costs

Description	US$/ore.t	LOM (US$ 000's)
Power	$1.40	$95,079
Consumables	$24.23	$1,651,104
Maintenance	$2.03	$138,203
Labor	$0.54	$36,524
Total Leach and Recovery Plant	$28.19	$1,920,910

18.18.4 Process G&A

The process G&A costs total US$455 million (US$6.54/ore.t) over the LOM (Table 18.18.4.1).

Table 18.18.4.1: Process General and Administration Costs

Description	US$/ore.t	LOM (US$ 000's)
Power	$3.34	$227,762
Consumables	$0.35	$23,677
Maintenance	$1.50	$101,965
Labor	$0.65	$44,152
Process Management	$0.69	$47,165
Metallurgical	$0.42	$28,419
Total Leach and Recovery Plant	$6.94	$473,140

18.18.5 General and Administration

The G&A costs do not include any contingency and total US$522 million (US$7.66/ore.t) over the LOM. Included in the G&A costs are the ongoing closure costs associated with the mining panels.

Table 18.18.5.1: General and Administrative Costs

Description	US$/ore.t	LOM (US$ 000's)
Labor	$1.64	111,769
Expenses	$5.83	397,139
Mine Panel Closure	$0.10	6,573
Civil Site Fleet	$0.10	6,507
Total G&A	**$7.66**	**521,987**

Please note that mine panel closure has been included as an overhead cost due to the progressive rehabilitation requirement that only 60 Ha of mine disturbance be open at any one time once operations commence.

Mine, process, and project G&A costs as presented throughout section 20.5 total US$1,046 million, or US$15.35/t. This value includes provisions for:

- Project management and support functions;
- Laboratories / metallurgical testing;
- Power and consumables related to process;

- Utility support equipment; and
- Mine closure costs.

18.19 Operations Labor

The Nkamouna Co-Ni-Mn Project will be the first operation of its kind in Cameroon and will require highly skilled and experienced personnel to manage the operation. The operation's manning structure is organized into six levels or categories. The organizational structure will include management expatriates, technical expatriates, local professionals, skilled labor, semi skilled labor and laborers.

Management expatriate staff will be used for senior management positions in the organization. These management positions will require personnel with mining management experience. It is envisaged that personnel with previous African mining experience will be recruited for these positions.

Technical and supervisory positions will be filled with a combination of technical expatriates and local professionals. Technical expatriates with experience in the base metal hydrometallurgical industry will be recruited to fill the technical expatriate positions. It is envisaged that local professional staff will be trained to move into the positions of the technical expatriates over time.

Cameroonian nationals will be recruited for the skilled and semi-skilled labor positions. People from the local communities will be recruited for the laborer positions. As this process plant is the first facility of its kind in Cameroon, it is recognized that inexperienced personnel will be employed to operate the processing plant. Plant operating personnel will be recruited early to facilitate the rigorous training process that will be required.

Key operations personnel will have to be recruited before the implementation phase of this project commences. These will include the following positions, Vice President Mining Operations, Vice President Finance, Chief Accountant, Head of Security and Vice President Plant Operations and Metallurgy.

Table 18.19.1 shows a summary of total manpower requirements for Nkamouna when operating at full capacity. Manning levels will change based on mining production profile on a as needed basis.

Table 18.19.1: Manning Levels - Total

Department	Management Expatriates	Technical Expatriates	Local Professional	Skilled Labor	Semi Skilled Labor	Laborer	Total
Administration	4	0	28	53	100	55	240
Mining	7	7	15	4	111	46	190
Processing and Metallurgy	5	6	4	37	78	114	244
Process Maintenance	1	3	8	37	2	41	92
Total	**17**	**16**	**55**	**131**	**291**	**256**	**766**

The majority of people in the Administration Department will be within Security (68 employees) and Catering and Camp Facilities (46 employees) sections. For the purpose of the study, Catering is assumed to be carried out by permanent personnel.

Manning levels for mining are based upon the assumption that the mine will be an owner operator mine.

18.20 Economic Analysis

The financial model (The Model) constructed by SRK combines project costs and revenues generated by the Feasibility Study contributors and applies them (project economics) over the proposed project life. The model is designed to form the basis of future financial analysis by third party investors and as a tool for optimization and/or project analysis.

Model results are at the project-level and do not account for the joint venture ownership. However as noted elsewhere, the project is held by GeoCam with Geovic's participation as a 60.5% direct corporate holding. The remaining 39.5% interest is held by the SNI and other Cameroonian investors represented by SNI.

18.20.1 Reliance on Information

SRK's opinion contained herein is based on data and information provided to SRK by GeoCam and its consultants throughout the course of the project work.

SRK used its experience to determine if the information from previous reports was suitable for inclusion in this volume and adjusted information that required amending. The level of detail utilized is appropriate for this level of study.

This volume includes technical information, which required subsequent calculations to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, SRK does not consider them material.

18.20.2 Markets

As with all projects, the market prices of cobalt, nickel and manganese are key parameters to Project success. Factors influencing the market prices include the following:

- International economic and political conditions;
- Expectations of inflation and international currency exchange rates;
- Interest rates;
- Global or regional consumptive patterns;
- Speculative activities, levels of supply and demand, increased production;
- Availability and costs of metal substitutes, metal stock levels maintained by producers and traders; and
- Inventory carrying costs.

Table 18.20.2.1 shows general market assumptions used in the economic analysis. The average 36 month cobalt price through December 31, 2010 is US$57,760 (US$26.20/lb). All product sales assume a reduction in market price for cobalt (85% of price) and nickel (82% of price) products to reflect the refining charges for mixed sulfide precipitate.

Table 18.20.2.1: Market Assumptions

Description	Value (US$)	Unit	Basis
Cobalt	$26.20	US$/lb	December 2010, 36 month average, Metals Bulletin High Grade Product
Nickel	$8.71	US$/lb	December 2010, 36 month average, LME
$MnCO_3$	$0.544	US$/lb	40% of Mn metal price ($3,000/t)
Mixed Sulfide Freight	$105.01	US$/t-MSP	Provided by Lycopodium
Manganese Carbonate Freight	$0.09	US$/lb	Provided by Lycopodium
Marketing and Insurance	$0.14	$/lb-Co	Provided by Lycopodium
Ad Valorem Tax	2.00%		Provided by SNI
Royalty	0.0		

18.20.3 Economic Considerations

The following sections describe the currency, financial assumptions, Value Added Tax (VAT) and project logistics inputs.

Currency

The analysis provisions for US dollar, Euro dollar, and Central African CFA Franc (XAF) currency exchange as most equipment and reagents are sourced from either Europe or the United States. All local labor is paid in XAF while expatriate labor is assumed to be paid in US dollars. Table 18.20.3.1 shows exchange rates used in the analysis.

Table 18.20.3.1: Exchange Rates

Currency	Exchange Rate per US$
US Dollars	$1.00
Euros	€ 0.75
CFA Francs	XAF 512

Financial Assumptions

SRK has modeled the project using 100% Project equity to eliminate the positive effect of gearing on Net Present Value (NPV) results. The working capital is assumed to be 20% of cash costs and is modeled to reflect periodic changes in operating cash-flow requirements.

Taxable income is estimated in the Model and is calculated by deducting a depreciation provision from the operating margin (earnings). The Model assumes an 8 year straight-line depreciation of all capital which provides an indicative estimate of allowable depreciation. The Model assumes allowance of loss carry-forward in the calculation of taxable income. The maximum tax rate is 38.5%. The Cameroonian government will allow a tax holiday for the Project and therefore a 50% reduction of corporate tax is levied during the first 5 years of the installation phase and 12 years of the exploitation phase. The full income tax rate is applied from year 13 onward.

The Model includes a financial analysis using the assumptions shown in Table 18.20.3.1.

Table 18.20.3.1: Financial Assumptions

Description	Value	Unit
Project Equity	100.0%	- (no gearing)
Working Capital Requirement	20.0%	of cash costs
Depreciation	8.0	year straight-line
Discount Rate	8.0%	
Corporate Income Tax Rate	38.5%	
Dividend Tax	0.0%	
Cameroonian Labor Deduction	25.0%	
Exported Product Deduction	0.5%	

18.20.4 Model Parameters

Assumptions used are discussed in detail throughout this section and are summarized in Table 18.20.4.1.

Table 18.20.4.1: Technical Economic Model Parameters

Model Parameter	Technical Input
Pre-production Period	36 months
Mine Life	11.0 years
Project Life	23 years
Cobalt Price (LOM Avg.)	US$26.20/lb
Nickel Price (LOM Avg.)	US$8.71/lb
$MnCO_3$ Price (LOM Avg.)	US$0.54/lb
Cobalt Produced (klb)	229,843
Nickel Produced (klb)	163,483
$MnCO_3$ Produced (klb)	2,478,929
MSP Produced (kt)	260,640
Discount Rate	8%

A 36 month pre-production period allows for mine development, tailings and process plant construction. The mine will have an estimated life of 11.0 years, followed by 12 years of stockpile processing, for a total treatment of 68.16 Mt of reserves through the process plant. An estimated 104,256 t (229,843,000 lb) of cobalt will produced over the LOM.

18.20.5 Project Financials

The financial analysis results, shown in Table 18.20.5.1, indicate an $NPV_{8\%}$ of US$669 million with an IRR of 22% (after estimated taxes). The estimated payback will be in 41 months (2Q 2017) from the start of production in 2014. The following provides the basis of the LOM plan and economics:

- Proven and probable reserves of 68.1 Mt are included;
- A mine life of 11.0 years, and Project life of 23 years;
- Overall average metallurgical recoveries of 58.66% cobalt, 16.43% nickel and 53.06% manganese over the LOM;
- A cash operating cost of US$25,500/t (US$11.57/pound) cobalt equivalent;

- Capital costs of US$839 million, comprised of initial capital costs of US$617 million, and sustaining capital over the LOM of US$222 million;
- Mine closure costs, included in the above sustaining capital estimate of US$51.2 million; and
- No provision for salvage value.

Table 18.20.5.1: Financial Model Results

Description	Units	Value	Unit Cost (US$/lb Co Eq)
Production			
ROM Ore Processed	kt	68,132	-
Cobalt Produced	klb	229,843	-
Nickel Produced	klb	163,483	-
$MnCO_3$ Produced	klb	2,478,929	-
Estimate of Cash Flow			
Cobalt Price	US$/lb	$26.20	-
Nickel Price	US$/lb	$8.71	-
$MnCO_3$ Price	US$/lb	$0.54	-
Gross Revenue	**US$000's**	7,635,149	$22.267
Freight and Marketing	US$000's	(282,651)	($0.824)
Net Revenue	**US$000's**	7,352,497	$21.442
Gross Income	**US$000's**	7,352,497	$21.442
Operating Costs			
Mining	US$000's	329,067	$0.960
PUG Plant	US$000's	288,272	$0.841
Leach and Recovery Plant	US$000's	1,920,910	$5.602
Process G&A	US$000's	473,140	$1.380
G&A	US$000's	521,987	$1.552
Ad Valorem Tax	US$000's	152,703	$0.445
Operating Costs	**US$000's**	3,686,079	**$10.750**
Cash Cost			$11.574
Operating Margin	**US$000's**	**3,666,418**	**$10.692**
Capital			
Mine Equipment	US$000's	112,944	-
PUG, Leach and Recovery	US$000's	565,775	-
TSF	US$000's	97,586	-
Owners Costs	US$000's	11,168	-
Mine Closure	US$000's	51,252	-
Total Capital	**US$000's**	**838,725**	-
Total Tax	US$000's	726,463	-
Cash Flow	**US$000's**	**2,139,019**	**-**
Present Value at 8%	US$000's	669,579	-
IRR	%	22%	-

18.20.6 Sensitivities

From the NPV project sensitivity analysis detailed in Table 18.20.6.1 and IRR analysis illustrated in Table 20.6.4, results suggest that the project is most sensitive to revenues (market prices), followed by capital costs and is least sensitive to operating costs.

Project results are materially affected by the planned 36 month pre-production period, followed by 5 months (of the planned 16 month) ramp-up period before positive cash flow is realized. This results in the high sensitivity to capital costs due to the time value of money.

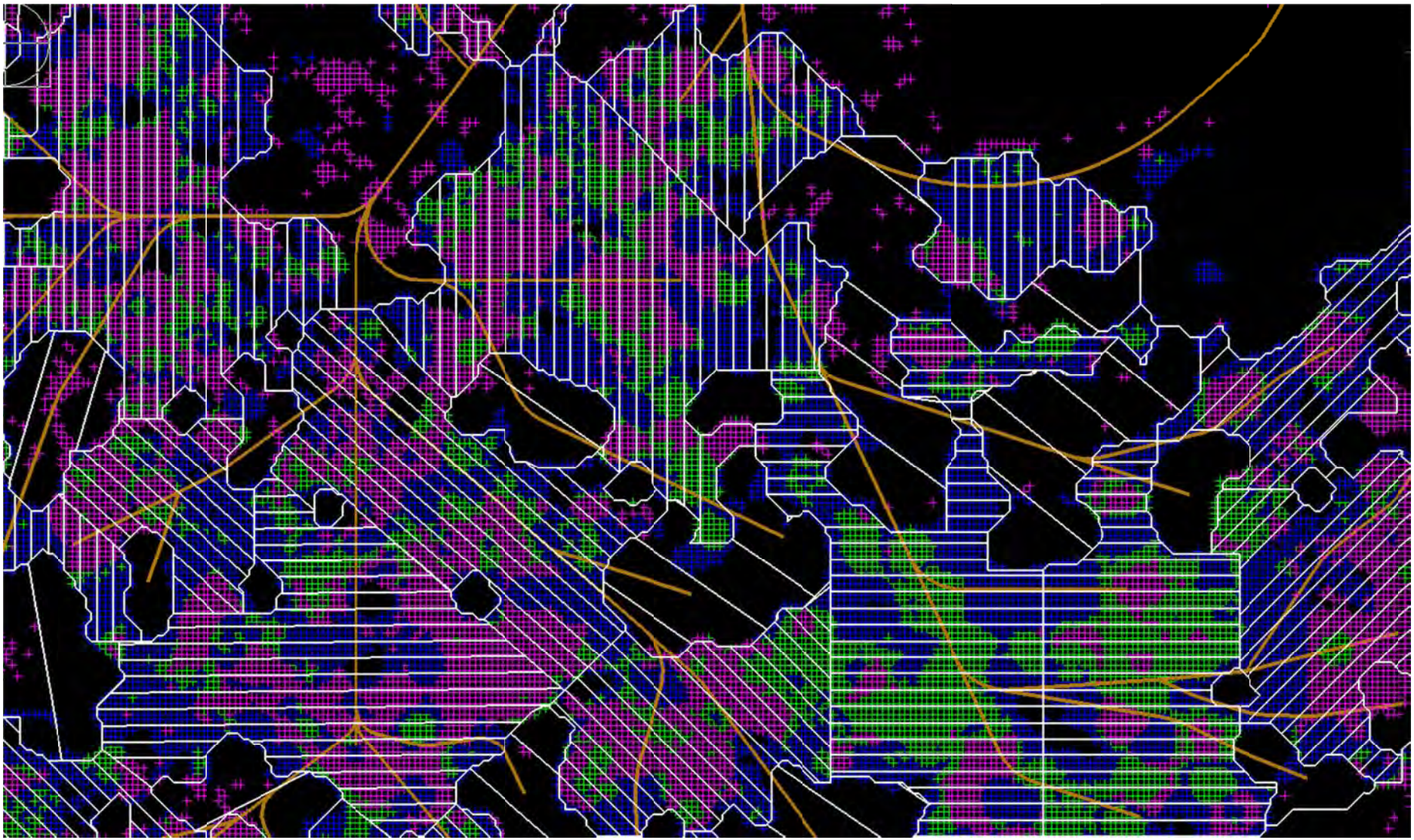
Table 18.20.6.1: Project Sensitivity (NPV8% US$ million)

NPV 8%	-15%	-10%	-5%	Base	5%	10%	15%
Revenues	316	434	552	670	787	904	1,021
Capital Costs	753	726	698	670	642	613	585
Operating Costs	718	702	686	670	654	637	621

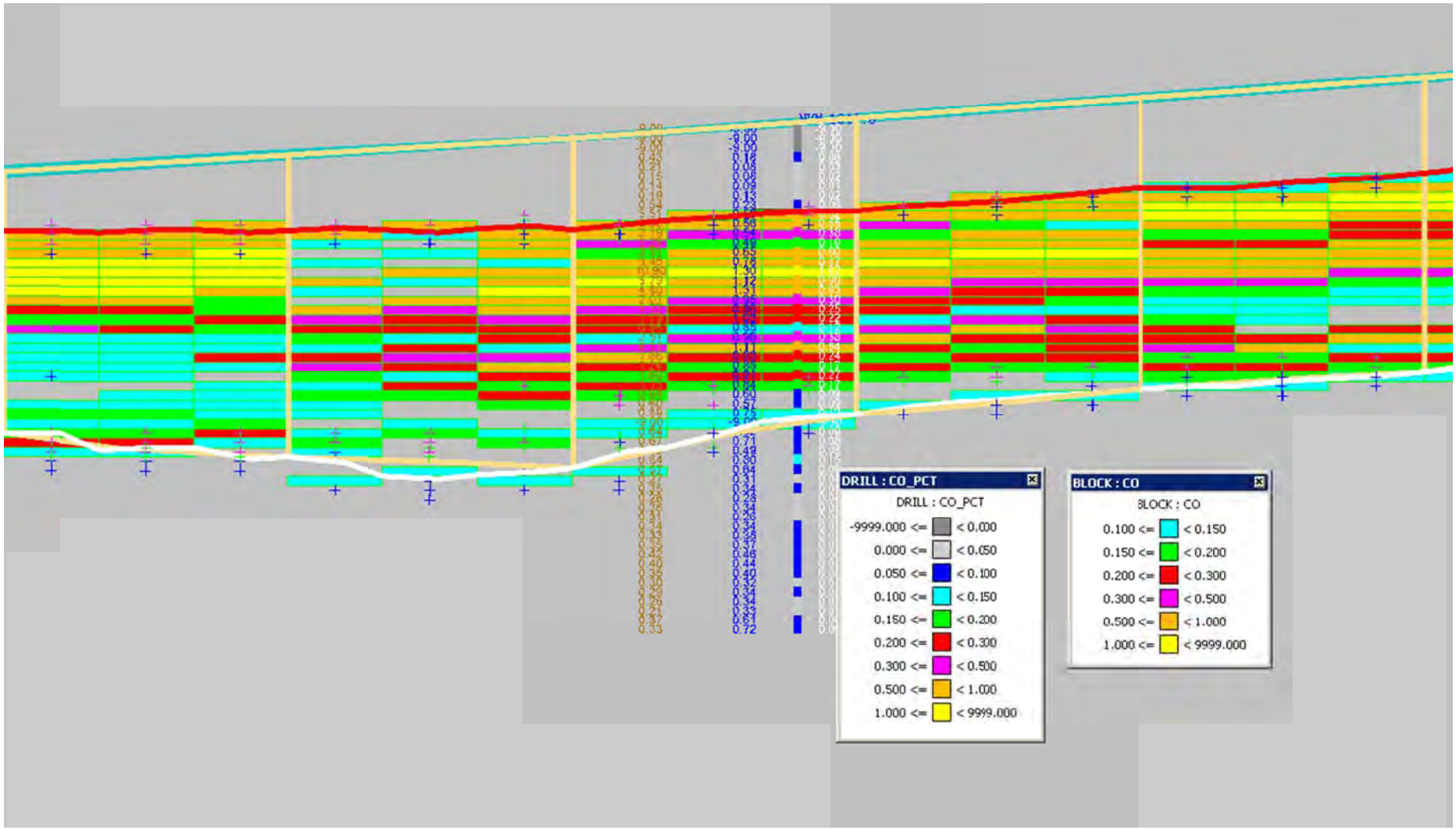
Table 18.20.6.2: Project Sensitivity (IRR)

IRR	-15%	-10%	-5%	Base	5%	10%	15%
Revenues	16%	18%	20%	22%	24%	25%	27%
Capital Costs	25%	24%	23%	22%	21%	20%	19%
Operating Costs	23%	22%	22%	22%	22%	21%	21%

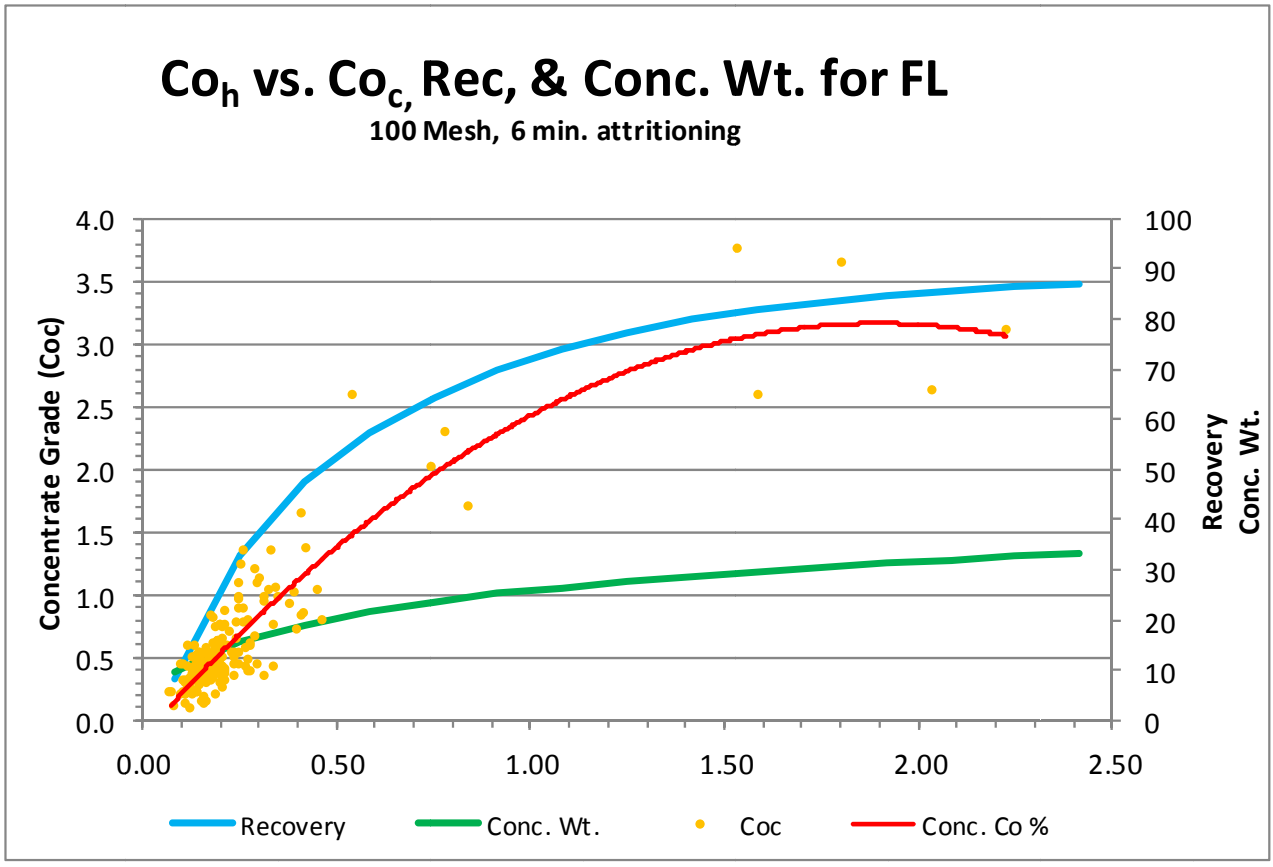
The sensitivity to operating costs is also relatively high. In fact, if the effect of the time value of money is removed, operating costs are as sensitive as capital costs. This is due to the high proportion of operating cost to revenue. LOM economic results suggest that operating costs represent 50% of revenues. Typically, this metric is on the order of 35 - 40% for similar mining operations. Inordinate freight, reagent, power, and G&A costs due the remoteness of the project are contributing factors.



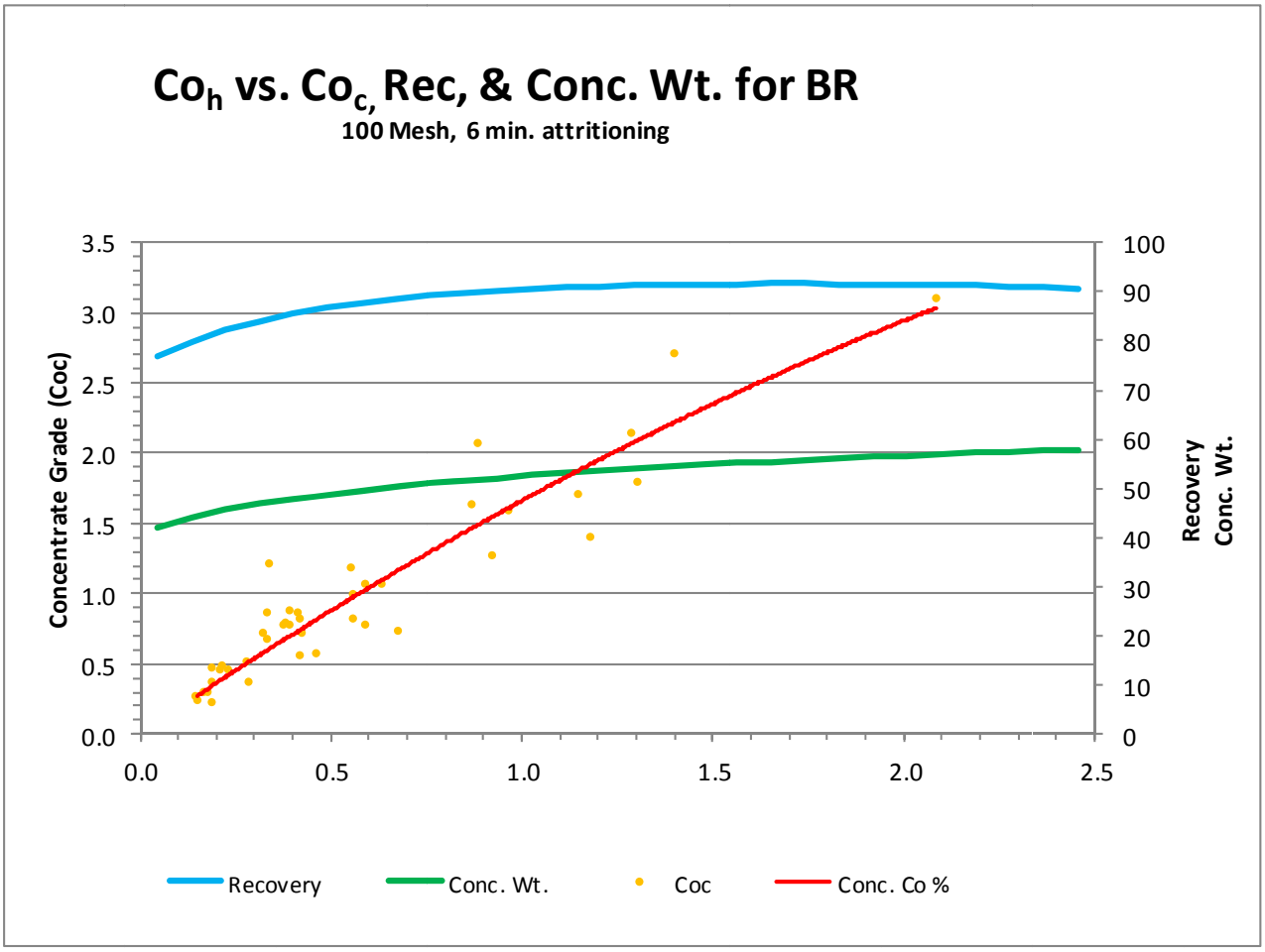
SRK Consulting Engineers and Scientists	**Nkamouna and Mada Deposits East Province of Cameroon, Africa**	**Pit Identification and Strip Layout**		
SRK Job No.: 195100.020				
File Name: Figure18-1.docx		Date: 07/28/2010	Approved: bcs	**Figure: 18-1**



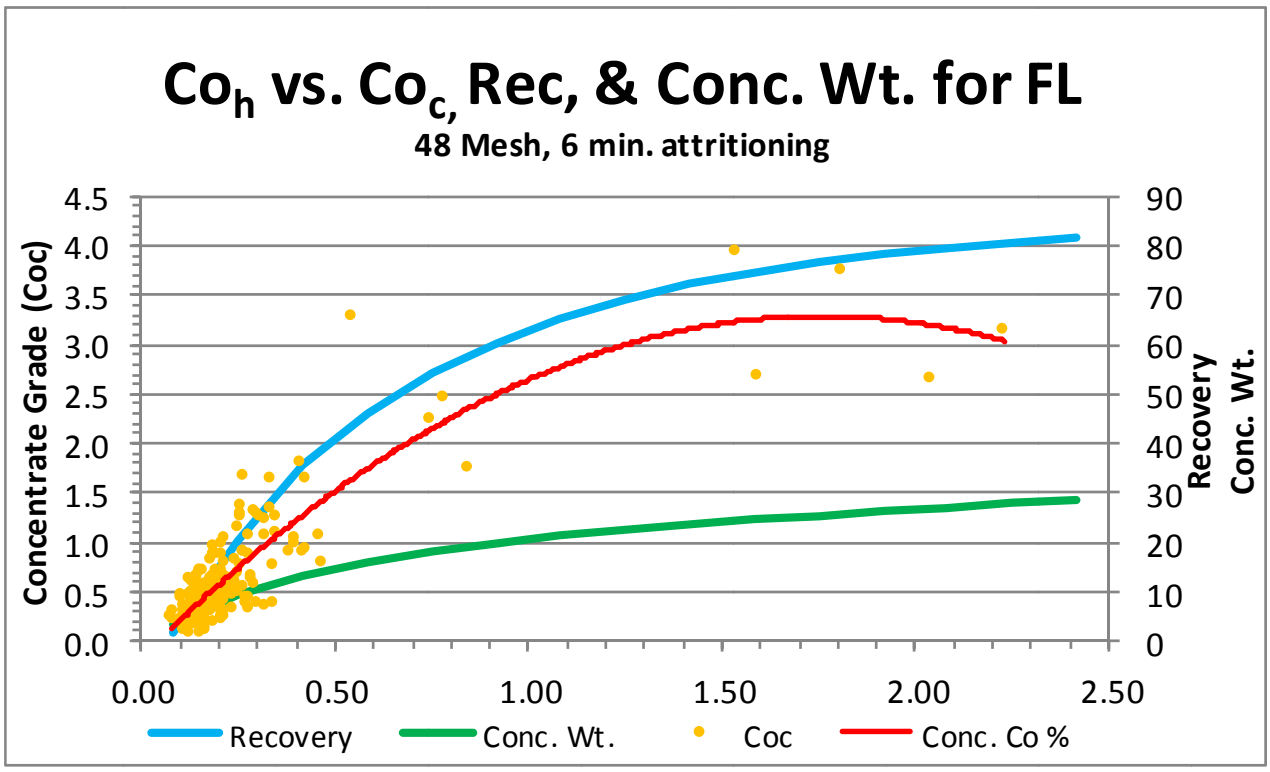
SRK Consulting Engineers and Scientists	**Nkamouna and Mada Deposits East Province of Cameroon, Africa**	**Typical Cross Section Strip Layout**
SRK Job No.: 195100.020		
File Name: Figure18-2.docx		Date: 07/28/2010 Approved: bcs **Figure: 18-2**



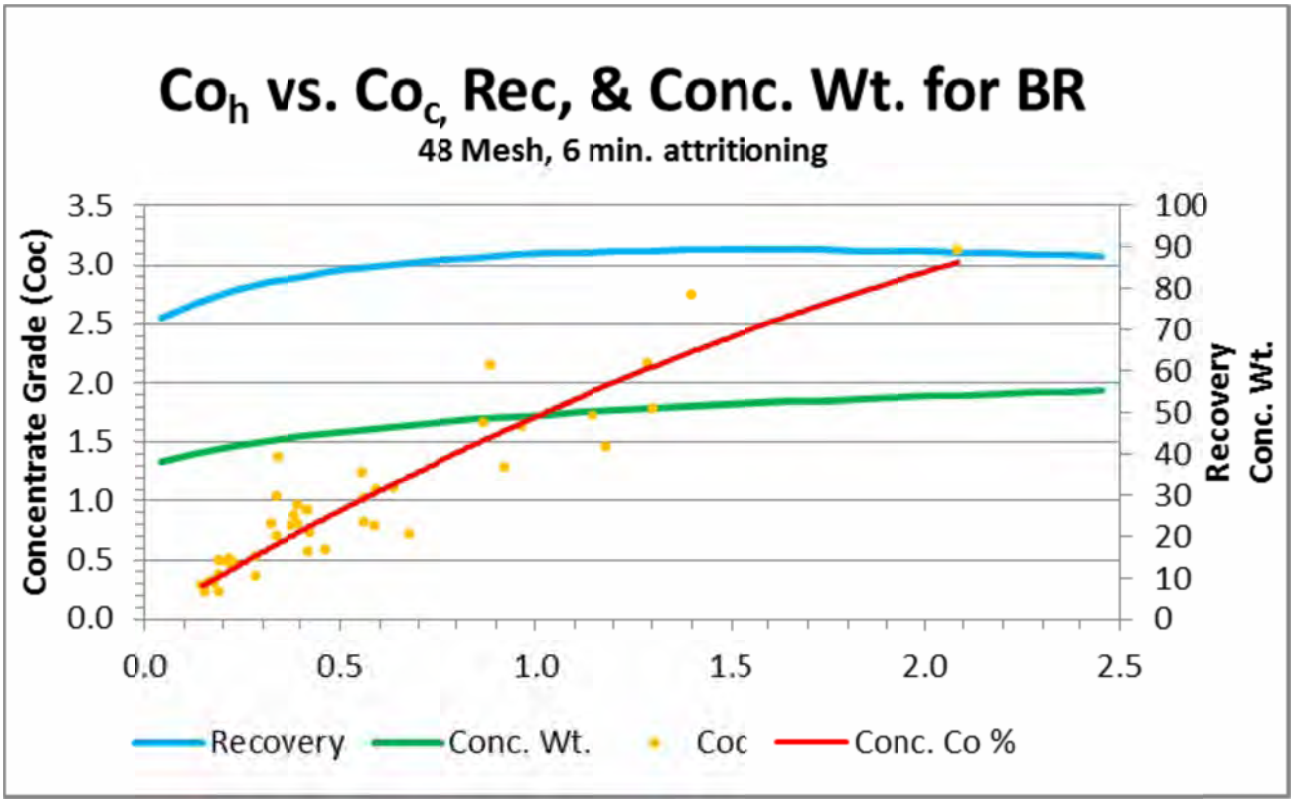
SRK Consulting Engineers and Scientists	Nikamouna and Mada Deposits East Province of Cameroon, Africa	100 Mesh Ferralite Regression Curves Zone
SRK Job No.: 195100.020		
File Name: Figure18-3.docx		Date: 05/11/2011 | Approved: bcs | **Figure: 18-3**



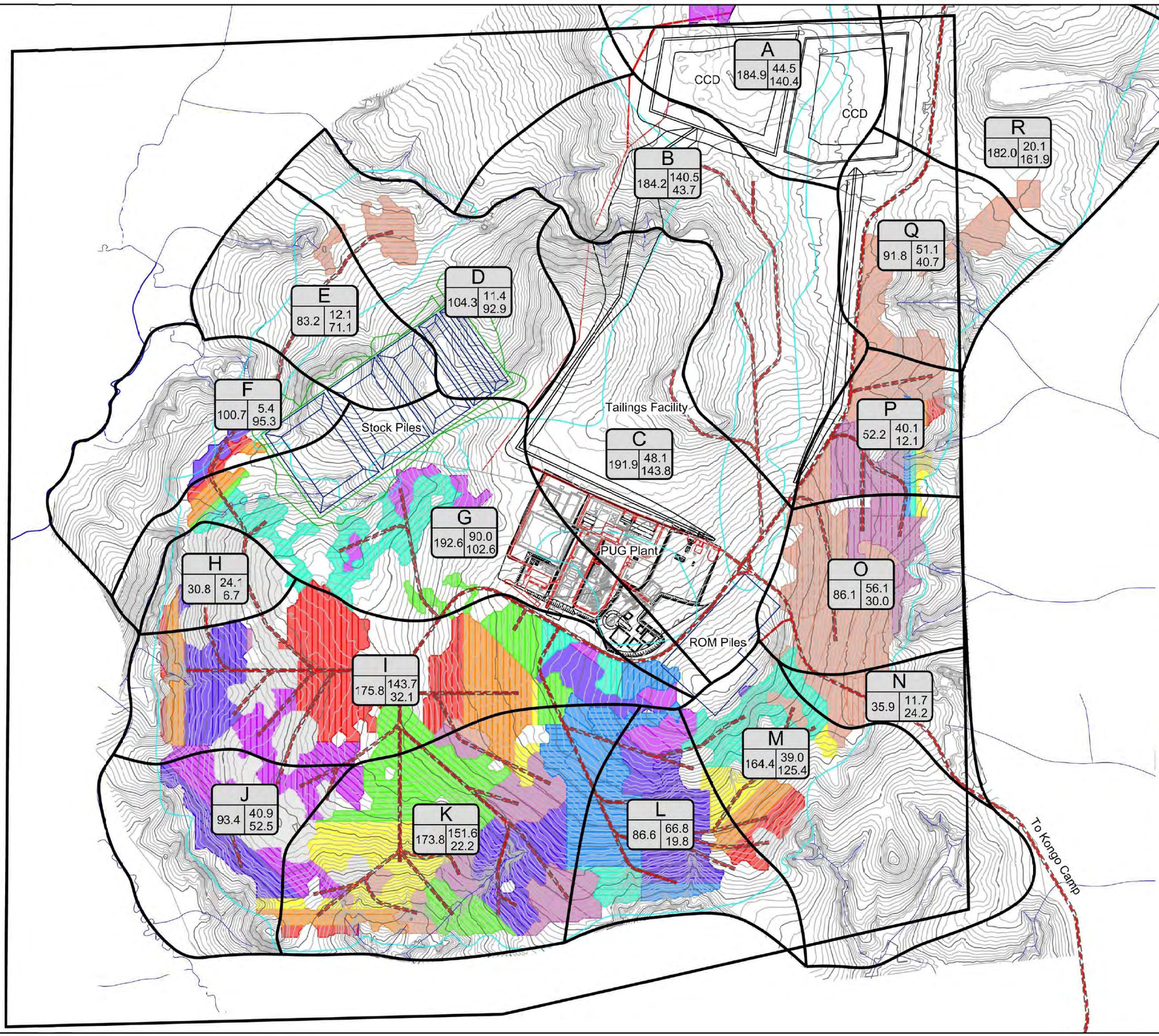
SRK Consulting Engineers and Scientists	Nikamouna and Mada Deposits East Province of Cameroon, Africa	100 Mesh Breccia Regression Curves
SRK Job No.: 195100.020		
File Name: Figure18-4.docx		Date: 05/11/2011 Approved: bcs **Figure: 18-4**



SRK Consulting Engineers and Scientists	Nikamouna and Mada Deposits East Province of Cameroon, Africa	48 Mesh Ferralite Regression Curves	
SRK Job No.: 195100.020			
File Name: Figure18-5.docx		Date: 5/11/2011 Approved: JAV	**Figure: 18-5**



SRK Consulting Engineers and Scientists	Nikamouna and Mada Deposits East Province of Cameroon, Africa	48 Mesh Breccia Regression Curves		
SRK Job No.: 195100.020				
File Name: Figure18-6.docx		Date: 5/11/2011	Approved: bcs	**Figure: 18-6**



N

1 : 20,000
Scale in Meters

200 0 200 400 600 800 1,000

Legend

Basin Designation

0 | 0 | 0

Potentially Disturbed

Basin Area in Hectares

Clean area

- Drainage Flow Arrow
- Catchement Basin Boundary
- Schist Boundary
- Rivers/Streams
- Lease Boundary

Geovic Cameroon PLC
Bastos - Youndé,
P.O. Box 11555, Cameroon
Office: +237.22.21.4518
Fax: +235.22.21.1802

Nkamouna Water Catchment Areas

System:	UTM 33 WGS 84	Prepared For:	Brian Briggs
Date:	30 March 2011	Drawn by:	Andrew Lovato
File:	Catchments Plan	Approved:	.
Scale:	1:20,000	Sheet No:	1 of 1

CCD

CCD

PUG TSF

Stock Piles

PUG Plant

Cell Tower

ROM Piles

To Kongo Camp

N

1 : 20,000
Scale in Meters

0 200 400 600 800 1,000

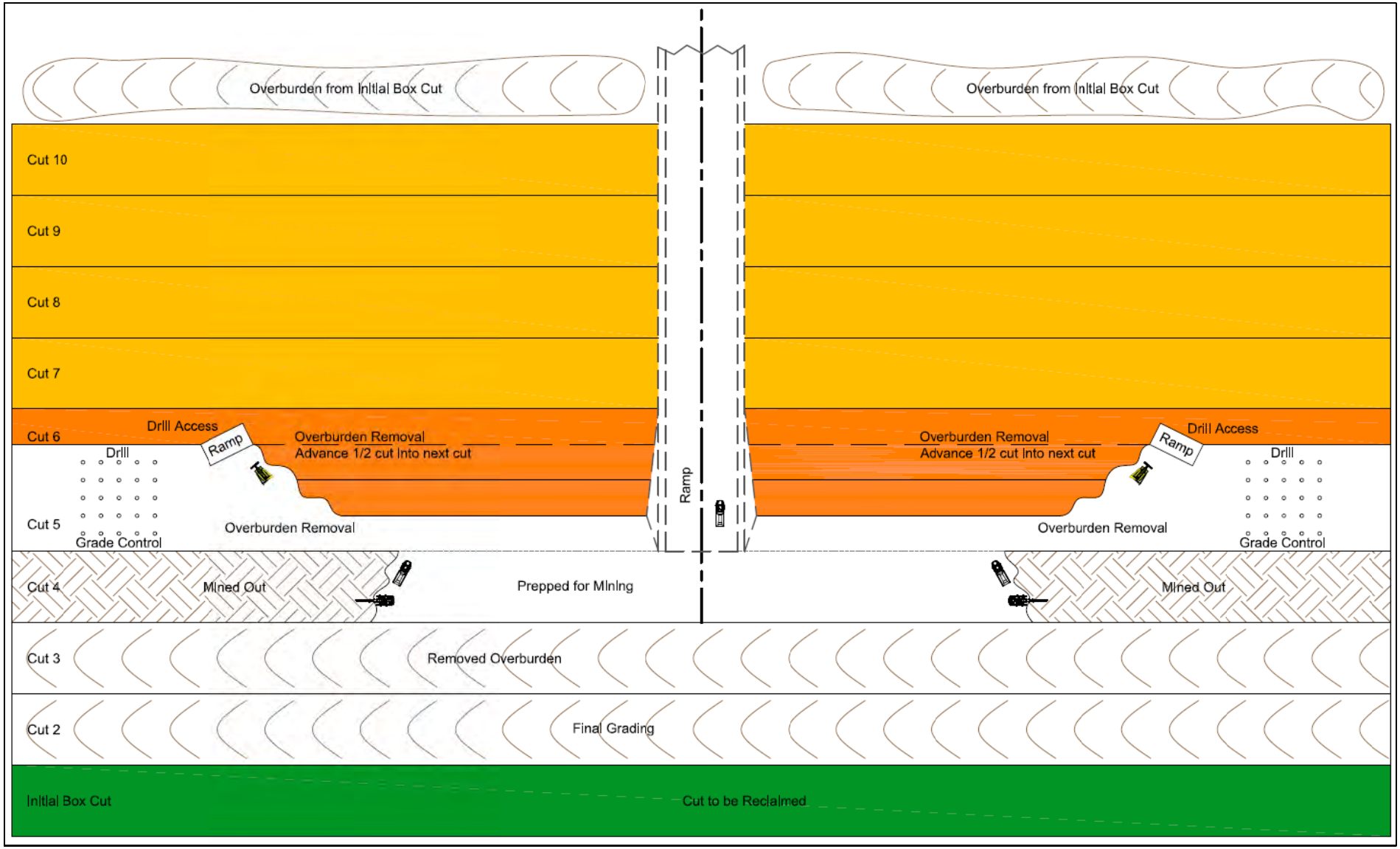
Legend

2017	
2016	
2015	
2014	
2013	Construction
Schist Boundary	
Rivers/Streams	
Lease Boundary	

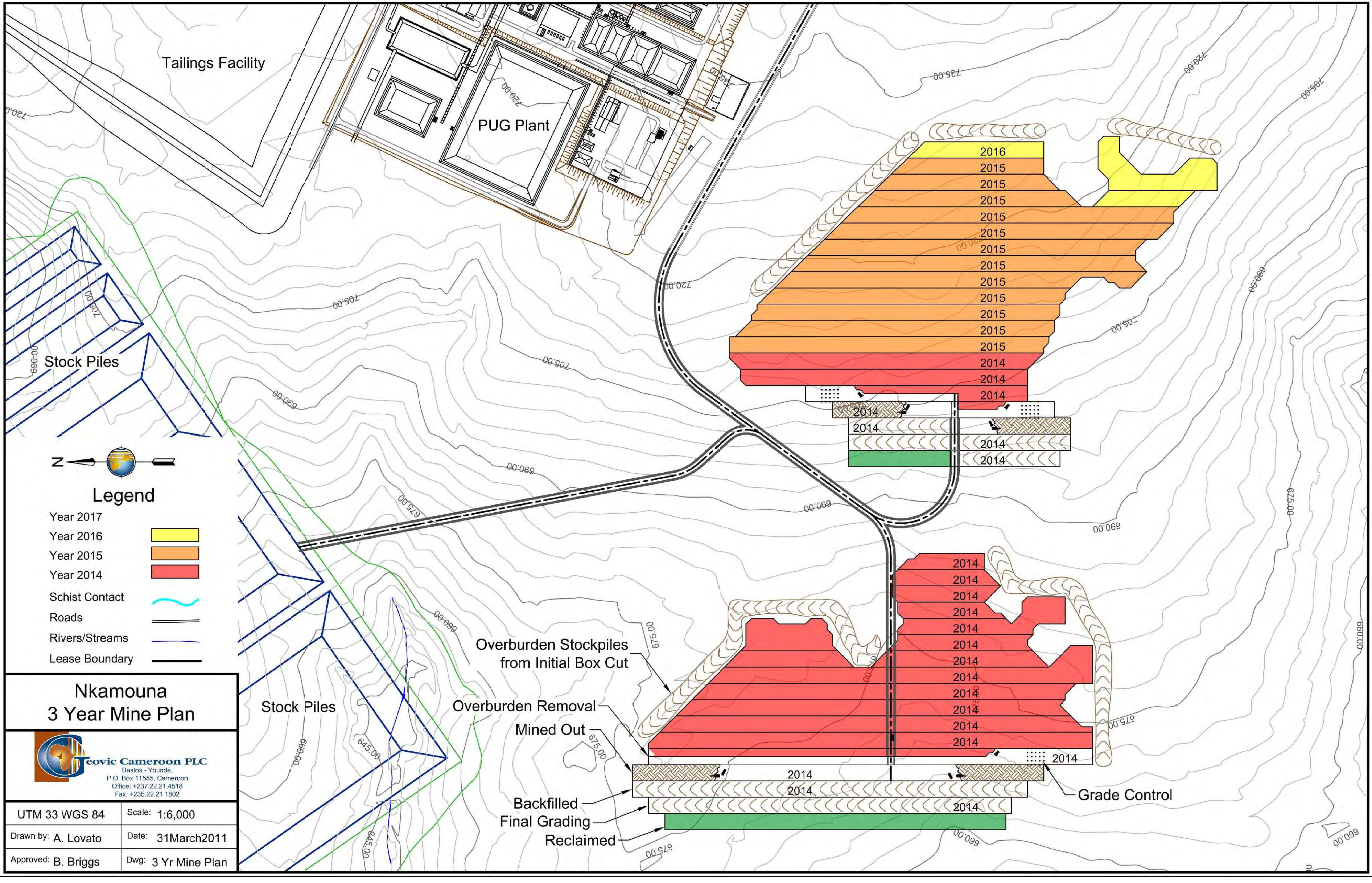
Geovic Cameroon PLC
Bastos - Youndé,
P.O. Box 11555, Cameroon
Office: +237.22.21.4518
Fax: +235.22.21.1802

Nkamouna
5 Year Infrastructure

System:	UTM 33 WGS 84	Prepared For:	Brian Briggs
Date:	30 March 2011	Drawn by:	Andrew Lovato
File:	5 Year Mine Plan	Approved:	.
Scale:	1:20,000	Sheet No:	1 of 1



SRK Consulting Engineers and Scientists	**Nkamouna and Mada Deposits** **East Province of Cameroon, Africa**	**Typical Pit Advance**
SRK Job No.: 195100.020	**Source: Geovic, 2010**	
File Name: Figure18-9.docx	Date: 07/28/2010 Approved: bcs	**Figure: 18-9**


Tailings Facility
PUG Plant
Stock Piles
Stock Piles
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2015
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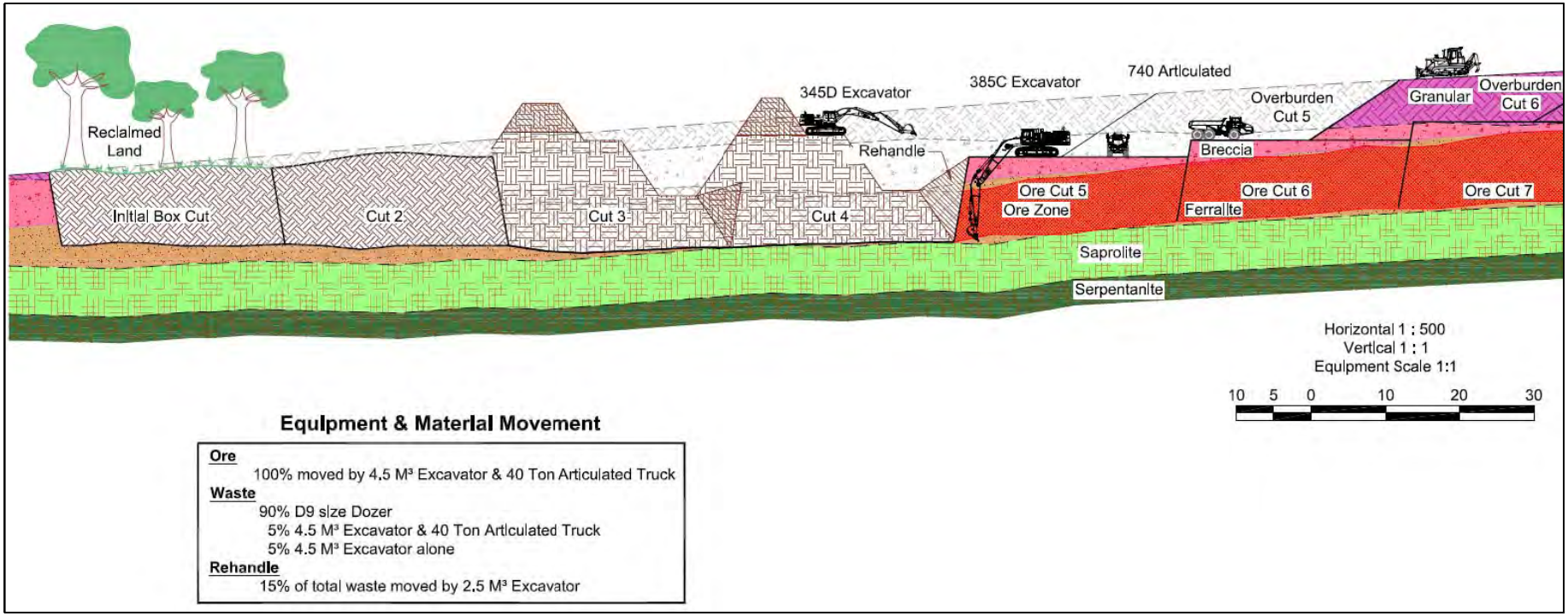
Overburden Stockpiles from Initial Box Cut
Overburden Removal
Mined Out
Backfilled
Final Grading
Reclaimed
Grade Control
720.00
735.00
705.00
690.00
675.00
660.00
645.00
N
Legend
Year 2017
Year 2016
Year 2015
Year 2014
Schist Contact
Roads
Rivers/Streams
Lease Boundary
Nkamouna
3 Year Mine Plan
Geovic Cameroon PLC
Bastos - Youndé,
P.O. Box 11555, Cameroon
Office: +237.22.21.4518
Fax: +235.22.21.1802
UTM 33 WGS 84
Scale: 1:6,000
Drawn by: A. Lovato
Date: 31March2011
Approved: B. Briggs
Dwg: 3 Yr Mine Plan



Equipment & Material Movement

Ore
100% moved by 4.5 M³ Excavator & 40 Ton Articulated Truck

Waste
90% D9 size Dozer
5% 4.5 M³ Excavator & 40 Ton Articulated Truck
5% 4.5 M³ Excavator alone

Rehandle
15% of total waste moved by 2.5 M³ Excavator

SRK Consulting
Engineers and Scientists

SRK Job No.: 195100.020

File Name: Figure18-11.docx

Nkamouna and Mada Deposits
East Province of Cameroon, Africa

Source: Geovic, 2010

Modified Open Cast Mining Method Cross Section

Date: 07/28/2010 | Approved: bcs | **Figure: 18-11**

ORE

ROM BIN

APRON FEEDER

PRIMARY CRUSHING
MINERAL SIZER

PADDLE MIXER
FEED CONVEYOR

PADDLE MIXER

VIBRATING
PRODUCT SCREEN

SECONDARY
CRUSHER

SECONDARY CRUSHER
FEED CONVEYOR

ATTRITIONING
CELLS FEED
HOPPER

ATTRITIONING
CELLS FEED PUMPS

ATTRITIONING CELLS (4 BANKS)

DEWATERING
CYCLONES (x4)

HYDROSIZER
(x2)

DEWATERING
SCREEN (x2)

PUG TAILINGS
HOPPER

PUG
TAILINGS
PUMPS

PUG TAILINGS
STORAGE FACILITY

PUG CONCENTRATE
DEAD STOCKPILES

PUG CONCENTRATE
LIVE STOCKPILE

RECLAIM
FEEDER (x3)

MILL FEED
CONVEYOR

BALL MILL

BALL MILL
DISCHARGE
HOPPER

BALL MILL
DISCHARGE
PUMPS

LEACH FEED TANK

LEACH FEED
PUMPS

LEACHING

DRAWN	CHECKED	DESIGN	SENIOR ENG	DESIGN APP'D	PROJ APP'D	CLIENT APP'D

DRG No	REFERENCE DRAWINGS
XX	XXX

No	DATE	REVISIONS	DRN	CHK'D	DESIGN	DESIGN APP'D	PROJ APP'D	CLIENT APP'D
C	14APR11	ISSUED FOR FEASIBILITY STUDY	VEC	AH	AH	HS	BO	
B	14JAN11	ISSUED FOR STUDY	HP	HS	HS		BO	
A	19NOV10	ISSUED FOR INTERNAL REVIEW	HP	HS	HS		BO	

CLIENT: GEOVIC CAMEROON P.L.C.

PROJECT: NKAMOUNA Co-Ni-Mn PROJECT

Lycopodium

Lycopodium Minerals Pty Ltd ABN: 34 055 880 209
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DRAWING TITLE
PHYSICAL UPGRADE PLANT (PUG)
CRUSHING & CLASSIFICATION
PROCESS FLOW DIAGRAM

SCALE	JOB No.	DWG No.	REV.
NTS	1637	000-F-001	C

DRAWN	DATE
HP	18NOV10

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A1

SODA ASH
-
SODIUM HYDROSULPHIDE
-
MILLED PYRITE
-
SULPHURIC ACID
-
MILLED PUG CONCENTRATE
-

LEACHING

PRIMARY PURIFICATION

SULPHIDE PRECIPITATION REACTORS

MIXED SULPHIDE THICKENER

COUNTER CURRENT DECANTATION CIRCUIT (6 HIGH DENSITY THICKENERS)

SECONDARY PURIFICATION

VACUUM PUMP

STEAM

CONDITIONING TANK

SECONDARY PURIFICATION CLARIFIER

OXYGEN STRIPPING TOWER

TO LEACHING

HEAT EXCHANGER

CONDENSATE

TOWER FILTER PRESS

FILTER FEED TANK

TAILINGS PROCESSING

CCD TAILINGS STORAGE FACILITY

MIXED SULPHIDE BAGGING PLANT

TO MARKET
-

MANGANESE CARBONATE PRECIPITATION 2 THICKENER

MANGANESE CARBONATE PRECIPITATION 1 THICKENER

GLAUBER'S SALT CRYSTALLISER

MANGANESE CARBONATE PRECIPITATION 2

MANGANESE CARBONATE PRECIPITATION 1

TERTIARY PURIFICATION CLARIFIER

TERTIARY PURIFICATION

TO LEACHING

BELT FILTER

BELT FILTER

FILTER FEED TANK

MANGANESE CARBONATE PRECIPITATION 2 REPULP TANK

TO PUG TAILINGS

FILTER FEED TANK

MANGANESE CARBONATE PRECIPITATION 1 BAGGING PLANT

TO MARKET
-

DRAWN	CHECKED	DESIGN	SENIOR ENG	DESIGN APP'D	PROJ APP'D	CLIENT APP'D

DRG No	REFERENCE DRAWINGS
XX	XXX

No	DATE	REVISIONS	DRN	CHK'D	DESIGN	DESIGN APP'D	PROJ APP'D	CLIENT APP'D
C	14APR11	ISSUED FOR FEASIBILITY STUDY	VEC	AH	AH	HS	BO	
B	14JAN11	ISSUED FOR STUDY	HP	HS	HS		BO	
A	19NOV10	ISSUED FOR INTERNAL REVIEW	HP	HS	HS		BO	

CLIENT: GEOVIC CAMEROON P.L.C.

PROJECT: NKAMOUNA Co-Ni-Mn PROJECT

Lycopodium
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DRAWING TITLE: LEACH AND RECOVERY CIRCUIT

PROCESS FLOW DIAGRAM

SCALE	JOB No.	DWG No.	REV.
NTS	1637	000-F-002	C

DRAWN: HP DATE: 18NOV10

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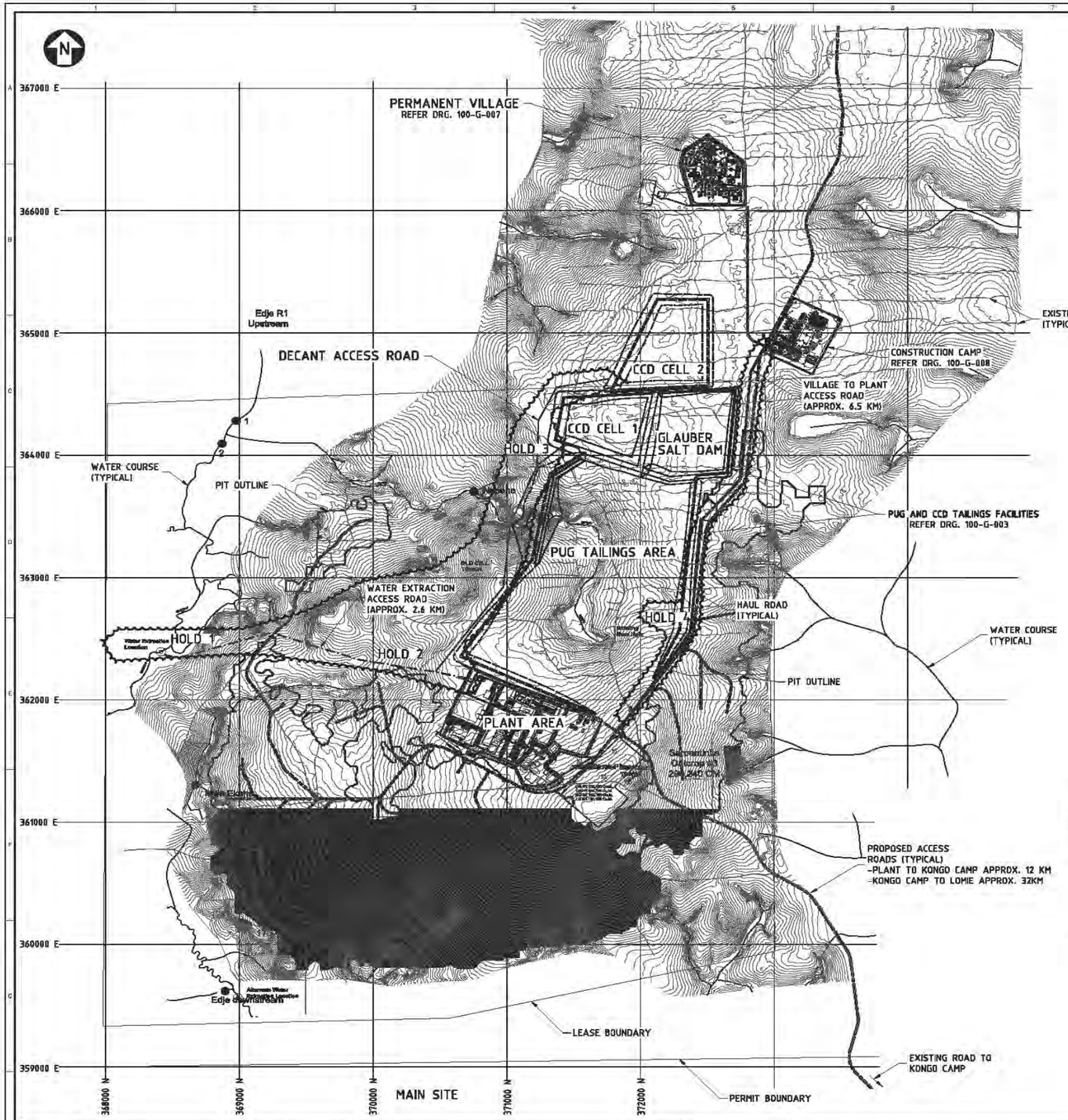
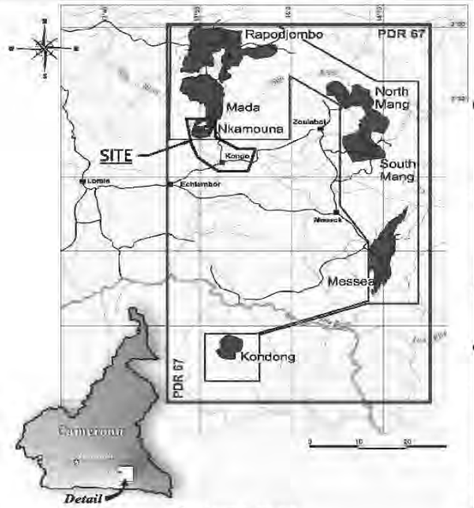


NOTES

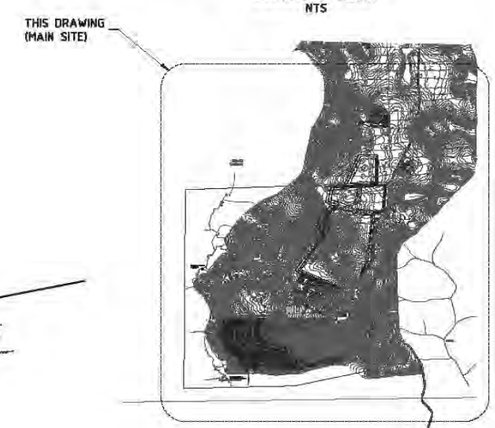
1. REFERENCE CO-ORDINATE SYSTEM - WGS84, ZONE 33 NORTH.

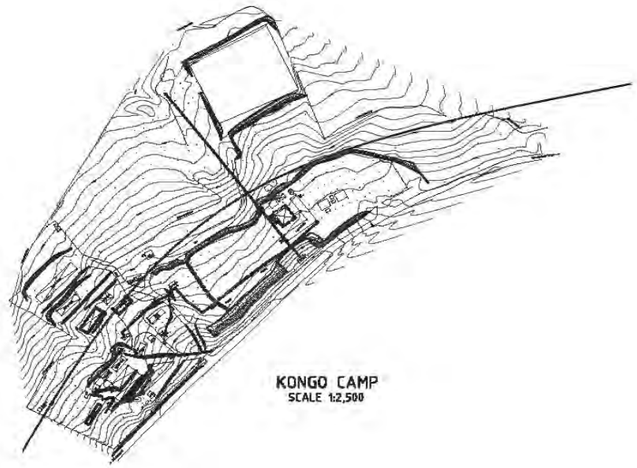
HOLDS

1. WATER EXTRACTION POINT TO BE CONFIRMED
2. ROAD ROUTE TO WATER EXTRACETION TBA
3. DECANT ROAD TO BE ADVISED
4. ACCESS ROAD TO BE CONFIRMED

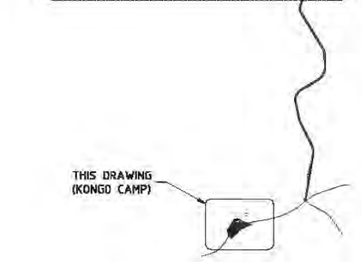


LOCALITY PLAN
NTS





KONGO CAMP
SCALE 1:2,500



SITE AREA
SCALE 1:60,000

300 0 300 600 900 1200 1500m

SCALE 1:15,000

DRAWN	CHECKED	DESIGN	SENIOR ENG	DESIGN APP'D	PROJ APP'D	CLIENT APP'D

DRG No	REFERENCE DRAWINGS
100-G-005	TREATMENT PLANT, PROCESS PLANT LAYOUT, PLAN
100-G-003	SITE AREA, TSF FACILITY INFRASTRUCTURE, PLAN
100-G-002	SITE AREA, GENERAL ARRANGEMENT, PLANT AREA AND TAILINGS

No	DATE	REVISIONS	DRN	CHKD	DESIGN	DESIGN APP'D	PROJ APP'D	CLIENT APP'D
C	07DEC10	ISSUED FOR STUDY	ALS		JS	JS		
B	29OCT10	ISSUED FOR CLIENT REVIEW	TK		JS	JS		
A	21OCT10	ISSUED FOR INTERNAL REVIEW	TK		JS	JS		

CLIENT: GEOVIC CAMEROON P.L.C.

PROJECT: NKAMOUNA Co-Ni-Mn PROJECT

Lycopodium

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T: +61(0) 8 6210 1222 www.lycopodium.com.au

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DRAWING TITLE
SITE AREA
GENERAL ARRANGEMENT
KEY PLAN

SCALE	JOB No.	DWG No.	REV.
1:15,000 U.N.O.	1637	100-G-001	C

DRAWN: TK DATE: 19OCT10

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B1

NORTH

CONSTRUCTION & LAYDOWN AREA

EPCM OFFICES

BUS DROP OFF PARK

CONTAINER HANDLING

MINE SERVICES AREA (BY OTHERS)

FENCE LINE

FACILITY LEGEND

PUG & HYDROMET PLANT:
1. 120-PRIMARY CRUSHING
2. 120-SECONDARY CRUSHING
3. 120-PADDLE MIXER & SCREENING
4. 120-ATTRITIONING & CLASSIFICATION
5. 120-PUG PRODUCT LIVE STOCKPILE
6. 120-PUG PRODUCT INTERMEDIATE STOCKPILE
7. 120-PUG PRODUCT STANDBY STOCKPILES
8. 120-PUG PRODUCT RECLAIM

9. 130-PUG PRODUCT MILLING & SLURRY STORAGE

10. 140-LEACHING & PRIMARY PURIFICATION
11. 140-COUNTER CURRENT DECANTATION
12. 140-PLS TANK
13. 140-PLS EVENT POND
14. 140-TAILINGS PROCESSING
15. 140-SECONDARY PURIFICATION

16. 150-SULPHIDE PRECIPITATION
17. 150-SULPHIDE FILTRATION & HANDLING
18. 150-SULPHIDE GRINDING
19. 150-TERTIARY PURIFICATION

20. 160-MANGANESE CARBONATE PRECIPITATION 1
21. 160-MANGANESE CARBONATE PRECIPITATION 2
22. 160-MANGANESE CARBONATE FILTRATION & HANDLING
23. 160- MANGANESE CARBONATE PURIFIED LIQUOR POND
24. 160-GLAUBER'S SALT PLANT
25. 160-MOTHER LIQUOR STORAGE POND

26. 170-SULPHUR STORAGE & HANDLING
27. 170-SULPHURIC ACID PLANT

28. 210-PYRITE STORAGE
29. 210-PYRITE MILLING
30. 210-FLOCCULANT
31. 210-SODIUM HYDROSULPHIDE STORAGE
32. 210-SODIUM HYDROSULPHIDE MIXING
33. 210-SODIUM CARBONATE STORAGE
34. 210-SODIUM CARBONATE MIXING
35. 210-NITROGEN PLANT
36. 210-WATER TREATMENT CHEMICAL STORAGE
37. 210-HYDROGEN PEROXIDE STORAGE

38. 230-WATER TREATMENT PLANT
39. 230-FRESH WATER SURGE POND
40. 230-PROCESS WATER POND
41. 230-SODIUM SULPHATE RICH WATER POND
42. 230-FIRE WATER
43. 230-PUG CIRCUIT WATER POND

44. 240-STEAM GENERATION
45. 240-SEWAGE TREATMENT
46. 240-DIESEL POWER GENERATION

47. 250-AIR SERVICES

48. 260-DIESEL STORAGE & DISTRIBUTION

49. 300-CAR PARK
50. 300-WEIGHBRIDGE

51. 310-EVENT POND 1
52. 310-EVENT POND 2

53. 350-SECURITY / RECEPTION / FIRST AID BUILDING
54. 350-ADMINISTRATION BUILDING
55. 350-MAIN CONTROL ROOM
56. 350-PROCESS PLANT CHANGE HOUSE
57. 350-CAFETERIA
58. 350-CLINIC & EMERGENCY RESPONSE BUILDING
59. 350-PROCESS PLANT WORKSHOP & WAREHOUSE
60. 350-PROCESS PLANT OFFICE BUILDING
61. 350-ABLUTION BUILDING
62. 350-VEHICLE WASH BAY

63. 360-LABORATORY

64. 420-ROM PAD

65. 430-SITE RUN-OFF SETTLEMENT POND

66. 450-MINE OFFICE BUILDING
67. 450-MINE CHANGE HOUSE
68. 450-MINE WORKSHOP & WAREHOUSE
69. 450-MINE TRUCK WASH BAY

25 0 25 50 75 100 125

SCALE 1:1250

REFERENCE DRAWINGS	
100-G-006	REFINERY LAYOUT
100-G-001	SITE AREA, KEY PLAN & GENERAL ARRANGEMENT

No.	DATE	REVISIONS	DRN	CHK'D	DESIGN	DESIGN APP'D	PROJ APP'D	CLIENT APP'D
-		—						
E	29MAR11	ISSUED FOR FEASIBILITY STUDY	JM			JL	HS	
D	16MAR11	ISSUED FOR INTERNAL REVIEW - LAYOUT UPDATED	QJ			RGR		
C	24DEC10	RE-ISSUED FOR STUDY DOCUMENT - GENERAL REVISION, REFINERY REMOVED	DW			RG	RG	
B	07DEC10	ISSUED FOR STUDY	ALS			EJ		
A	22NOV10	ISSUED FOR INTERNAL REVIEW	KLL			DGB/EJ		

CLIENT: GEOVIC CAMEROON P.L.C.

PROJECT: NKAMOUNA Co-Ni-Mn PROJECT

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DRAWN	CHECKED	DESIGN	SENIOR ENG	DESIGN APP'D	PROJ APP'D	CLIENT APP'D

DRAWING TITLE: TREATMENT PLANT PROCESS PLANT LAYOUT PLAN

SCALE	JOB No.	DWG No.	REV.
1:1250	1637	100-G-005	E

DRAWN: K L L DATE: 21OCT10

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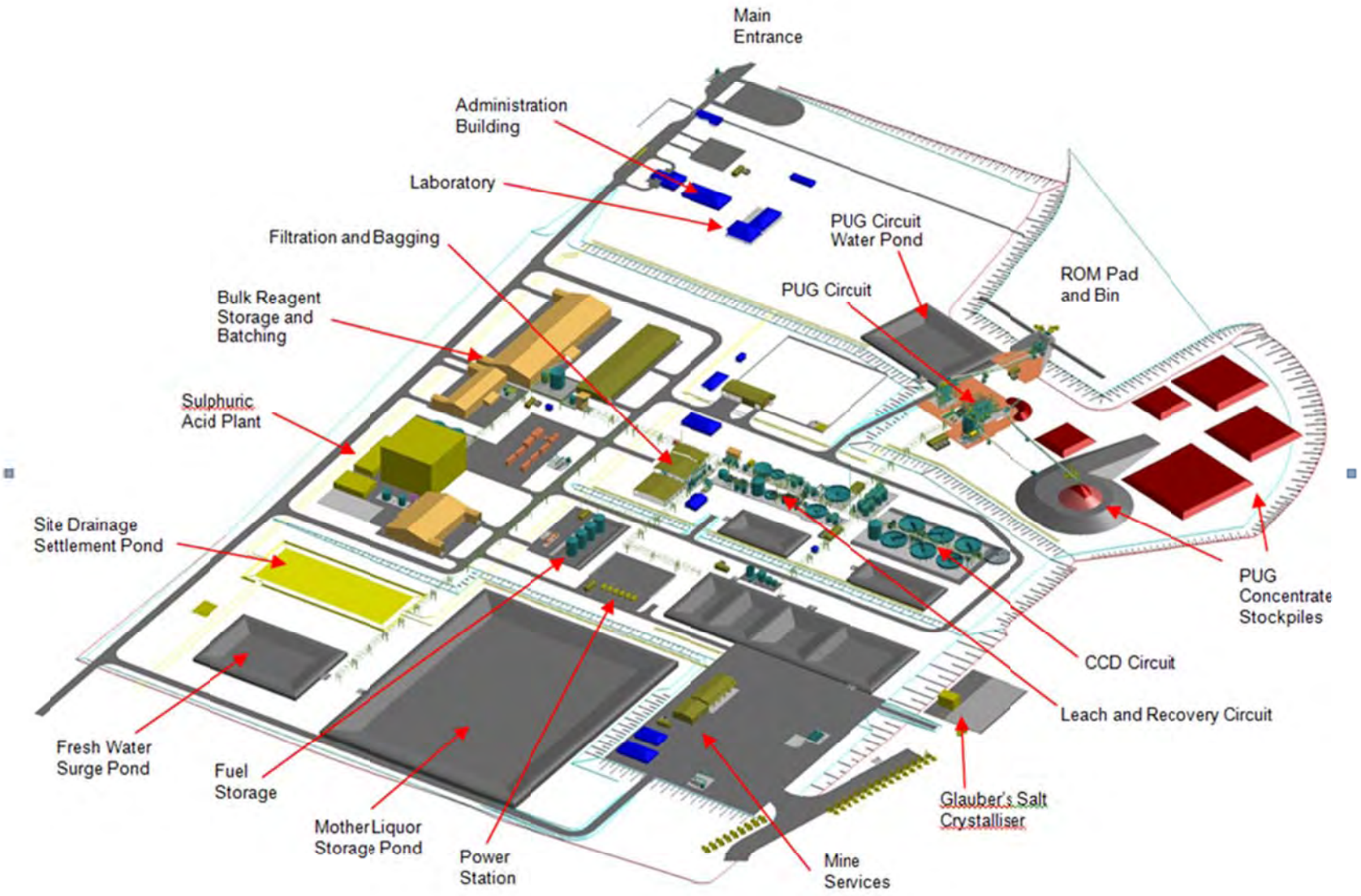
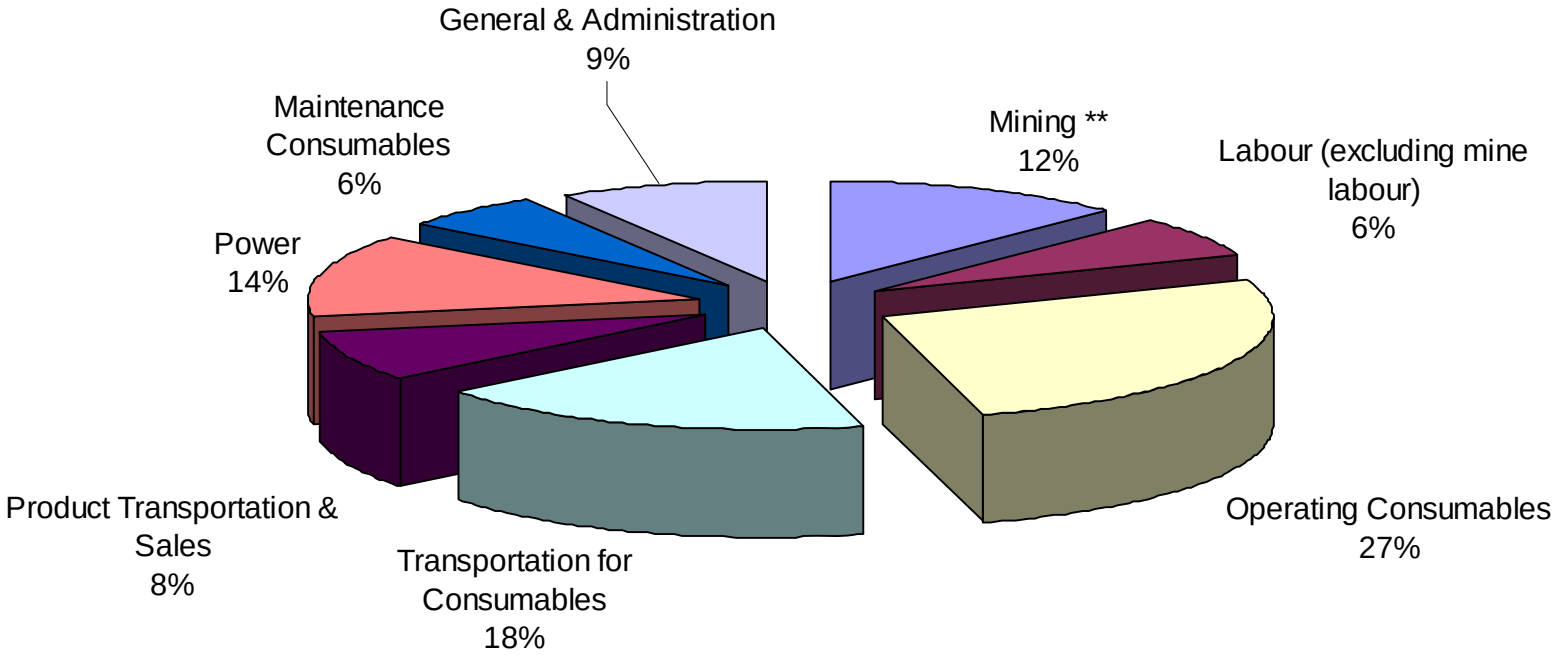
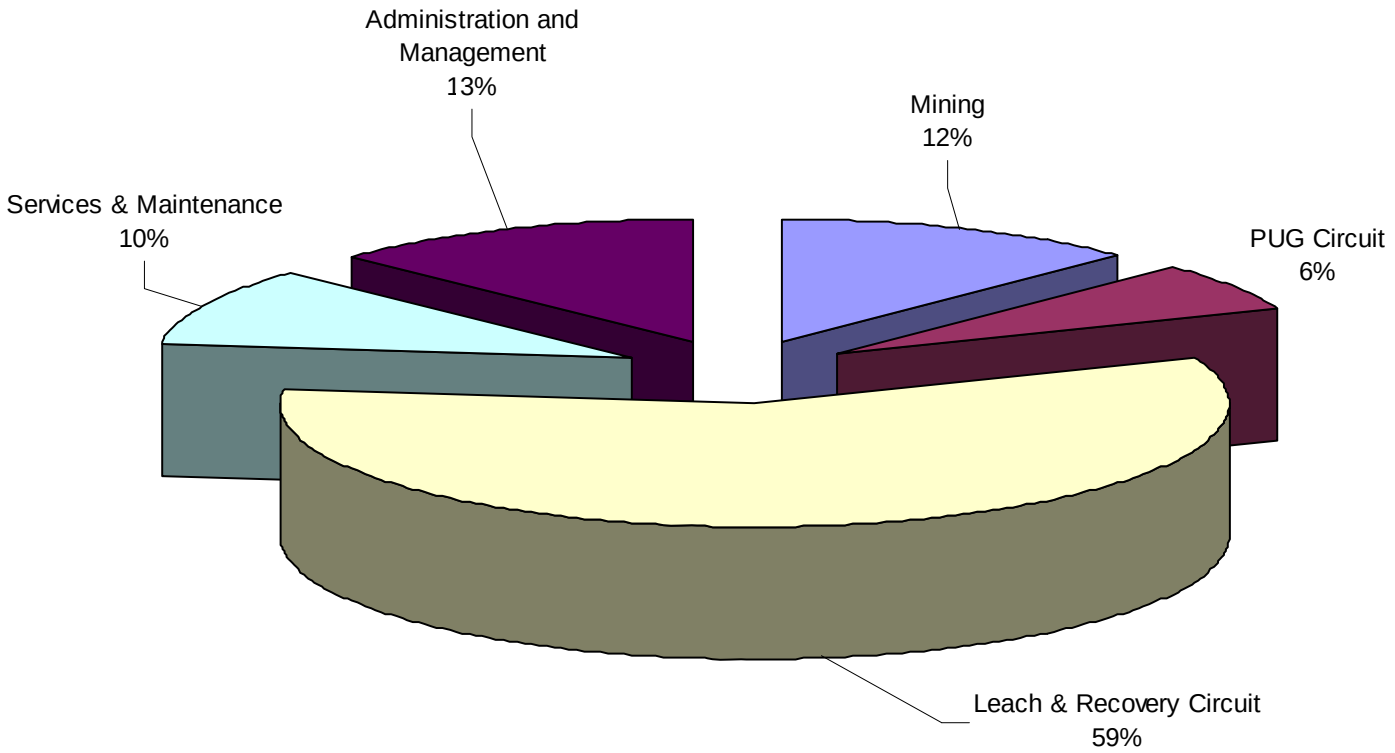


SRK Consulting Engineers and Scientists	**Nkamouna and Mada Deposits East Province of Cameroon, Africa**	**Process Plant Layout**
SRK Job No.: 195100.020		
File Name: Figure18-16.docx		Date: 07/28/2010 Approved: bcs **Figure: 18-16**



SRK Consulting Engineers and Scientists	**Nkamouna and Mada Deposits East Province of Cameroon, Africa**	**Steady State Operating Cost Breakdown by Department**
SRK Job No.: 195100.020		
File Name: Figure18-17.docx		Date: 07/28/2010 Approved: bcs **Figure: 18-17**



SRK Consulting Engineers and Scientists	**Nkamouna and Mada Deposits East Province of Cameroon, Africa**	**Steady State Operating Cost Breakdown by Plant Circuit**
SRK Job No.: 195100.020		
File Name: Figure18-18.docx		Date: 07/28/2010 Approved: bcs **Figure: 18-18**

19 Interpretation and Conclusions (Item 21)

19.1 Tailings Storage Facilities

The PUG and CCD tailing storage facilities have been designed to a feasibility study-level. The PUG and CCD TSFs have the capacity to store approximately 53.1 Mt (million dry tonnes) and 15.4 Mt of tailing at final buildout, respectively. The Glauber's Salt storage facilities, namely the GS Pad and alternative GSF, have been designed to a conceptual level. The GS Pad has the capacity to store four years of Glauber's Salt production. If the GS Pad is converted into the alternative GSF, the combined GS Pad and alternative GSF have the capacity to store Glauber's Salt production over the mine life (23.3 years, totaling 4.8 Mt). Further project phases will include detail design of the aforementioned project structures. Section 20 includes recommendations, threats, and opportunities associated with the tailing and Glauber's Salt management systems and the related storage facility structures, and includes additional studies to advance these from the current design level.

19.2 Geology and Resource Estimation Conclusions

SRK has reviewed the sample preparation and analytic methods utilized during the various drilling and sampling campaigns conducted at the project, and is of the opinion that the results are generally of high quality and suitable for use in resource estimation. SRK has also conducted a thorough audit of the 2009 resource model that forms the basis for current Mineral Resources, and is of the opinion that the model has been constructed using industry accepted methods, and is suitable as a basis for a Feasibility level of study.

19.3 Mining Conclusions

SRK is of the opinion that the mine plan and associated production schedule has been developed to a level of detail suitable for a feasibility study.

The mine operations plan fully exploits the up dip, free-dig nature and of the deposit with the need for progressive rehabilitation through the use of bulldozers for overburden relocation and excavator/ADT equipment fleet for ore liberation. The mine equipment has suitable size for load and haul operations and expected production rates.

The purchase of the mine production fleet for pre-production construction activities allow the capital cost of tailings and plant site construction to be reduced. The synergy between these activities combined with the expected mine life and size of equipment (low maintenance cost) confirm owner operations will reduce overall mine operations cost.

SRK has estimated the unit mining cost to be US$0.82/t (including rehandle) of material moved. This estimate includes mine administration, labor and consumable cost combined with fleet productivity and performance. The overall tonnage includes low cost dozer operations and rehandle of stockpile material.

For pre-production activities, SRK has estimated the average cost of bulk earthwork, compaction and grading to average US$2.04/$m^3$ of material moved.

With the effective implementation of the mitigation measures and monitoring programs defined in the ESIA and with periodic updates to the ESAP and selected component plans, GeoCam can prevent or minimize its Project impacts.

19.4 Metallurgical

Elemental Deportment

Despite the limitations of this elemental deportment review, interpretation of the analytical data from the pilot testwork campaigns has enabled an estimate to be made of the likely composition of the various products (MSP, MnCP1, MnCP2, and Glauber's salt) and of the tailings residue. In addition the composition of the final effluent stream for environmental discharge has also been estimated.

Based on interpretation of the available data, obtained from leaching and impurity removal testwork of a representative concentrate sample, there appears to be no significant elemental accumulation within the Nkamouna circuit that could adversely affect product quality, and the quality of the final effluent stream.

The SysCAD mass and energy balance, as used for the design of the Nkamouna processing facility, has been adapted to assist with the elemental deportment evaluation and supports the conclusion that there appears to be insignificant build-up of elements within the circuit.

Test Work

Extensive small and large scale testwork has been conducted at reputable testwork laboratories over the duration of the project to date, culminating in the completion of multiple pilot plant testwork programs to demonstrate the operation of the selected process flowsheet and provide reagent consumption data for process design requirements. The results obtained from the pilot plant testwork programs demonstrate the robustness of the process flowsheet selected.

The pilot plant testwork programs have been conducted by competent and internationally renown laboratories to a standard suitable for incorporation into the level of Feasibility Study conducted to determine the process design, as well as capital and operating costs presented in this report.

19.5 Environmental

The ESA identifies the specific actions required to assure that the health, safety, social, and environmental program performances are consistent with international mining best practice.

The ESIA has identified several potential environmental and social impacts for which the mitigation measures may be broadly categorized as; (1) reclamation of disturbed land to forest/wildlife habitat, (2) protecting and enhancing biodiversity, (3) protecting air quality, (4) ensuring adequate water supplies, and (5) economic diversification and sustainable community development. The ESAP has defined the specific actions that must occur to implement these mitigation measures and who is responsible for their implementation. It also defines specific monitoring programs aimed at documenting the implementation and adequacy of control systems and mitigation measures and the reporting that is required to assure transparency.

With the effective implementation of the mitigation measures and monitoring programs defined herein and with periodic updates to the ESAP and selected component plans, GeoCam can prevent or minimize its Project impacts.

Feasibility Study

SRK is of the opinion that an appropriate level of geological modeling, mine planning, metallurgical test work, metallurgical modeling, infrastructure design, tailings design,

environmental planning, cost modeling and economic analysis to support a feasibility level study and associated resource and reserve statement. The accumulated data and information contributed by Lycopodium, Knight Piésold, Geovic and SRK summarized in this report met, in the opinion of SRK, the requirements for a Feasibility level and further development of the project.

20 Recommendations (Items 22)

20.1 Sampling

Specific SRK recommendations relating to geological assay sampling are listed below:

- Discontinue the use of drillhole and interval for sample identification (currently being addressed by GeoCam by the incorporation of Fusion software into the QA/QC program);
- Discontinue placing the duplicate in sequence after the original sample (currently being addressed by GeoCam by the incorporation of Fusion software into the QA/QC program);
- When creating a standard for use, have the standard analyzed by more than one laboratory or have site specific standards created by a third party such as Geostats Pty Ltd or Rocklabs;
- Submit a percentage of pulp splits to a second laboratory for analytical crosscheck (currently being addressed by GeoCam by submitting pulps from previous programs to ALS Chemex);
- Continually monitor QA/QC during exploration programs;
- coarse blanks be added to the on-site sample preparation program to monitor potential cross contamination during on-site preparation;
- GeoCam correct / address the missing assay certificate issues and correct the database for future resource estimates; and
- Rerunning some check samples at ALS Chemex using an ore specific method such as ME-OG62 to see if this changes the results.

20.2 Grade Estimation

Based on the SRK audit of the resource estimation process and resource classification procedures, the following activities are recommended:

- Given the current resource classification parameters and the highly nuggetty distribution of grade in the breccia units, SRK recommends that the next model iteration attempt to subdivide the various breccia units that were combined for the 2009 resource update;
- In order to convert the current resource to higher confidence levels, SRK recommends infill drilling to a spacing of 25 m x 25 m for the following year mining and drilling the subsequent next three years planned production at a nominal spacing of 50 m x 50 m. This will allow adequate and timely definition of overburden thickness and depth to bedrock, as well as provide additional assay data for use in short range model construction to allow for better a better estimate of tonnage and grade in the short-medium term production schedule; and
- The current grade estimation methodology, while scientifically defendable, is a reasonably complex and time consuming process that is likely not suitable once the project is in production. SRK is of the opinion that the process can be simplified, as well as designed to cater more directly to the needs of mine production, including detailed mine design and mine reconciliation.

20.3 Mining Recommendations

- SRK recommends that the geological block model be redesigned to incorporate lithological interpretations of breccia and ferralite. In conjunction with Tetra modeling (lithologically controlled modification of estimation search ellipse) for better grade estimation, a new projected mine design should be constructed to more accurately determine overburden, interburden and ore volumes. A requirement for the new model will be infill drilling to within 25 m;
- SRK recommends a full geotechnical and resource evaluation of Serpentinite quarry rock and associated gravel and/or aggregate sources in the region to be quantified and evaluated. Mine plans, production schedules and crusher requirements need to be assessed to ensure supply of construction materials and road course during pre-production; and
- SRK is of the opinion that the rehandle stockpiles are unlikely to be built during operations. However, this is only likely if a regional exploration program targets high grade ore intercepts from both Mada and other satellite deposits. SRK recommends further ore delineation of high grade be carried out as soon as mine operations begin so the size of the required stockpiles can be minimized. This would reduce the quantity of lower grade bins being stockpiled and would be wasted at the mine face if increased reserves quantums are brought into the plan.

20.4 Metallurgical Testwork

- Lycopodium strongly recommends that a PUG pilot plant test be run on a representative sample of Nkamouna ore (for the first nine years of operation) to provide final design and operating data for the dewatering cyclones and hydrosizer equipment, to confirm the required attrition energy input, and to evaluate the beneficial impact of the SG differential on the size separation step. This work, to support detailed engineering design, should be completely scoped and agreed in advance of commencing the work between the client and the selected EPC engineering contractor for the Project. This work should be over-seen by a suitable engineering representative of the client and/or engineering contractor to ensure that it is undertaken to the required standards of quality and completeness.

20.5 Additional Studies – TSF, Glauber's Salt, Edjé River Water Supply, Plant Site

Work to be performed in support of the final design of the tailing and Glauber's Salt storage facilities, the plant site, and evaluation of water supply from the Edjé River would include, but not be limited to, the following:

- Final design-level subsurface site investigations in select areas of the TSF, Glauber's Salt storage facility, and Plant site;
- Procurement of baseline Edjé River flow quantity measurements;
- Procurement of updated climate data and update the climate analysis report;
- Tailing waste characterization of tailing produced under field conditions, particularly to support design values for discharge of waters to the environment;

- Glauber's Salt waste characterization;
- Design of Glauber's Salt bagging and bag handling/storage systems for design of the GS Pad operations;
- Finalization of the embankment sections including foundation excavation for use in final designs;
- Final design of other project features such as the tailing distribution system, water reclaim systems, and excess water discharge systems. The latter includes discharges from the PUG TSF, and controlled discharge system for waters from the GS Plant, via the buffer pond, into the Edjé River, including design features for the mixing zone;
- Acquisition of land related to excess water discharge systems and the mixing zone;
- Investigate the proposed PUG TSF closure outlet channel;
- Conduct a dambreak analysis in potentially impacted drainages;
- Preparation of technical specifications and Construction Quality Assurance and Quality Control (CQA/QC) plans for the TSF and Glauber's Salt storage facilities;
- Preparation of a TSF instrumentation and monitoring program;
- Preparation of an Emergency Action Plan (EAP);
- Preparation of an operations manual for operation of the tailing delivery system; and
- Preparation of an operations manual for operation of the water reclaim and water discharge systems.

The final design phase will include preparation of design drawings in sufficient detail for use during construction and an updated quantity and cost estimate. Services during construction would include resident engineering, home office support, and CQA/QC services.

20.6 Threat Assessment

The following paragraphs summarize the threats identified by the Project team to date.

Political Threats associated with the Project are generally believed to be of low threat primarily because of the political stability of the country and the strong desire of the current establishment to foster the development of the mining and minerals processing industry for diversification of the country's economy. The open and accessible nature of the political system at all levels ensures that open dialogue can occur between GeoCam personnel and government officials enabling the interests of this emerging mining sector to be presented in such a way as to reduce the likelihood of policies being developed that would restrict its growth.

Financial / marketing threats are centered primarily on the requirements for confirmation of the sale price of the products against a yet to be fully defined product specification. Depending upon its ultimate use the presence of minor impurities can be a significant factor in the value of the product and the lack of a fully transparent market such as the London Metal Exchange where the spot prices are in the public domain the current market for both MSP and manganese carbonate is difficult to confirm without negotiated Letters of intent or similar commercial arrangements being in place. Delays in Project schedule invariably result in financial impacts and the areas identified in Section 21 need to be closely monitored.

The technological threat profile for this Project is not believed to be of concern based on current information. The threat should not, however, be dismissed until a review of the intended Project product specifications have been confirmed. Further chemical assaying of the ore may be required to be undertaken if tight tolerances on minor contaminants are demanded by the potential customers as these can generate significant technical issues and specialist contaminant stripping circuits.

The supply of Serpentinite is based on limited data. Geotechnical and resource drilling during the pre-production period will sure up the supply of aggregates for site construction activities. No blasting quotations or local contractor has been approached. Problems can be alleviated through sourcing local aggregate suppliers, research into local gravel resources and use of ferricrete.

Reserves have not been diluted (due to lack of lithological modeling and density of drilling) and assume a SMU of 10 m x 10 m x 1 m. The effect of dilution will be reliant on grade control and operator practices employed by the mine. A large cost allocation for grade control and re-modeling in 2011 has been applied to the economic model.

There is a 60 Ha limit on mine disturbance at any one time. A detailed production schedule (to be completed in 2011) will identify if the 60 Ha limit is breached. Current estimates are based on life of mine detail. Changing the maximum disturbance schedule per period will alleviate the problem.

There is a threat regarding the supply of ore to the PUG / leach and recovery plant at a constant grade of material which will lead to excessive fluctuations in reagents consumptions and product variability / quality issues and variability in operating costs. In this regards significant surge stockpile capacity has been allocated for the ROM stockpile and intermediate concentrate stockpiles to handle the nuggetty nature of the ore. With adequate sampling and reconciliation processes in place, the plant will be able to be fed with a consistent grade.

The current lack of a developed mining / minerals processing industry in the Cameroon exacerbates the operational and human resources threats associated with the inability to recruit suitably experienced personnel for the Project. It is envisaged that it will be necessary to utilize the services of a higher percentage of non Cameroon nationals in the initial years of operation until suitable training can be undertaken and experience built up. The high levels of general education in the country coupled with the likelihood of experienced Cameroon nationals returning from other countries as jobs become available will reduce the impact of this issue.

The environmental / geotechnical threats related to the tailing and Glauber's Salt storage facilities are the lack of sufficient information about the Glauber's Salt and its behavior in the storage facilities and environment, and the threat of insufficient water availability in the Edjé River for abstraction or to receive Glauber's Salt plant effluent into the river. This includes limited CCD tailings and Glauber's Salt waste characterization information to-date. It includes, as related to sodium sulfate/Glauber's Salt inclusion in the waste streams, for example, potential impacts on achieved density of the CCD tailings; the potential for plugging the underdrain systems in the CCD TSF, GS Pads, or alternative GSF; the potential for plugging the water recovery systems in the CCD TSF or alternative GSF; and dust development on the CCD TSF or alternative GSF during the dry season. This will need to be evaluated in a further design phase. Additional water storage in the PUG TSF will be provided to limit the threat of insufficient Edjé

River water availability, however, additional Edjé River baseline flow data and updated Project climate data will be required to measure and predict Edjé River water availability.

Specific actions have been identified in the ESIA and ESAP that are required so that the health, safety, social, and environmental program performances are in line with international mining best practice.

20.7 Opportunities

The primary opportunities are similarly centered on the potential to reduce capital and operating costs.

The PUG circuit as currently nominated is based on the circuit prepared by previous studies. Recent test work indicates that it may be possible to simplify this circuit and reduce the capital cost and operating costs for the circuit without adversely affecting its operational performance;

The current equipment list and layouts are based on the use of very high quality sampling techniques and although sample quality may suffer to some extent if alternate methods were employed a substantial capital savings may result in this less rigorous approach without appreciable threat to product quality assurance;

The layout of the current stockpile area is based on the design criteria which allows for blending, a radial stacker sampling and significant live and dead stockpiles. There is potential to remove capital from this area with minimal loss of operational flexibility but which may add some additional operational costs under certain operational scenarios;

Consider the economic implication of the replacement of concrete bunded areas under thickeners and its replacement with HDPE lined containment bunds and significantly smaller concrete bunds;

GeoCam has been discussing with the local and federal authorities in country over positive actions by the government to facilitate execution of the Project. This would significantly reduce the Project capital cost and improve the economics;

Review the potential to remove the manganese carbonate precipitation 2 circuit. This may have an adverse impact on product quality and potentially threaten the percentage of LME sale price achievable but may be economic;

SRK is of the opinion that the large rehandle stockpiles are unlikely to be built to their full size during operations. However, this is only likely if a regional exploration program targets high grade ore intercepts from both Mada and other satellite deposits;

With the commencement of mining at Mada, stripping ratios increase as Nkamouna resource is exhausted. This, combined with extended haul lengths, mean the number of primary earth moving equipment increases significantly. By sacrificing reserves and shortening the mine life, the sustaining capital and mine unit cost can be reduced and ROM stockpile sizes reduced. In practice, there will be eight years to find alternate low strip ratio high grade deposits within the mining exploration lease;

A conservative overall 54% application of available shift hours has been applied to the unit mining cost calculation and production capacity. This was done to account for elements such as weather, operator efficiency, and maintenance and equipment performance;

The environmental / geotechnical opportunities related to the PUG TSF include the potential for increased PUG tailing density even with some air drying. This provides an impetus to control tailing deposition and the decant water pond, and to work toward developing a tailing beach followed by the installation of a thin layer, rotational tailing deposition system. This potentially benefits later stages of the PUG TSF to increase tailing densities, thus increase storage capacity, and to accelerate closure and reclamation activities. The use of a more effective flocculation mix system to enhance tailing settlement and consolidation characteristics, and speed up water release from the tailing can lead to improved tailing density. Additional water storage is provided in the PUG TSF to limit the threat of insufficient water availability in the Edjé River (See Threats, Section 2.21.1). This could negate the opportunity for increased PUG tailing density discussed above;

The opportunity exists to provide additional PUG tailing capacity by mining in the PUG TSF basin; encroachment into zones of the dam foundations needed for stability is to be avoided, and can be further evaluated in further design phases;

PUG TSF closure channel modifications may result in a simplified closure channel relative to that included in the feasibility study. Site conditions are as yet unknown along the proposed channel location;

The potential use of 60-mil vs. 80-mil HDPE geomembrane in the geocomposite liner systems of the CCD TSF and GS Pads. The reduction in thickness could provide cost savings, and can be considered in further design; and

The sale of the Glauber's salt and removal off-site would eliminate the need for the GS Pads and could also realize a potential income.

21 References (Item 23)

Evans, D.J.I., Shoemaker, R.S., and Veltman, H (eds.), (1979), *International Laterite Symposium: Society of Mining Engineers,* 688 pages.

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22 Glossary

22.1 Mineral Resources

The mineral resources and mineral reserves have been classified according to the "CIM Standards on Mineral Resources and Reserves: Definitions and Guidelines" (November 27, 2010). Accordingly, the Resources have been classified as Measured, Indicated or Inferred, the Reserves have been classified as Proven, and Probable based on the Measured and Indicated Resources as defined below.

A Mineral Resource is a concentration or occurrence of natural, solid, inorganic or fossilized organic material in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge.

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

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

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

22.2 Mineral Reserves

A Mineral Reserve is the economically mineable part of a Measured or Indicated Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified. A Mineral Reserve includes diluting materials and allowances for losses that may occur when the material is mined.

A 'Probable Mineral Reserve' is the economically mineable part of an Indicated, and in some circumstances a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical,

economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified.

A 'Proven Mineral Reserve' is the economically mineable part of a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction is justified.

22.3 Glossary

The following general mining terms may be used in this report.

Table 22.3.1: Glossary

Term	Definition
Assay:	The chemical analysis of mineral samples to determine the metal content.
Capital Expenditure:	All other expenditures not classified as operating costs.
Composite:	Combining more than one sample result to give an average result over a larger distance.
Concentrate:	A metal-rich product resulting from a mineral enrichment process such as gravity concentration or flotation, in which most of the desired mineral has been separated from the waste material in the ore.
Crushing:	Initial process of reducing ore particle size to render it more amenable for further processing.
Cut-off Grade (CoG):	The grade of mineralized rock, which determines as to whether or not it is economic to recover its gold content by further concentration.
Dilution:	Waste, which is unavoidably mined with ore.
Dip:	Angle of inclination of a geological feature/rock from the horizontal.
Fault:	The surface of a fracture along which movement has occurred.
Footwall:	The underlying side of an orebody or stope.
Gangue:	Non-valuable components of the ore.
Grade:	The measure of concentration of gold within mineralized rock.
Hangingwall:	The overlying side of an orebody or slope.
Haulage:	A horizontal underground excavation which is used to transport mined ore.
Hydrocyclone:	A process whereby material is graded according to size by exploiting centrifugal forces of particulate materials.
Igneous:	Primary crystalline rock formed by the solidification of magma.
Kriging:	An interpolation method of assigning values from samples to blocks that minimizes the estimation error.
Level:	Horizontal tunnel the primary purpose is the transportation of personnel and materials.
Lithological:	Geological description pertaining to different rock types.
LoM Plans:	Life-of-Mine plans.
LRP:	Long Range Plan.
Material Properties:	Mine properties.
Milling:	A general term used to describe the process in which the ore is crushed and ground and subjected to physical or chemical treatment to extract the valuable metals to a concentrate or finished product.
Mineral/Mining Lease:	A lease area for which mineral rights are held.
Mining Assets:	The Material Properties and Significant Exploration Properties.
Ongoing Capital:	Capital estimates of a routine nature, which is necessary for sustaining operations.
Ore Reserve:	See Mineral Reserve.
Pillar:	Rock left behind to help support the excavations in an underground mine.
RoM:	Run-of-Mine.
Sedimentary:	Pertaining to rocks formed by the accumulation of sediments, formed by the erosion of other rocks.
Shaft:	An opening cut downwards from the surface for transporting personnel, equipment,

Term	Definition
	supplies, ore and waste.
Sill:	A thin, tabular, horizontal to sub-horizontal body of igneous rock formed by the injection of magma into planar zones of weakness.
Smelting:	A high temperature pyrometallurgical operation conducted in a furnace, in which the valuable metal is collected to a molten matte or doré phase and separated from the gangue components that accumulate in a less dense molten slag phase.
Stope:	Underground void created by mining.
Stratigraphy:	The study of stratified rocks in terms of time and space.
Strike:	Direction of line formed by the intersection of strata surfaces with the horizontal plane, always perpendicular to the dip direction.
Sulfide:	A sulfur bearing mineral.
Tailings:	Finely ground waste rock from which valuable minerals or metals have been extracted.
Thickening:	The process of concentrating solid particles in suspension.
Total Expenditure:	All expenditures including those of an operating and capital nature.
Variogram:	A statistical representation of the characteristics (usually grade).

22.4 Abbreviations

The following abbreviations may be used in this report.

Table 22.4.1: Abbreviations

Abbreviation	Unit or Term
AA	atomic absorption
Ag	silver
Au	gold
AuEq	gold equivalent grade
°C	degrees Centigrade
CCD	counter-current decantation
CoG	cut-off grade
cm	centimeter
cm^2	square centimeter
cm^3	cubic centimeter
cfm	cubic feet per minute
°	degree (degrees)
ft	foot (feet)
ft^2	square foot (feet)
ft^3	cubic foot (feet)
g	gram
g/L	gram per liter
g/t	grams per tonne
ha	hectares
HDPE	Height Density Polyethylene
hp	horsepower
HTW	horizontal true width
ICP	induced couple plasma
ID2	inverse-distance squared
ID3	inverse-distance cubed
IFC	International Finance Corporation
ILS	Intermediate Leach Solution
kg	kilograms
km	kilometer
km^2	square kilometer
koz	thousand troy ounce

Abbreviation	Unit or Term
kt	thousand tonnes
kt/d	thousand tonnes per day
kt/y	thousand tonnes per year
kV	kilovolt
kW	kilowatt
kWh	kilowatt-hour
kWh/t	kilowatt-hour per metric tonne
L	liter
L/sec	liters per second
L/sec/m	liters per second per meter
lb	pound
LHD	Long-Haul Dump truck
LLDDP	Linear Low Density Polyethylene Plastic
LOI	Loss On Ignition
LoM	Life-of-Mine
m	meter
m^2	square meter
m^3	cubic meter
m^3/h	Cubic meter per hour
Ma	million years
masl	meters above sea level
MARN	Ministry of the Environment and Natural Resources
MDA	Mine Development Associates
mg/L	milligrams/liter
mm	millimeter
mm^2	square millimeter
mm^3	cubic millimeter
Mm^3	million cubic millimeters
MME	Mine & Mill Engineering
Moz	million troy ounces
Mt	million tonnes
MTW	measured true width
MW	million watts
NGO	non-governmental organization
NI 43-101	Canadian National Instrument 43-101
OSC	Ontario Securities Commission
oz	troy ounce
%	percent
PLC	Programmable Logic Controller
PLS	Pregnant Leach Solution
PMF	probable maximum flood
ppb	parts per billion
ppm	parts per million
QA/QC	Quality Assurance/Quality Control
RC	rotary circulation drilling
RoM	Run-of-Mine
RQD	Rock Quality Description
SEC	U.S. Securities & Exchange Commission
sec	second
SG	specific gravity
SPT	standard penetration testing
st	short ton (2,000 pounds)
t	tonne (metric ton) (2,204.6 pounds)
t/h	tonnes per hour
t/m^3	tonnes per cubic meter

Abbreviation	Unit or Term
t/d	tonnes per day
t/y	tonnes per year
TSF	tailings storage facility
TSP	total suspended particulates
µm	micron or microns
V	volts
VFD	variable frequency drive
W	watt
XRD	x-ray diffraction
y	year

Appendix A

Certificate of Author

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CERTIFICATE OF AUTHOR

I, Jeffrey Volk, CPG, FAusIMM, MSc, do hereby certify that:

1. I am Principal Resource Geologist with:

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2. I graduated with a Master of Science degree in Structural Geology from the Washington State University in 1986. In addition, I have obtained a Bachelor of Arts degree in geology from the University of Vermont in 1983.

3. I am a fellow of the Society of Economic Geologists and a Certified Professional Geologist and member of the American Institute of Professional Geologists (AIPG). I am also a fellow and member of the Australian Institute of Mining and Metallurgy (FAusIMM).

4. I have worked as a geologist for a total of 25 years since my graduation from university.

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

6. I am responsible for Sections 4,5,6,7,8,9,10,11,12,13 of the Technical Report titled "NI 43-101 Technical Report, Geovic Mining Corp., Nkamouna and Mada Deposits, East Province of Cameroon, Africa" and dated June 2, 2011 (the "Technical Report"). I visited the Nkamouna and Mada property on November 10, 2009 for 1 day.

7. I have not had prior involvement with the property that is the subject of this Technical Report.

8. I am independent of the issuer applying all of the tests in section 1.4 of National Instrument 43-101.

9. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

10. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

11. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 2nd Day of June, 2011.

"Signed" *"Sealed"*

_____

Jeffrey Volk, CPG, FAusIMM, MSc CPG#10835

Lycopodium Minerals Pty Ltd
ABN: 34 055 880 209

Level 5, 1 Adelaide Terrace
East Perth, Western Australia 6004
Australia

PO Box 6832
East Perth, Western Australia 6892
Australia

T: +61 (0) 8 6210 5222
www.lycopodium.com.au

CERTIFICATE OF QUALIFIED PERSON

BRETT MALCOLM CROSSLEY

I, Brett Malcolm Crossley do hereby certify that:

1. I am a Principal Metallurgist with Lycopodium Minerals Pty Ltd. My office address is Level 5, 1 Adelaide Terrace, East Perth, Western Australia 6004.

2. I am a graduate of the University of Western Australia in 1985 with a Bachelor of Applied Science degree in Chemistry.

3. I am a member of the Australasian Institute of Mining and Metallurgy (AusIMM), membership number 108328.

4. I have worked as an industrial chemist, metallurgist and process engineer for a total of 26 years since my graduation.

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

6. I am responsible for mineral processing and metallurgy and the preparation of Section 14 of the Technical Report titled "NI 43-101 Technical Report, Geovic Mining Corp., Nkamouna and Mada Deposits, East Province of Cameroon, Africa" and dated June 02, 2011 (the "Technical Report"). I have not visited the Nkamouna and Mada property to date.

7. I have not had prior involvement with the property that is the subject of the Technical Report.

8. As of the date of the Certificate, to the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

9. I am independent of the Issuer applying all of the tests set out in Section 1.5 of NI 43-101.

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

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 2nd day of June, 2011

Original Signed by

Signed

B. M. Crossley

SRK Consulting (U.S.), Inc.
7175 West Jefferson Avenue, Suite 3000
Lakewood, CO
USA 80235

denver@srk.com
www.srk.com

Tel: 303.985.1333
Fax: 303.985.9947

CERTIFICATE of AUTHOR

I, Bret C. Swanson, BE (Mining), MAusIMM #112411 do hereby certify that:

1. I am Senior Mining Engineer of:

SRK Consulting (U.S.), Inc.
7175 W. Jefferson Ave, Suite 3000
Denver, CO, USA, 80235

2. I graduated with a degree in Bachelor of Engineering In Mining Engineering from the University of Wollongong in 1997.

3. I am a current member of the Australian Institute of Mining and Metallurgy, #112411.

4. I have worked as a Mining Engineer for a total of 14 since my graduation from university.

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

6. I am responsible for mineral reserves and the compilation and editing of all Sections of the Technical Report titled "NI 43-101 Technical Report, Geovic Mining Corp., Nkamouna and Mada Deposits, East Province of Cameroon, Africa" and dated June 2, 2011 (the "Technical Report"). I visited the Nkamouna and Mada property on November 10, 2009 and September 17, 2010.

7. I have not had prior involvement with the property that is the subject of the Technical Report.

8. As of the date of the certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

9. I am independent of the issuer applying all of the tests in Section 1.4 of National Instrument 43-101.

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

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 2nd Day of June, 2011.

"Signed"

_____

Bret C. Swanson, MAusIMM **MAusIMM #112411**

Appendix B

Mining Decree

RÉPUBLIQUE DU CAMEROUN — PAIX – TRAVAIL – PATRIE

DECRET N° 2003/077 DU 11 AVR. 2003
portant institution d'un permis d'exploitation valable pour cobalt, nickel et substances connexes.-

LE PRESIDENT DE LA REPUBLIQUE,

VU la Constitution ;
VU l'ordonnance n° 74/2 du 6 juillet 1974 fixant le régime domanial ;
VU la loi n°96/012 du 5 août 1996 portant loi cadre relative à la gestion de l'environnement ;
VU la loi n° 98/015 du 14 juillet 1998 relative aux établissements classés dangereux, insalubres ou incommodes ;
VU la loi n° 001 du 16 avril 2001 portant code minier ;
VU le décret n° 96/227 du 1er octobre 1996 portant organisation du Ministère des Mines, de l'Eau et de l'Energie ;
VU le décret n° 99/014/PR du 27 janvier 1999 portant institution d'un permis de recherches valable pour cobalt, nickel et substances connexes ;
VU le décret n° 2002/216 du 24 août 2002 portant réorganisation du Gouvernement ;
VU le décret n° 2002/648/PM du 26 mars 2002 fixant les modalités d'application de la loi n° 001 du 16 avril 2001 portant code minier ;

DECRETE :

ARTICLE 1er.- Il est accordé à la société GEOVIC CAMEROON S.A., BP 5839 Douala et dans les conditions prévues par les textes en vigueur, un permis d'exploitation valable pour cobalt, nickel et substances connexes.

ARTICLE 2.- (1) Le permis d'exploitation institué au profit de GEOVIC CAMEROON S.A. est constitué d'un seul bloc de forme polygonale dont les coordonnées géographiques sont les suivantes :

POINTS	COORDONNEES GEOGRAPHIQUES	
	EST	NORD
1	13°59'43,7''	3°15'00,3''
2	13°47'03,4''	3°14'49,6''
3	13°47'02,5''	3°29'57,7''
4	13°58'37,6''	3°29'56''
5	14°09'21,3''	3°22'46,6''

6	14°13'45,9''	3°22'46,8''
7	14°13'57,7''	2°53'46,1''
8	14°08'11,9''	2°53'40''
9	13°56'54,8''	2°49'19,6''
10	13°56'54,5''	2°42'41,9''
11	13°51'41,3''	2°42'24,9
12	13°51'05,7''	2°49'21,6''
13	13°56'55,3''	2°49'22,5''
14	14°08'12,4''	2°53'41,9''
15	14°08'10,1''	3°01'32,3''
16	14°05'25''	3°05'43,9''
17	14°05'22,3''	3°19'27,3''
18	13°59'43,1''	3°24'33,7

(2) La superficie concernée par le permis d'exploitation GEOVIC est réputée égale à 1250 km².

ARTICLE 3.- Le permis d'exploitation attribué à GEOVIC, inscrit sous le numéro 33 dans le registre spécial de la Direction des Mines et de la Géologie, dans la rubrique des titres miniers d'exploitation, est valable pour une période de vingt cinq (25) ans renouvelable.

ARTICLE 4.- Avant le démarrage des activités d'exploitation sur le permis d'exploitation n° 33, l'Administration mettra au préalable à la disposition de GEOVIC CAMEROON S.A. les terrains nécessaires à son activité sur la base d'un levé topographique réalisé par un géomètre assermenté commis par l'opérateur à cet effet.

ARTICLE 5.- (1) Au terme des quatre (4) premières années de validité du permis d'exploitation n° 33, l'opérateur tiendra à la République du Cameroun un plan de développement de tous les gisements mis en évidence à l'intérieur du périmètre d'exploitation.

(2) S'agissant des substances connexes au cobalt et au nickel l'opérateur soumettra à l'approbation du Ministre chargé des mines, un plan de développement et d'exploitation des substances concernées.

ARTICLE 6.- Durant la validité du permis d'exploitation n° 33, la société GEOVIC CAMEROON S.A. doit mettre à la disposition des populations riveraines les infrastructures sociales, sportives, éducatives et sanitaires pour favoriser leu épanouissement.

ARTICLE 7.- Pendant l'exercice de ses activités d'exploitation, la société GEOVI CAMEROON S.A. devra faire parvenir au Ministre chargé des mines un rappo d'activités semestriel et un rapport d'activités annuel.

ARTICLE 8.- (1) La société GEOVIC CAMEROON S.A. doit se conformer aux recommandations de l'étude d'impact sur l'environnement menée préalablement à l'exploitation et remettre en l'état initial les sites d'exploitation.

(2) Au démarrage des activités d'exploitation, l'opérateur ouvrira un compte de réhabilitation de l'environnement, domicilié dans un compte séquestre auprès d'une banque agréée de la place pour garantir la réhabilitation du site lors de sa fermeture en fin d'exploitation.

ARTICLE 9.- (1) Avant le démarrage des activités d'exploitation, l'opérateur assurera la mise en place de la caution devant permettre de couvrir les paiements dus en vertu du code minier.

(2) Le montant de ladite caution est égal à deux et demi pour cent (2,5%) de l'investissement total requis avant la première production commerciale.

ARTICLE 10.- Les rapports d'études et les résultats d'analyses issus des travaux d'exploitation constituent des secrets industriels. La société GEOVIC CAMEROON S.A. est tenue de les faire parvenir systématiquement au Ministre chargé des mines. Ceux-ci, propriété de la République du Cameroun, demeureront confidentiels pendant la durée de validité du permis d'exploitation.

ARTICLE 11.- Le présent décret sera enregistré et publié suivant la procédure d'urgence, puis inséré au Journal Officiel en français et en anglais./-

YAOUNDE, le 11 AVR. 2003

LE PRESIDENT DE LA REPUBLIQUE,

PAUL BIYA

REPUBLIC OF CAMEROON

PEACE - WORK- FATHERLAND

DECREE N° 20003/077 of 11 April 2003
TO INSTITUTE A MINING PERMIT VALID FOR COBALT, NICKEL ANDASSOCIATED SUBSTANCES

THE PRESIDENT OF THE REPUBLIC,

MINDFUL of the Constitution ;
MINDFUL of Ordinance N° 74/2 of July 1974 to regulate the land system ;
MINDFUL of Law N°96/012 of 5 August 1996 to set the framework law relating to the management of the environment ;
MINDFUL of Law N° 98/015 of 14 July 1998 relating to establishments considered as dangerous, unsanitary or uncomfortable ;
MINDFUL of Law N°001 of 16 April 2001 to lay down the Mining code ;
MINDFUL of Decree N° 96/227 of October 1996 to organise the Ministry of Mines, Water Resources and Energy ;
MINDFUL of Decree N°99/014/PR of 27 January 1999 to institute a research permit valid for cobalt, nickel and associated substances ;
MINDFUL of Decree N°20002/216 of 24 August 2002 to reorganise the Government ;
MINDFUL of Decree N°2002/648/PM of 26 March 2002 to implement law N°001 of 16 April 2002 to lay down the Mining code,

HEREBY DECREES AS FOLLOWS:

Article 1 : A mining permit valid for cobalt, nickel and related substances is granted to GEOVIC CAMEROON S.A., P.O. Box 5839 Douala, under the conditions set forth by the law in force.

Article 2 : 1) The mining permit granted to GEOVIC CAMEROON S.A. shall constitute a single polygonal block whose geographic references shall be as follows:

POINTS	GEOGRAPHIC REFERENCES	
	EAST	NORTH
1	13°59'43,7''	3°15'00,3''
2	13°47'03 ,4''	3°14'49,6''
3	13°47'02,5''	3°29'57,7''
4	13°58'37,6''	3°29'56''
5	14°09'21,3''	3°22'46,6''
6	14°13'45,9''	3°22'46,8''
7	14°13'57,7''	2°53'46,1''
8	14°08'11,9''	2°53'40''
9	13°56'54,8''	2°49'19,6''
10	13°56'54,5''	2°42'41,9''

11	13°51'41,3'	2°42'24,6''
12	13°51'05,7''	2°49'21,6''
13	13°56'55,3''	2°4922,5''
14	14°08'12,4''	2°53'41,9''
15	14°08'10,1''	3°01'32,3''
16	14°05'25''	~~3°01'32,3''~~ 3°05'43,9''
17	14°05'23,3''	3°19'27,3''
18	13°59'43,1''	3°24'33,7''

2) The surface area covered by the mining permit shall be equal to 1600 km^2. 1643.95 km^2

Article3 : The mining permit granted to GEOVIC, which is registered under No 33 in the special records of the Department of Mines and Geology, under the heading reserved for mining exploitation permits, shall be valid for a renewable period of twenty-five (25) years.

Article 4 : Prior to the launching of mining activities under mining permit No 33, Government shall provide GEOVIC CAMEROON S.A. with the land necessary for such activities on the basis of a topographic map drawn up by a recognised geometer designated by the operator to this effect.

Article 5 : 1)At the end of the first four (4) years of validity of mining permit No 33, the operator shall submit to the Republic of Cameroon a development plan for all deposits discovered within the mining area.

2)Regarding cobalt and nickel associated substances, the operator shall submit a development and mining plan of such substances to the Minister in charge of Mines for approval.

Article 6 : During the validity of mining permit No 33, GEOVIC CAMEROON S.A. shall provide the neighbouring population with social, sports, education and health infrastructure to promote their well-being.

Article 7 : In the course of its mining activities, GEOVIC CAMEROON S.A. shall forward a semi-annual and an annual activity report to the Minister in charge of Mines.

Article 8 : 1) GEOVIC CAMEROON S.A. shall comply with the recommendations of the environmental impact study carried out prior to the mining and shall rehabilitate exploitation sites.

2) Upon launching its mining activities, the operator shall open an environment rehabilitation account which shall be subject to sequestration with a chartered bank in the country to ensure that the site is rehabilitated at the end of the mining.

Article 9 : 1) Prior to the launching of mining activities, the operator shall pay a caution fee to cover the expenditure due by virtue of the Mining Code.

2) The amount of such caution fee shall represent two point five per cent (2.5 %) of the total investment required before the first commercial production.

Article 10 : The study reports and analysis results from the mining activities shall be industrial secrets. GEOVIC CAMEROON S.A. shall be bound to forward them systematically to the Minister in charge of Mines. Such reports and results shall be the property of the Republic of Cameroon and remain confidential during the validity of the mining permit.

Article 11 : This Decree shall be registered and published in the Official Gazette in English and French according to the emergency procedure.

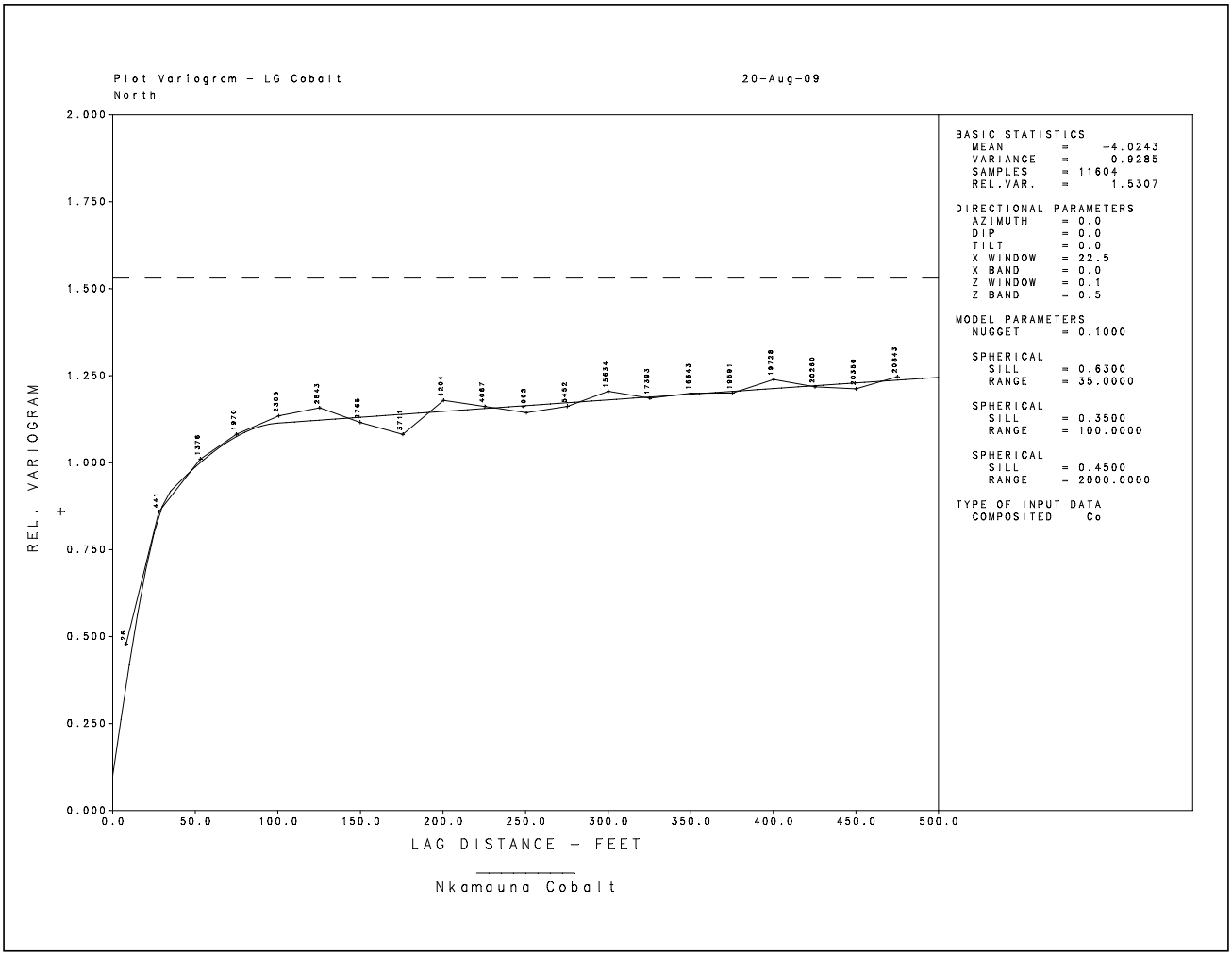
Yaounde, 11 April 2003

PAUL BIYA

PRESIDENT OF THE REPUBLIC

Appendix C

Variograms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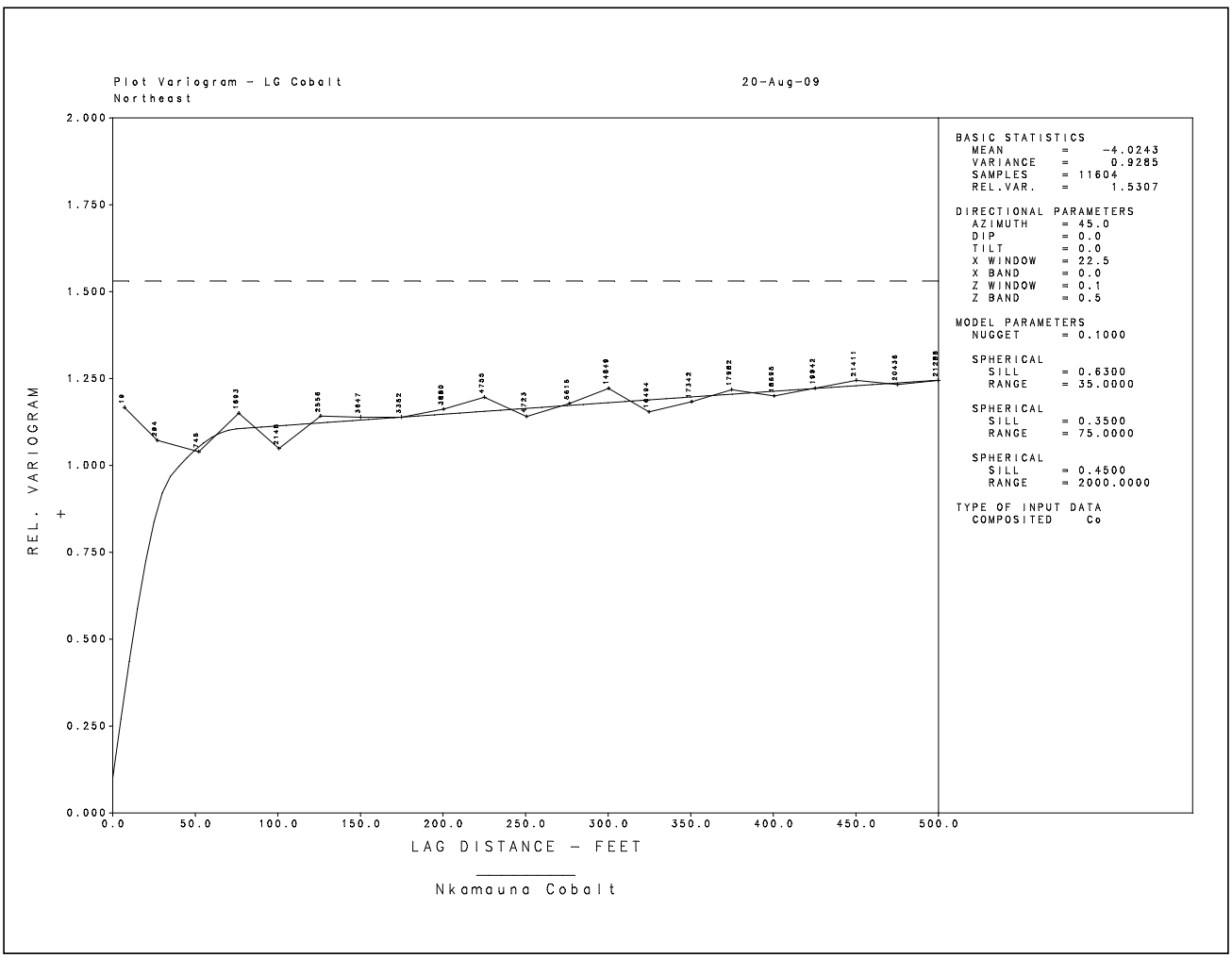
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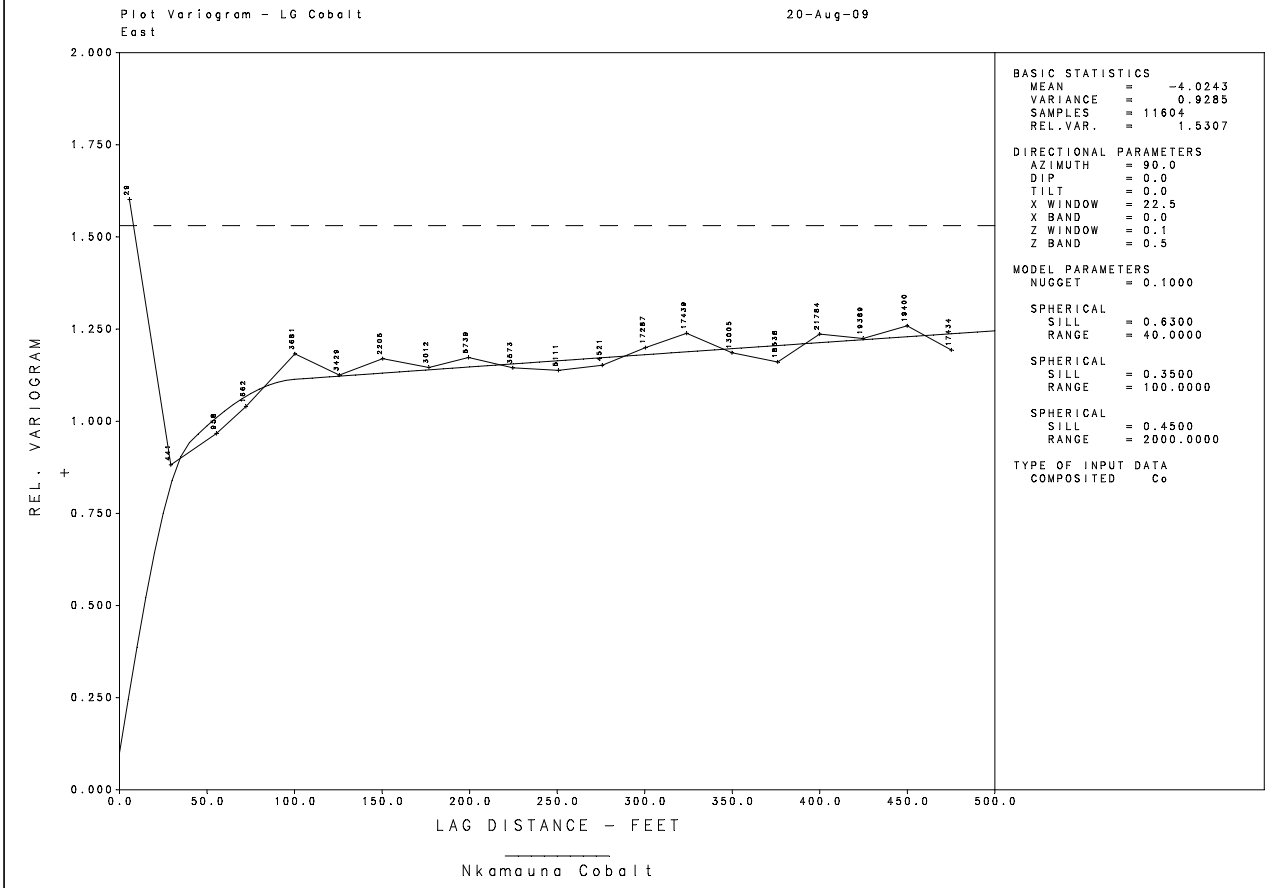
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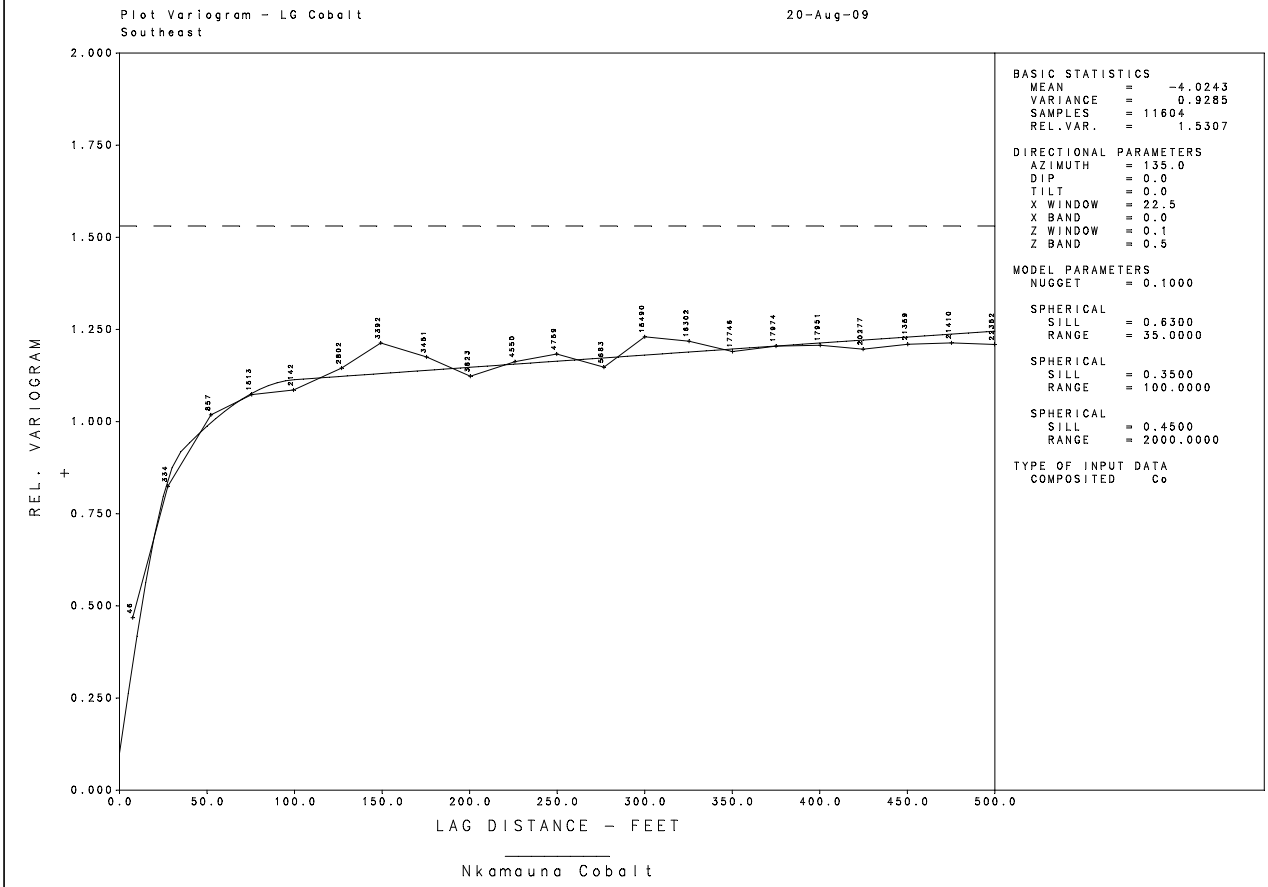
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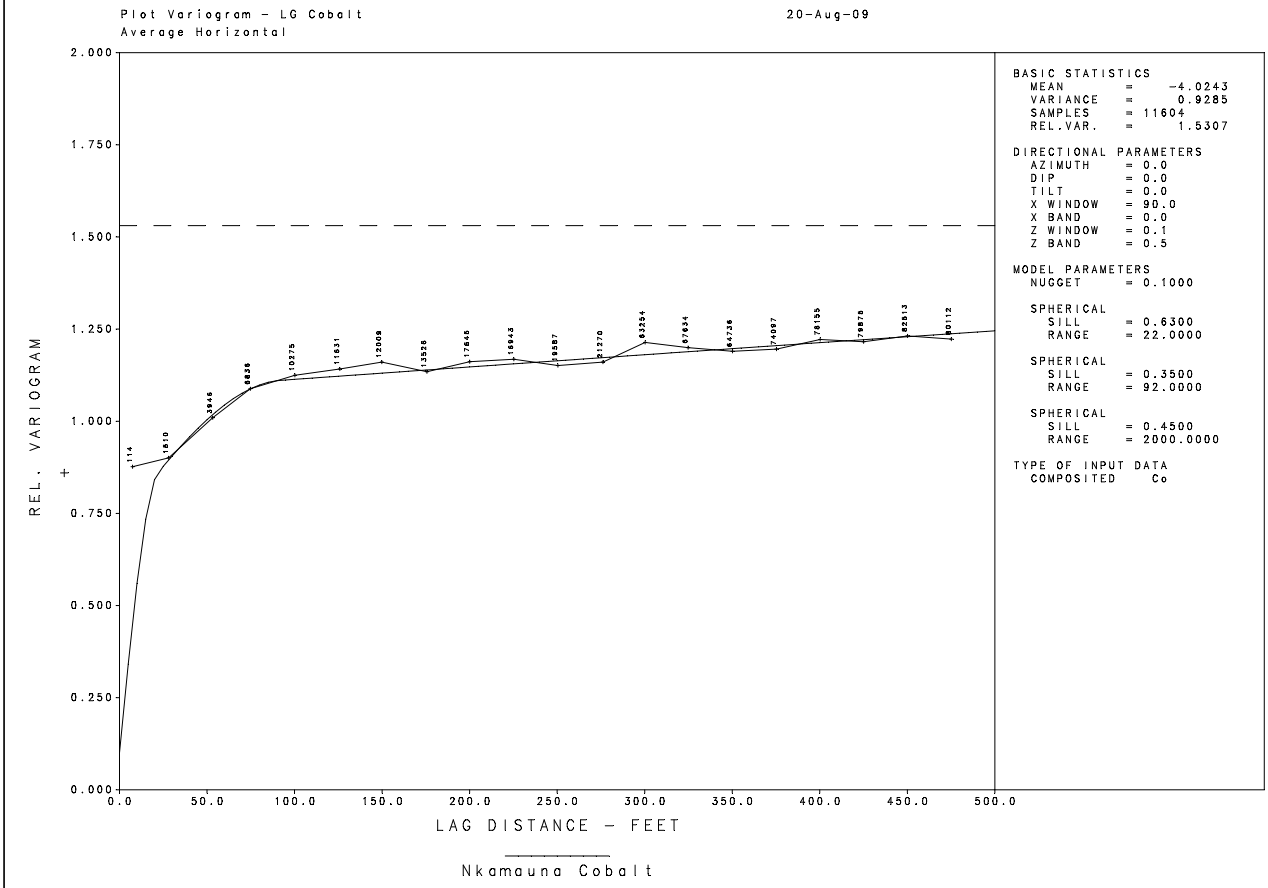
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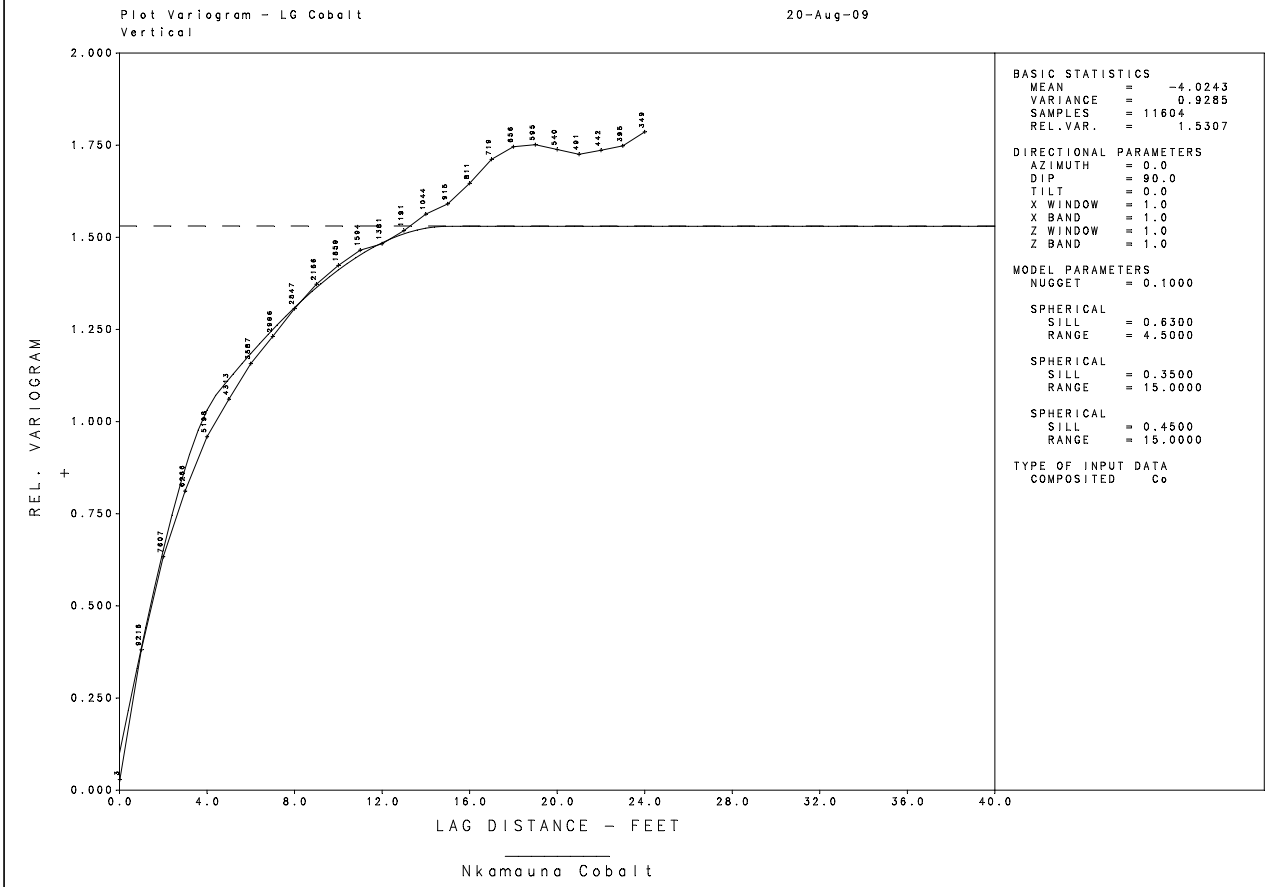
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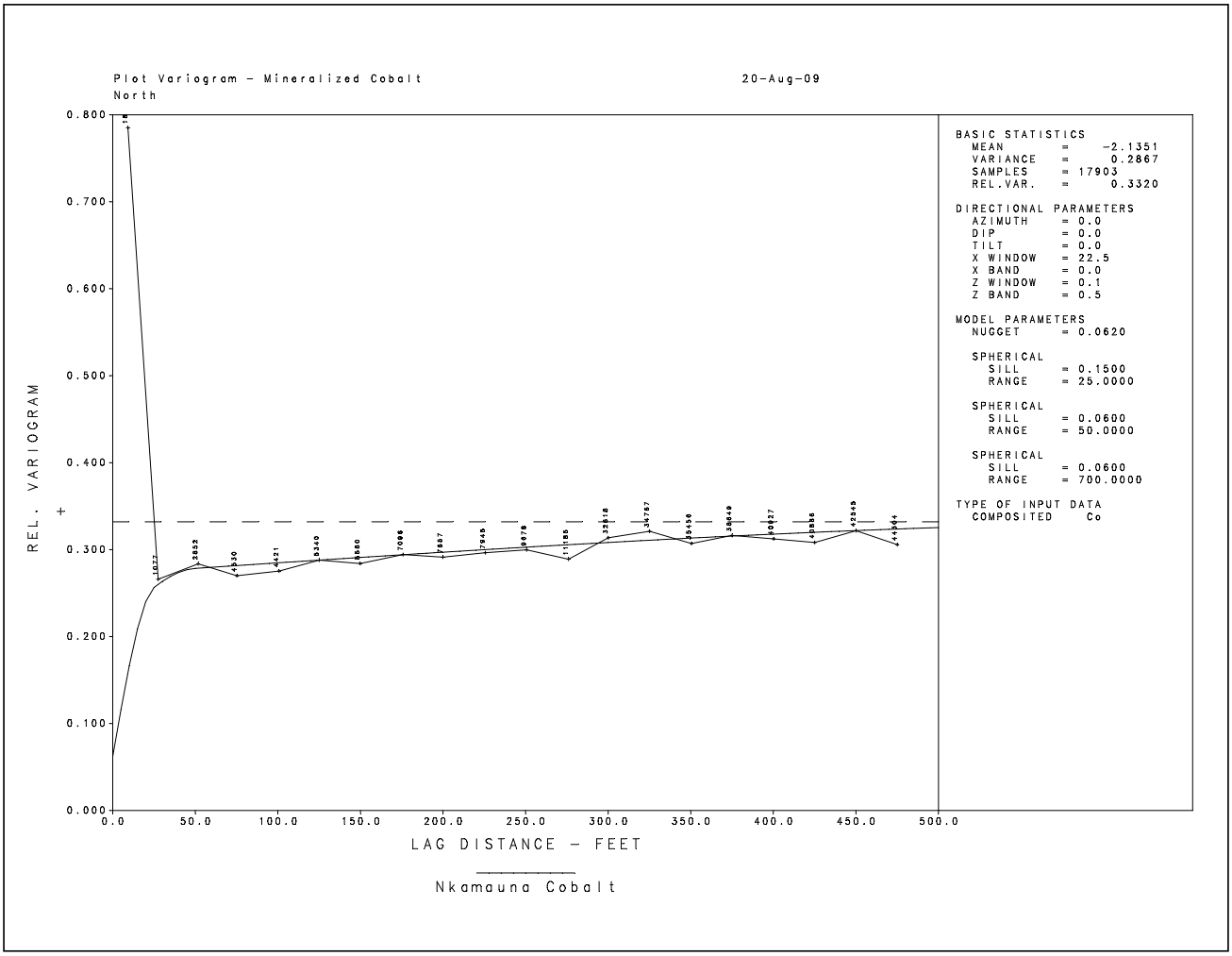
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20-Aug-09
North
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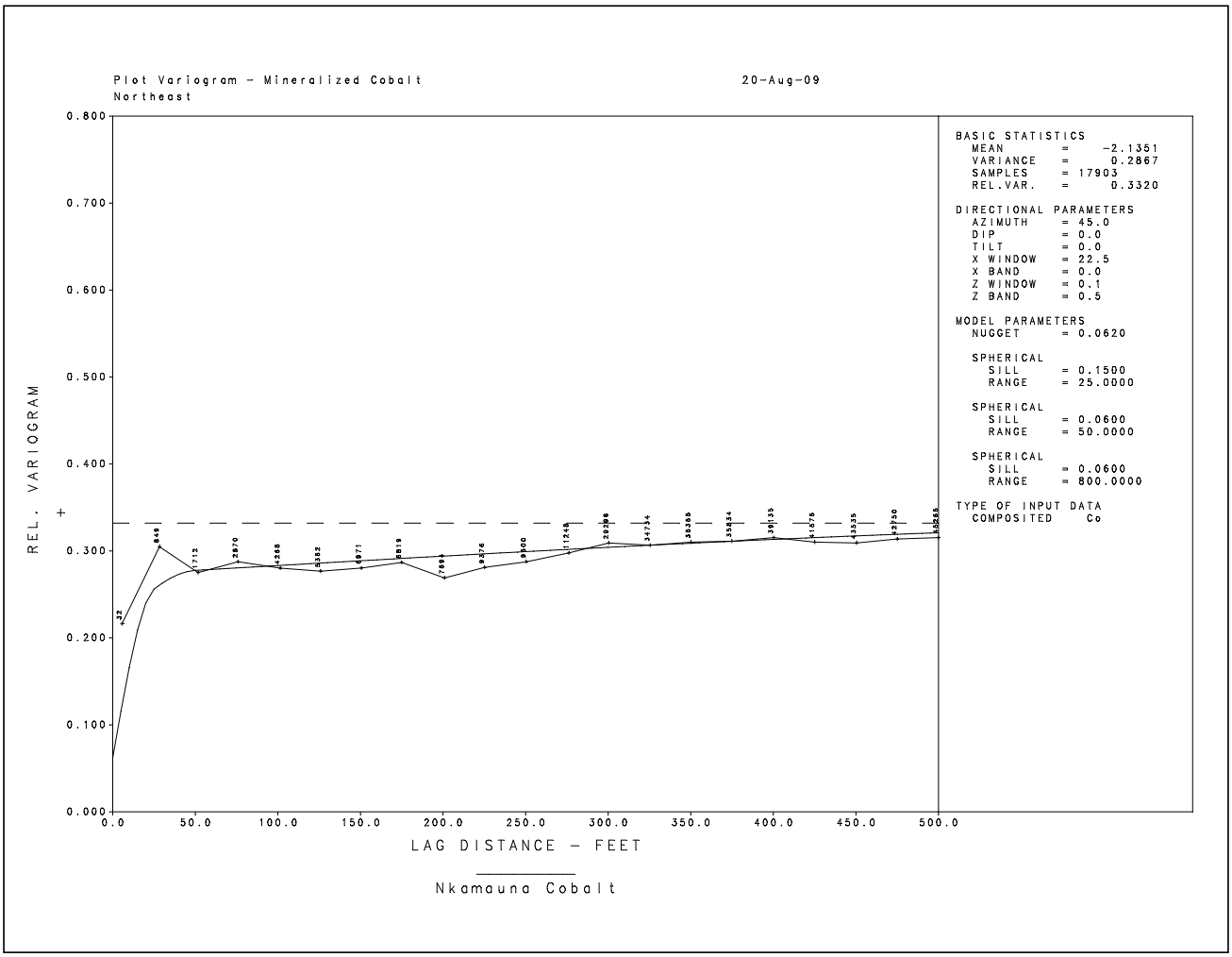
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Northeast
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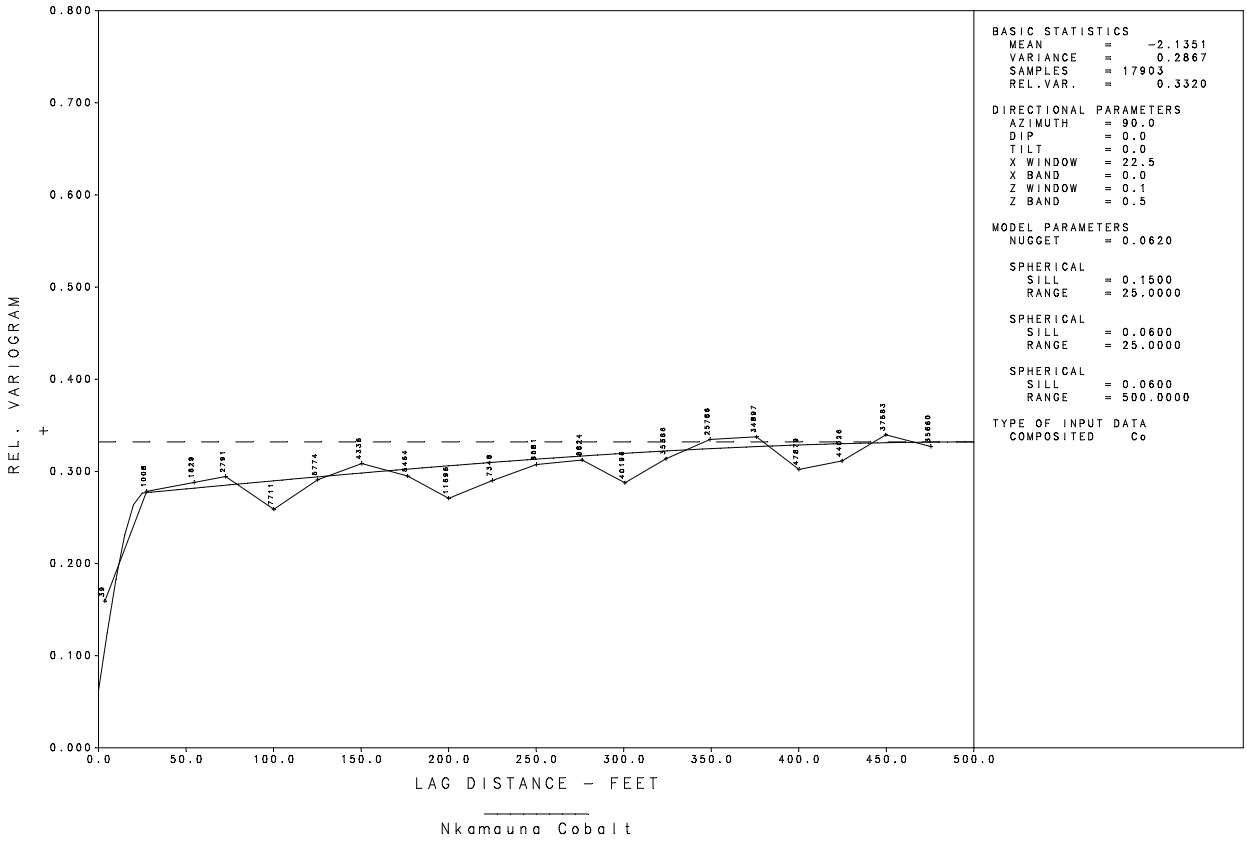
East

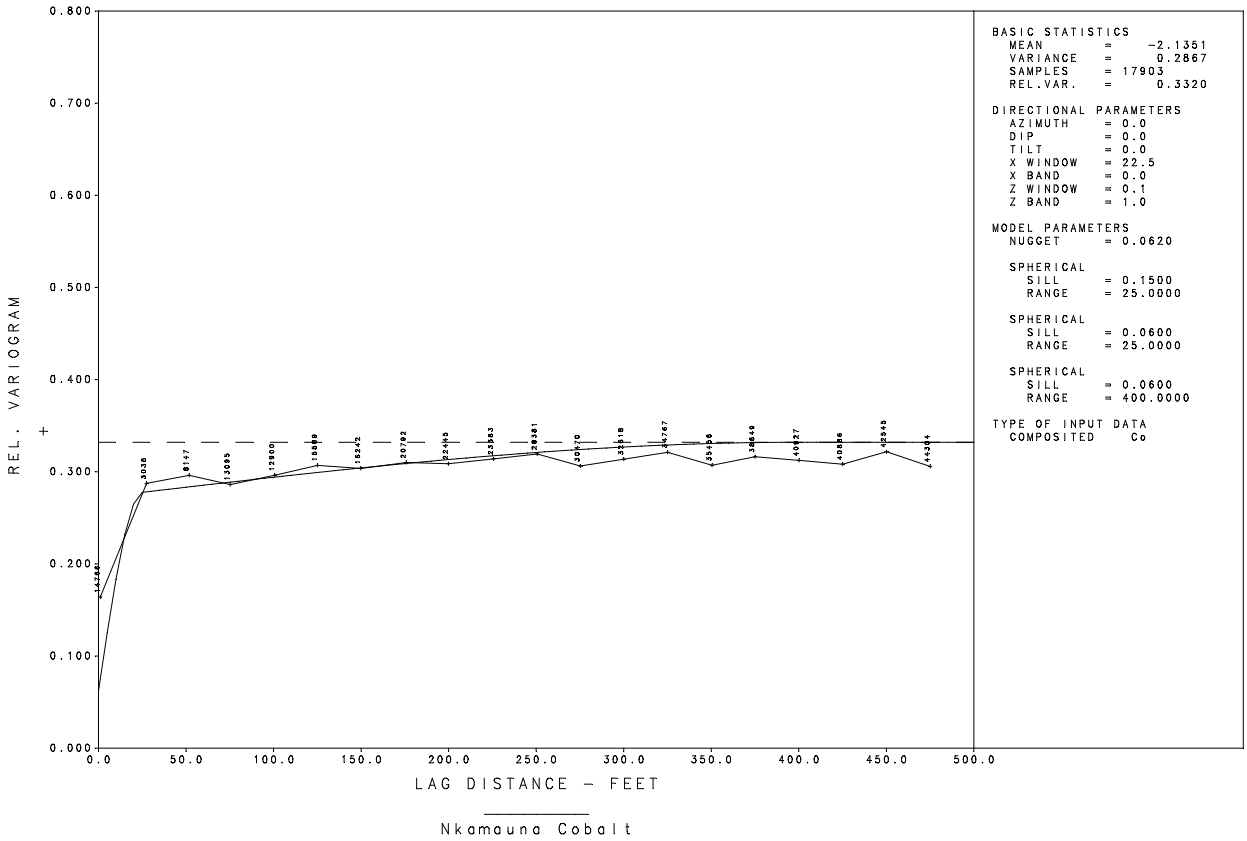
20-Aug-09



Nkamauna Cobalt

Plot Variogram - Mineralized Cobalt
Southeast

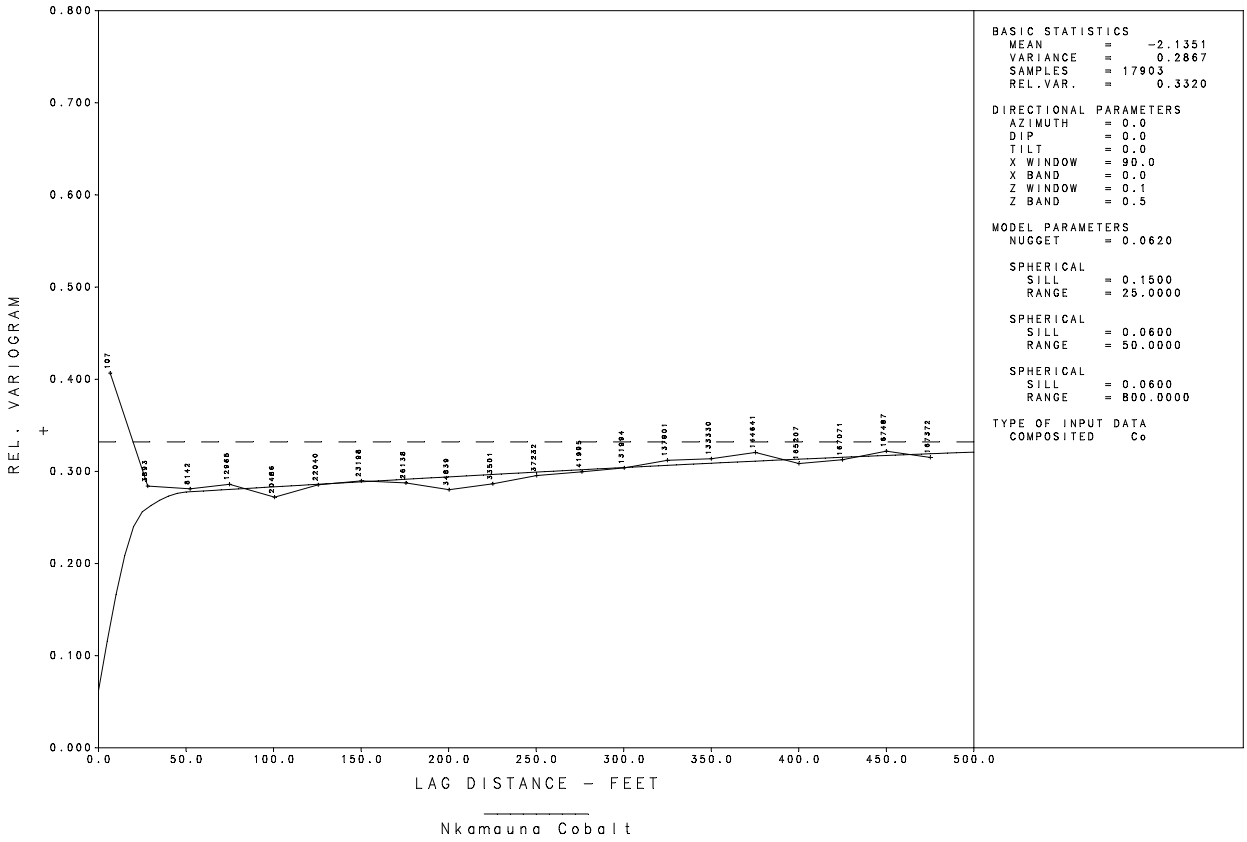
20-Aug-09



Nkamauna Cobalt

Plot Variogram - Mineralized Cobalt
Average Horizontal

20-Aug-09



BASIC STATISTICS
MEAN = -2.1351
VARIANCE = 0.2867
SAMPLES = 17903
REL.VAR. = 0.3320

DIRECTIONAL PARAMETERS
AZIMUTH = 0.0
DIP = 0.0
TILT = 0.0
X WINDOW = 90.0
X BAND = 0.0
Z WINDOW = 0.1
Z BAND = 0.5

MODEL PARAMETERS
NUGGET = 0.0620

SPHERICAL
SILL = 0.1500
RANGE = 25.0000

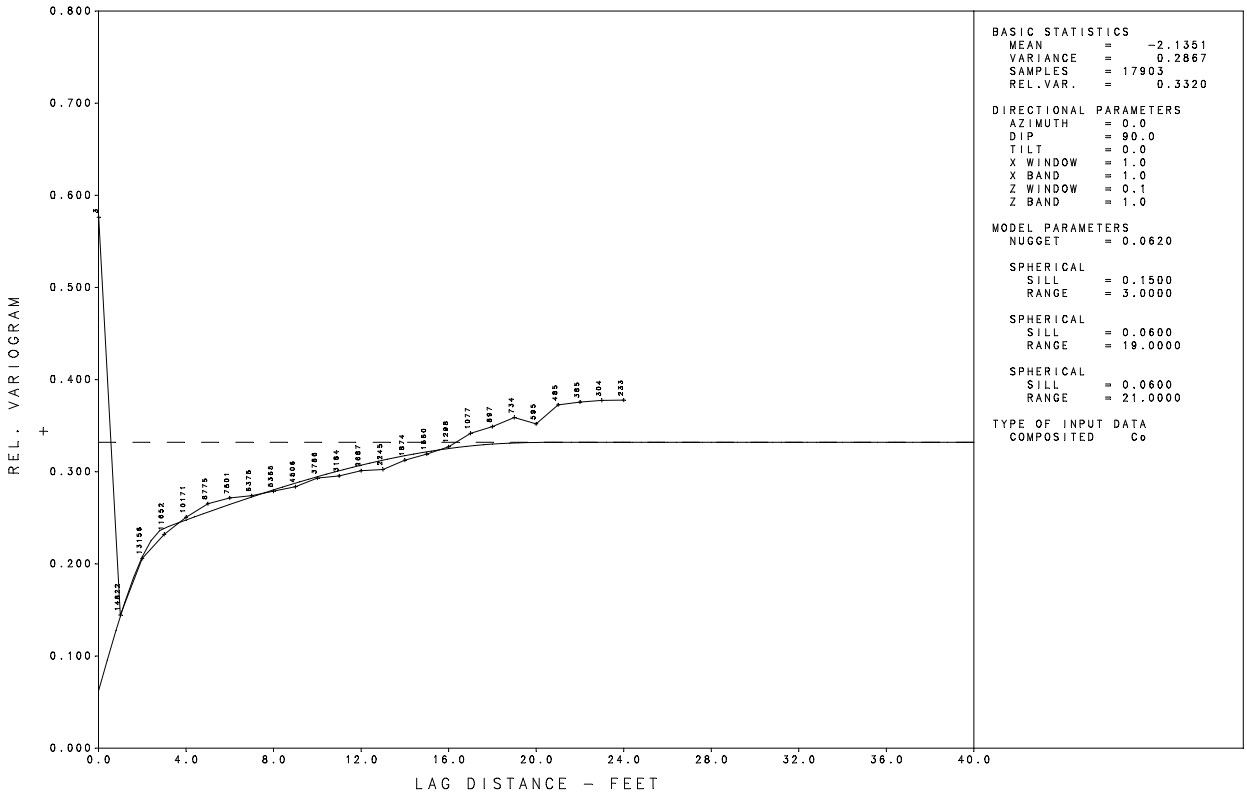
SPHERICAL
SILL = 0.0600
RANGE = 50.0000

SPHERICAL
SILL = 0.0600
RANGE = 800.0000

TYPE OF INPUT DATA
COMPOSITED Co

Plot Variogram - Mineralized Cobalt
Vertical

20-Aug-09



BASIC STATISTICS
MEAN = -2.1351
VARIANCE = 0.2867
SAMPLES = 17903
REL.VAR. = 0.3320

DIRECTIONAL PARAMETERS
AZIMUTH = 0.0
DIP = 90.0
TILT = 0.0
X WINDOW = 1.0
X BAND = 1.0
Z WINDOW = 0.1
Z BAND = 1.0

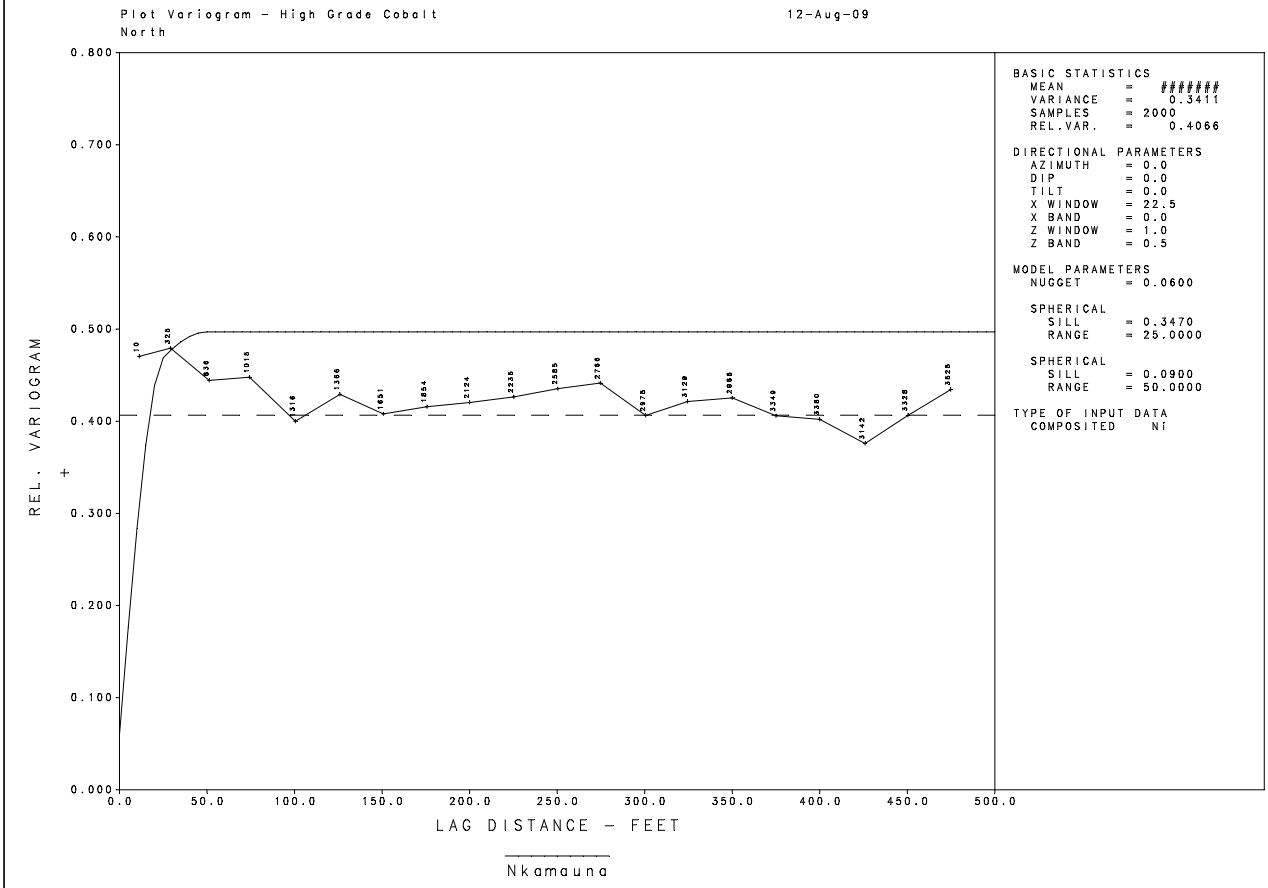
MODEL PARAMETERS
NUGGET = 0.0620

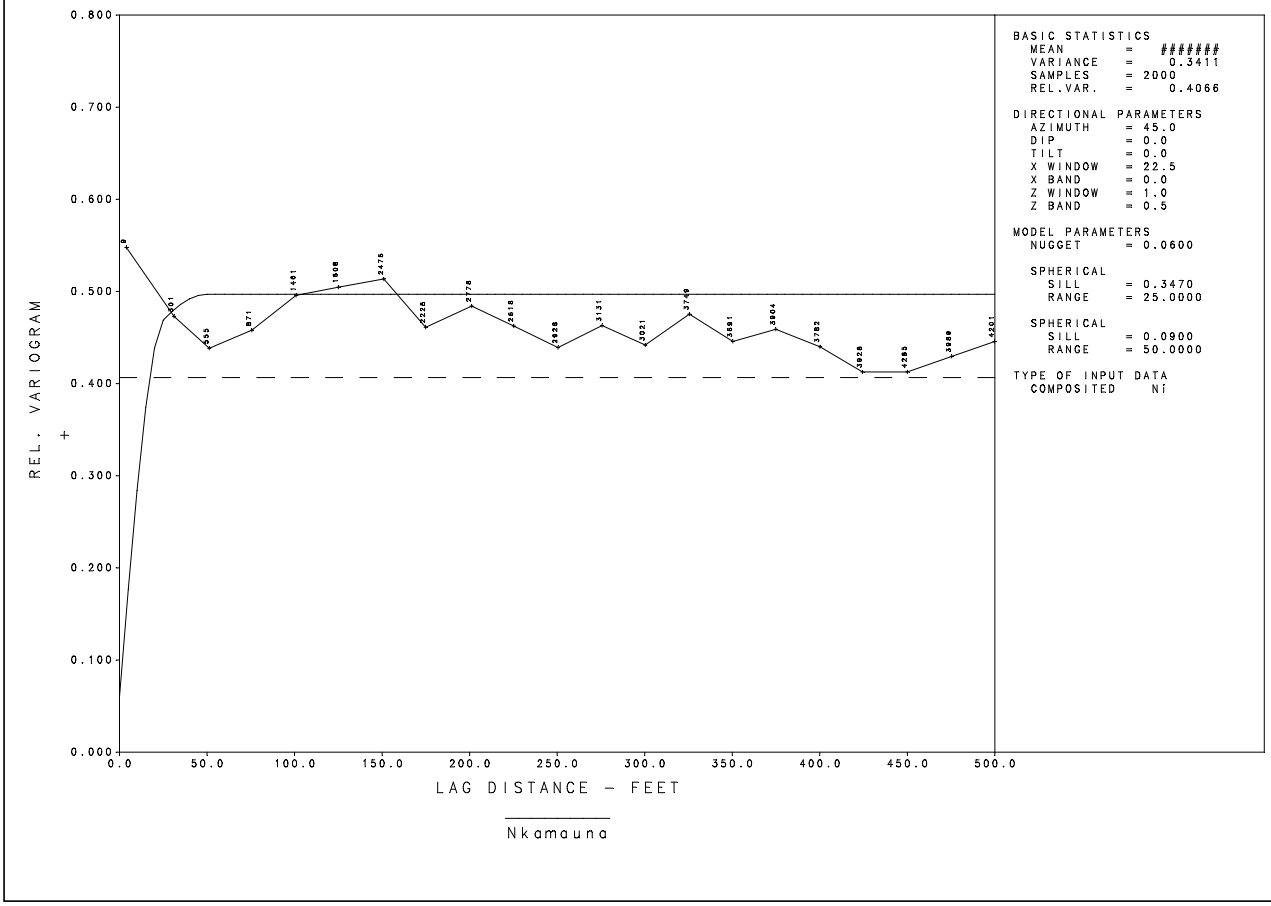
SPHERICAL
SILL = 0.1500
RANGE = 3.0000

SPHERICAL
SILL = 0.0600
RANGE = 19.0000

SPHERICAL
SILL = 0.0600
RANGE = 21.0000

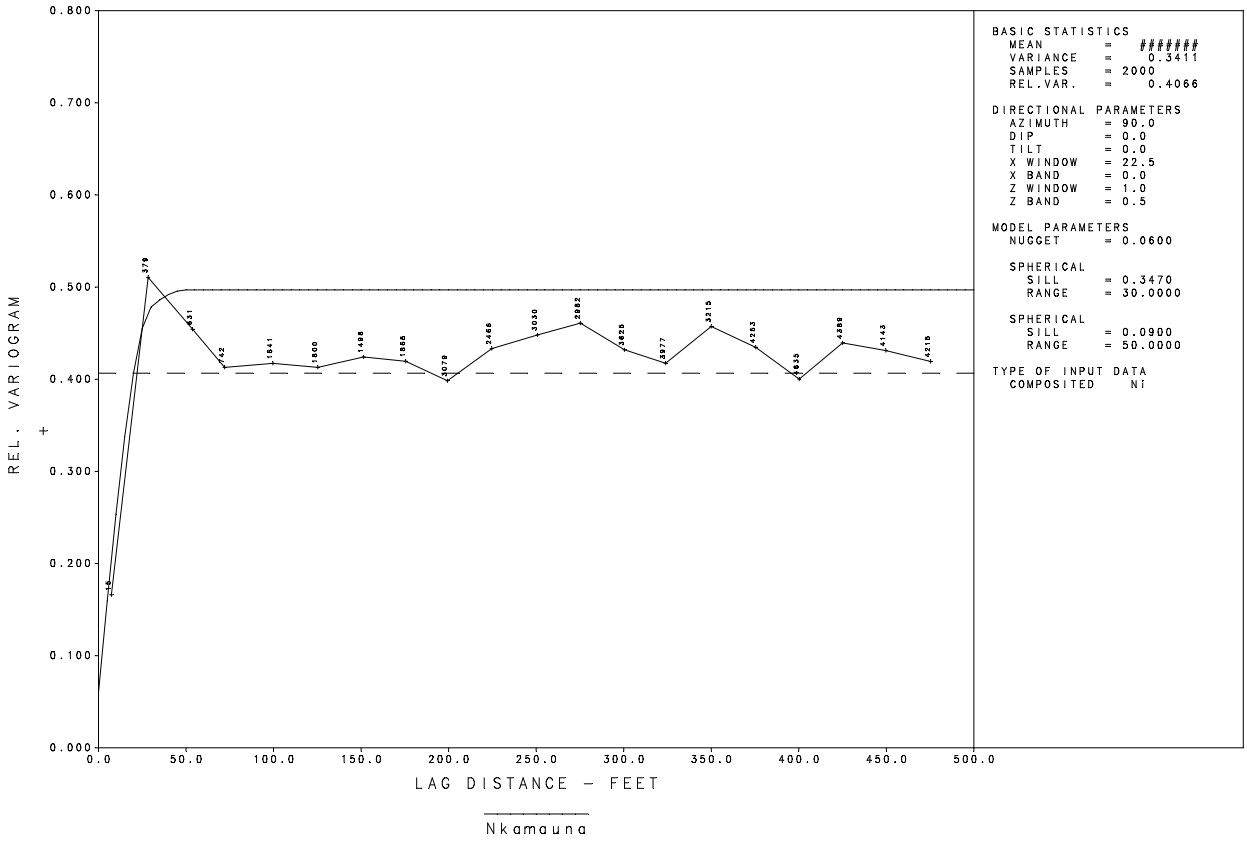
TYPE OF INPUT DATA
COMPOSITED Co


Plot Variogram - High Grade Cobalt
North
12-Aug-09
REL. VARIOGRAM
LAG DISTANCE - FEET
0.000
0.100
0.200
0.300
0.400
0.500
0.600
0.700
0.800
0.0
50.0
100.0
150.0
200.0
250.0
300.0
350.0
400.0
450.0
500.0
BASIC STATISTICS
MEAN = #######
VARIANCE = 0.3411
SAMPLES = 2000
REL.VAR. = 0.4066
DIRECTIONAL PARAMETERS
AZIMUTH = 0.0
DIP = 0.0
TILT = 0.0
X WINDOW = 22.5
X BAND = 0.0
Z WINDOW = 1.0
Z BAND = 0.5
MODEL PARAMETERS
NUGGET = 0.0600
SPHERICAL
SILL = 0.3470
RANGE = 25.0000
SPHERICAL
SILL = 0.0900
RANGE = 50.0000
TYPE OF INPUT DATA
COMPOSITED Ni
Nkamauna


Plot Variogram - High Grade Cobalt
Northeast
12-Aug-09
REL. VARIOGRAM
LAG DISTANCE - FEET
0.000
0.100
0.200
0.300
0.400
0.500
0.600
0.700
0.800
0.0
50.0
100.0
150.0
200.0
250.0
300.0
350.0
400.0
450.0
500.0
BASIC STATISTICS
MEAN = #######
VARIANCE = 0.3411
SAMPLES = 2000
REL.VAR. = 0.4066
DIRECTIONAL PARAMETERS
AZIMUTH = 45.0
DIP = 0.0
TILT = 0.0
X WINDOW = 22.5
X BAND = 0.0
Z WINDOW = 1.0
Z BAND = 0.5
MODEL PARAMETERS
NUGGET = 0.0600
SPHERICAL
SILL = 0.3470
RANGE = 25.0000
SPHERICAL
SILL = 0.0900
RANGE = 50.0000
TYPE OF INPUT DATA
COMPOSITED Ni
Nkamauna

Plot Variogram - High Grade Cobalt
East

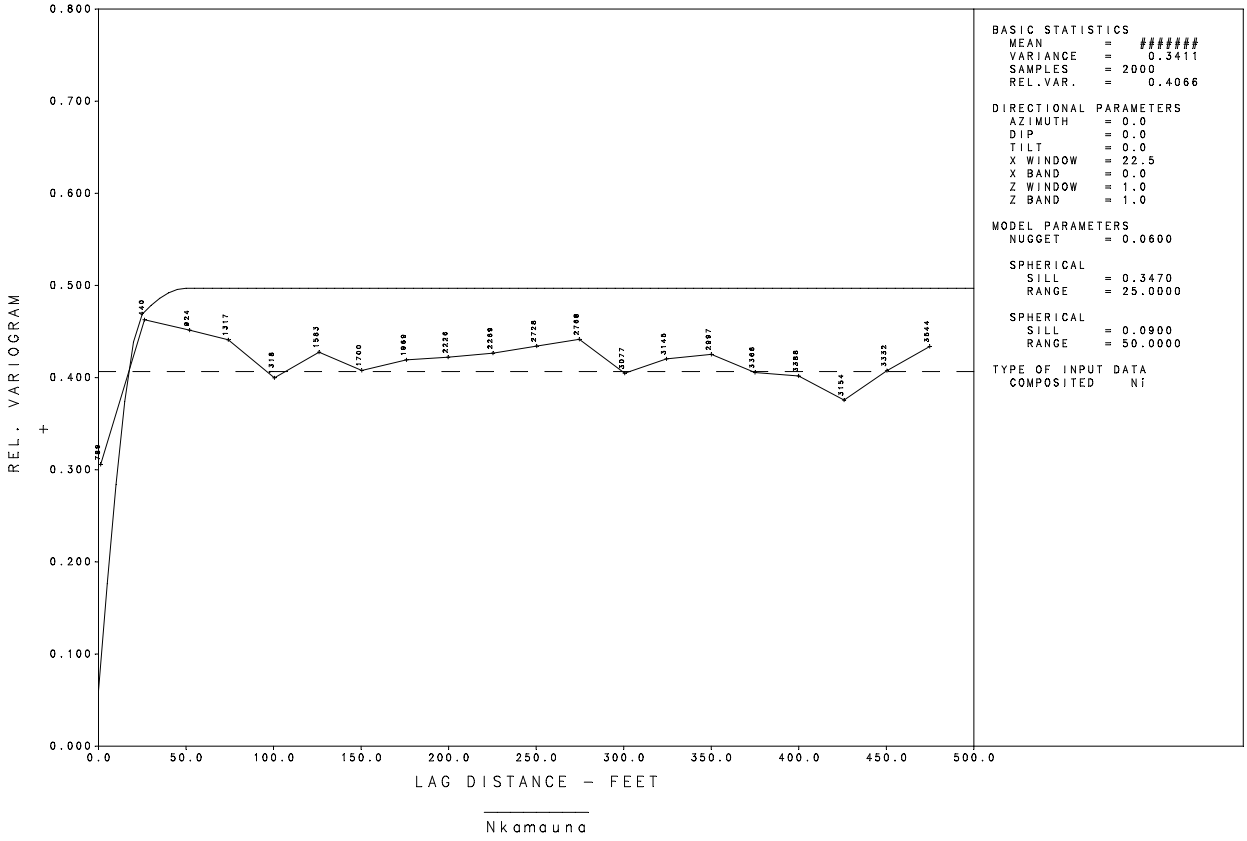
12-Aug-09



Nkamauna

Plot Variogram - High Grade Cobalt
Southeast

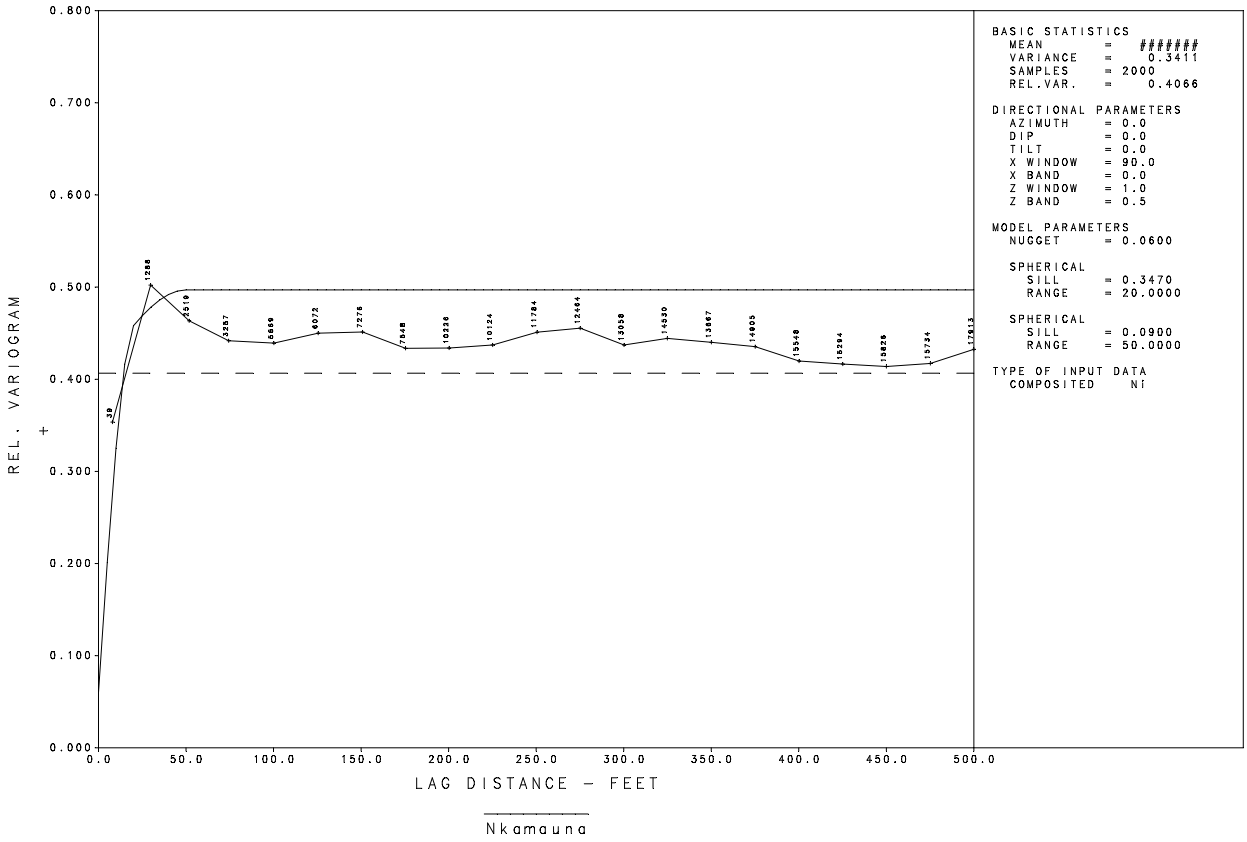
12-Aug-09



Nkamauna

Plot Variogram - High Grade Cobalt
Average Horizontal

12-Aug-09



BASIC STATISTICS
MEAN = #######
VARIANCE = 0.3411
SAMPLES = 2000
REL.VAR. = 0.4066

DIRECTIONAL PARAMETERS
AZIMUTH = 0.0
DIP = 0.0
TILT = 0.0
X WINDOW = 90.0
X BAND = 0.0
Z WINDOW = 1.0
Z BAND = 0.5

MODEL PARAMETERS
NUGGET = 0.0600

SPHERICAL
SILL = 0.3470
RANGE = 20.0000

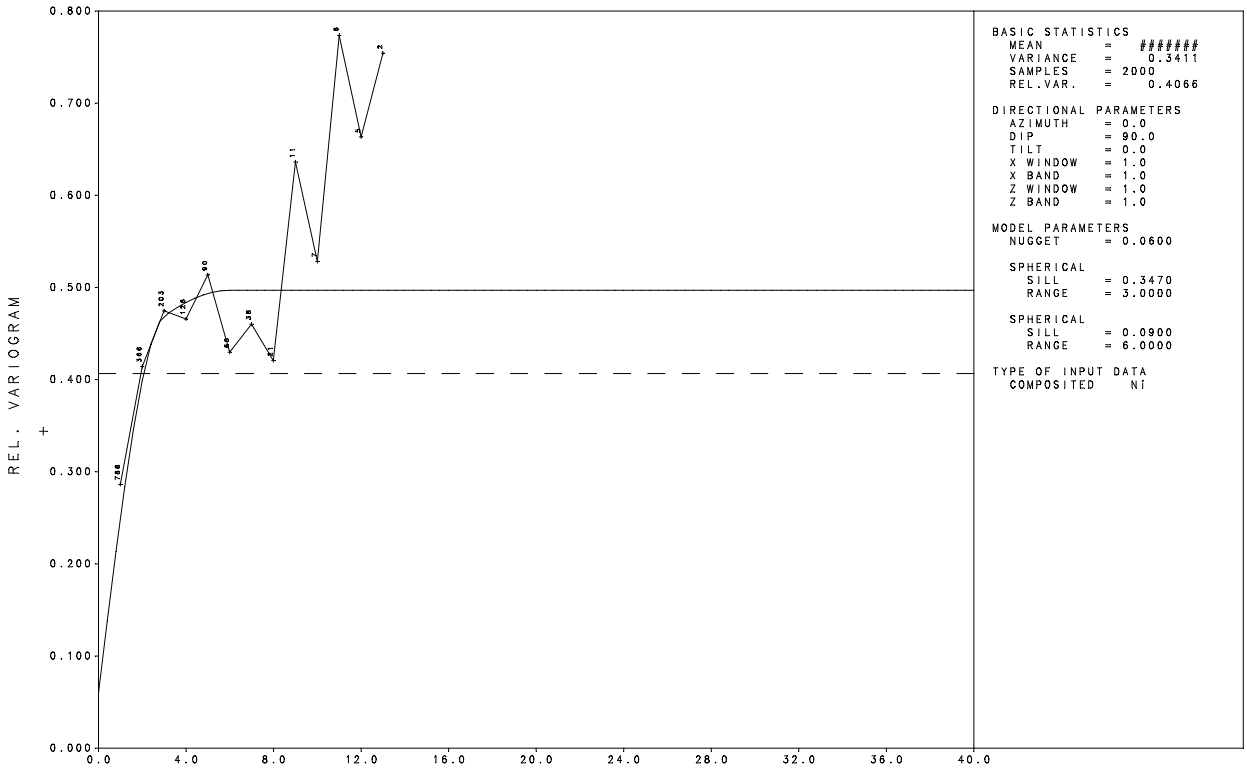
SPHERICAL
SILL = 0.0900
RANGE = 50.0000

TYPE OF INPUT DATA
COMPOSITED Ni

Nkamauna

Plot Variogram - High Grade Cobalt
Vertical

12-Aug-09



BASIC STATISTICS
MEAN = #######
VARIANCE = 0.3411
SAMPLES = 2000
REL.VAR. = 0.4066

DIRECTIONAL PARAMETERS
AZIMUTH = 0.0
DIP = 90.0
TILT = 0.0
X WINDOW = 1.0
X BAND = 1.0
Z WINDOW = 1.0
Z BAND = 1.0

MODEL PARAMETERS
NUGGET = 0.0600

SPHERICAL
SILL = 0.3470
RANGE = 3.0000

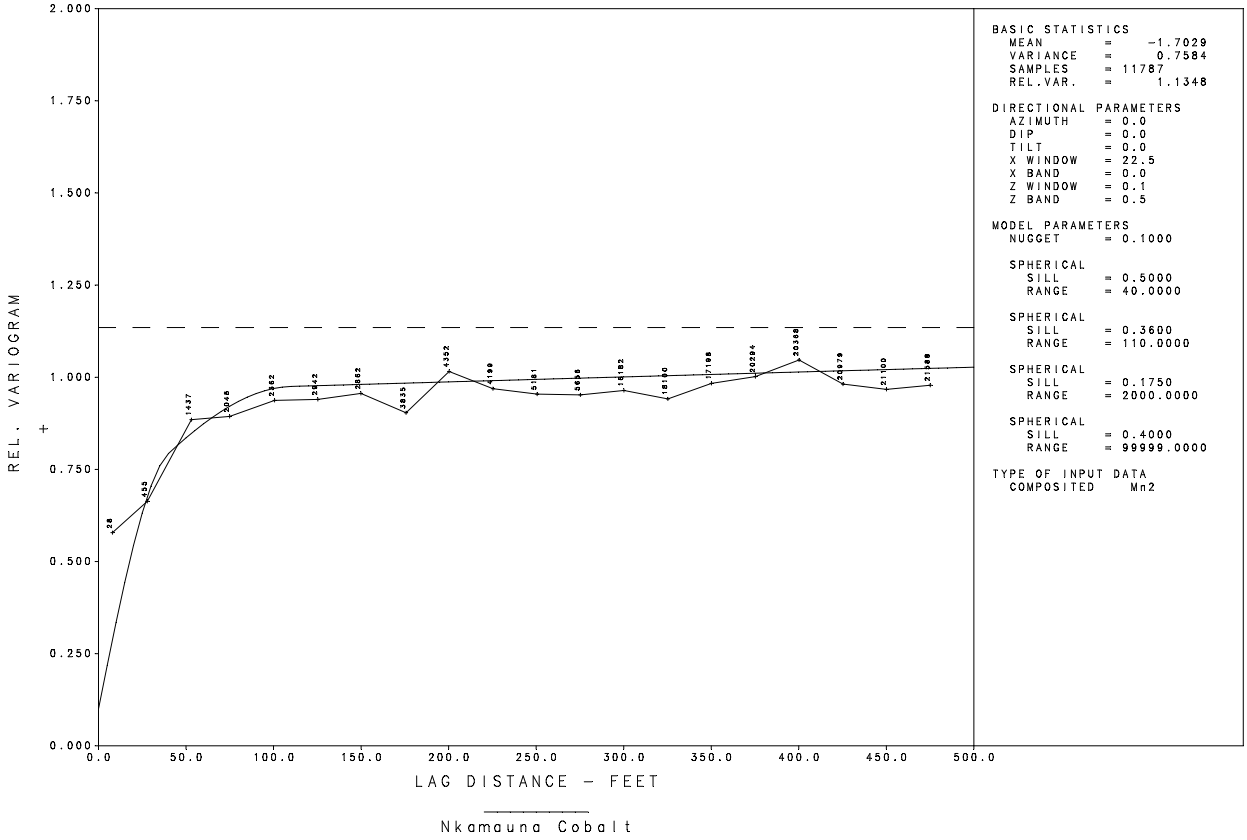
SPHERICAL
SILL = 0.0900
RANGE = 6.0000

TYPE OF INPUT DATA
COMPOSITED Ni

Nkamauna

Plot Variogram - LG Manganese
North

20-Aug-09



BASIC STATISTICS
- MEAN = -1.7029
- VARIANCE = 0.7584
- SAMPLES = 11787
- REL.VAR. = 1.1348

DIRECTIONAL PARAMETERS
- AZIMUTH = 0.0
- DIP = 0.0
- TILT = 0.0
- X WINDOW = 22.5
- X BAND = 0.0
- Z WINDOW = 0.1
- Z BAND = 0.5

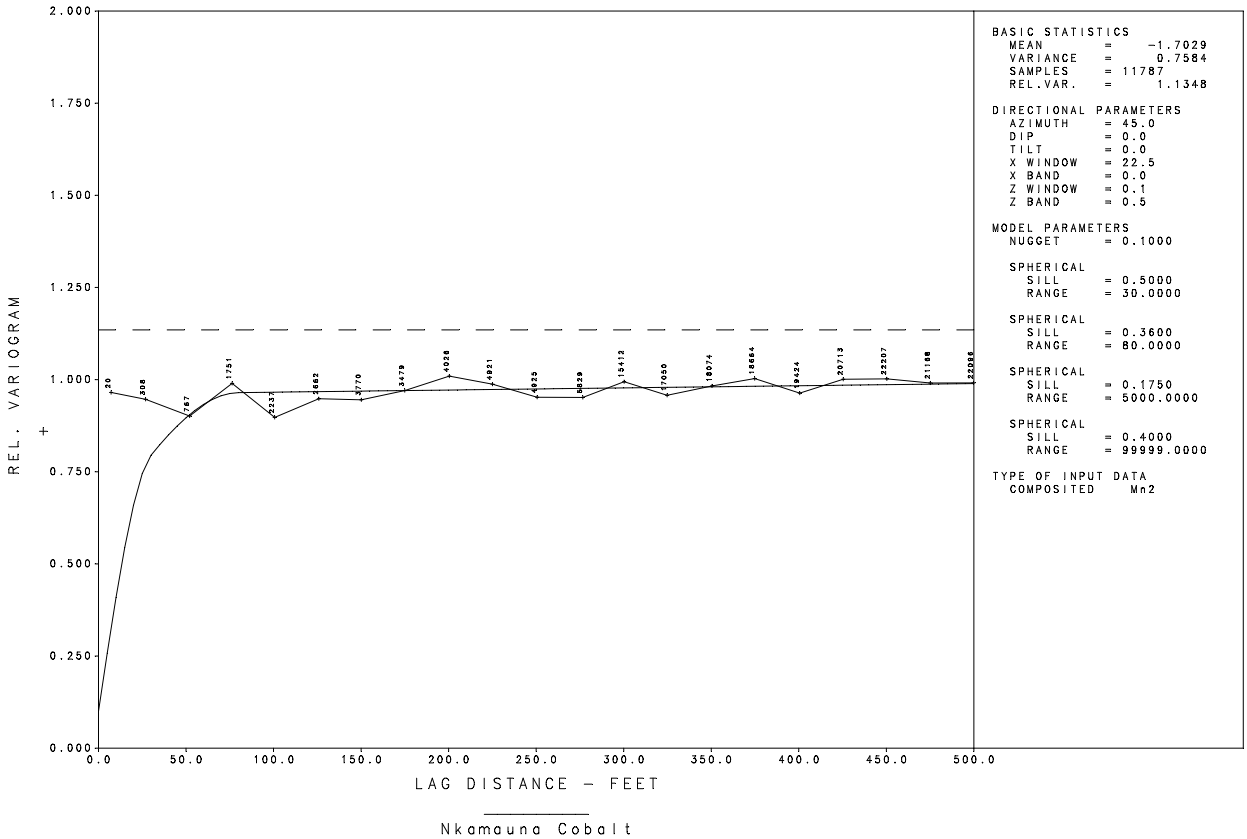
MODEL PARAMETERS
- NUGGET = 0.1000
- SPHERICAL
  - SILL = 0.5000
  - RANGE = 40.0000
- SPHERICAL
  - SILL = 0.3600
  - RANGE = 110.0000
- SPHERICAL
  - SILL = 0.1750
  - RANGE = 2000.0000
- SPHERICAL
  - SILL = 0.4000
  - RANGE = 99999.0000

TYPE OF INPUT DATA
- COMPOSITED Mn2

Nkamauna Cobalt

Plot Variogram - LG Manganese
Northeast

20-Aug-09



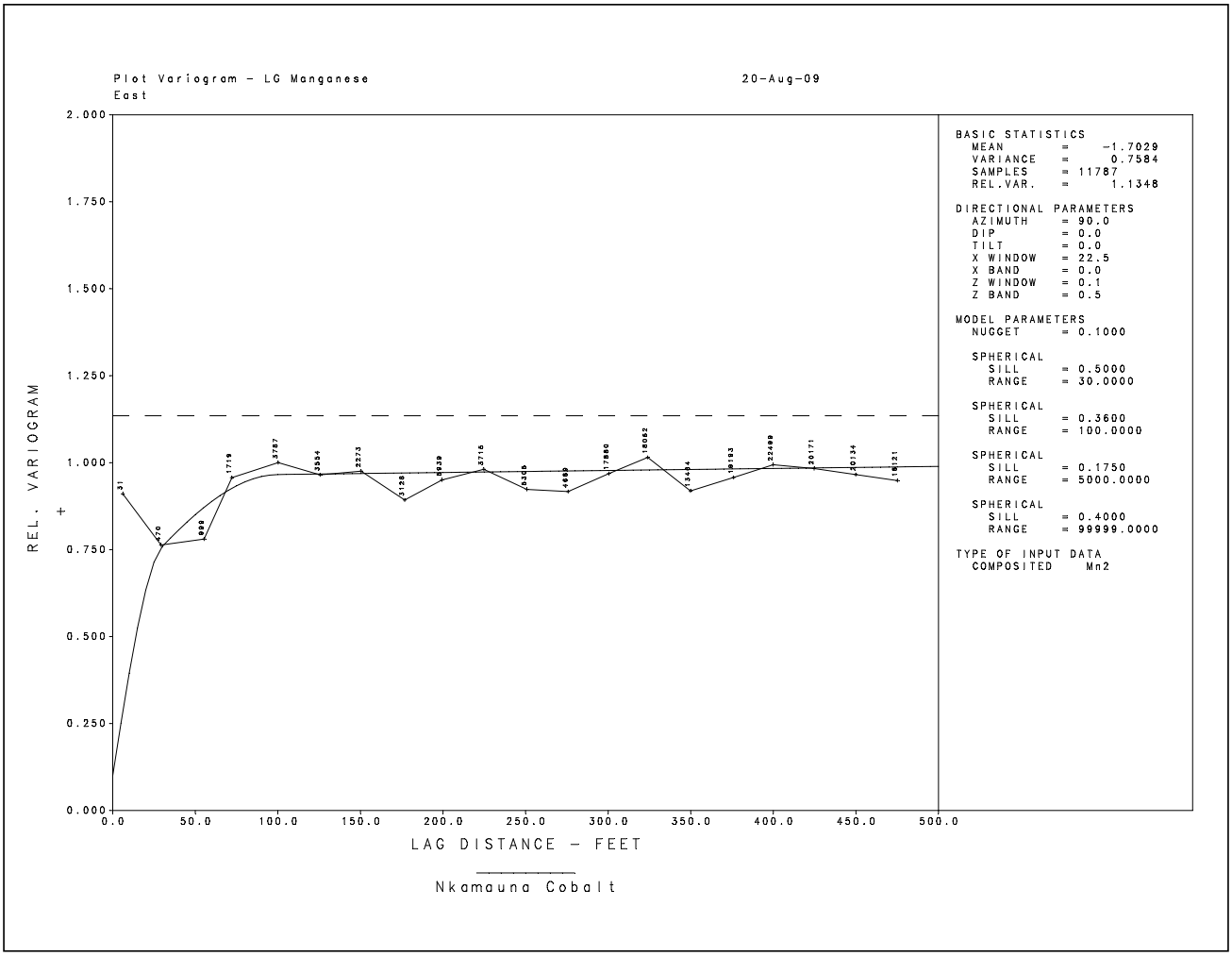
BASIC STATISTICS
- MEAN = -1.7029
- VARIANCE = 0.7584
- SAMPLES = 11787
- REL.VAR. = 1.1348

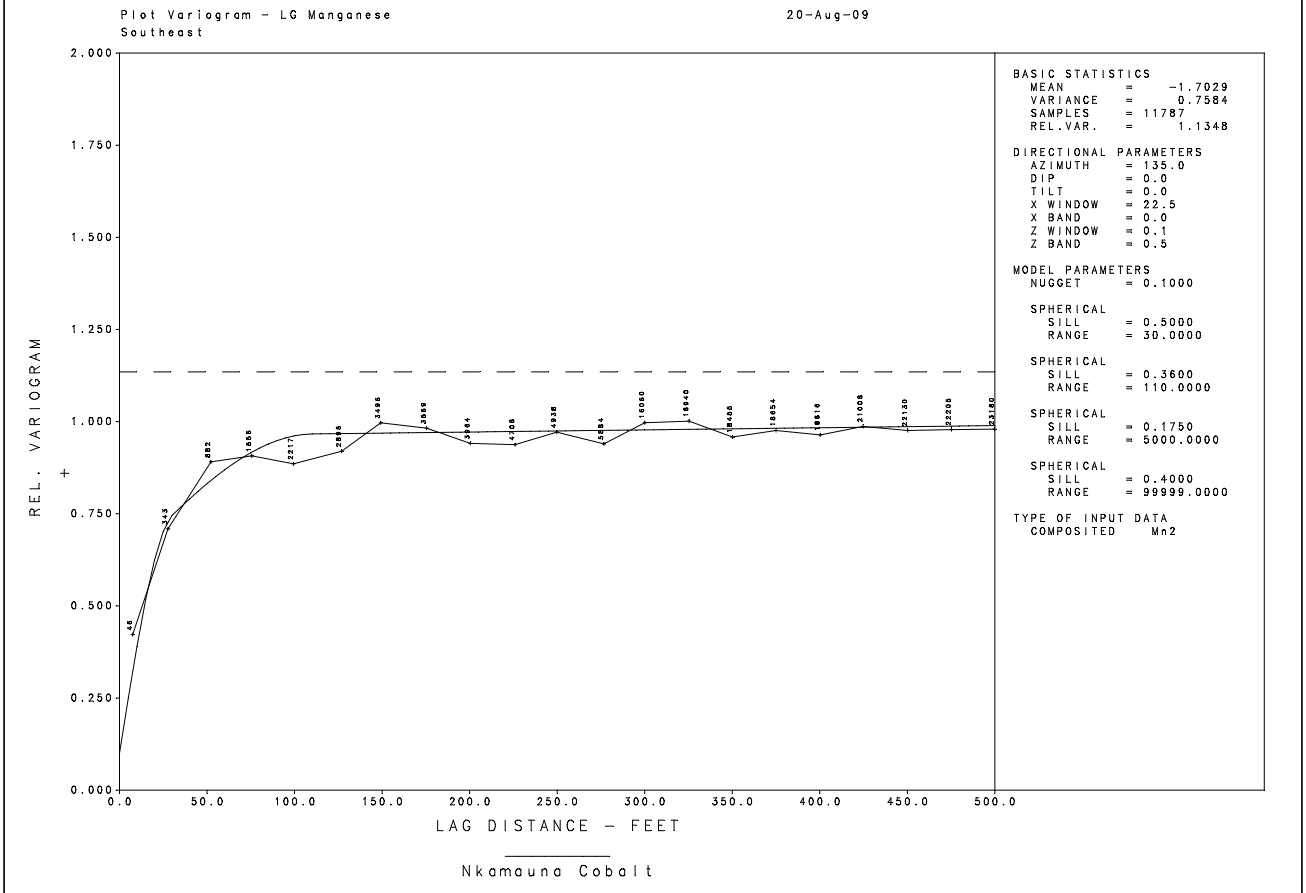
DIRECTIONAL PARAMETERS
- AZIMUTH = 45.0
- DIP = 0.0
- TILT = 0.0
- X WINDOW = 22.5
- X BAND = 0.0
- Z WINDOW = 0.1
- Z BAND = 0.5

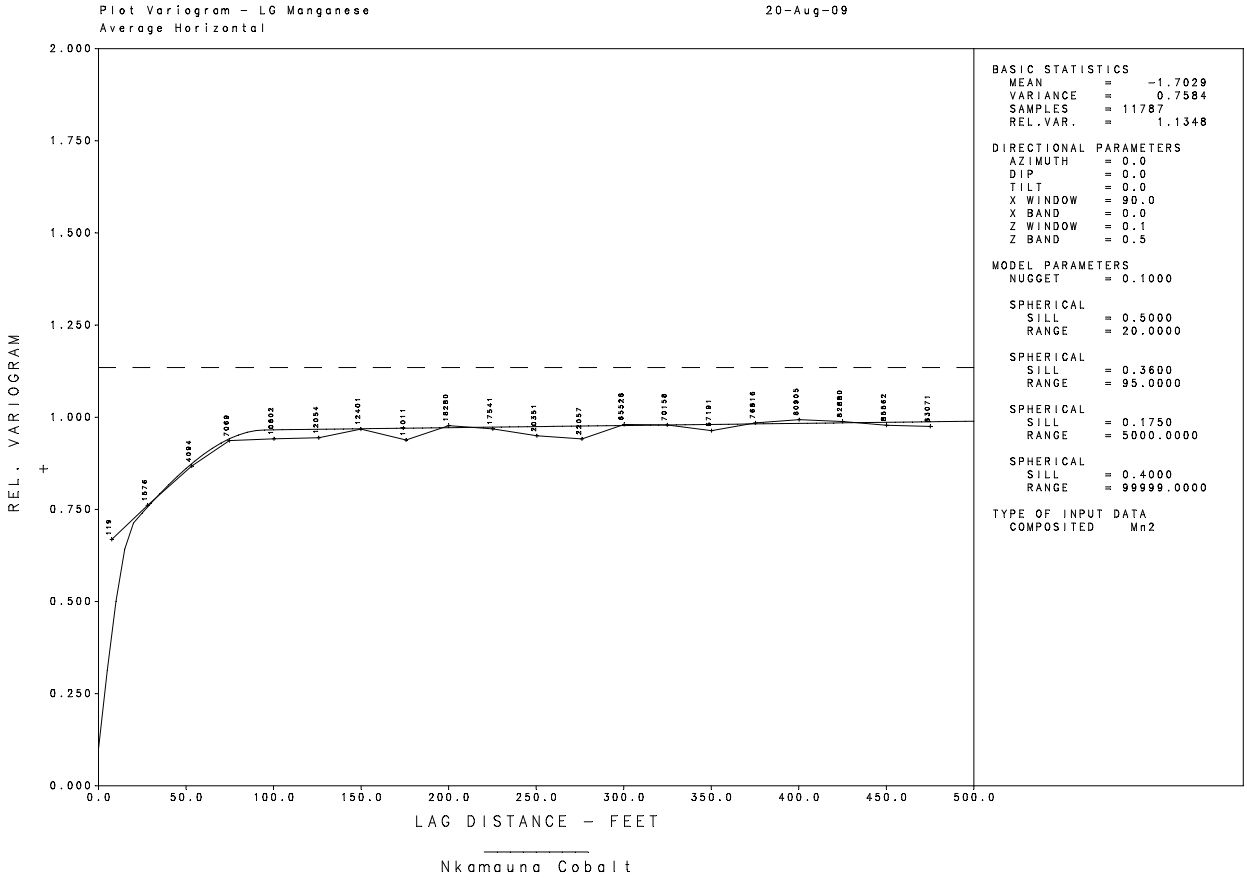
MODEL PARAMETERS
- NUGGET = 0.1000
- SPHERICAL
  - SILL = 0.5000
  - RANGE = 30.0000
- SPHERICAL
  - SILL = 0.3600
  - RANGE = 80.0000
- SPHERICAL
  - SILL = 0.1750
  - RANGE = 5000.0000
- SPHERICAL
  - SILL = 0.4000
  - RANGE = 99999.0000

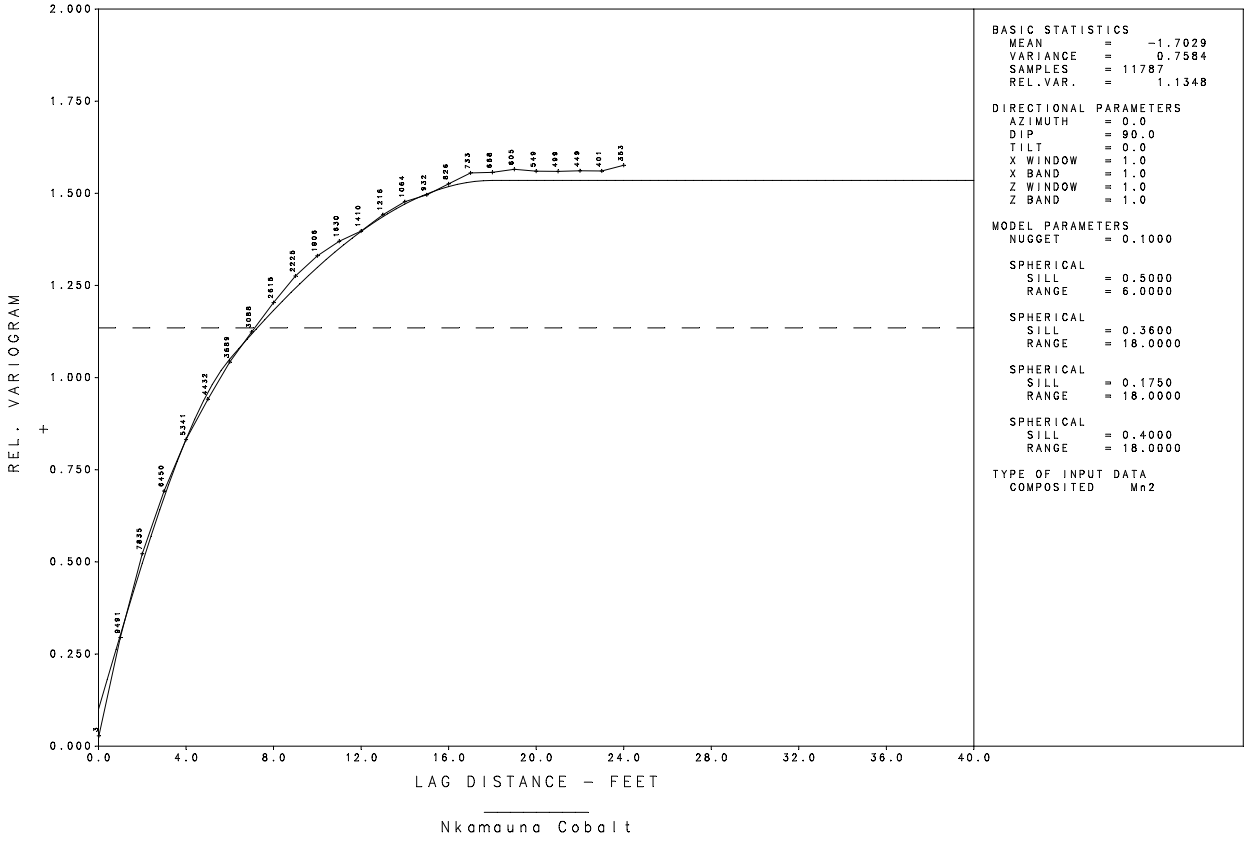
TYPE OF INPUT DATA
- COMPOSITED Mn2

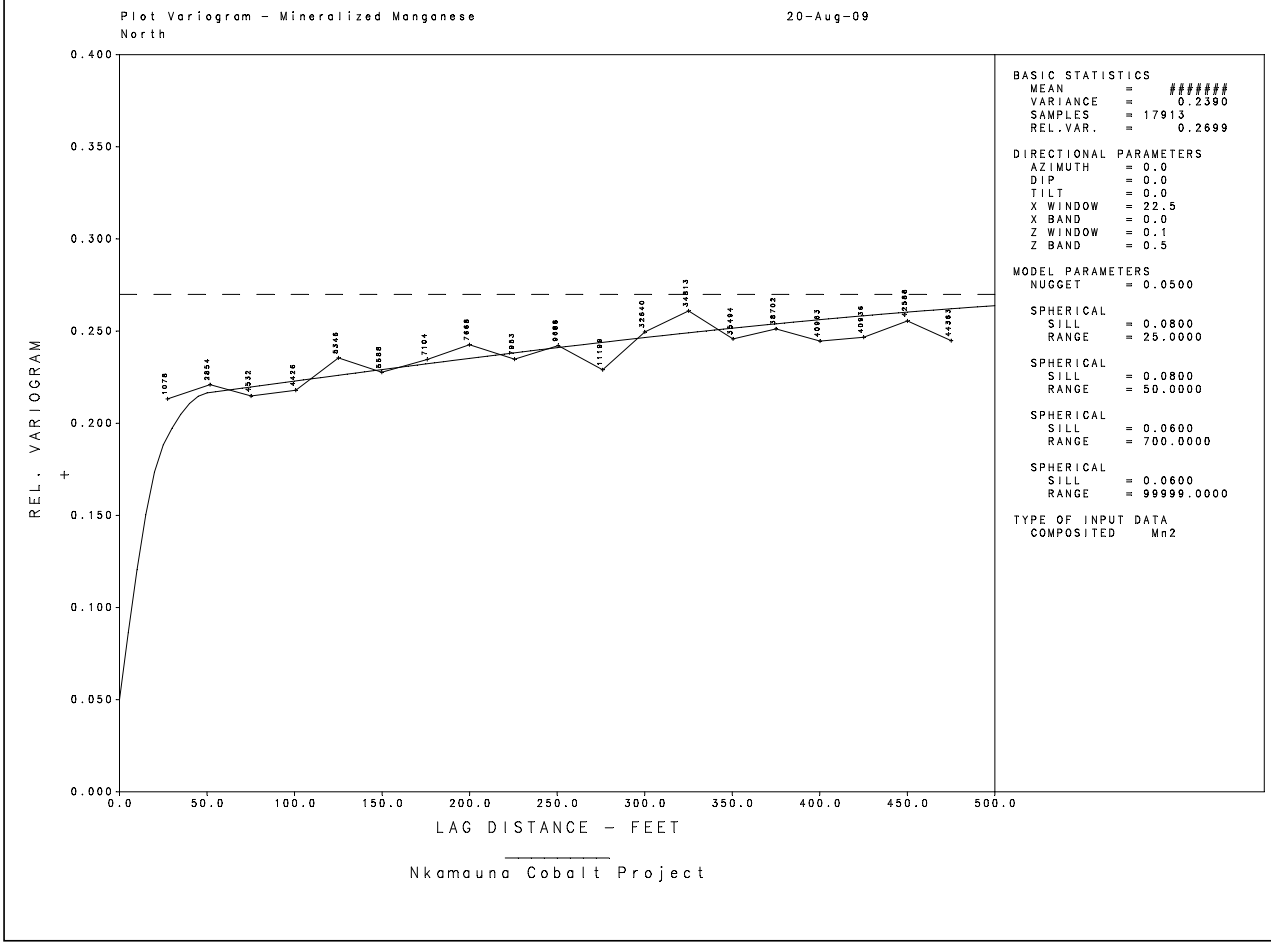
Nkamauna Cobalt

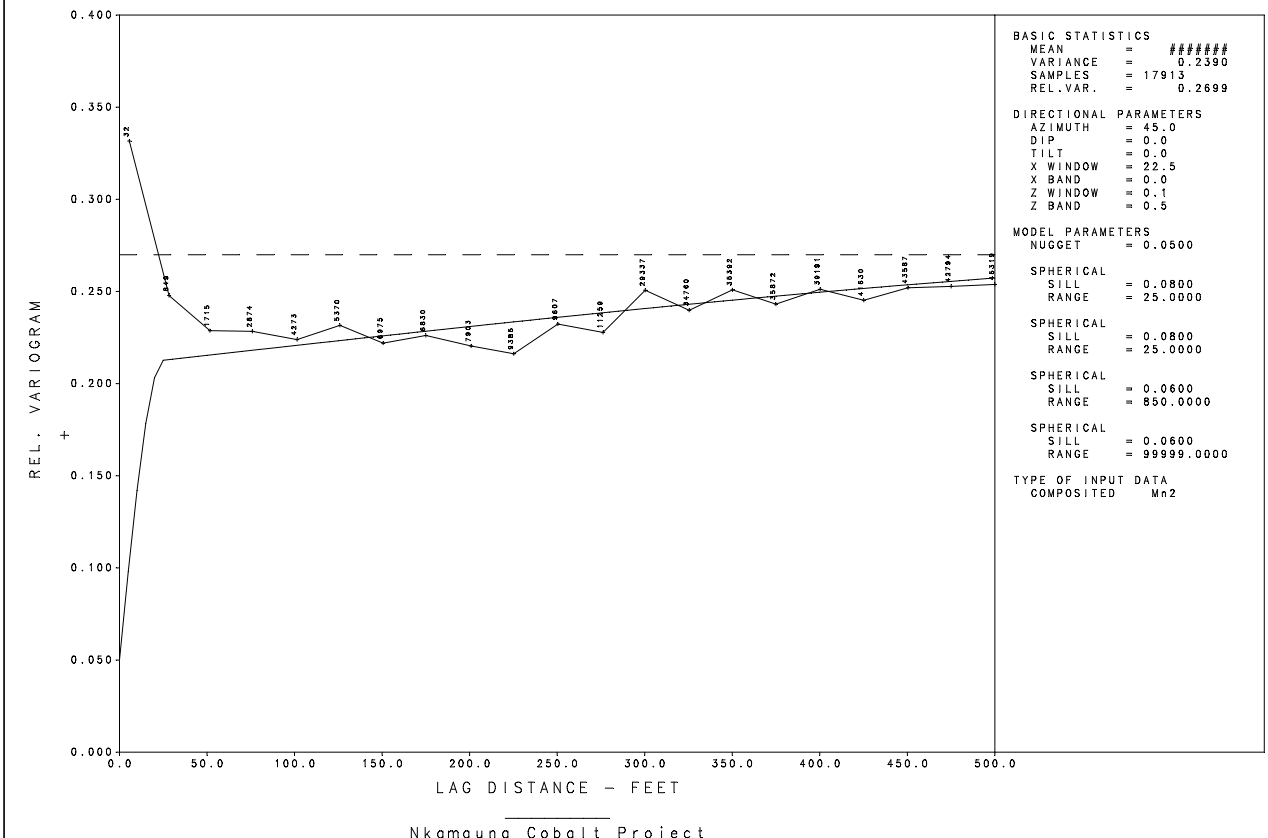

Plot Variogram - LG Manganese
East
20-Aug-09
REL. VARIOGRAM
LAG DISTANCE - FEET
Nkamauna Cobalt
BASIC STATISTICS
MEAN = -1.7029
VARIANCE = 0.7584
SAMPLES = 11787
REL.VAR. = 1.1348
DIRECTIONAL PARAMETERS
AZIMUTH = 90.0
DIP = 0.0
TILT = 0.0
X WINDOW = 22.5
X BAND = 0.0
Z WINDOW = 0.1
Z BAND = 0.5
MODEL PARAMETERS
NUGGET = 0.1000
SPHERICAL
SILL = 0.5000
RANGE = 30.0000
SPHERICAL
SILL = 0.3600
RANGE = 100.0000
SPHERICAL
SILL = 0.1750
RANGE = 5000.0000
SPHERICAL
SILL = 0.4000
RANGE = 99999.0000
TYPE OF INPUT DATA
COMPOSITED Mn2


Plot Variogram - LG Manganese
Southeast
20-Aug-09
REL. VARIOGRAM
LAG DISTANCE - FEET
Nkamauna Cobalt
BASIC STATISTICS
MEAN = -1.7029
VARIANCE = 0.7584
SAMPLES = 11787
REL.VAR. = 1.1348
DIRECTIONAL PARAMETERS
AZIMUTH = 135.0
DIP = 0.0
TILT = 0.0
X WINDOW = 22.5
X BAND = 0.0
Z WINDOW = 0.1
Z BAND = 0.5
MODEL PARAMETERS
NUGGET = 0.1000
SPHERICAL
SILL = 0.5000
RANGE = 30.0000
SPHERICAL
SILL = 0.3600
RANGE = 110.0000
SPHERICAL
SILL = 0.1750
RANGE = 5000.0000
SPHERICAL
SILL = 0.4000
RANGE = 99999.0000
TYPE OF INPUT DATA
COMPOSITED Mn2

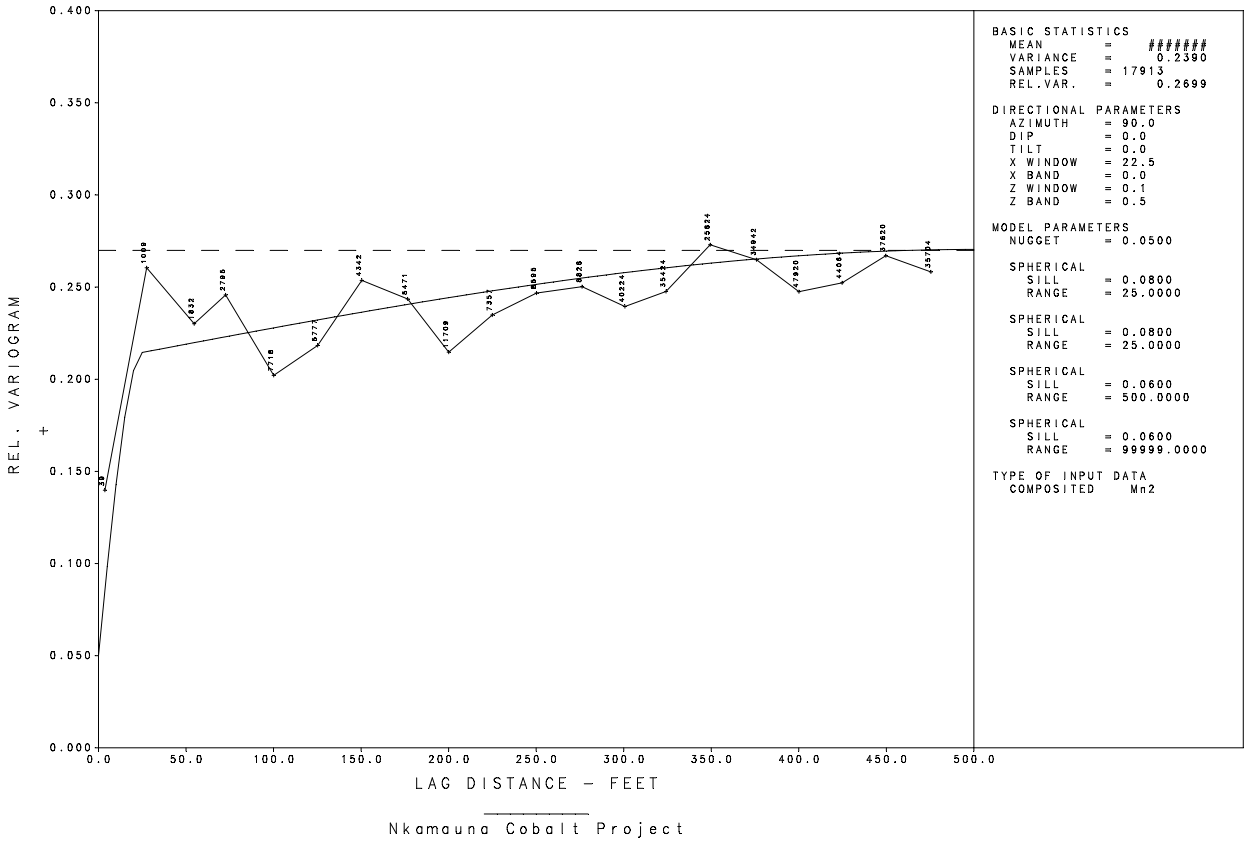

Plot Variogram - LG Manganese
Average Horizontal
20-Aug-09
REL. VARIOGRAM
LAG DISTANCE - FEET
Nkamauna Cobalt
BASIC STATISTICS
MEAN = -1.7029
VARIANCE = 0.7584
SAMPLES = 11787
REL.VAR. = 1.1348
DIRECTIONAL PARAMETERS
AZIMUTH = 0.0
DIP = 0.0
TILT = 0.0
X WINDOW = 90.0
X BAND = 0.0
Z WINDOW = 0.1
Z BAND = 0.5
MODEL PARAMETERS
NUGGET = 0.1000
SPHERICAL
SILL = 0.5000
RANGE = 20.0000
SPHERICAL
SILL = 0.3600
RANGE = 95.0000
SPHERICAL
SILL = 0.1750
RANGE = 5000.0000
SPHERICAL
SILL = 0.4000
RANGE = 99999.0000
TYPE OF INPUT DATA
COMPOSITED Mn2


Plot Variogram - LG Manganese
Vertical
20-Aug-09
REL. VARIOGRAM
LAG DISTANCE - FEET
Nkamauna Cobalt
BASIC STATISTICS
MEAN = -1.7029
VARIANCE = 0.7584
SAMPLES = 11787
REL.VAR. = 1.1348
DIRECTIONAL PARAMETERS
AZIMUTH = 0.0
DIP = 90.0
TILT = 0.0
X WINDOW = 1.0
X BAND = 1.0
Z WINDOW = 1.0
Z BAND = 1.0
MODEL PARAMETERS
NUGGET = 0.1000
SPHERICAL
SILL = 0.5000
RANGE = 6.0000
SPHERICAL
SILL = 0.3600
RANGE = 18.0000
SPHERICAL
SILL = 0.1750
RANGE = 18.0000
SPHERICAL
SILL = 0.4000
RANGE = 18.0000
TYPE OF INPUT DATA
COMPOSITED Mn2


Plot Variogram - Mineralized Manganese
North
20-Aug-09
REL. VARIOGRAM
LAG DISTANCE - FEET
0.400
0.350
0.300
0.250
0.200
0.150
0.100
0.050
0.000
0.0
50.0
100.0
150.0
200.0
250.0
300.0
350.0
400.0
450.0
500.0
1078
2654
4932
4426
5348
5588
7104
7668
7953
8688
11189
12640
14813
15444
15702
16993
16995
17588
14363
BASIC STATISTICS
MEAN = #######
VARIANCE = 0.2390
SAMPLES = 17913
REL.VAR. = 0.2699
DIRECTIONAL PARAMETERS
AZIMUTH = 0.0
DIP = 0.0
TILT = 0.0
X WINDOW = 22.5
X BAND = 0.0
Z WINDOW = 0.1
Z BAND = 0.5
MODEL PARAMETERS
NUGGET = 0.0500
SPHERICAL
SILL = 0.0800
RANGE = 25.0000
SPHERICAL
SILL = 0.0800
RANGE = 50.0000
SPHERICAL
SILL = 0.0600
RANGE = 700.0000
SPHERICAL
SILL = 0.0600
RANGE = 99999.0000
TYPE OF INPUT DATA
COMPOSITED Mn2
Nkamauna Cobalt Project


Plot Variogram - Mineralized Manganese
Northeast
20-Aug-09
REL. VARIOGRAM
LAG DISTANCE - FEET
0.400
0.350
0.300
0.250
0.200
0.150
0.100
0.050
0.000
0.0
50.0
100.0
150.0
200.0
250.0
300.0
350.0
400.0
450.0
500.0
32
848
1715
2874
4373
5370
6975
6830
7943
8386
9607
11359
12337
14740
15872
16991
17830
17587
17784
18318
BASIC STATISTICS
MEAN = #######
VARIANCE = 0.2390
SAMPLES = 17913
REL.VAR. = 0.2699
DIRECTIONAL PARAMETERS
AZIMUTH = 45.0
DIP = 0.0
TILT = 0.0
X WINDOW = 22.5
X BAND = 0.0
Z WINDOW = 0.1
Z BAND = 0.5
MODEL PARAMETERS
NUGGET = 0.0500
SPHERICAL
SILL = 0.0800
RANGE = 25.0000
SPHERICAL
SILL = 0.0800
RANGE = 25.0000
SPHERICAL
SILL = 0.0600
RANGE = 850.0000
SPHERICAL
SILL = 0.0600
RANGE = 99999.0000
TYPE OF INPUT DATA
COMPOSITED Mn2
Nkamauna Cobalt Project

Plot Variogram - Mineralized Manganese 20-Aug-09
East



BASIC STATISTICS
MEAN = #######
VARIANCE = 0.2390
SAMPLES = 17913
REL.VAR. = 0.2699

DIRECTIONAL PARAMETERS
AZIMUTH = 90.0
DIP = 0.0
TILT = 0.0
X WINDOW = 22.5
X BAND = 0.0
Z WINDOW = 0.1
Z BAND = 0.5

MODEL PARAMETERS
NUGGET = 0.0500

SPHERICAL
SILL = 0.0800
RANGE = 25.0000

SPHERICAL
SILL = 0.0800
RANGE = 25.0000

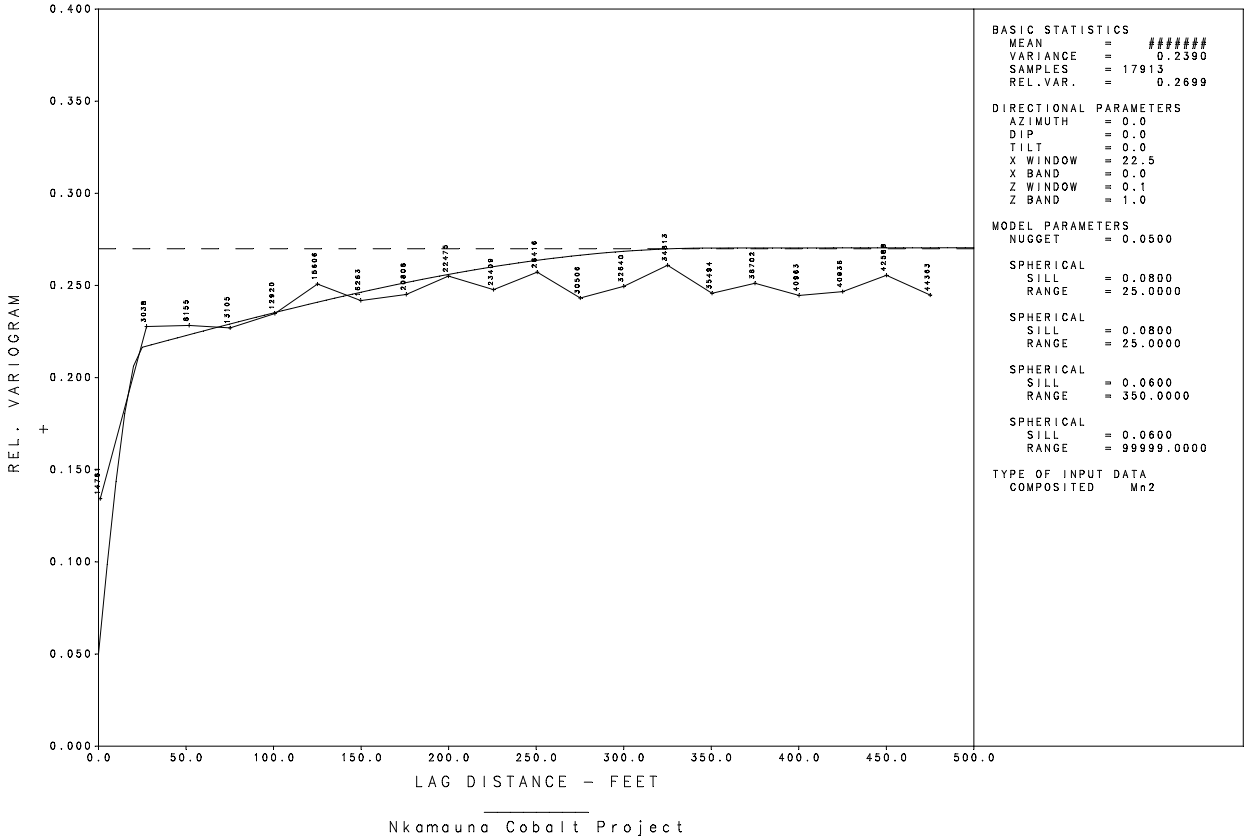
SPHERICAL
SILL = 0.0600
RANGE = 500.0000

SPHERICAL
SILL = 0.0600
RANGE = 99999.0000

TYPE OF INPUT DATA
COMPOSITED Mn2

Nkamauna Cobalt Project

Plot Variogram - Mineralized Manganese 20-Aug-09
Southeast



BASIC STATISTICS
MEAN = #######
VARIANCE = 0.2390
SAMPLES = 17913
REL.VAR. = 0.2699

DIRECTIONAL PARAMETERS
AZIMUTH = 0.0
DIP = 0.0
TILT = 0.0
X WINDOW = 22.5
X BAND = 0.0
Z WINDOW = 0.1
Z BAND = 1.0

MODEL PARAMETERS
NUGGET = 0.0500

SPHERICAL
SILL = 0.0800
RANGE = 25.0000

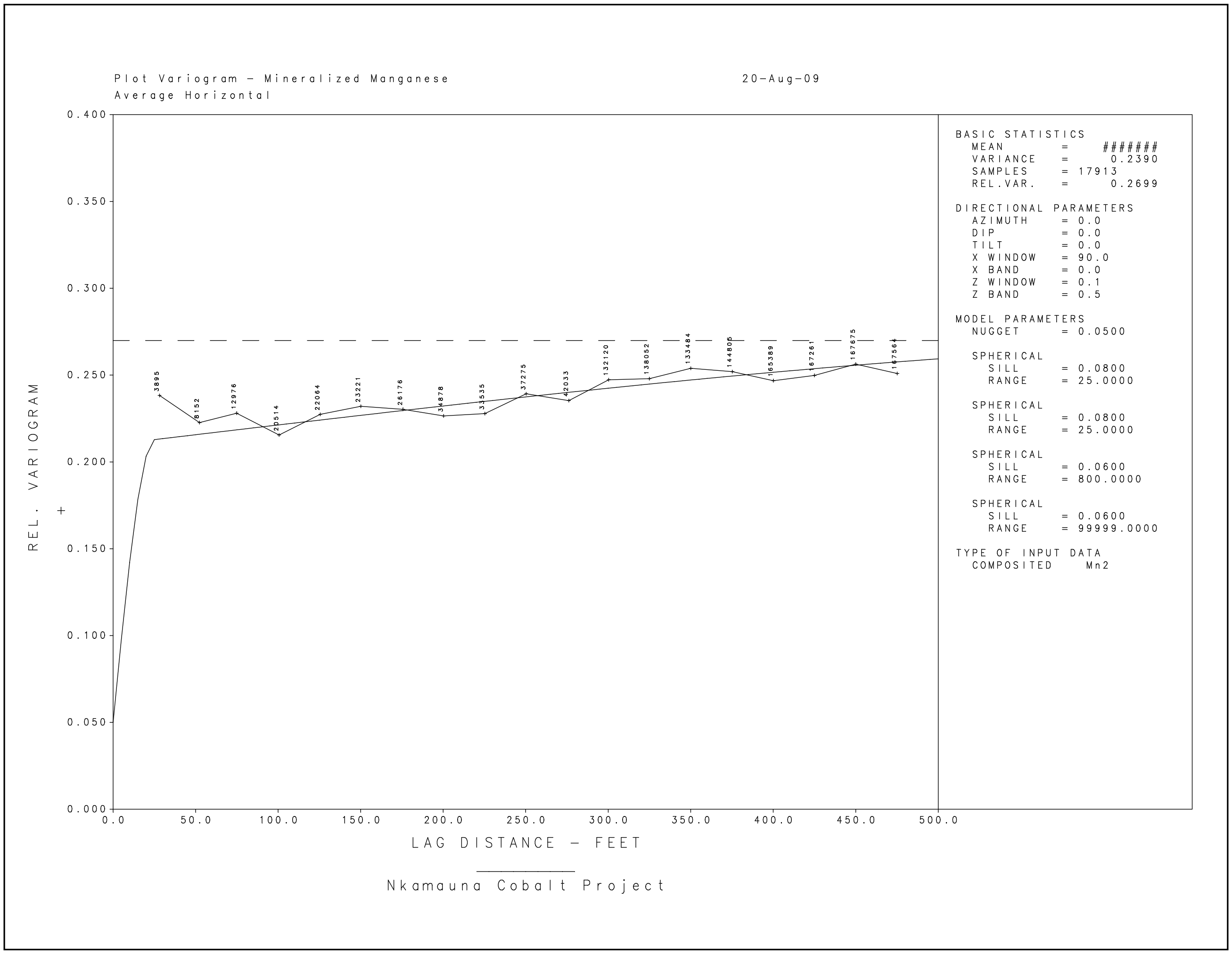
SPHERICAL
SILL = 0.0800
RANGE = 25.0000

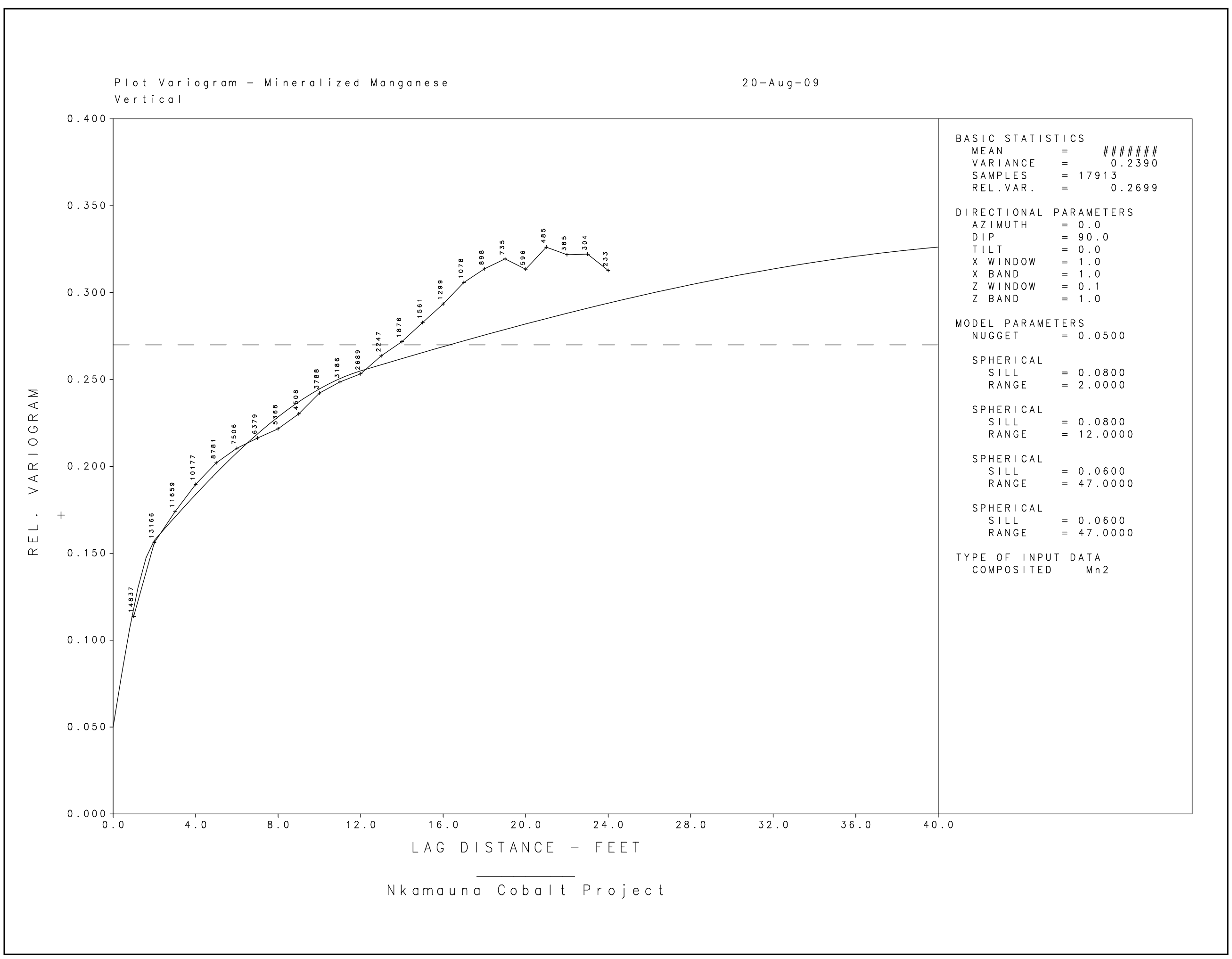
SPHERICAL
SILL = 0.0600
RANGE = 350.0000

SPHERICAL
SILL = 0.0600
RANGE = 99999.0000

TYPE OF INPUT DATA
COMPOSITED Mn2

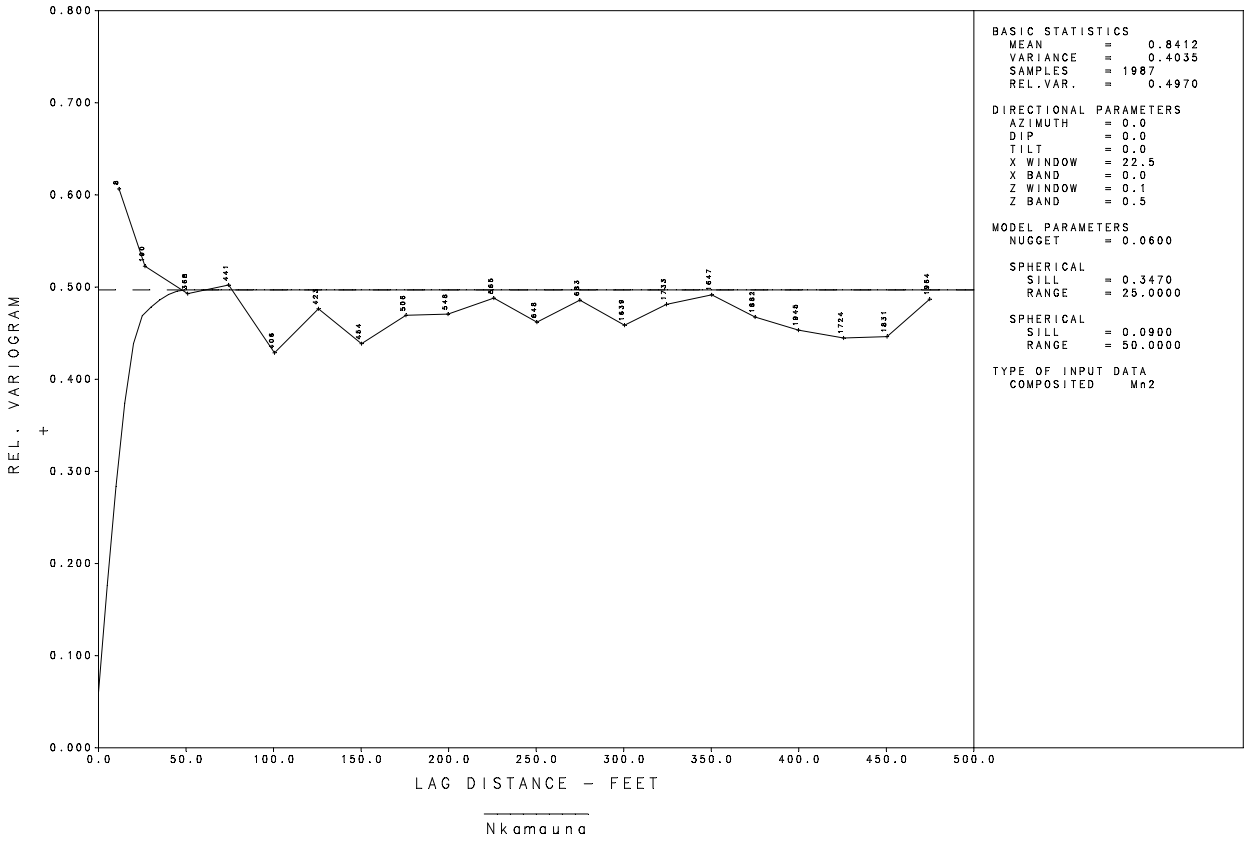
Nkamauna Cobalt Project


Plot Variogram - Mineralized Manganese
20-Aug-09
Average Horizontal
REL. VARIOGRAM
LAG DISTANCE - FEET
BASIC STATISTICS
MEAN = #######
VARIANCE = 0.2390
SAMPLES = 17913
REL.VAR. = 0.2699
DIRECTIONAL PARAMETERS
AZIMUTH = 0.0
DIP = 0.0
TILT = 0.0
X WINDOW = 90.0
X BAND = 0.0
Z WINDOW = 0.1
Z BAND = 0.5
MODEL PARAMETERS
NUGGET = 0.0500
SPHERICAL
SILL = 0.0800
RANGE = 25.0000
SPHERICAL
SILL = 0.0800
RANGE = 25.0000
SPHERICAL
SILL = 0.0600
RANGE = 800.0000
SPHERICAL
SILL = 0.0600
RANGE = 99999.0000
TYPE OF INPUT DATA
COMPOSITED Mn2
Nkamauna Cobalt Project


Plot Variogram - Mineralized Manganese
20-Aug-09
Vertical
REL. VARIOGRAM
LAG DISTANCE - FEET
BASIC STATISTICS
MEAN = #######
VARIANCE = 0.2390
SAMPLES = 17913
REL.VAR. = 0.2699
DIRECTIONAL PARAMETERS
AZIMUTH = 0.0
DIP = 90.0
TILT = 0.0
X WINDOW = 1.0
X BAND = 1.0
Z WINDOW = 0.1
Z BAND = 1.0
MODEL PARAMETERS
NUGGET = 0.0500
SPHERICAL
SILL = 0.0800
RANGE = 2.0000
SPHERICAL
SILL = 0.0800
RANGE = 12.0000
SPHERICAL
SILL = 0.0600
RANGE = 47.0000
SPHERICAL
SILL = 0.0600
RANGE = 47.0000
TYPE OF INPUT DATA
COMPOSITED Mn2
Nkamauna Cobalt Project

Plot Variogram - High Grade Manganese
North

20-Aug-09



BASIC STATISTICS
- MEAN = 0.8412
- VARIANCE = 0.4035
- SAMPLES = 1987
- REL.VAR. = 0.4970

DIRECTIONAL PARAMETERS
- AZIMUTH = 0.0
- DIP = 0.0
- TILT = 0.0
- X WINDOW = 22.5
- X BAND = 0.0
- Z WINDOW = 0.1
- Z BAND = 0.5

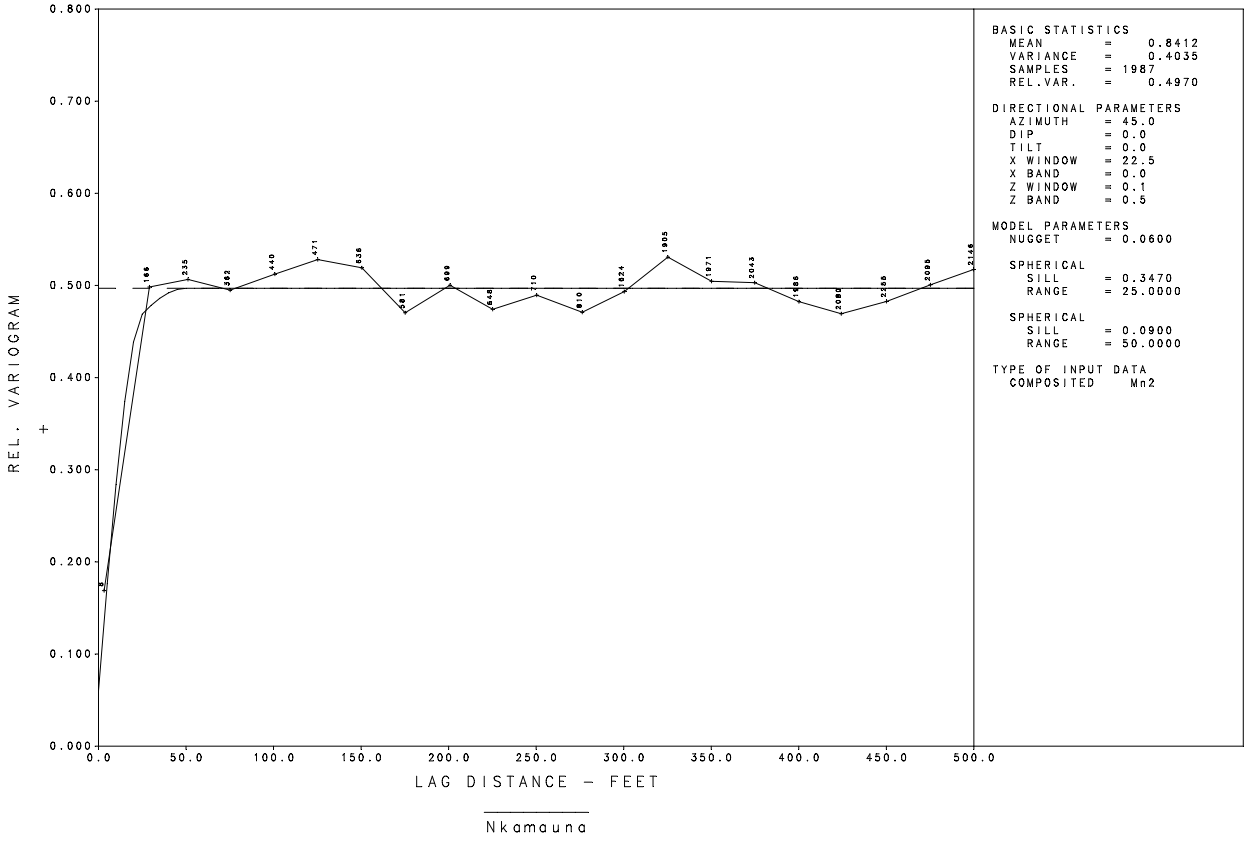
MODEL PARAMETERS
- NUGGET = 0.0600
- SPHERICAL
  - SILL = 0.3470
  - RANGE = 25.0000
- SPHERICAL
  - SILL = 0.0900
  - RANGE = 50.0000

TYPE OF INPUT DATA
- COMPOSITED Mn2

Nkamauna

Plot Variogram - High Grade Manganese
Northeast

20-Aug-09



BASIC STATISTICS
- MEAN = 0.8412
- VARIANCE = 0.4035
- SAMPLES = 1987
- REL.VAR. = 0.4970

DIRECTIONAL PARAMETERS
- AZIMUTH = 45.0
- DIP = 0.0
- TILT = 0.0
- X WINDOW = 22.5
- X BAND = 0.0
- Z WINDOW = 0.1
- Z BAND = 0.5

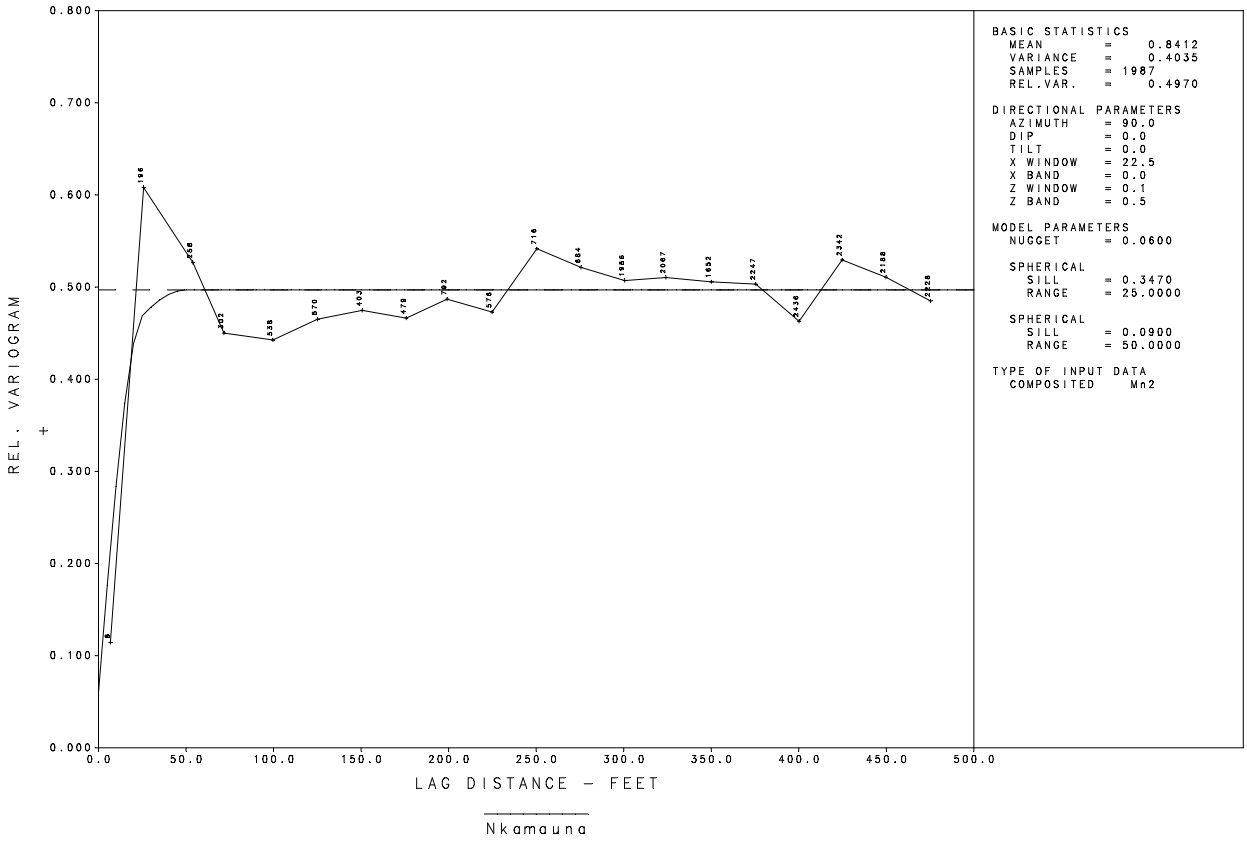
MODEL PARAMETERS
- NUGGET = 0.0600
- SPHERICAL
  - SILL = 0.3470
  - RANGE = 25.0000
- SPHERICAL
  - SILL = 0.0900
  - RANGE = 50.0000

TYPE OF INPUT DATA
- COMPOSITED Mn2

Nkamauna

Plot Variogram - High Grade Manganese
East

20-Aug-09



BASIC STATISTICS
- MEAN = 0.8412
- VARIANCE = 0.4035
- SAMPLES = 1987
- REL.VAR. = 0.4970

DIRECTIONAL PARAMETERS
- AZIMUTH = 90.0
- DIP = 0.0
- TILT = 0.0
- X WINDOW = 22.5
- X BAND = 0.0
- Z WINDOW = 0.1
- Z BAND = 0.5

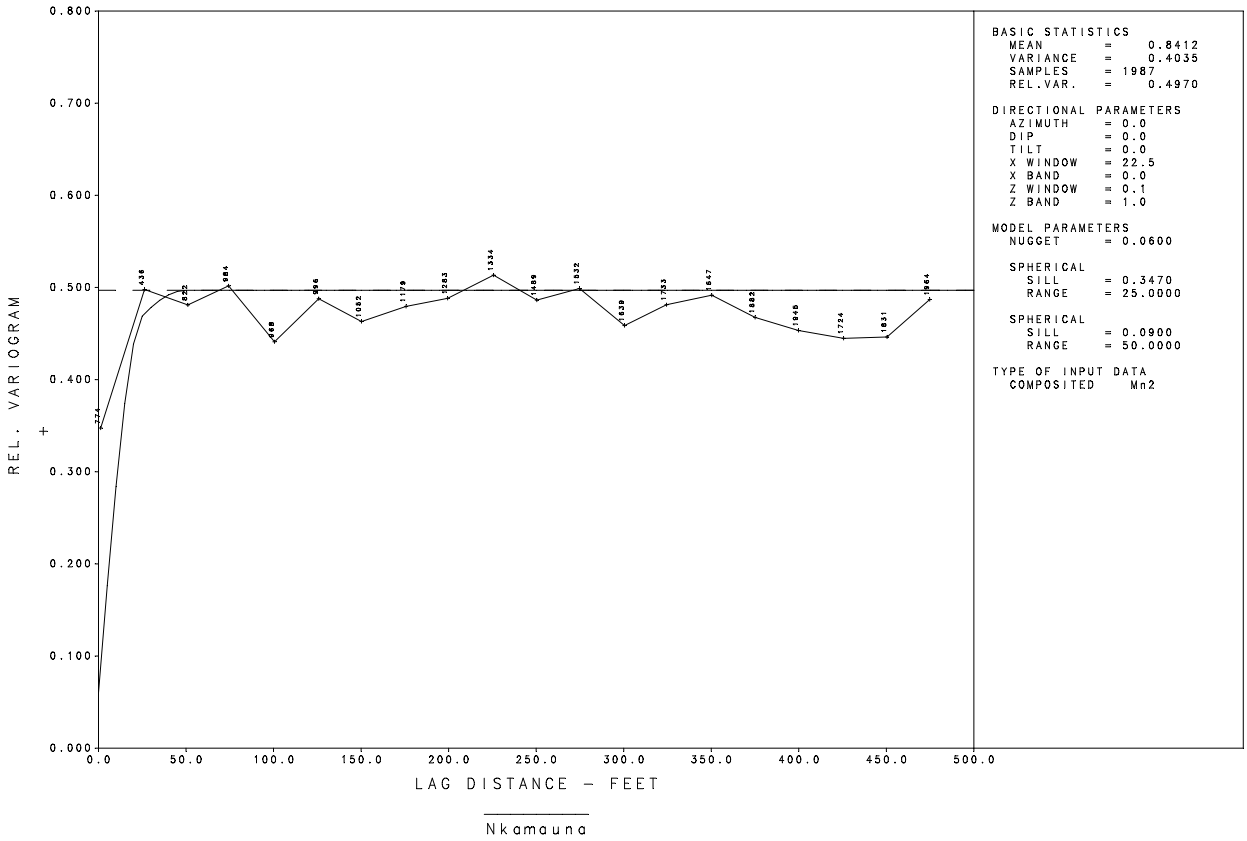
MODEL PARAMETERS
- NUGGET = 0.0600
- SPHERICAL
  - SILL = 0.3470
  - RANGE = 25.0000
- SPHERICAL
  - SILL = 0.0900
  - RANGE = 50.0000

TYPE OF INPUT DATA
- COMPOSITED Mn2

Nkamauna

Plot Variogram - High Grade Manganese
Southeast

20-Aug-09



BASIC STATISTICS
- MEAN = 0.8412
- VARIANCE = 0.4035
- SAMPLES = 1987
- REL.VAR. = 0.4970

DIRECTIONAL PARAMETERS
- AZIMUTH = 0.0
- DIP = 0.0
- TILT = 0.0
- X WINDOW = 22.5
- X BAND = 0.0
- Z WINDOW = 0.1
- Z BAND = 1.0

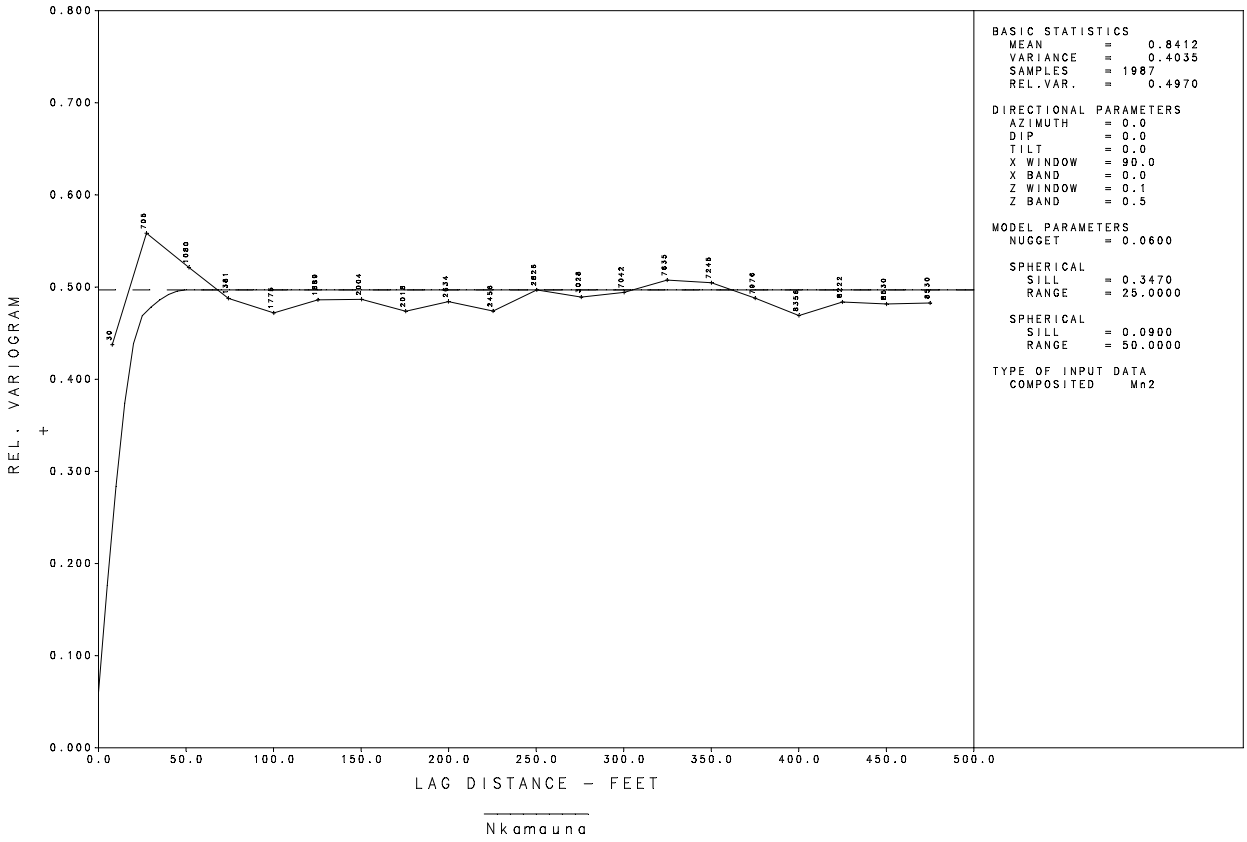
MODEL PARAMETERS
- NUGGET = 0.0600
- SPHERICAL
  - SILL = 0.3470
  - RANGE = 25.0000
- SPHERICAL
  - SILL = 0.0900
  - RANGE = 50.0000

TYPE OF INPUT DATA
- COMPOSITED Mn2

Nkamauna

Plot Variogram - High Grade Manganese
Average Horizontal

20-Aug-09



BASIC STATISTICS
MEAN = 0.8412
VARIANCE = 0.4035
SAMPLES = 1987
REL.VAR. = 0.4970

DIRECTIONAL PARAMETERS
AZIMUTH = 0.0
DIP = 0.0
TILT = 0.0
X WINDOW = 90.0
X BAND = 0.0
Z WINDOW = 0.1
Z BAND = 0.5

MODEL PARAMETERS
NUGGET = 0.0600

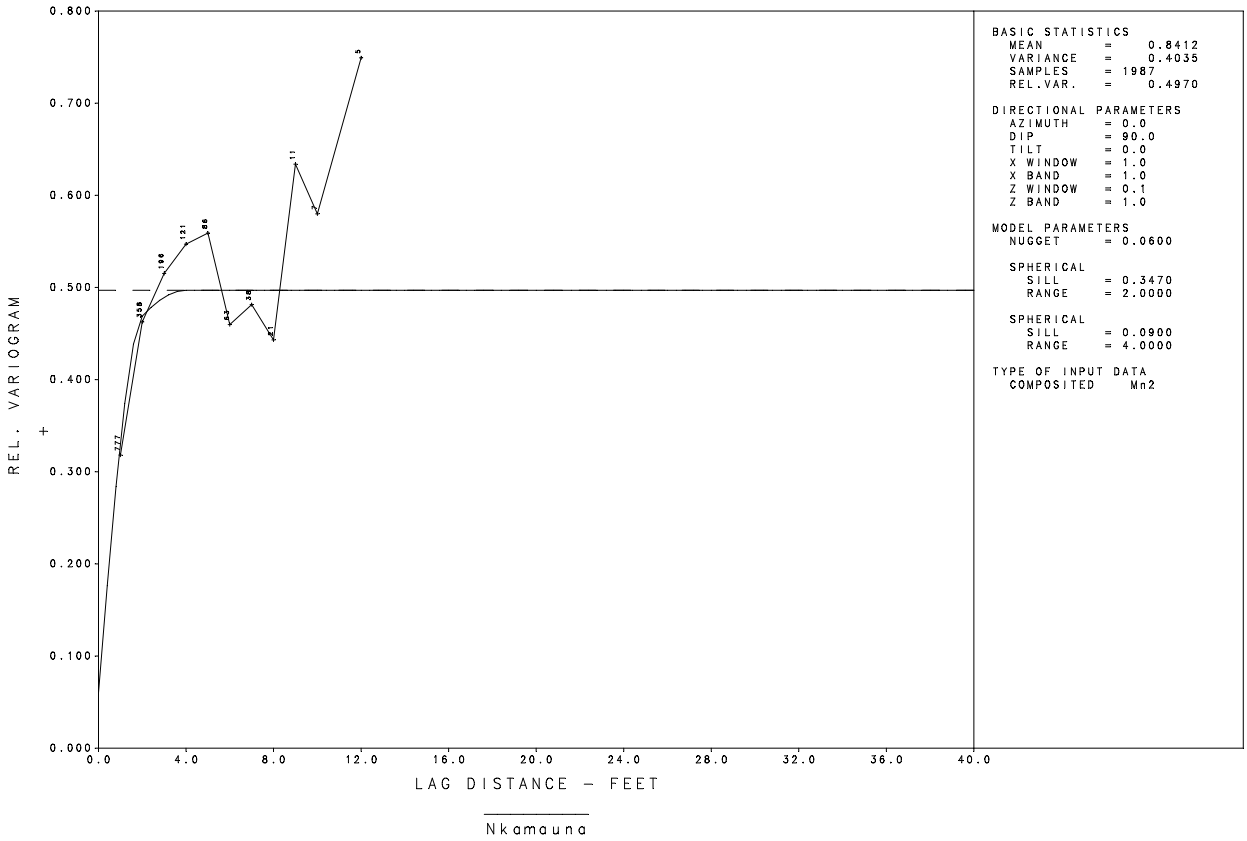
SPHERICAL
SILL = 0.3470
RANGE = 25.0000

SPHERICAL
SILL = 0.0900
RANGE = 50.0000

TYPE OF INPUT DATA
COMPOSITED Mn2

Plot Variogram - High Grade Manganese
Vertical

20-Aug-09



BASIC STATISTICS
MEAN = 0.8412
VARIANCE = 0.4035
SAMPLES = 1987
REL.VAR. = 0.4970

DIRECTIONAL PARAMETERS
AZIMUTH = 0.0
DIP = 90.0
TILT = 0.0
X WINDOW = 1.0
X BAND = 1.0
Z WINDOW = 0.1
Z BAND = 1.0

MODEL PARAMETERS
NUGGET = 0.0600

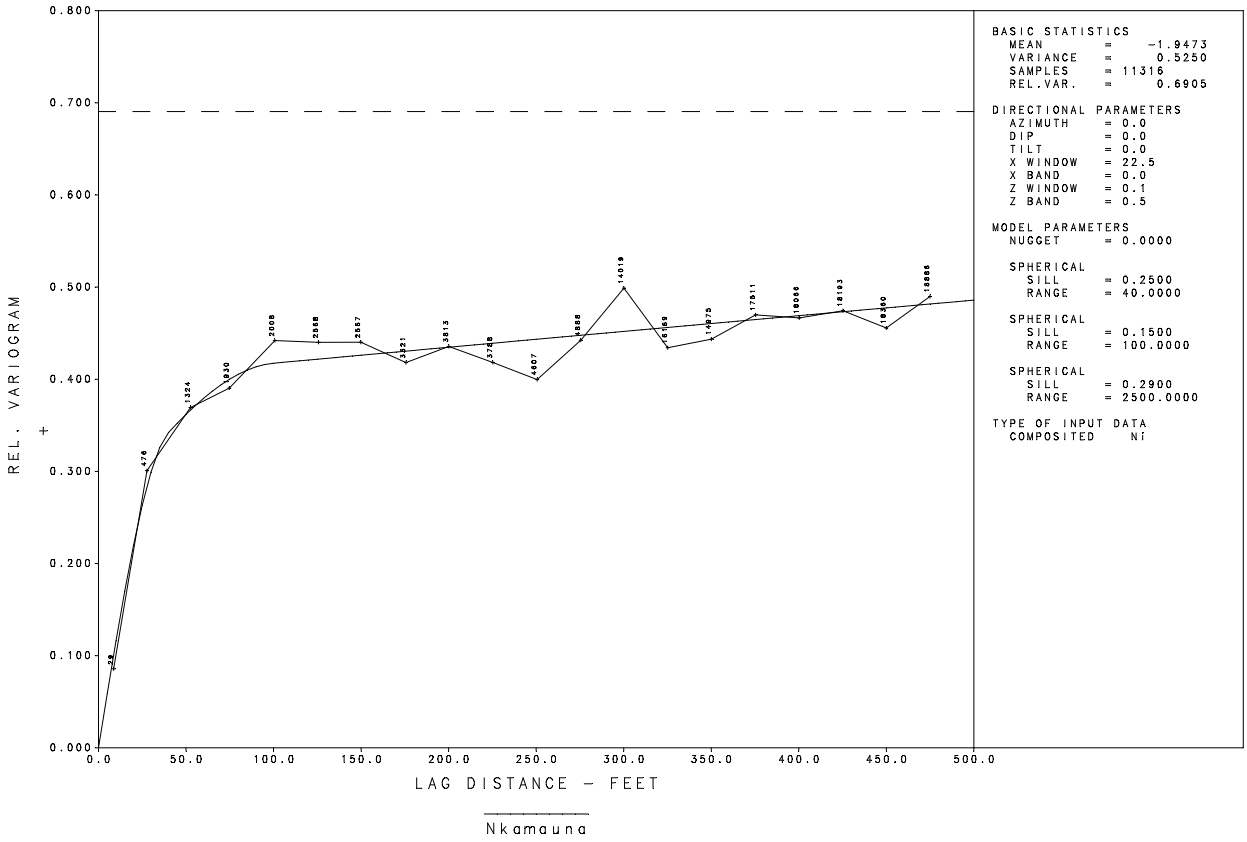
SPHERICAL
SILL = 0.3470
RANGE = 2.0000

SPHERICAL
SILL = 0.0900
RANGE = 4.0000

TYPE OF INPUT DATA
COMPOSITED Mn2

Plot Variogram - LG Nickel
North

20-Aug-09



BASIC STATISTICS
- MEAN = -1.9473
- VARIANCE = 0.5250
- SAMPLES = 11316
- REL.VAR. = 0.6905

DIRECTIONAL PARAMETERS
- AZIMUTH = 0.0
- DIP = 0.0
- TILT = 0.0
- X WINDOW = 22.5
- X BAND = 0.0
- Z WINDOW = 0.1
- Z BAND = 0.5

MODEL PARAMETERS
- NUGGET = 0.0000

SPHERICAL
- SILL = 0.2500
- RANGE = 40.0000

SPHERICAL
- SILL = 0.1500
- RANGE = 100.0000

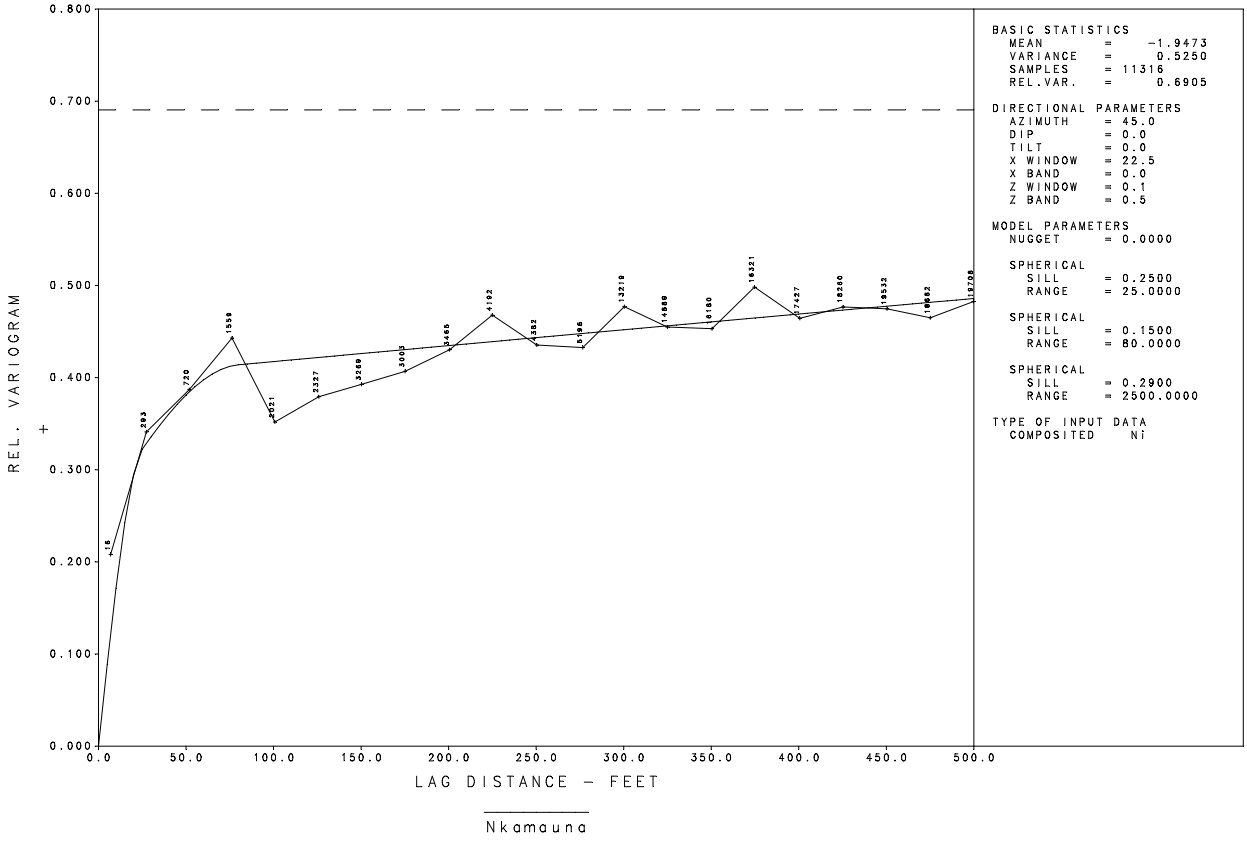
SPHERICAL
- SILL = 0.2900
- RANGE = 2500.0000

TYPE OF INPUT DATA
- COMPOSITED Ni

Nkamauna

Plot Variogram - LG Nickel
Northeast

20-Aug-09



BASIC STATISTICS
- MEAN = -1.9473
- VARIANCE = 0.5250
- SAMPLES = 11316
- REL.VAR. = 0.6905

DIRECTIONAL PARAMETERS
- AZIMUTH = 45.0
- DIP = 0.0
- TILT = 0.0
- X WINDOW = 22.5
- X BAND = 0.0
- Z WINDOW = 0.1
- Z BAND = 0.5

MODEL PARAMETERS
- NUGGET = 0.0000

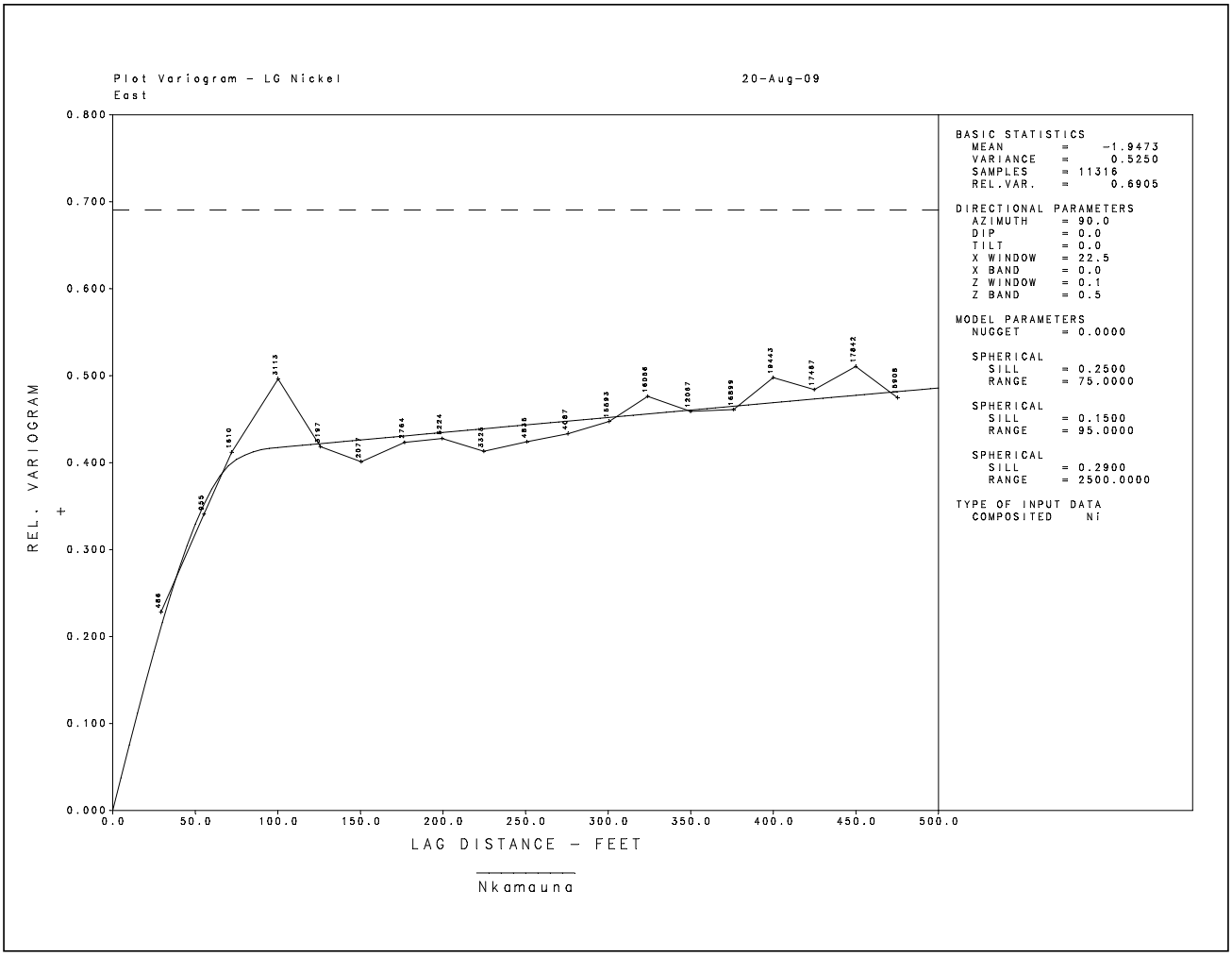
SPHERICAL
- SILL = 0.2500
- RANGE = 25.0000

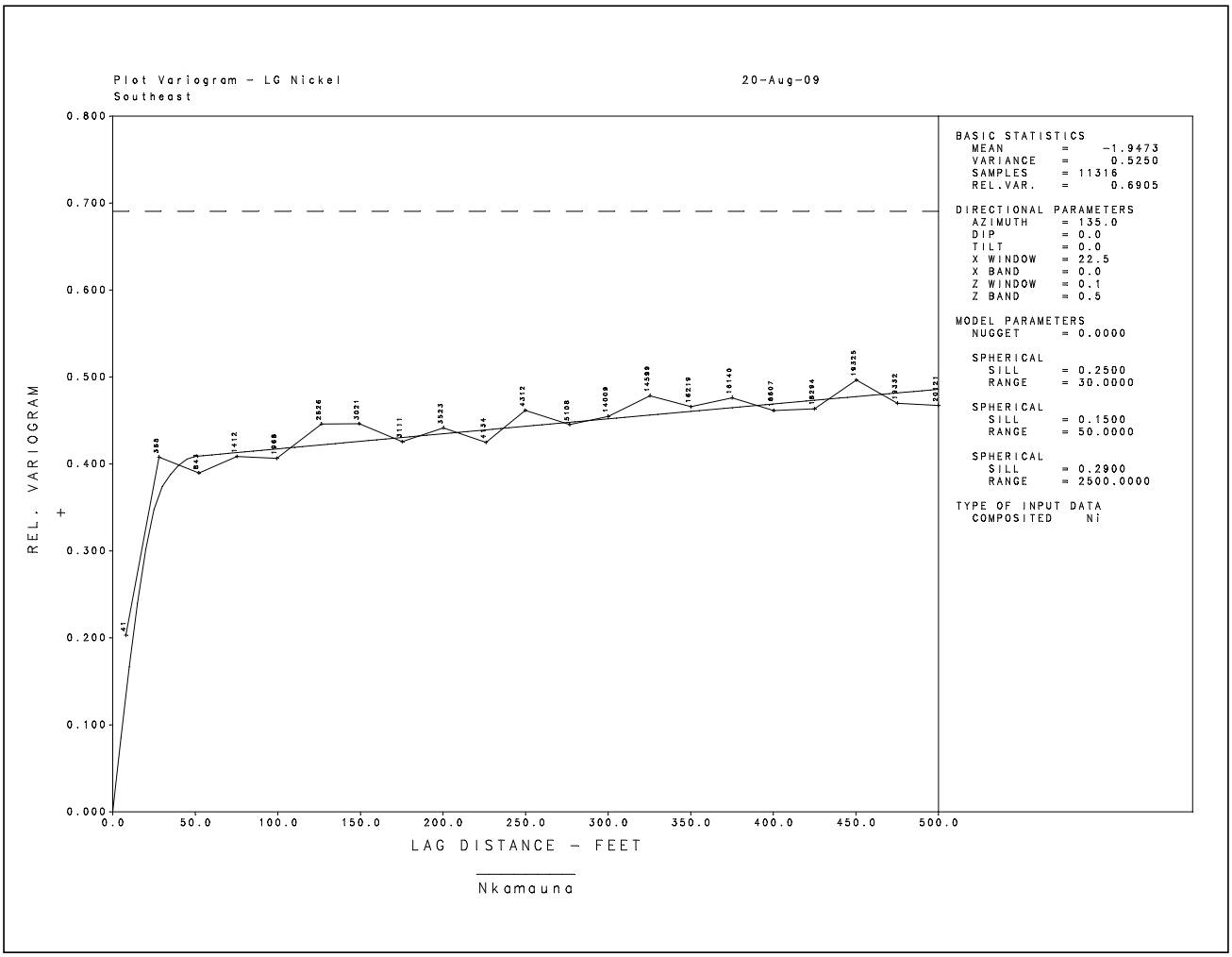
SPHERICAL
- SILL = 0.1500
- RANGE = 80.0000

SPHERICAL
- SILL = 0.2900
- RANGE = 2500.0000

TYPE OF INPUT DATA
- COMPOSITED Ni

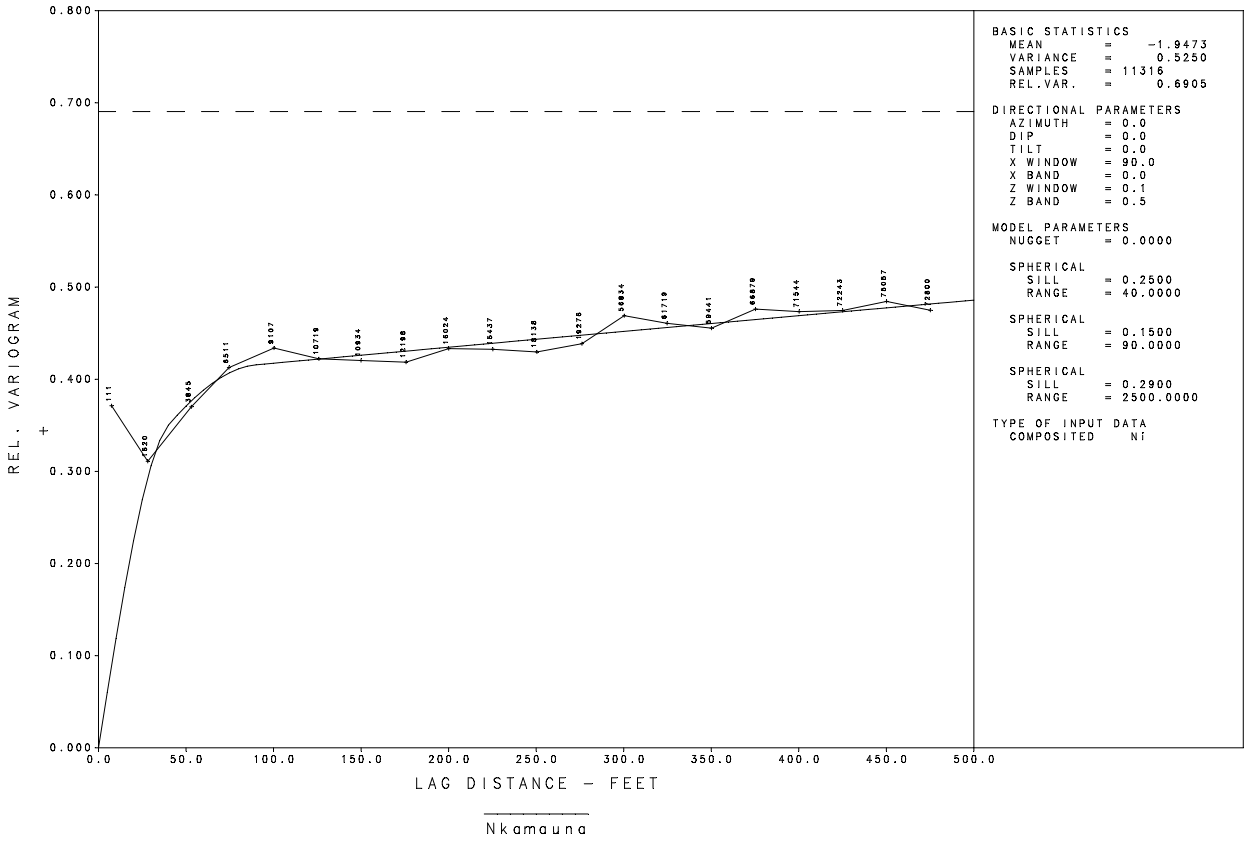
Nkamauna


Plot Variogram - LG Nickel
East
20-Aug-09
REL. VARIOGRAM
LAG DISTANCE - FEET
0.000
0.100
0.200
0.300
0.400
0.500
0.600
0.700
0.800
0.0
50.0
100.0
150.0
200.0
250.0
300.0
350.0
400.0
450.0
500.0
BASIC STATISTICS
MEAN = -1.9473
VARIANCE = 0.5250
SAMPLES = 11316
REL.VAR. = 0.6905
DIRECTIONAL PARAMETERS
AZIMUTH = 90.0
DIP = 0.0
TILT = 0.0
X WINDOW = 22.5
X BAND = 0.0
Z WINDOW = 0.1
Z BAND = 0.5
MODEL PARAMETERS
NUGGET = 0.0000
SPHERICAL
SILL = 0.2500
RANGE = 75.0000
SPHERICAL
SILL = 0.1500
RANGE = 95.0000
SPHERICAL
SILL = 0.2900
RANGE = 2500.0000
TYPE OF INPUT DATA
COMPOSITED Ni
Nkamauna


Plot Variogram - LG Nickel
Southeast
20-Aug-09
REL. VARIOGRAM
LAG DISTANCE - FEET
0.000
0.100
0.200
0.300
0.400
0.500
0.600
0.700
0.800
0.0
50.0
100.0
150.0
200.0
250.0
300.0
350.0
400.0
450.0
500.0
BASIC STATISTICS
MEAN = -1.9473
VARIANCE = 0.5250
SAMPLES = 11316
REL.VAR. = 0.6905
DIRECTIONAL PARAMETERS
AZIMUTH = 135.0
DIP = 0.0
TILT = 0.0
X WINDOW = 22.5
X BAND = 0.0
Z WINDOW = 0.1
Z BAND = 0.5
MODEL PARAMETERS
NUGGET = 0.0000
SPHERICAL
SILL = 0.2500
RANGE = 30.0000
SPHERICAL
SILL = 0.1500
RANGE = 50.0000
SPHERICAL
SILL = 0.2900
RANGE = 2500.0000
TYPE OF INPUT DATA
COMPOSITED Ni
Nkamauna

Plot Variogram - LG Nickel
Average Horizontal

20-Aug-09



BASIC STATISTICS
MEAN = -1.9473
VARIANCE = 0.5250
SAMPLES = 11316
REL.VAR. = 0.6905

DIRECTIONAL PARAMETERS
AZIMUTH = 0.0
DIP = 0.0
TILT = 0.0
X WINDOW = 90.0
X BAND = 0.0
Z WINDOW = 0.1
Z BAND = 0.5

MODEL PARAMETERS
NUGGET = 0.0000

SPHERICAL
SILL = 0.2500
RANGE = 40.0000

SPHERICAL
SILL = 0.1500
RANGE = 90.0000

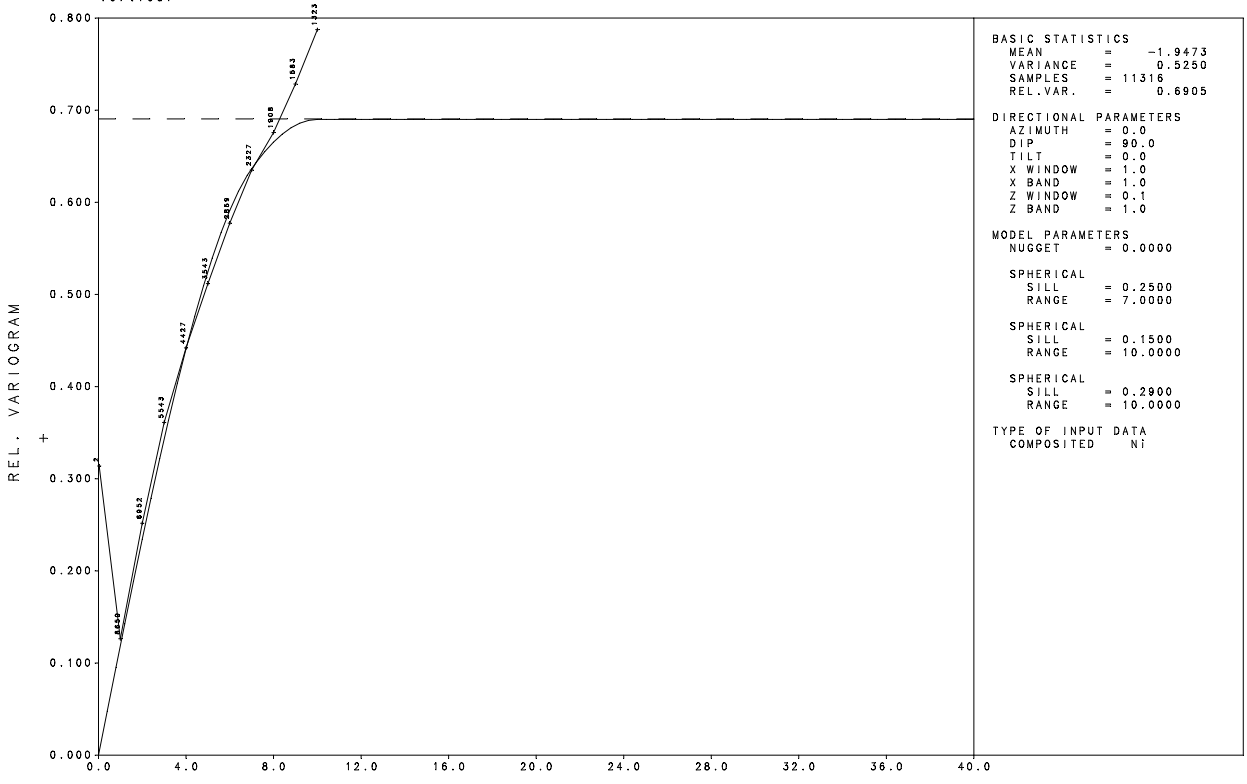
SPHERICAL
SILL = 0.2900
RANGE = 2500.0000

TYPE OF INPUT DATA
COMPOSITED Ni

Nkamauna

Plot Variogram - LG Nickel
Vertical

20-Aug-09



BASIC STATISTICS
MEAN = -1.9473
VARIANCE = 0.5250
SAMPLES = 11316
REL.VAR. = 0.6905

DIRECTIONAL PARAMETERS
AZIMUTH = 0.0
DIP = 90.0
TILT = 0.0
X WINDOW = 1.0
X BAND = 1.0
Z WINDOW = 0.1
Z BAND = 1.0

MODEL PARAMETERS
NUGGET = 0.0000

SPHERICAL
SILL = 0.2500
RANGE = 7.0000

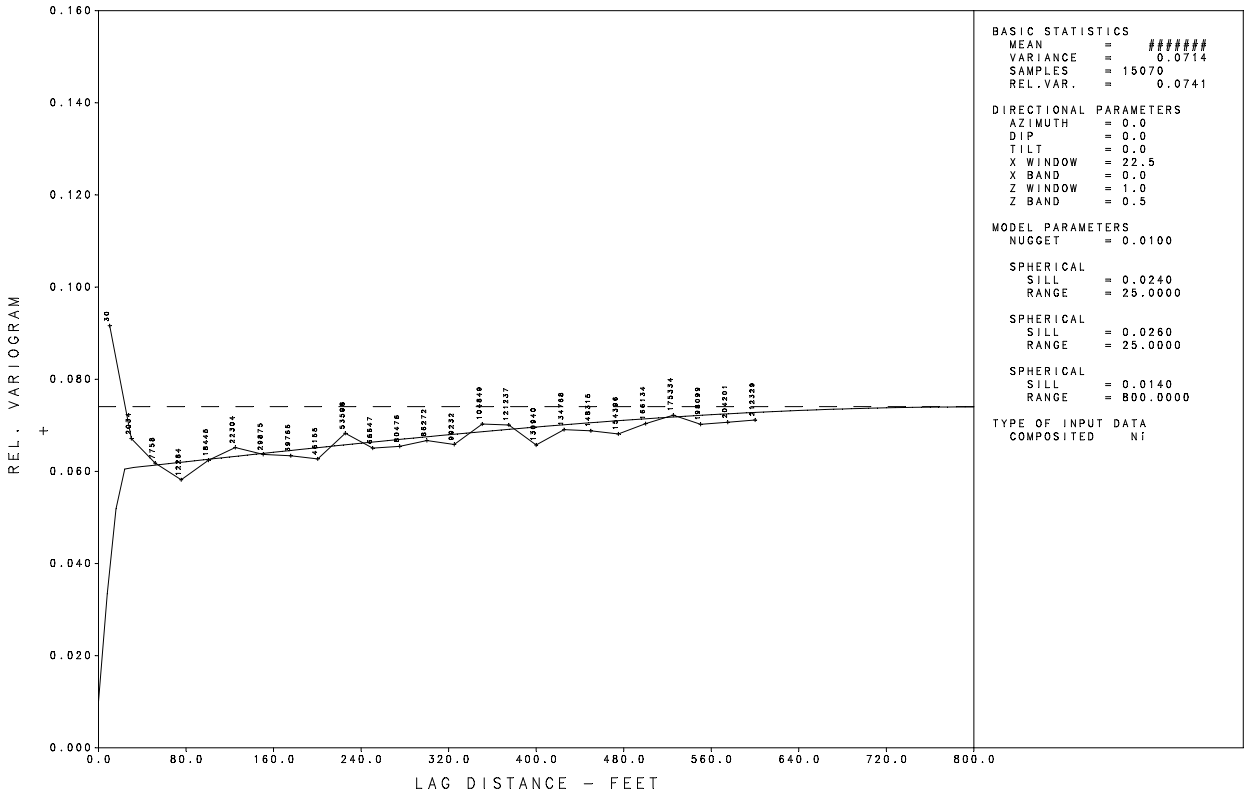
SPHERICAL
SILL = 0.1500
RANGE = 10.0000

SPHERICAL
SILL = 0.2900
RANGE = 10.0000

TYPE OF INPUT DATA
COMPOSITED Ni

Nkamauna

Plot Variogram - Mineralized Nickel 12-Aug-09
North



BASIC STATISTICS
MEAN = #######
VARIANCE = 0.0714
SAMPLES = 15070
REL.VAR. = 0.0741

DIRECTIONAL PARAMETERS
AZIMUTH = 0.0
DIP = 0.0
TILT = 0.0
X WINDOW = 22.5
X BAND = 0.0
Z WINDOW = 1.0
Z BAND = 0.5

MODEL PARAMETERS
NUGGET = 0.0100

SPHERICAL
SILL = 0.0240
RANGE = 25.0000

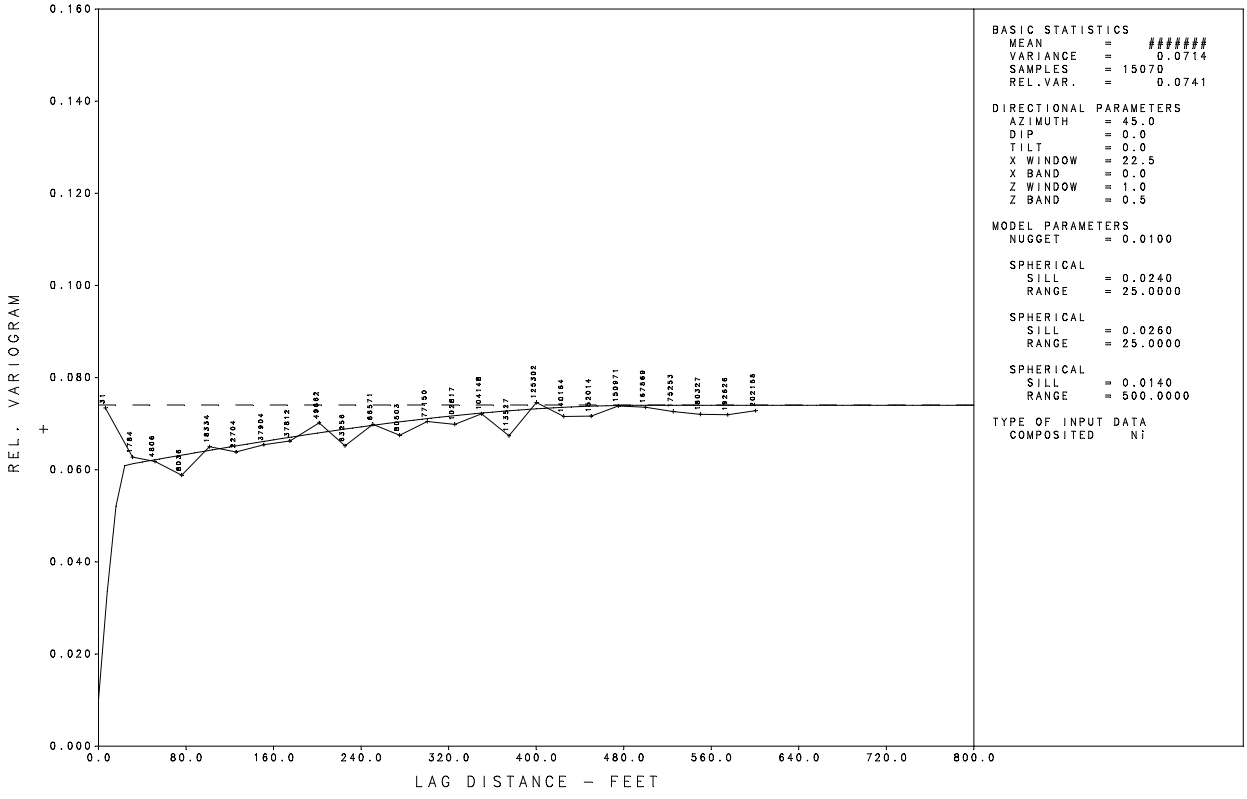
SPHERICAL
SILL = 0.0260
RANGE = 25.0000

SPHERICAL
SILL = 0.0140
RANGE = 800.0000

TYPE OF INPUT DATA
COMPOSITED Ni

Nkamauna

Plot Variogram - Mineralized Nickel 12-Aug-09
Northeast



BASIC STATISTICS
MEAN = #######
VARIANCE = 0.0714
SAMPLES = 15070
REL.VAR. = 0.0741

DIRECTIONAL PARAMETERS
AZIMUTH = 45.0
DIP = 0.0
TILT = 0.0
X WINDOW = 22.5
X BAND = 0.0
Z WINDOW = 1.0
Z BAND = 0.5

MODEL PARAMETERS
NUGGET = 0.0100

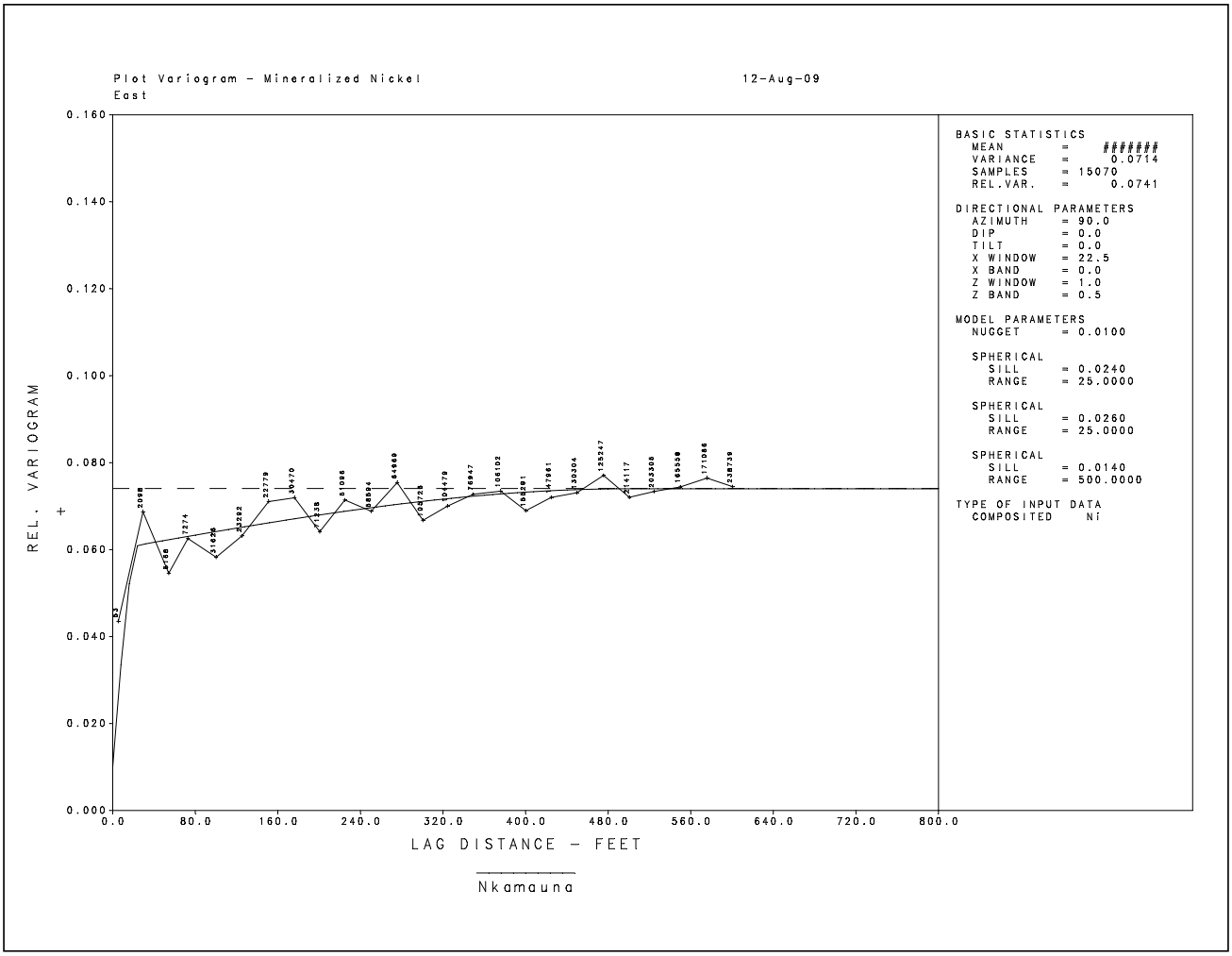
SPHERICAL
SILL = 0.0240
RANGE = 25.0000

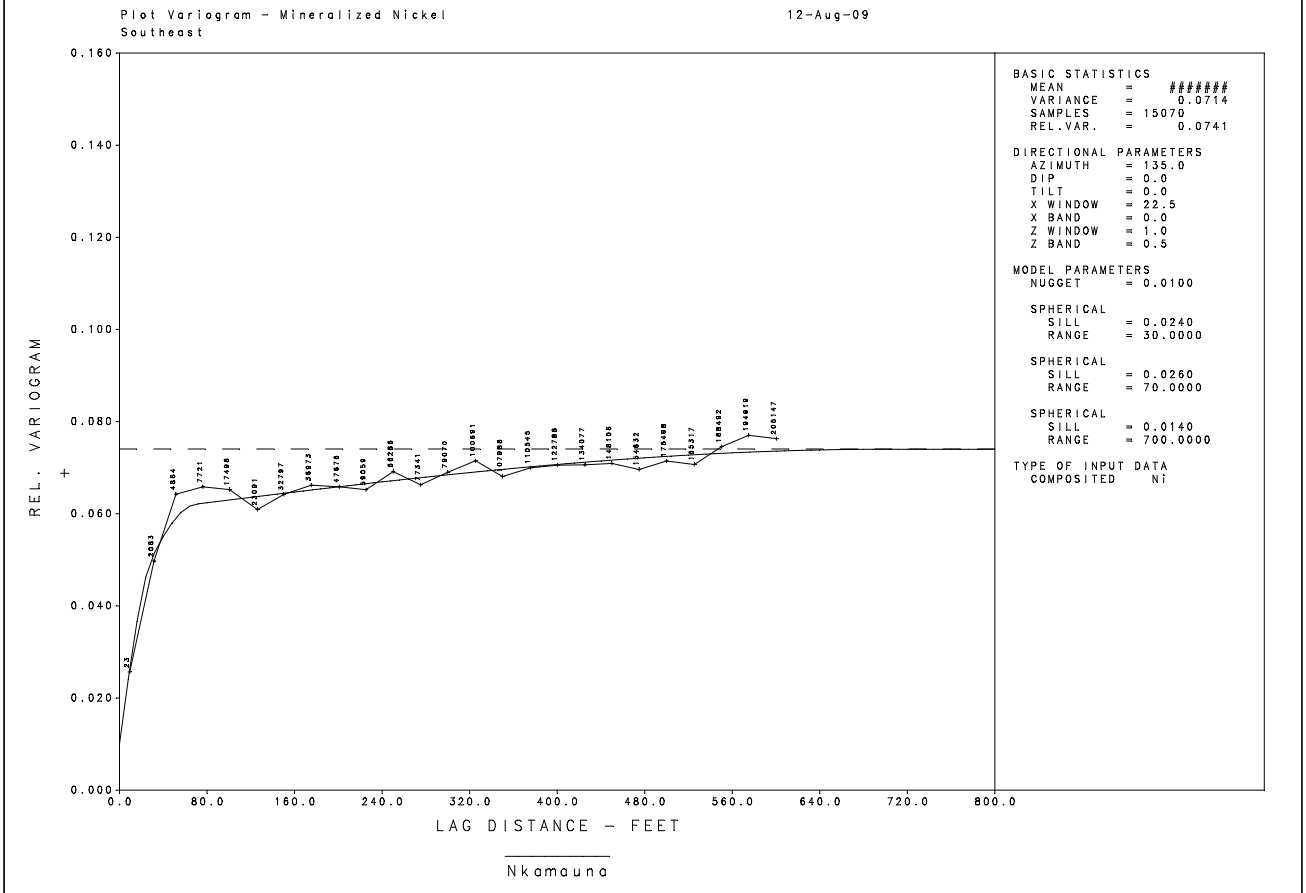
SPHERICAL
SILL = 0.0260
RANGE = 25.0000

SPHERICAL
SILL = 0.0140
RANGE = 500.0000

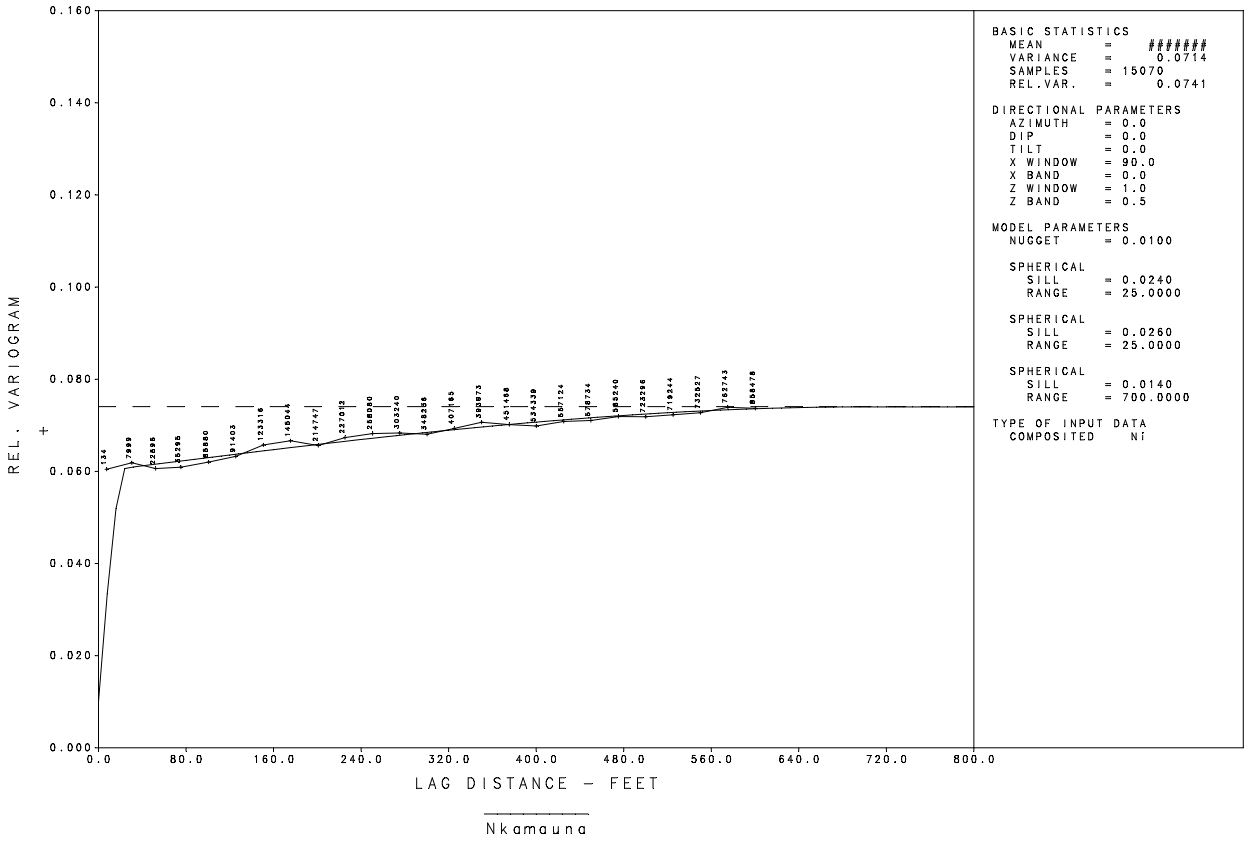
TYPE OF INPUT DATA
COMPOSITED Ni

Nkamauna


Plot Variogram - Mineralized Nickel
East
12-Aug-09
REL. VARIOGRAM
LAG DISTANCE - FEET
Nkamauna
BASIC STATISTICS
MEAN = #######
VARIANCE = 0.0714
SAMPLES = 15070
REL.VAR. = 0.0741
DIRECTIONAL PARAMETERS
AZIMUTH = 90.0
DIP = 0.0
TILT = 0.0
X WINDOW = 22.5
X BAND = 0.0
Z WINDOW = 1.0
Z BAND = 0.5
MODEL PARAMETERS
NUGGET = 0.0100
SPHERICAL
SILL = 0.0240
RANGE = 25.0000
SPHERICAL
SILL = 0.0260
RANGE = 25.0000
SPHERICAL
SILL = 0.0140
RANGE = 500.0000
TYPE OF INPUT DATA
COMPOSITED Ni


Plot Variogram - Mineralized Nickel
Southeast
12-Aug-09
REL. VARIOGRAM
LAG DISTANCE - FEET
Nkamauna
BASIC STATISTICS
MEAN = #######
VARIANCE = 0.0714
SAMPLES = 15070
REL.VAR. = 0.0741
DIRECTIONAL PARAMETERS
AZIMUTH = 135.0
DIP = 0.0
TILT = 0.0
X WINDOW = 22.5
X BAND = 0.0
Z WINDOW = 1.0
Z BAND = 0.5
MODEL PARAMETERS
NUGGET = 0.0100
SPHERICAL
SILL = 0.0240
RANGE = 30.0000
SPHERICAL
SILL = 0.0260
RANGE = 70.0000
SPHERICAL
SILL = 0.0140
RANGE = 700.0000
TYPE OF INPUT DATA
COMPOSITED Ni

Plot Variogram - Mineralized Nickel 12-Aug-09
Average Horizontal



BASIC STATISTICS
MEAN = #######
VARIANCE = 0.0714
SAMPLES = 15070
REL.VAR. = 0.0741

DIRECTIONAL PARAMETERS
AZIMUTH = 0.0
DIP = 0.0
TILT = 0.0
X WINDOW = 90.0
X BAND = 0.0
Z WINDOW = 1.0
Z BAND = 0.5

MODEL PARAMETERS
NUGGET = 0.0100

SPHERICAL
SILL = 0.0240
RANGE = 25.0000

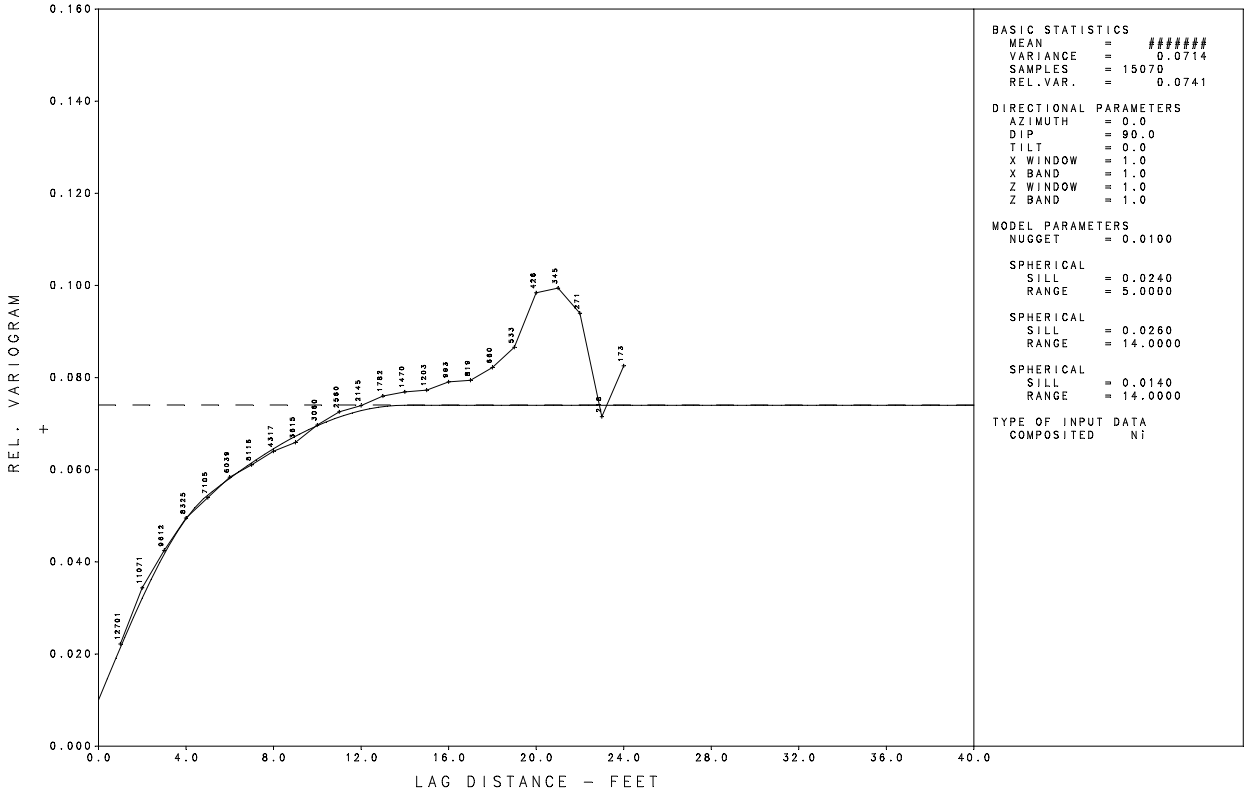
SPHERICAL
SILL = 0.0260
RANGE = 25.0000

SPHERICAL
SILL = 0.0140
RANGE = 700.0000

TYPE OF INPUT DATA
COMPOSITED Ni

Nkamauna

Plot Variogram - Mineralized Nickel 12-Aug-09
Vertical



BASIC STATISTICS
MEAN = #######
VARIANCE = 0.0714
SAMPLES = 15070
REL.VAR. = 0.0741

DIRECTIONAL PARAMETERS
AZIMUTH = 0.0
DIP = 90.0
TILT = 0.0
X WINDOW = 1.0
X BAND = 1.0
Z WINDOW = 1.0
Z BAND = 1.0

MODEL PARAMETERS
NUGGET = 0.0100

SPHERICAL
SILL = 0.0240
RANGE = 5.0000

SPHERICAL
SILL = 0.0260
RANGE = 14.0000

SPHERICAL
SILL = 0.0140
RANGE = 14.0000

TYPE OF INPUT DATA
COMPOSITED Ni

Nkamauna

Item 24

NI 43-101 Technical Report, Geovic Mining Corp., Nkamouna and Mada Deposits, East Province of Cameroon, Africa, (Effective Date: December 31, 2010)

Dated this 2nd Day of June, 2011.

"Signed"

Jeffrey Volk, CPG, FAusIMM, MSc

"Signed"

Brett Malcolm Crossley, BAS (Chemistry), MAusIMM

"Signed"

Brett Swanson, BE Mining, MAusIMM