# GORO NICKEL-COBALT PROJECT

# LOCATED IN THE FRENCH OVERSEAS TERRITORIAL COMMUNITY

# (COLLECTIVITÉ TERRITORIALE) OF NEW CALEDONIA

## TECHNICAL REPORT

### PURSUANT TO NATIONAL INSTRUMENT 43-101 OF THE CANADIAN SECURITIES ADMINISTRATORS


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Toronto, Canada

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| Dr. Lawrence B. Cochrane | Director, Mines Exploration, Inco Technical Services Limited |
| Stuart F. Feiner | Executive Vice-President, General Counsel & Secretary, Inco Limited |
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## 3. SUMMARY

This technical report (the "Report") is intended to cover the 85% owned Goro nickel-cobalt project (the "Goro Project") of Inco Limited ("Inco") and meet the requirements of National Instrument 43-101, "Standards of Disclosure for Mineral Projects" and Form 43-101F1, Technical Report of the Canadian Securities Administrators. Unless otherwise indicated, all dollar amounts are in U.S. dollars.

Goro Nickel, S.A. ("Goro Nickel"), an indirect subsidiary of Inco and the project company for the Goro Project, intends to develop fully integrated facilities for the mining, processing and sale of commercial products containing nickel and cobalt from deposits in the French Overseas Territorial Community (*Collectivité Territoriale*) of New Caledonia ("New Caledonia"). The Goro Project is currently planned to produce 55,800 metric tonnes of nickel as nickel oxide (78% nickel) and 4,500 metric tonnes of cobalt as cobalt carbonate (47% cobalt) annually. This Report is intended to cover the initial 14 year life of the Goro Project, based upon only the current mineral reserve estimate as of December 31, 2002. However, mineral resources contained within the Goro Project concessions are considered to be sufficient for a substantially longer project operating life and are expected to provide potential feed for future expansions.

Goro Nickel was incorporated in 1992 to develop and commercialize certain nickel and cobalt deposits in New Caledonia which had been identified by Inco and other companies (reference is made to Item 8, "History" of this Report) (the "Goro Deposits"). It is a French société anonyme with two principal shareholders. At present, 85% of Goro Nickel's shares are controlled by Inco. As noted in Item 8 of this Report, Inco has been involved in various mining and exploration projects in New Caledonia for close to 100 years. The other 15% of Goro Nickel is owned by Selani, a société anonyme incorporated under the laws of France and a wholly owned subsidiary of Bureau de Recherches Géologiques et Minières ("BRGM"), a French government agency.

During 2002, Inco held negotiations with a Japanese consortium to be led by Sumitomo Metal Mining Co., Ltd. concerning the consortium's continuing interest in acquiring a 25% interest in the Goro Project. Given the review of the Goro Project currently being undertaken (reference is made to "Review of the Goro Project" below), formal negotiations with this consortium were suspended in late 2002 pending the completion of the review, but Inco has continued to have discussions with the Japanese consortium regarding their continued interest in this acquisition. Inco and Goro Nickel have also requested the deferral of the $350 million in tax advantaged *Paul Act* financing for the Goro Project to which the French authorities had granted their consent in principle in 2002. While Inco and Goro Nickel believe that this financing will be available for the Goro Project, there can be no assurance that this request will be granted. During 2002, Inco also entered into negotiations covering the acquisition of BRGM's 15% interest in Goro Nickel and reached an agreement in principle with BRGM on such an acquisition. Given the review of the Goro Project referred to below, and that such an acquisition was linked to the Japanese consortium's acquisition of a 25% interest in Goro Nickel, these negotiations were suspended pending the completion of the Goro Project review. Inco also held discussions during 2002 with representatives of the Government of New Caledonia concerning the terms under which the government or an agency thereof would receive a 5% interest in Goro Nickel and have certain rights to acquire an additional 5% interest once the Goro Project had met certain financial returns for its principal shareholders.

The planned principal parts or locations of the Goro Project are expected to be:

- the mine and feed preparation plant together with tailings and overburden disposal and water storages located in the Rivière Kwe drainage basin;
- the process plant on a plateau to the east of Baie du Prony;
- accommodation complexes near the process plant; and
- the port on the eastern shore of Baie du Prony.

A system of roads, pipelines, power lines, phone communications, a conveyor and related infrastructure are expected to link these locations.

The Goro Deposits occur in "laterites". These are residual soils formed in hot, wet tropical climates from weathering of a rock called peridotite that contains low values of nickel. The hills surrounding the Goro Deposits consist of peridotite and are typically devoid of laterite. Laterites in the area of the Goro Project consist of five distinct layers that overlie the peridotite bedrock at depths varying between 20 and 90 metres.

Goro Nickel has undertaken drilling programs with a drill hole density sufficient to outline estimated measured and indicated mineral resources of 95.1 million tonnes with an average grade of 1.58% nickel and 0.13% cobalt and inferred mineral resources of 143.8 million tonnes with an average grade of 1.7% nickel and 0.12% cobalt. In addition to these mineral resources, the proven and probable mineral reserves are estimated to be 56.9 million tonnes with an average grade of 1.52% nickel and 0.12% cobalt. The mineral resources and mineral reserves are shown in Table 3.1 and Table 3.2.

**Table 3.1: Goro Deposits Estimated Mineral Resources**

| **Mineral Resources @ 1.20 % nickel cut-off, -50.8mm screening size** | | | | |
|---|---|---|---|---|
| **Classification** | **Tonnes Million** | **Ni %** | **Co %** | **Fe %** |
| Measured | 56.0 | 1.40 | 0.14 | 47.1 |
| Indicated | 39.1 | 1.84 | 0.12 | 31.4 |
| **Total** | **95.1** | **1.58** | **0.13** | **40.7** |
| **Inferred Mineral @ 1.20 % nickel cut-off, -50.8mm screening size** | | | | |
| Inferred (Krige) | 21.0 | 2.1 | 0.10 | 21.38 |
| Inferred (Nearest) | 122.8 | 1.6 | 0.13 | 39.70 |
| **Total** | **143.8** | **1.7** | **0.12** | **37.0** |

**Table 3.2: Goro Deposits Estimated Mineral Reserves**

| Mineral Reserves @ 1.20 % nickel cut-off, -50.8mm. screening size | | | | |
|---|---|---|---|---|
| Classification | Tonnes Million | Ni % | Co % | Fe % |
| Proven | 43.9 | 1.41 | 0.13 | 46.4 |
| Probable | 13.0 | 1.92 | 0.08 | 17.0 |
| **Total** | **56.9** | **1.52** | **0.12** | **39.6** |

**Review of the Goro Project**

In September 2002, at the time the Goro Project was experiencing certain labour disruptions, Goro Nickel and Inco initiated an update of the status of certain key aspects of the Goro Project, including the necessary permitting, capital cost estimate, schedule and organization. Work over the September – November 2002 period on certain critical parts of the Goro Project, including engineering, continued during this update process. On December 5, 2002, Goro Nickel and Inco announced that they would be undertaking a comprehensive review of the Goro Project. This action had been based upon information from the Goro Project's principal firms providing project engineering, procurement and construction management services that Goro Nickel and Inco had received that, if confirmed, would indicate an increase in the capital cost for the Goro Project in the range of 30 to 45 per cent above the then current capital cost estimate of $1,450 million. The objective of this comprehensive review is to assess all information on the Goro Project, including the various cost estimates and trends, and determine what changes in the capital cost estimate and the Goro Project can be made to maintain the Goro Project's economic feasibility. The review of the capital cost estimate will cover what downward adjustments can be made in such estimate through scope or design changes, modifications to construction and related plans and civil and other contractual arrangements, and alternative project execution strategies. Since that announcement, Goro Nickel and Inco have been evaluating what onsite and offsite work should be curtailed or stopped and what work should be continued while this review is ongoing. Based upon this ongoing evaluation, Goro Nickel and Inco have also been reviewing various contractual and other arrangements covering construction and other work relating to the Goro Project and implementing certain actions to suspend or terminate certain of those contractual arrangements.

Since this review process is still in its preliminary stages given its planned scope, Goro Nickel and Inco do not currently expect to be in a position to report on the results of this review, including an updated capital cost estimate for the Goro Project and the additional effect, if any, that this review could have on the timing and scope of the Goro Project, until at least the end of the second quarter or early in the third quarter of 2003. Goro Nickel and Inco have been working with various parties to assist in the review process. As discussed above, the key objective of this comprehensive review is to implement such actions and steps, if required, to enable the Goro Project to meet an acceptable rate of return on the investment made and to be made.

The estimate of mineral reserves included in this Report for the Goro Project has been prepared based on certain information updated from that contained in the March 2001 *Goro Nickel Project Bankable Feasibility Study* prepared by Hatch Associates (the "BFS"), including with respect to capital costs. The capital cost estimate for the Goro Project used for the mineral reserve estimate and for other estimates referred to in Item 20 of this Report has been increased by approximately 15 per cent from the $1.45

billion estimate based upon the BFS. While, as noted above, Inco has received from the principal firms responsible for the engineering and other services relating to the construction of the Goro Project information that could indicate that the Goro Project's capital costs may be higher than the capital cost estimate used for the purpose of preparing the mineral reserve estimate and for the economic analysis and other estimates referred to in Item 20 of this Report, Inco, Goro Nickel and the qualified persons who have prepared this Report believe that the capital cost estimate used for such purposes is reasonable based on available data as of December 31, 2002.

While this review of all of the key aspects of the Goro Project is being conducted, both Inco and Goro Nickel plan to continue working with the New Caledonian authorities to complete the necessary permitting and to seek the remaining local approvals required for the Goro Project.

As at December 31, 2002, Inco estimated that it had spent approximately $350 million on the detailed engineering and early construction phase of the Goro Project since the project was formally launched on July 1, 2001. Work since July 1, 2001 has consisted of exploration, testwork, preparation of engineering and environmental studies, development of mine production schedules, design and engineering, procurement and initial construction and related work. Reference is made to Item 12 of this Report which includes a detailed discussion of exploration undertaken to date in connection with the Goro Project.

**Cautionary Statement on Forward Looking Statements**

This Report may include certain forward-looking statements (within the meaning of applicable U.S. securities laws) relating to Goro Nickel and the Goro Project, including mineral reserve and mineral resource estimates, new process technologies, production capacities, operating and capital costs, exchange rates, products to be produced and demand for such products and political conditions. Inherent in such forward-looking statements are risks and uncertainties well beyond the ability of Goro Nickel or Inco to predict or control. Actual results and developments are likely to differ, and may differ materially, from those expressed or implied by the forward-looking statements contained in this Report. These forward-looking statements should not be relied upon as representing any party's views as of any date subsequent to the effective date of this Report.

# 4. INTRODUCTION AND TERMS OF REFERENCE

## 4.1 Terms of Reference

As noted in the Summary, this Technical Report has been prepared pursuant to National Instrument 43-101, "Standards of Disclosure for Mineral Projects" and Form 43-101F1, Technical Report of the Canadian Securities Administrators.

## 4.2 Technical Abbreviations and Acronyms

The following sets forth a number of abbreviations and acronyms used in this Report:

| | |
|---|---|
| Al | Aluminum |
| $Al_2O_3$ | Alumina |
| Asbolane | A Chrome-manganese hydroxide |
| Becker | Reverse circulation drill hole that produces a chipped sample |
| Benoto | Large diameter (0.8m) round shaft holes with casing |
| BFS | Goro Nickel Project Bankable Feasibility Study dated March 2001 prepared by Hatch Associates |
| Block model | A two or three dimensional shape composed of multiple blocks, used in mineral resource estimation. |
| Ca | Calcium |
| CaO | Calcium Oxide |
| CCD | Counter Current Decantation |
| cm | Centimeter |
| Co | Cobalt |
| Core hole | Drilling method that recovers sample consisting of a core of rock or soil |
| Cr | Chromium |
| $Cr_2O_3$. | Chromium Oxide |
| Cu | Copper |
| Cut-off | The minimum grade of mineralized material that would qualify as mineral resources or mineral reserves |
| °C | Degree Celsius |
| DTF | Dry tonnage factor – tonnes per cubic meter of dried material |
| ERTK | 20cm core size |
| Fe | Iron |
| Floating Cone | A computational technique in mining to evaluate the optimum pit design based on economic parameters |
| FPP | Feed Preparation Plant |
| Garnierite | A green mineral that contains nickel, magnesium and silicon |
| Geostatistics | A branch of applied statistics used in the analysis of the distribution of metal content in rock |
| Goethite | Iron hydroxide mineral of red-brown colour |
| Hematite | Iron rich oxide mineral that occurs in limonitic soil |
| Hopper | A storage bin or a funnel that is loaded from the top and discharges through a door or a chute in the bottom |
| HQ | 63.5 mm drill core size |
| ICP | Induced coupled plasma technique of chemical analysis of solids or liquids |
| ITSL | Inco Technical Services Limited, the engineering and technology services unit of Inco Limited |
| K | Potassium |

| | |
|---|---|
| $K_2O$ | Potassium Oxide |
| Kg/h | Kilogram per hour |
| Kg/t | Kilogram per tonne |
| Km | Kilometre |
| Kriging, kriged: | Geostatistical technique employed in estimating grade and tonnage of mineral resources in a block from sampling data |
| Ktonnes/yr | Kilotonne (metric) per year |
| KW | Kilowatt |
| Lb | Pound |
| LIM | Limonite |
| Limonite | Generic term used to describe yellow or red clay soils that are rich in iron |
| LOB | Limonite Overburden |
| LOI | Loss on ignition: The loss in weight that results from heating a sample to a high temperature to determine the amount of free water |
| m | Meter |
| Mg | Magnesium |
| MgO | Magnesia |
| Mlbs | Million of pounds |
| Mlbs/yr | Million of pounds per year |
| Mn | Manganese |
| MnO | Manganese Oxide |
| Mm | Millimetres |
| MW/hr | Megawatt per hour |
| Nearest Neighbour Model | Technique used in grade and tonnage estimation of mineral resources whereby the closest sample value is assigned to a block |
| Ni | Nickel |
| NQ | 47.6mm core size |
| °C | Degree Celsius |
| Pisolite, pisolitc | Spherical, concretion or aggregate of mainly goethite mineral |
| Peridotite | Rock mainly composed of iron and magnesium silicate mineral olivine and may contain on average 0.3% nickel |
| ROM | Run Of Mine |
| Run of Mine | Mineralized material excavated without treatment |
| Seam model | A two or three dimentional model where data is interpolated relative to a reference plane. |
| Serpentinisation | Alteration of ultramafic rocks, such as peridotite |
| Shelby Tube | Thin walled soil sampling tube, 30.5 to 76.2cm long, used in geotechnical investigations to collect undisturbed samples |
| Si | Silicon |
| $SiO_2$ | Silica |
| Stochastic Simulation | A statistical random process that yields multiple possible results based on sampling data statistics |
| SX | Solvent extraction |
| T, or t | Metric tonne (2,204.62 Lb) |
| t/h | Metric tonnes per hour |
| Ti | Titanium |
| $TiO_2$ | Titanium Oxide |
| Trivelsonda | Large diameter (1.0m) round shaft holes with casing |
| Ultramafic | General name for rock composed mainly of iron and magnesium silicate minerals. Dunite, peridotite and pyroxenite are ultramafic rocks. |
| Unfolding | A mathematical process that transforms three dimensional data to a relative reference plane |
| UTM | Universal Trans Mercator geographic reference system |
| Variogram | Statistical parameter used to measure correlation between sample points |

| | |
|---|---|
| Wobbler | Equipment used in earth moving to sort material based on its size |
| wt. | Weight |
| WTF | Wet tonnage factor, tonnes per cubic meter, including moisture |
| XRF | X-ray fluorescence, a non destructive technique for elemental analysis of solids and liquids that uses x-rays as a source of radiation |
| Zn | Zinc |

### 4.3 Purpose of Technical Report

Since the Goro Project has reached the stage in its development where it could be viewed as representing a mineral project or a property that could be material to Inco, Inco has prepared this Report pursuant to National Instrument 43-101.

### 4.4 Sources of Information

Primary documents referred to in this Report include, but are not limited to, the *Goro Nickel Project Bankable Feasibility Study* ("BFS") dated March 2001, an addendum to Section 11 thereof dated May 2002, related corporate approvals, the Goro Nickel Project Mine Plans and Mine Production schedules, December 31, 2002 Report prepared by Independent Mining Consultant ("IMC") March 2003 and a number of other documents listed in the detailed References provided in Item 23 of this Report.

### 4.5 Field Involvement of Qualified Persons

R. A. Horn, P. Geo, *Vice-President, Exploration Inco Limited*, has been associated with the Goro Project since 2001 and visited the Goro Deposits August 13 to 20, 2001 and March 24 to 28, 2002.

Dr. W. Gordon Bacon, P. Eng, FIMM, *Vice-President, Technology and Engineering, Inco Limited,* has been associated with the Goro Project since 1998 and has visited the Goro Deposits various times between 1998 and 2002. Dr. Bacon also visited the pilot plant during its operation in 2000 and 2001.

## 5. DISCLAIMER

The authors wish to make clear that their respective areas of expertise for the purposes of this Report are limited to the areas identified in their Certificates of Qualified Person submitted with this Report to the Ontario Securities Commission. The authors have relied, and believe that they had a reasonable basis to rely, upon the following individuals as having contributed supporting technical documentation or other assistance in connection with this Report, as noted below:

| | |
|---|---|
| Ted Bassett<br>Goro Project Project Director | Capital Cost Estimate referred to in Item 20 |
| Stuart F. Feiner<br>Executive Vice-President, General Counsel & Secretary, Inco Limited | Review of Goro Project in Item 3 and related areas in this Report |
| Peter J. Goudie<br>Executive Vice-President, Marketing, Inco Limited | Marketing of nickel and cobalt and product forms and contracts in Item 20 |
| Peter C. Jones<br>President and Chief Operating Officer, Inco Limited | Capital Cost Estimate referred to in Item 20 |
| William A. Napier<br>Vice-President, Environment & Health, Inco Limited | Environmental Liabilities, Remediation and related areas in Item 6 and Item 20 |
| Brent A. Rochon<br>Assistant Vice-President, Marketing, Inco Limited | Marketing of nickel and cobalt and product forms and contracts in Item 20 |
| Larry Smith<br>Project Evaluation Consultant to Inco Limited | Economic and Sensitivity Analyses and Operating Cost Estimates in Item 20 |
| Dr. Ric Stratton-Crawley<br>Directeur General et Chef des Operations, Goro Nickel S.A. | General discussion of Goro Nickel, its facilities and planned operations and data from the BFS; Economic and Sensitivity Analyses and Operating Cost Estimates in Item 20 |

The authors have also relied upon IMC to prepare the mine production schedule and the mine plans.

The authors wish to state that information in this Report regarding the Goro Project that predates 1992 has been considered historical information developed by Inco or third parties. The authors have assumed this historical information to be correct and have listed such information in the References provided in Item 23 of this Report. In some cases, the authors or personnel under their responsibility or supervision have undertaken verifications of records and made site visits to observe physical evidence of past exploration activities as reported in such historical information.

# 6. PROPERTY DESCRIPTION AND LOCATION

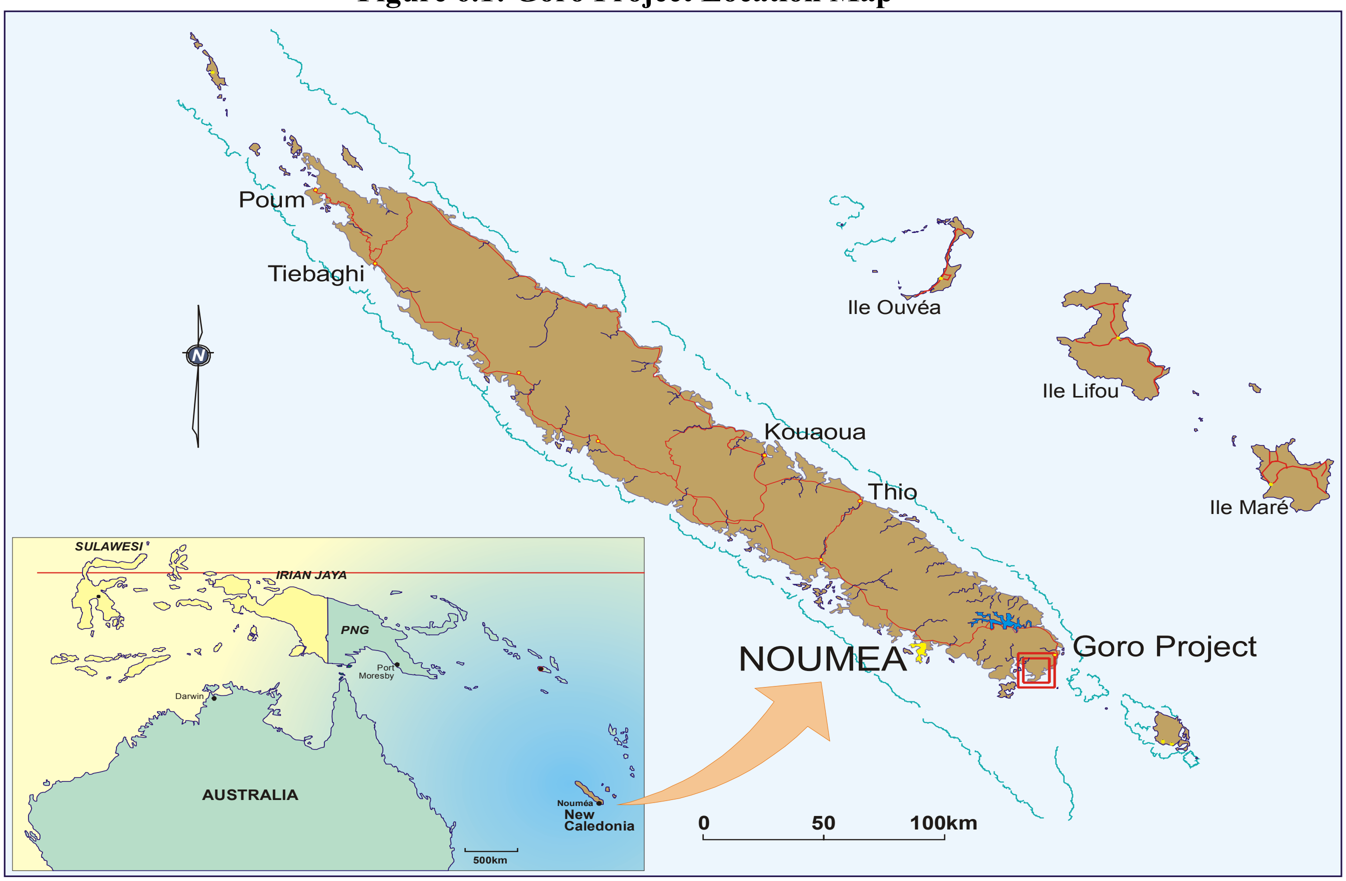
## 6.1 Goro Project Location

New Caledonia is currently an overseas territorial community (*collectivité territoriale*) of the Republic of France having special legal status under the French constitution. It is located approximately 1,500km east of Australia and approximately 2,300km northwest of New Zealand.

New Caledonia is divided politically into three provinces: Province Nord, Province Sud and Province des Iles. Province Nord and Province Sud are the northern and southern parts of Grande Terre, the main island of New Caledonia. The Province des Iles is the island group to the east of Grande Terre. The capital of New Caledonia is Nouméa, located on the south end of Grande Terre in Province Sud.

The Goro Project, as shown in Figure 6.1, is located on the southeast end of Grande Terre in Province Sud, about 60km east of Nouméa at approximate latitude 22 º, 15' south and longitude 167º east.

**Figure 6.1: Goro Project Location Map**



### 6.2 Goro Project Description

Goro Nickel holds 69 nickel, cobalt and chromium mining concessions and approximately 19,500 hectares of surface rights in Province Sud. Of these 69 mining concessions, the Goro Project comprises 6,042 hectares in 7 mining concessions. Reference is made to the description and location of these concessions in Table 6.2 and Figure 6.2 below. A number of other Goro Nickel concessions contain possible nickel laterite mineral deposits that were not considered in this Report. Four of the Goro Project mining concessions are perpetual and three others, as shown in Table 6.2 below, may be renewed for an additional 25-year period after their initial expiry date. Goro Nickel expects that these mining concessions will be renewed prior to their expiry.

**Table 6.2: Goro Project Concessions**

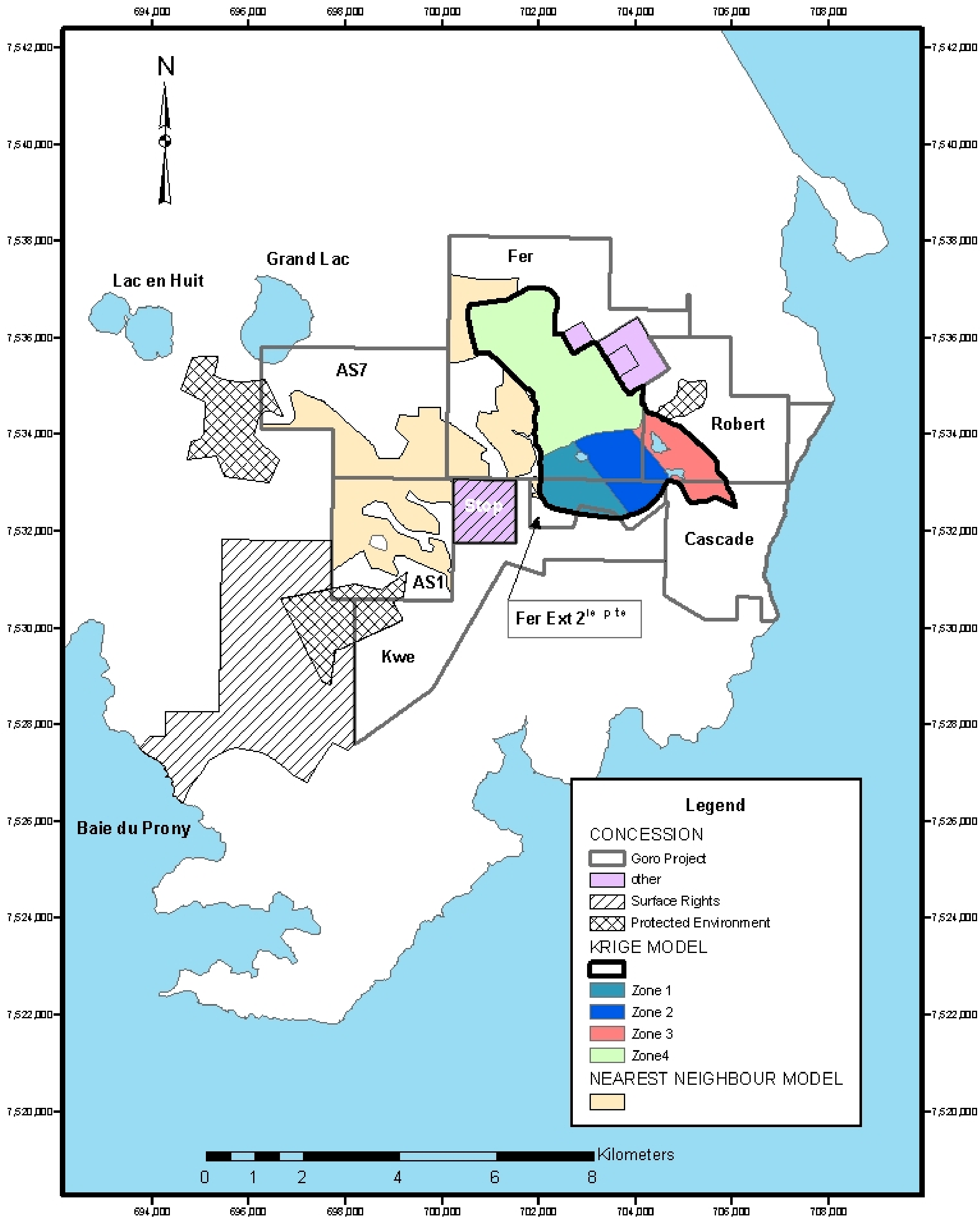
| Concession | Permit | Area(Ha) | Rights | Expiry |
|---|---|---|---|---|
| AS1 | 2415 | 625.00 | Mining and Surface | 15/10/2016 |
| AS7 | 2426 | 890.29 | Mining and Surface | 15/10/2016 |
| Fer | 1681 | 1857.14 | Mining and Surface | Perpetual |
| Fer Ext. 2 Pte | 1682 | 360.06 | Mining and Surface | Perpetual |
| Robert | 1646 | 725.48 | Mining and Surface | Perpetual |
| Cascade | 1687 | 680.75 | Mining and Surface | Perpetual |
| Kwe | 3433/PS | 903.22 | Mining and Surface | 07/08/2051 |

In addition to the seven mining concessions referred to above, Goro Nickel intends to include an eighth mining concession known as “Stop” as part of the Goro Project. The “Stop” concession is currently owned by Société Minière Georges Montagnat and Société Le Nickel, who have agreed in principle to exchange the Stop concession for one of the other mining concessions held by Goro Nickel outside the Goro Project. While this exchange has not been formally completed, Goro Nickel is not aware of any conditions that would prevent this agreement in principle from becoming effective on a timely basis.

Permits or concessions are not physically surveyed in New Caledonia. Instead, the permits or concessions requested are established by the property UTM coordinates on the maps of Institut Géographique Nationale (IGN) and the topographical maps from Service Topographique, a New Caledonian government land survey department that oversees land surveys and tenements status. These reference points are then transferred to the Government Mining Bureau for filing. When the concession is granted, the coordinates are published in the “Journal Officiel” within the “Arrêté”, pursuant to applicable legislation that officially formalizes the granting of rights to the area covered by the permit or concession.

The mineral resources and mineral reserves estimates for the Goro Project are located primarily within the Fer mining concession, with portions also located within the Fer Ext. 2 Pte and the Robert mining concessions (Figure 6.2). All of the seven mining concessions listed in Table 6.2 above are in good standing

## Figure 6.2: Goro Project Concessions Location Map



### 6.3 Environmental Liabilities

The site of the Goro Project is generally considered a 'greenfield' site because no previous mining has been conducted. However, exploration activity, test mining, a nursery and vegetation trials and land clearing have been conducted integrated processing facilities and accommodation complexes and supporting infrastructure will be constructed at the site. The appropriate regulatory authorities have approved all development activities conducted to date. Since 1994, a series of baseline studies have been undertaken to characterize the terrestrial, air, and aquatic (marine freshwater and groundwater) environments and provide information to assist in the design of the Goro Project. The Installation Classées discussed below in Section 6.4 (reference is made to www.inco.com/projects/goro/default.asp which provides a link to the French language version of the Installation Classées application) outlines the current known environmental liabilities and the potential environmental effects of the Goro Project. Environmental components of the Goro Project will also be addressed on Goro Nickel's website, www.goronickel.com.

### 6.4 Permitting

As at December 31, 2002, Goro Nickel had identified over 100 permits required for construction and operation of the Goro Project. Many of these permits have already been obtained, while many others have been applied for or will be applied for by Goro Nickel.

The most significant permits necessary to the Goro Project are (1) the Demande d'autorisation pour exploiter des Installations Classées (IC), which is the main operating permit for the commercial facilities (overall process and mining activities) and (2) the Déclaration Minière permit which covers the mining operation itself. The government of Province Sud accepted Goro Nickel's IC permit application on January 15, 2002 and the application for the Déclaration Minière will be submitted on a timely basis. Goro Nickel expects to obtain the IC and the Déclaration Minière on a timely basis.

# 7. ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

## 7.1 Topography and Elevation

The Goro Project covers a series of drainage basins. These drainage basins are separated by ridges underlain by the peridotite rocks of the Massif du Sud. The Goro Deposits are located on a gently stepped plateau (the Goro plateau) rising from about 200 metres in the south to about 330 metres in the north over 5 kilometres. The Wadjana river drains to the eastern coast while the Kwe and the Rivière Bleue drain to the southern coast. The Kwe river drains the Goro plateau through a natural break in the southern ridgeline through to the Port-Boisé basin.

The Plaine des Lacs Basin lying to the north is outside the immediate area of the Goro Project but appears to influence the groundwater in the Goro Project area through minor geological structures that seasonally drain from the north. The Prony Est basin to the west is planned to be the location of the process plant and wharf. The Cascade basin to the east beyond the proposed open pit footprint is planned to be the location of the east overburden dump. The general site plan of the Goro Project is illustrated in Figure 7.1.

## 7.2 Vegetation

The vegetation on the Goro plateau and adjacent basins is diverse in terms of endemic plant species. Both geological isolation and nutritional deficiencies of the laterite profile overlying the peridotite bedrock on the Goro plateau have contributed to the development of a specific flora tolerant to the nickel-cobalt-iron rich soil. Since 1995, floral inventories have outlined several vegetation types on the Goro plateau, including woody "maquis" (scrub heath land) on iron cap plateaus, swamp sedge land around sinkholes and patches of rainforest in valleys. To date, approximately 600 plant species have been identified in the area of the Goro Project.

Two forest reserves are located at the western boundary of the Goro Deposits on or near the peridotite ridges that have no nickel laterite potential.

Since 1996, Goro Nickel has been testing a revegetation strategy with the objective of re-establishing the native plant ecosystem and preserving the diversity of the flora. Floral inventories, species selection for reforestation, seed collection and propagation are ongoing.

## 7.3 Access

The Goro Project will be accessible by provincial roads and by sea from New Caledonia's capital, Nouméa. Land access to the Goro Project is from Nouméa, 40km by the paved Route Principal No. 3 to la Capture and then by a well-maintained dirt road to the Goro Project site. By sea, access to the Goro Project can be made from Nouméa via Canal Woodin to Baie du Prony, where a wharf was upgraded in 1998 to service Goro Nickel's pilot plant operation. Almost all bulk supplies will be shipped by sea freight to be handled at a new marine port facility at the site of the present Prony wharf. Goro Nickel plans to operate an ocean-going vessel to provide transportation between the Prony wharf and areas near Nouméa.

## 7.4 Climate

The climate in the area of the Goro Project is subtropical. Seasonal temperatures range from about 30°C maximum to about 15°C minimum, with a mean annual temperature of about 23°C.

The tropical cyclone season extending from December to March will likely impact mining activity because intermittently heavy rainfall (on occasions daily downpours have been recorded in excess of 400mm) and high winds have been recorded during this period. However, Goro Nickel has made provisions to the Goro Project's mine schedule, road network, mine drainage and stockpile management to minimize any such delays due to inclement weather. Annual rainfall varies in the area of the Goro Project from in excess of 3,000mm on the Goro plateau to less than 2,500mm at Baie du Prony.

It is planned that the Goro Project's process plant will operate throughout the year while the mine operation may be affected by bad weather for approximately 21 days per year.

## 7.5 Local Infrastructure

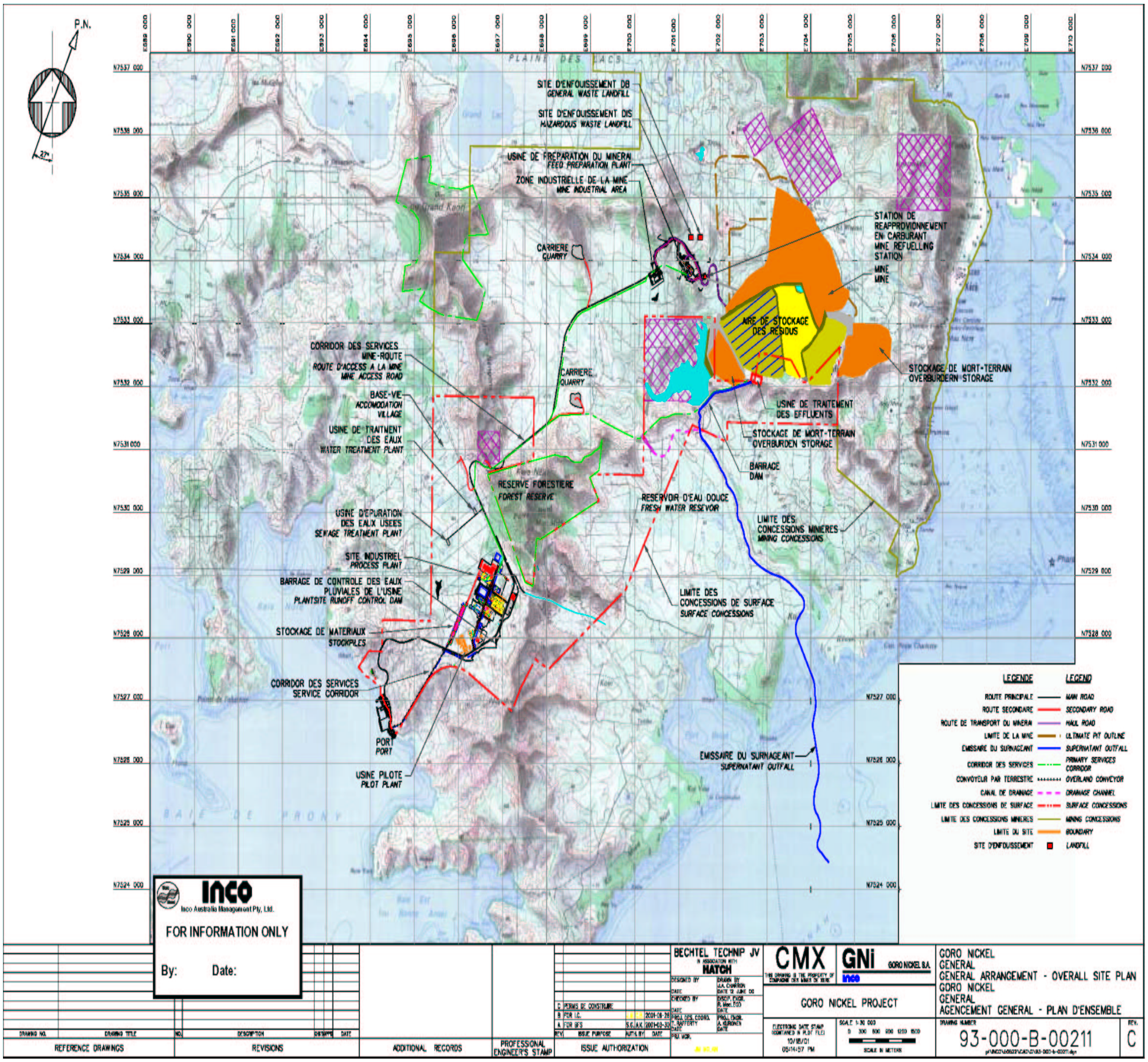
The relative remoteness of the Goro Project requires the establishment of all support services and infrastructure at the site. A port facility, located on the north east shore of Baie du Prony, will be utilized during the construction and operational phases. A wharf will be constructed to load and unload supplies as well as to provide berthing for the ocean transport of the local workforce from the Nouméa area. A lay-down area will be available at the port facility to store containers.

It is planned that the process water and potable water for the Goro Project will be obtained from a fresh water reservoir to be constructed on the West Kwe River.

An oil fired steam and power plant will be constructed to supply high pressure steam for the process plant and to supply 182,000MWhr per year, or approximately 1/3 of the electrical power required for the Goro Project. The balance of the power required for the Goro Project is expected to be supplied by a coal fired power plant.

Goro Nickel has secured concessions and surface rights to support the Goro Project and, as referred to in Item 6.2 of this Report, Goro Nickel is in the process of acquiring mining rights in the Stop concession in addition to the surface rights Goro Nickel already holds in the Stop concession. The Stop concession will be the location of the process water dam and reservoir shown in Figure 7.1.

## Figure 7.1: General Site Plan of the Goro Project



# 8. HISTORY

## 8.1 Project to Date

The current Goro Project was initiated in mid-1992, when a predecessor company to Goro Nickel, Compagnie des Mines de Xéré (CMX), was incorporated to develop and commercialize the Goro Deposits. Work to date with respect to the Goro Project has consisted principally of exploration, pilot plant and testwork, engineering, design, environmental and other studies and initial construction work and development of mine production schedules.

## 8.2 Prior Ownership

Inco has been involved in various projects in New Caledonia for close to 100 years and has been exploring nickel laterite deposits in southern New Caledonia since the late 1950's. In October 1967, the Government of France agreed to the formation of a new company, Compagnie Française Industrielle et Minière du Pacifique ("COFIMPAC"), with Inco as the foreign partner and Société Auxiliaire Minière du Pacifique ("SAMIPAC"), representing French interests, to develop unexploited nickel laterite deposits. The mining properties on which this project, the COFIMPAC Project, was based included the Goro Deposits. The COFIMPAC Project terminated in 1971.

From 1972 to 1988, BRGM jointly with Compagnie Française d'Entreprises Minières, Métallurgiques et d'Investissements ("COFREMMI") held mining concessions covering the Goro Deposits. In 1986, BRGM formed a joint venture with Queensland Nickel Pty. ("QNI"), an Australian nickel mining company, to study the viability of exporting ore from the Goro Deposits to Yabulu, QNI's nickel refinery located near Townsville, Australia. QNI withdrew from this venture in 1990.

In 1989, Inco initiated discussions with BRGM with respect to the acquisition by Inco of all of BRGM's mining concessions in southern New Caledonia, including those covering the Goro Deposits. An agreement was signed in 1991 between Inco and BRGM and in 1992 existing BRGM mining concessions in southern New Caledonia were transferred to a newly created subsidiary of BRGM, Société de Promotion des Mines S.A. ("SOPROMINES"). Inco purchased SOPROMINES in 1992 and renamed the company Compagnie des Mines de Xéré ("CMX"). On December 31, 2001, CMX was merged into Goro Nickel. Immediately after the merger, CMX changed its name to Goro Nickel.

As previously discussed, at present Goro Nickel is 85% owned by Inco and 15% owned by Selani, a wholly owned subsidiary of BRGM.

## 8.3 Previous Exploration Work

### 8.3.1 COFIMPAC (1969-1971)

The main focus of exploration during the COFIMPAC Project was on the limonite portion of the laterite and not on the saprolite (see Item 11 below). Between 1969 and 1971, 1,237 reverse circulation hammer holes were drilled, at 200m x 100m average spacing, for a total of 44,951m. Part of this drilling was carried out at close spacing for statistical investigations. Bulk samples were excavated and a mining test was carried out. A number of other geological, geotechnical and mining studies were carried out during this period.

In 1970, Inco presented a feasibility study of the COFIMPAC Project to COFIMPAC. The feasibility study concluded that the zone called COFIMPAC 3 (comprising the Goro Deposits) was the most favourable of all properties investigated and this zone could support an annual production of 100Mlbs of nickel per year, at an average grade of 1.57% nickel. Other reports developed during this period indicated that the irregularities of the Goro Deposits were such that it would not have been possible to mine ore (as defined by the 10 metre spaced holes) without taking some waste and leaving some ore behind.

At that time, extensive testing was conducted at Inco's Canadian research laboratory in Mississauga, Ontario and at Inco's pilot plant in Port Colborne, Ontario on samples from the Goro Deposits, including two bulk sample shipments made in 1969 and 1971. The bulk samples allowed an approximate comparison to be made between the smaller drill samples and the larger bulk samples.

Reference is made to references 1, 2 and 3 in Item 23 of this Report for citations related to exploration work during the COFIMPAC Project.

#### 8.3.2 BRGM/COFREMMI (1972-1988)

In 1972, COFREMMI core drilled 43 holes within the Goro Deposits at sites of existing Becker drill holes.

In 1982, 64 HQ and NQ core size holes were drilled by BRGM on a 200m grid in a test mine area.

In 1986, the joint venture between BRGM and QNI began a detailed sampling program for a feasibility study that included 25 core holes. In 1988, this joint venture also performed geostatistical tests of 32 five-metre spaced holes and excavated eight Benoto 0.8 m diameter shafts through to the top of the saprolite layer.

In 1988, 60 NQ core size holes were drilled on a 10 m grid in preparation for a 30,000-tonnes bulk sample shipment to the QNI Yabulu refinery. The overall grade of the shipment to the Yabulu refinery was 1.55% nickel, 0.17% cobalt, 41% iron and 43.9% $H_2O$. However, the quantity and grade of saprolite estimated prior to shipment was not consistent with what was actually drilled and shipped. A report prepared for SOPROMINES concluded that the close-spaced core drilling and/or the type of core sampling conducted failed to anticipate the grade and chemistry of the saprolite mining profile. Reference is made to references 4, 5, 6, 7 and 8 in Item 23 of this Report for citations related to BRGM/COFREMMI exploration work.

### 8.4 Previously Reported Mineral Resource and Mineral Reserve Estimates

Prior to the effectiveness of National Instrument 43-101, mineral resources and mineral reserves estimates for the Goro Deposits reported in several Inco Annual Reports to Shareholders included inferred mineral resources that were not shown separate from, but were included together with, measured and indicated resources. From 1995 to 1997, mineral resources estimates for the Goro Deposits were reported in Inco Annual Reports to Shareholders as 165 million tonnes containing 1.6% nickel. The 1998 Inco Annual Report to Shareholders reported that the Goro Deposits contained potentially mineable resources estimated to be in excess of 200 million dry tonnes of laterite averaging 1.60% nickel and 0.17% cobalt. The 2000 Inco Annual Report to Shareholders reported a total mineral resources estimate for the Goro Deposits of 219 million tonnes containing 1.57% nickel, including 162 million tonnes of a inferred mineral resources estimate containing 1.55% nickel.

Beginning in 2001, Inco began reporting in its Annual Report to Shareholders mineral resources and mineral reserves estimates pursuant to the definitions set out in the Canadian Institute of Mining, Metallurgy and Petroleum (the "CIM") Standards on Mineral Resources and Mineral Reserves Definitions and Guidelines (the "CIM Standards") adopted by the CIM Council on August 20, 2000 and provided for under National Instrument 43-101. The 2001 Inco Annual Report to Shareholders reported 99 million tonnes of measured and indicated mineral resources estimates for the Goro Project containing 1.58% nickel and 0.14% cobalt and inferred mineral resources of 143 million tonnes containing 1.7% nickel and 0.12% cobalt. In addition to these mineral resources estimates, the 2001 Inco Annual Report to Shareholders reported proven and probable mineral reserves estimates of 54 million tonnes containing 1.53% nickel and 0.12% cobalt.

## 9. GEOLOGICAL SETTING

### 9.1 Regional Geology

New Caledonia is situated on the inner Melanesian zone of a major geological unit called the Austral-Indian crustal plate. The islands of New Caledonia are located on the north-south Norfolk Ridge surrounded by the Basin of New Caledonia to the west and the Loyalty Basin to the east, forming a series of ridges and basins between the Australian continent to the west and the geologically active island arc to the northeast. New Caledonia is composed of three structural zones: the Central Chain, the Western and Northern Coast, and the overlying ultramafic rocks.

Ultramafic rocks underlie almost 7,000 square kilometres and are the main components of the 5,000 square kilometres Massif du Sud located at the southeastern tip of the island. The remainder of the ultramafic bodies are scattered mostly along the western coast. The Goro Deposits are located in the Massif du Sud.

**Figure 9.1: Location of Ultramafic Intrusions in New Caledonia**

### 9.2 Local Geology

The Massif du Sud can be divided into two major domains. The northern part consists predominantly of high hills with scattered thin laterite deposits on slope terraces. The southern part is composed of large topographic basins (200 to 300 metre elevation) enclosed by elongated ridges. The topographic basins are deeply weathered and covered by thick limonite and underlying saprolite.

### 9.3 Property Geology

The Goro Deposits are located within the Kwe basin, an area of approximately 30 square kilometres. The laterite profile thickness is approximately 40 metres but is extremely variable, ranging from 20 to 90 metres or more as described in detail in Section 11 of this report.

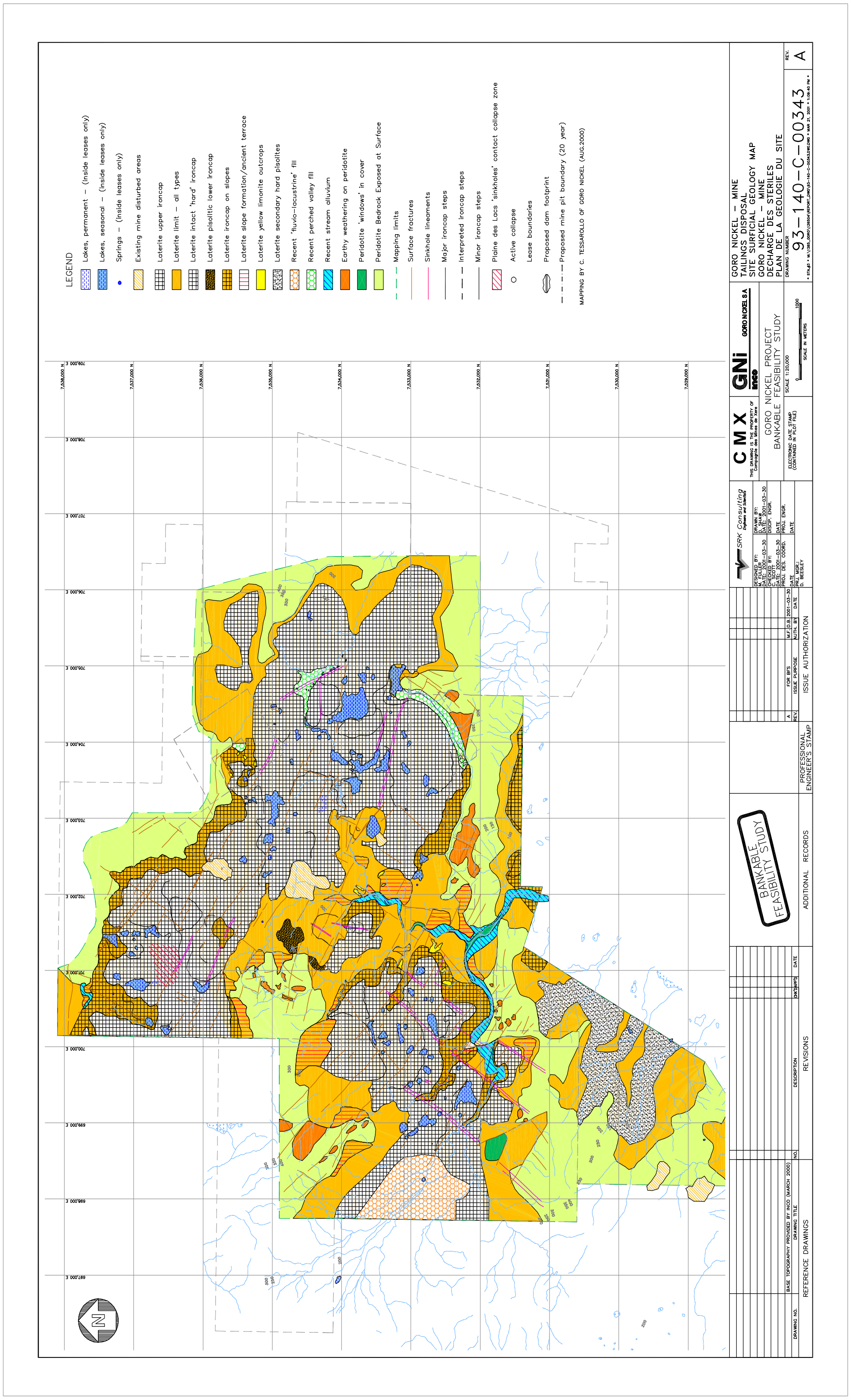
The laterite profile is well developed on plateaus and is protected from erosion by an iron cap. However, in some places the iron cap has been "dismantled" or breached by active drainage exposing the underlying laterite, as shown in Figure 9.2 below.

The regional picture of basins contained within a network of ridges is also repeated on a local scale, with small sub-basins enclosed by elevated areas composed of underlying rocks that are masked by the iron cap. The overall development of the laterite profile is controlled by the local land form and influenced by major geological structures and variations in the composition of the bedrock. These structures originate in the bedrock but are reflected through to the surface, as steps in the iron cap, development of cavities or "sinkholes", local depressions (dolines) and faults.

There are two predominant sets of steeply dipping faults, trending northwest southeast and northeast southwest. Many secondary faults reflect the compressive and tensile forces present during the movement of geological plates in the earth's crust and the later deformation of the ultramafic sheets.

The typical river basin profile is steep at its margins and becomes more shallow towards the valley centre. Laterite here is interspersed with blocks of iron cap. Erosion in some places has removed the limonite and exposed saprolite or bedrock. Laterite on the steep slopes forming the highlands around the basins is either absent or very poorly developed. Where the laterite is developed, a thin rocky saprolite is predominant which may have no limonite cover.

# Figure 9.2: Goro Property Geological Map



## 10. DEPOSIT TYPES AND GENESIS

The Goro Deposits occur in nickel laterite soils that form over ultramafic rocks in a tropical to semi-tropical climate in areas of heavy rainfall. Lateralization is a chemical weathering process whereby atmospheric, groundwater and biological processes interact on exposed rock surfaces and fractures in the bedrock to decompose the primary rock minerals and form chemically stable mineral phases in the weathering zone. Weathering begins on joints and fractures, initially forming large blocks or "boulders" within a clay matrix. As weathering progresses, the more mobile elements, silicon and magnesium, are leached and deposited at the base of the laterite profile or are flushed out.

Iron, and to some extent alumina, are less soluble and are enriched in the upper portion of laterite forming the limonite layer rich in goethite which may collapse under its own weight as leaching removes large quantities of the more soluble elements. As weathering progresses, the original fabric or texture of the rock is destroyed. Some re-crystallization occurs and an "iron cap" is formed at surface. Nickel, cobalt and manganese are enriched in the limonite layer, especially at the interface of the limonite and saprolite.

During further weathering, the peridotite boulders form saprolitised crusts as the weathering front advances from the surface of the boulder to the core. During this process, the relic boulders are reduced in size and the clay component, typical of the saprolite layer, becomes dominant. In the saprolite clay-boulder mix the structure of the parent rock is preserved. The saprolite profile consists of a complex mixture of decomposed rocks containing fine-grained nickel-manganese minerals that is the ore and altered boulders, which may be enriched in nickel on their surfaces and along internal fractures or may be barren.

The silicate nickel content in the original ultrafamic rock is about 0.3% or less, contained within the mineral olivine. The nickel is liberated during the leaching process and migrates downward, carried by the groundwater, in the laterite profile. This process is called supergene enrichment.

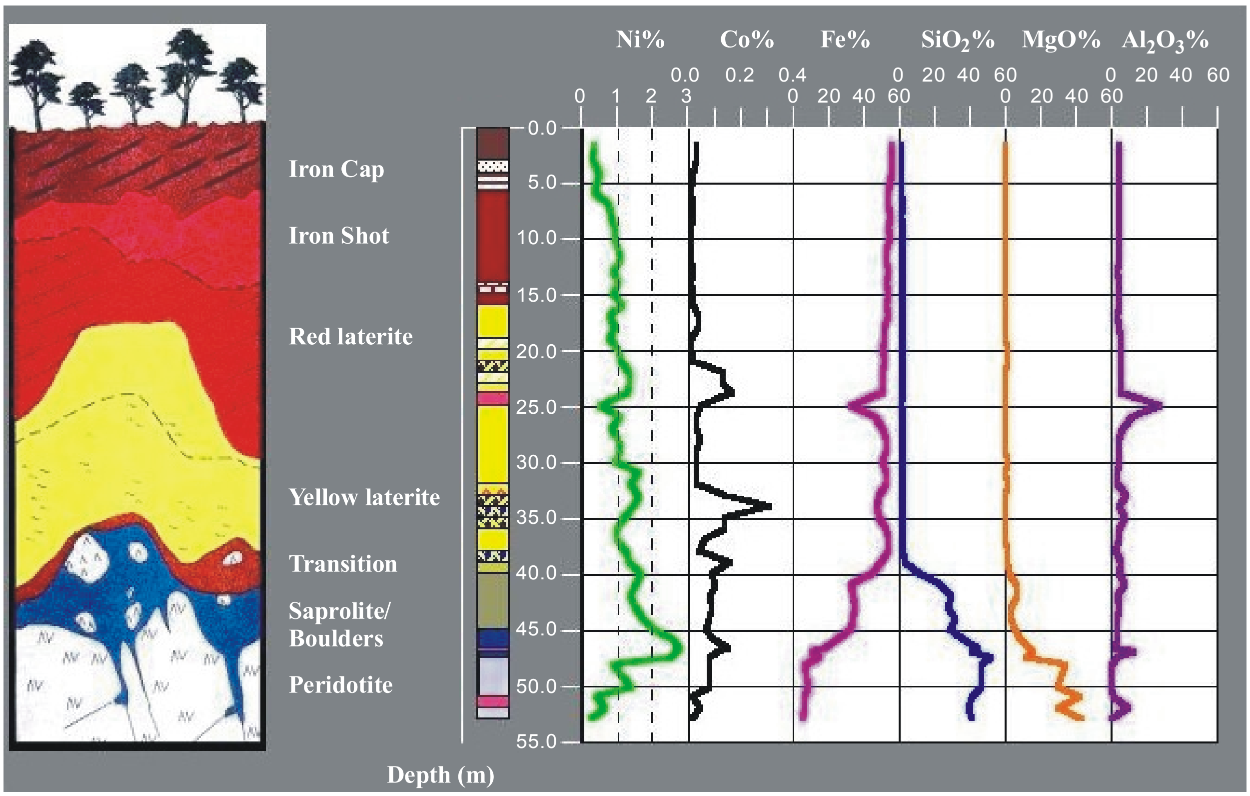
The cobalt remains with the iron hydroxides but may also migrate downward to become concentrated, particularly at the transition zone between the saprolite and the limonite layers. Almost invariably cobalt is associated with manganese in the mineral asbolane.

The formation of each laterite deposit is influenced by the original bedrock, the degree of serpentinisation, the local bedrock structures, topography and the local climate. There is significant variation between different laterite profiles and within each laterite profile.

## 11. MINERALIZATION

As illustrated in Figure 11.1, six layers are recognized in the laterite profile overlying the Goro Deposits ultramafic bedrock. Figure 11.1 also shows the variation in the nickel and cobalt content and the chemistry through the profile. The mineral reserves and mineral resources are located within three layers: the yellow laterite, the transition and the saprolite/boulders. The yellow laterite and the transition layers are collectively referred to as the limonite ore.

**Figure 11.1: Goro Deposits Generalized Laterite Profile And Chemistry**




Each of the layers in the profile is described below.

**Iron Cap (ICP):** The iron cap located at the top of the weathering profile comprises massive to broken blocks of secondary hematite. It has a low water content. The iron cap once covered most of the Goro plateau but now is dissected because of the collapse of the laterite profile and the step-like features that are the surface expression of weathering deep in the profile and ground water movement along faults. The iron cap is dismantled at the edge of the Goro plateau due to more recent erosion of the profile along the more significant active rivers. The iron cap is expected to provide a suitable material for civil construction work such as road ballast.

**Iron Shot (ISH):** The iron shot is located immediately below the iron cap and is a pisolitic soil composed of a mixture of remobilized and partly transported iron cap and red limonite and hematite accretions. The base of the iron cap or the iron shot is the location of significant instantaneous water flows during torrential rainfall. Small voids occur at the base of the iron cap or iron shot. The accretion-rich portion of the iron shot may be useful for civil construction work.

**Red Laterite (LATR):** This hematite-rich, reddish soil is characterized by a moisture content of generally less than 40%, low-grade nickel (less than 1%) and iron content greater than 40%. In the Becker holes it is defined as the layer above the first occurrence of 1% nickel located below the iron shot or iron cap layer. The red laterite frequently is exposed on the eroded flanks of the Goro plateau. The relatively low moisture content and the favourable geotechnical properties of this layer make red laterite suitable for various civil earthworks.

**Yellow Laterite (LATJ):** Containing a higher concentration of nickel and cobalt than the layers above it, yellow laterite is very fine-grained with goethite the predominant mineral as opposed to hematite in the overlying red laterite. Cobalt and manganese tend to occur preferably along structures that are observed as black streaks in the profile. The layer is chemically defined in the database of Becker holes as a nickel concentration higher than 1% and a magnesia content of less than 2%. In the core, yellow laterite is recognized by its colour and mottled appearance.

**Transition (TRN):** The transition zone is highly variable in thickness at the boundary of the yellow laterite and saprolite. It contains significant groundwater flow and during drill sampling of this zone core may be lost. The presence of asbolane and semi-liquid mud is used by the geologist for identifying the transition zone in the core. Some boulders may be present but this is not characteristic. In the database of the Becker holes, the magnesia concentration between 2% and 8% is a good indicator of the transition layer. Because the identification of this zone by geologists is essentially subjective and reflects the liquid nature of the material, significant discrepancies occur between the determination by visual inspection of the core and that from the chemistry.

**Saprolite (SAP):** The saprolite lies immediately above the bedrock and is characterized by pebble to boulder – size, partially weathered ultramafic rocks in a clay matrix. The magnesia content is higher than in the transition zone and can locally reach 30% or more. Silica concentration is about 15% to 30% and the iron content is below 40%. Nickel grades reach their highest values in this zone. In the database of Becker holes, the layer is defined as greater than 8% magnesia, iron greater than 20% and nickel greater than 0.9%. In the core holes the saprolite layer is recognized by the heterogeneous boulder-clay mixture and the unweathered bedrock at the base of the profile. Saprolite can be upgraded by screening out the coarser lower grade boulders with the added benefit that the magnesia content of the plant feed also decreases in the process because the boulder contain high levels of magnesium. The saprolite layer has two recognizable zones:

- **Soft Saprolite:** This zone is identified in core by its predominant clay matrix and low boulder content, reflecting a more mature state of the weathering process frequently associated with a higher degree of serpentinisation of the boulders.

- **Hard Saprolite:** This zone has a high pebble to boulder content, in a clay matrix. Usually, the hard saprolite is found closest to the bedrock although in some instances it is located higher in the saprolite layer, reflecting a less mature degree of weathering frequently associated with a lower degree of serpentinisation in the parent rock. At the highly irregular base of the saprolite, weathering extends through fractures into the partially weathered bedrock forming a series of irregular shoots of saprolite and ribs of bedrock.

**Bedrock (BRK):** The bedrock is variable but is predominantly composed of peridotite rock. The silicate nickel content of the bedrock in the olivine mineral is 0.3% nickel or less. Iron is usually below 10%, while silica and magnesia content is about 40% or less. Bedrock is classified in the Becker hammer database by a nickel content of less that 0.9% and iron less than 20%.

## 12. EXPLORATION

The present phase of exploration by Inco of the Goro Deposits commenced in 1992 when rights to the Goro Deposits were transferred to CMX. The 1992-2002 period can be best described as "mine definition" and not "exploration". Inco was aware of the potential value of the Goro Deposits from exploration information that was available from campaigns that began in 1968. Exploration data from work after the COFIMPAC Project had been made available to Inco.

The objective of the early phase of the Goro Project beginning in 1992 was to assess the value of the data from the extensive prior exploration and, where required, use limited additional exploration expenditures to conduct validation and infill sampling and bring the Goro Project to the stage where a study could be made of the economic feasibility of mining the Goro Deposits (a "feasibility study"). Towards this objective, Inco undertook an extensive compilation and review of the COFIMPAC and the BRGM-COFREMMI exploration files and data and relied on this information to guide its future sampling.

Exploration work conducted since 1992 has consisted of mine-definition drilling necessary to provide information on the chemistry of the feed to the mini-pilot plant in Canada and the integrated pilot plant constructed by Inco in New Caledonia to provide information on the metallurgical process to support a feasibility study. The major tasks undertaken were oriented towards the chemical definition of the lateritic layers, the variability of the layers, the most appropriate saprolite size fraction to process, the weather conditions prevailing at site, hydrogeology and geotechnical investigations to define the strength of the soils. Other studies were conducted on bedrock petrography as well as laterite layer profiling using geophysical surveys. More than five bulk samples have been taken by means of drilling for metallurgical testing either at the mini pilot plant or at the New Caledonia integrated pilot plant.

The quality of information collected during this period meets recognized industry standards. Since 1998, core holes have been drilled and have been systematically screened, weighed, and measured for recovery. The characteristics of the laterite profile are, accordingly, better defined within the zones that were recently drilled, which correspond to the area of the current mineral reserves estimates. Prior to 1992, aluminium, chromium, manganese, copper and zinc were not routinely chemically analyzed, but these elements have been routinely included in analyses since 1992.

Exploration activities in support of the various engineering studies and pilot plant feed campaigns were conducted partially by Goro Nickel staff and partially by local contractors familiar with the mining industry in New Caledonia. In the case of pre-feasibility and feasibility studies, the principal contractors have subcontracted to overseas specialists as required. Since 1992, the following contractors worked on the Goro Project in areas related to exploration:

- Société Photogrammétrique du Pacifique (SPP) for aerial photogrammetry and various generation of topographic maps;
- M. Grand Cabinet de Géomètre for borehole and exploration general surveying support;
- Agence pour l'Eau et Environnement du Pacifique (A2EP) for environmental and hydrology studies ;
- Laboratorie du Bâtiment et des Travaux Publiques de Nouvelle Caledonies (LBTP); A2EP; Golder Associates; SRK Consultants; and MAJM Corporation individually for geotechnical studies ;

- Geophysics GPR International, Associated Mining Consultants Limited, and Golder Associates independently for ground penetrating radar surveys, and or seismic, and/or electric surveys;

- FORAPAC, a subsidiary of FORACO, of France, conducted all of the HQ core size core and 20cm diameter core drilling and bulk sampling by 1 meter diameter Trivelsonda shaft sinking conducted between 1992 and 2002; and

- Geostat Systems International, SRK International, Micon International, and IMC undertook independent audits of mineral resources and/or mineral reserves estimates and/or statements of mineral reserves in separate studies conducted since 1996.

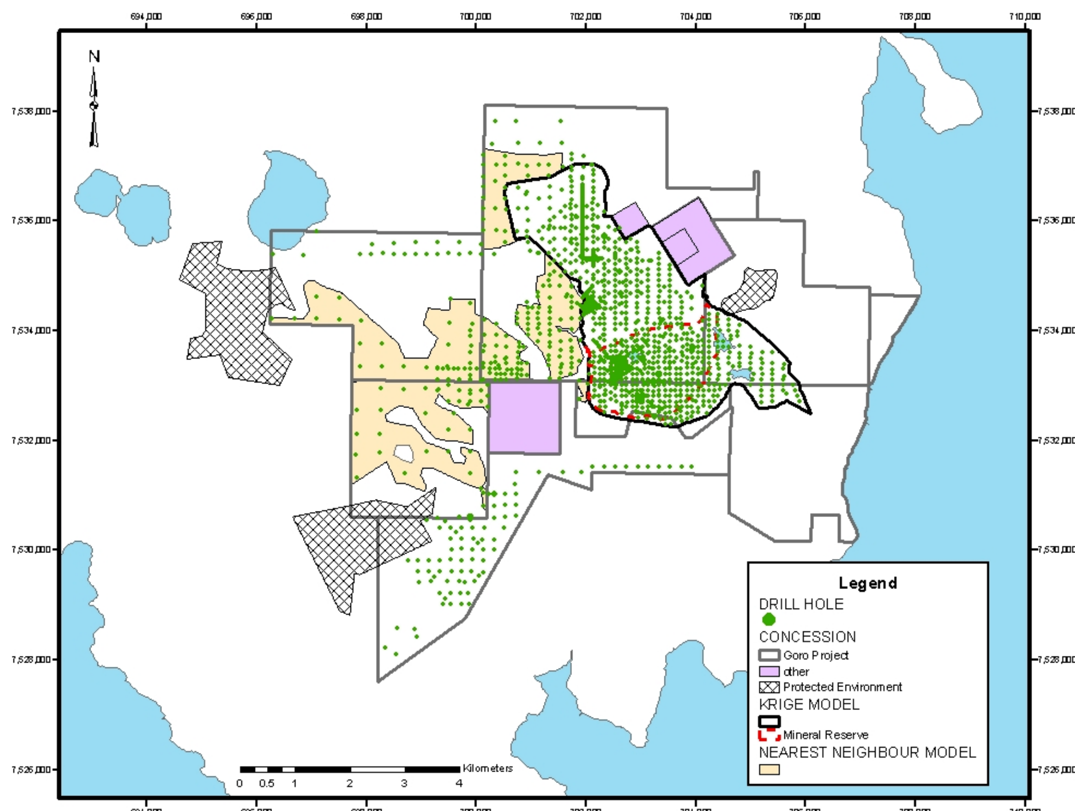
It is the authors' opinion that the exploration data acquired since the initiation of the Goro Project is of good quality and meets recognized industry standards. All external companies involved with exploration for the Goro Project are recognized companies with experience in their respective fields. Reference is made to reference 17 in Item 23 of this Report for a citation related to the present phase of exploration by Inco of the Goro Deposits.

## 13. DRILLING

This section includes a description of the drilling campaigns used for mineral resource and mineral reserve estimation and lists the drilling statistics based on the four different databases used for the mineral resources and mineral reserves estimates for the Goro Project.

Figure 13.1 illustrates the Goro Project concession drill hole location map inclusive of the COFIMPAC (mostly Becker holes, COFREMMI, NQ core size) and the Goro Nickel exploration campaigns of HQ core size and 20cm diameter core. All holes were drilled vertically and no surveys were done in the holes as the majority of them did not exceed 50m in length.

**Figure 13.1: Goro Project Drill Hole Location Map**



## 13.1 Historical Drilling (Pre 1992)

### 13.1.1 COFIMPAC 1968 to 1971

In the drilling campaigns carried out between 1968 and 1971 by COFIMPAC, truck-mounted Becker drills were used instead of core drills because the focus was on the limonite and not the saprolite portion of the profile. A feasibility study undertaken by COFIMPAC indicated that the Becker drills provided rapid and efficient sampling of the complete laterite profile from surface to bedrock. Samples were taken for each meter of drill advance, with material being brought to surface within the pipe stem by compressed air and discharged into a container. Sample recovery was reported to be good and contamination was not believed to be a problem.

The Becker method of sampling/drilling was used throughout the concessions of the Goro Deposits during this period at varying spacing ranging from an 800 m grid to 2.5m detailed line drilling for geostatistical tests. The total number of destructive holes within the seven concessions was 1237, reflecting a total of 44,951m drilled.

### 13.1.2 BRGM-COFREMMI JVS

Between 1971 and 1988, BRGM alone or in association with other partners continued the exploration program, using HQ and NQ core size drilling by FORAPAC. The drill hole data from this work was supplied to Inco as an electronic file.

This method of sampling/drilling was used at varying grid spacing ranging from 800m down to 10m. The total number of core holes drilled by BRGM within the seven concessions representing the Goro Deposits was 225 covering in total 8,339m drilled.

In addition to the HQ and NQ core size drilling eight Benoto ~0.8m external diameter shafts were excavated with a total length of 241m. These provided a bulk sample for metallurgical testing in 1988.

## 13.2 Inco Drilling (Post 1992)

### 13.2.1 HQ Core Size Drilling Practices

During 1996-98 and 2000-2001, a geological technician on site supervised HQ core size drilling and core handling by FORAPAC. The use of polymers in the drilling fluid and the use of controlled drilling practices under the supervision of Goro Nickel technicians ensured that high quality of core recovery was maintained. From 1992 to the end of 2002, some 740 HQ core holes were drilled on a grid of 100 by 200m down to 2m locally for a total of 37,376m to cover the area of the Goro Deposits assumed in the BFS and the footprint for the first 14 years of mining.

In 2002, a pre-development drilling campaign for mine opening and grade control purposes was completed. The drilling grid varied from 12.5m over an area of 200 by 200m at a location near the first months of operation to 25m and 50m over an area covering the initial 18 month of mine production.

### 13.2.2 Larger Diameter Drilling

Goro Nickel began periodic drilling campaigns with a special design of a 20cm - diameter core barrel in 1994. A total of ten holes, five in the central area of the Goro Deposits and five in the southern area of the Goro Deposits, were drilled for a total of 448m. The objective of the first campaign was to provide Inco with a bulk sample containing limonite and saprolite for process metallurgical testing. Geotechnical tests were also conducted on the cores and engineering properties such as detailed density and moisture calculations were obtained to derive the wet and dry tonnage factors. Super polymer drilling fluid was used to maximize core recovery.

In 1996, a second campaign of 20cm diameter core drilling covering five-hole and 164m, referred in the Goro Project as “ERTK”, was conducted on the Kwe concession. The objective of this campaign was to provide a bulk sample for mini pilot plant testing.

In 2001, a third campaign of 20cm holes was initiated to verify the saprolite screen recoveries. Holes were located in areas believed to have the thickest saprolite (principally in Zones 1 and 2, as shown in Figure 6.2). A total of 25 holes were drilled for 1143m on an irregular grid of about 400m, covering the planned footprint of the first 14 years of mining.

#### 13.2.3 Trivelsonda Shaft Sinking 2000 – 2001

In 2000 and 2001, metallurgical bulk samples were collected at various locations within the Goro Deposits for processing at the integrated pilot plant constructed in 1999 and located near the Baie du Prony. The bulk samples were excavated by a shaft-sinking Trivelsonda rig from FORAPAC. One meter diameter shafts were dug at three locations selected on the basis of the chemistry of the laterite. The locations included areas of high alumina and high magnesia that both consume acid and are therefore deleterious in the metallurgical process. The shafts were sunk on a 12.5m grid of 25 holes at each site. Before the shaft excavation took place, HQ size pilot holes were drilled to ascertain the presence of saprolite and determine its chemistry. The Trivelsonda rig was limited to a penetration depth of about 45m and was also stopped by excessive water and boulders larger than 75cm in diameter. Most of the 84 shafts, given these limitations, did not sample the full saprolite profile.

### 13.3 Drilling Statistics

The information recorded in Goro Nickel databases comes from ten different types of drill holes. The information gathered from each type of drilling varies significantly and it is the authors’ opinion that the more recent drilling (post 1998) yields the most valuable information for mineral resources modeling.

The following table identifies different drilling techniques used within the Goro Deposits:

**Table 13.3: Drill Hole Statistics**

| HOLE TYPE | No. of Holes | Length (m) |
|---|---|---|
| Destructive | 16 | 542 |
| 20 cm. Diameter core | 40 | 1755 |
| Becker | 1221 | 44,407 |
| Inco core 1998 few are not screened | 740 | 37,376 |
| BRGM Core | 225 | 8,359 |
| Benoto holes (0.8m diameter) | 8 | 241 |
| Trivelsonda (1.0m diameter) | 84 | 3042 |
| TOTAL | **2334** | **95,722** |

All recent drilling campaigns have been rigorously monitored and supervised, logged, photographed and labelled by Goro Nickel representatives. Several visits by personnel under the responsibility of R.A. Horn have witnessed the drilling activities sporadically and confirm that recent drilling techniques and procedures in use at the Goro Project meet, and in some cases, exceed industry standards.

# 14. SAMPLING METHOD AND APPROACH

A substantial part of the area of nickel laterite in the Goro Deposits has been drilled as shown in Figure 13.1.

## 14.1 Sampling Methods

### 14.1.1 COFIMPAC 1968 to 1971

During this period, samples were taken at every meter of drill advance. The sample recovery was good and contamination was not reported as being a problem. At the drill site, the cuttings from each meter of advance were collected in a tub (30Kg), dumped on a metal-plate, mixed, and split twice; the reject from each split was discarded, and one quarter of the remaining lot was bagged for shipment to a laboratory. The quartered samples were then dried and pulverized. The adequacy of the sampling was verified by extensive tests done by Inco personnel. The drilling was assumed to sample effectively the –150mm size fraction but not the +150mm material. This assumption was supported by the fewer boulders encountered in the destructive drilling when compared to core drilling. Because no weights were recorded for each drilling pass the assays could only be weighted by volume instead of mass. Furthermore, since the Becker samples were mainly chip samples, no screening was possible for the transition or the saprolite to test the possibility of beneficiation.

### 14.1.2 BRGM-COFREMMI 1972 – 1988

There are no original assay sheets available to check the database for data entry errors of the BRGM-COFREMMI core drilling campaigns. No sample preparation/assaying protocol could be found. There is very little core left at site that could be examined. Moreover, it is believed that succeeding geological teams through the years have so heavily sampled the remaining core boxes that it is pointless to draw any conclusions from this material, and that re-assaying of the core will not yield any useful conclusions.

The original BRGM database had reported the analyses as elemental values for Mg, Si, Al, Mn, and Cr instead of the actual format of analytical results that are expressed as oxides for MgO, $SiO_2$, $A1_2O_3$ and $Cr_2O_3$. The original values were thus modified to oxides. The sample depths and Fe grade were rounded to one decimal place. Based on a review of the electronic database, it can be seen that the sampling interval conformed to the geology contacts with samples of up to 5 meters in the limonite zone and down to 10cm in the saprolite. None of the samples were screened.

For the eight Benoto shafts sunk in 1988, the first 10 to 15m of non-mineralized material was not sampled. In the mineralized portion of the profile, samples were taken every meter and bagged in 1 cubic metre bags. A small part of the sample was quartered on a plate to obtain three samples of 2Kg each. Wall sloughing and water inflow were noted during drilling. This may have resulted in some cross-contamination. The Benoto shafts generally stopped in the transition layer.

### 14.1.3 Inco HQ Core Size Drilling Sampling Practices 1996-98

Between 1996 and 1998, a geological technician on site supervised the HQ core size drilling by FORAPAC and the core handling by Goro Nickel. The technician logged the core and completed a log for each sample interval that contained the geological description, core recovery, sample interval and sample tag. The core was immediately boxed and delivered to the central core handling and sample preparation facility.

All sampling intervals from the exploration drilling were divided on geological contacts. In the absence of geological discontinuities the sampling intervals were usually 1 meter in length.

Upon receipt of core for logging, the geologist verified the meterage tags in the core boxes and the recovery length, described the core and marked the sampling intervals. The data was written on a preformatted sheet for an electronic database. The sampling intervals were marked on row dividers on the core box.

The core samples were collected from half-core splits, one half being kept for historical record and one half being used for wet and dry tonnage factor estimation, grade and chemistry estimations. Only a few holes were wet screened for size analysis.

#### 14.1.4 Inco Sampling Post 1998

Similar logging and sampling methodology as the 1996-1998 HQ core drilling campaign has been conducted by Inco since 1998, with the exception of wet screening of transition and saprolite samples for size fraction analysis. From 1998 onward, all sample intervals that had more than one size fraction were systematically screened at 150mm, 50.8mm and 6.35mm.

#### 14.1.5 Larger Diameter Drilling (ERTK) Sampling Practice

The sampling frequency of the Goro Nickel 2 cm diameter core drilling was 1m in the limonite. Different types of laterite were sampled separately. The sampling interval was often less than one meter in the transition and saprolite layers as the sampling interval was intended to reflect more details in the complex mixture of clay matrix and boulders.

For the 20cm diameter core, two "V" cuts were taken on opposite sides of the core. The volume of the "V" cut samples represented 13.4% of the core volume. In sections where "V" cuts could not be taken, such as bedrock or in boulders, transverse slices were taken instead. Goro Nickel dried and split the samples and sent them to the Service des Mines laboratory in Nouméa, Als Chemex in Brisbane, or Amdel Lab in Adelaide. For each sampling interval, two analytical results were available. The arithmetic average of the two results was entered into the database.

In 2001, sampling procedures differed from previous sampling procedures because the entire sample was wet screened using a vibratory screen. Sample intervals were determined by lithology contacts. The screens used were 150mm, 100mm, 50.8mm, 25.4mm, 19.05mm, 12.7mm, and 6.35mm. Only the +150mm and the –6.35mm were reduced to 25% because of the excessive amount of material or water within this size range. All other size fractions were first crushed to 5mm before being riffled and pulverized.

#### 14.1.6 Trivelsonda Shaft Sinking Sampling Practice 2000 – 2001

Large samples for metallurgical testing at the pilot plant were individually bagged into 1 to 1.5 tonne bags. A 15cm wide horizontal sampling channel bar was placed across the hopper lip into which the excavated material was dumped into ~1 tonne bags for feed to the pilot plant. Approximately two channels per sample were collected for each ~1 tonne sample bag. The transition and saprolite samples were wet-screened for size recovery and chemical analysis. A total of 1,158 dry tonnes of material was collected during these campaigns.

### 14.2 Sample Quality

The following six factors can materially impact the quality of the sampling results:

1. Screen recovery factors
2. Layer chemistry bias in destructive drilling
3. Drill recovery
4. Density (tonnage factor)
5. Truncated holes
6. Core size

The following represents a discussion of each of these six factors.

#### 14.2.1 Screen Recovery Factors

In laterite deposits developed over certain types of bedrock, substantial gains in quality of the material fed to a process plant can be achieved by rejecting the coarse fraction of the excavated saprolite. As described earlier, the saprolite layer, and to some extent the transition zone of the Goro Deposits, are composed of pebble to boulder - size bedrock fragments in a silt to clay matrix. The coarse material is high in magnesia and usually low in nickel, while the fine matrix is low in magnesia and high in nickel content. Therefore, by mechanical treatment, stationary or vibrating screens or by wobblers, the relative concentration of boulders can be reduced and the content of elements of economic value, nickel and cobalt in the screened material, can be increased.

Without further information, it is assumed for the purpose of mineral resource estimation that the core sample represents the equivalent of the Run of Mine ("ROM"). Therefore, with some limitation on the largest boulders in the profile, the screen size distribution of the core is assumed to represent the equivalent screen size distribution of the material to be delivered to the process plant.

Beginning in 1994, studies on saprolite beneficiation of core were carried out on the Goro Deposits. As of December 2002, 765 core holes had been screened and analyzed at 150mm, 50.8mm and 6.35mm size fractions. This data is sufficient to recognize four distinct zones within the Goro plateau and to derive the optimum size fraction to be fed to the process plant. The four zones (see Figure 6.2) are:

- Zone 1, located in the southwest corner of the Goro plateau, has the highest nickel grade and second highest cobalt grade with the highest –50.8mm screen recovery;

- Zone 2, located immediately to the East of Zone 1 and to the West of Zone 3, has the lowest screen recovery, nickel grade and cobalt grade, and the highest magnesia content;

- Zone 3, located essentially on Robert and Cascade concessions, has the highest cobalt and second best nickel grades and screen recovery; and

- Zone 4, located North of Zones 1, 2 and 3, is an area that has very little sampling information on screen size fractions. The screen size fractions in Zone 4 are derived from factors applied to the destructive and to the non-screened core holes.

In Zone 4, in the absence of better information, the mineral resources estimates of the different size fractions is factored from the ROM grade using the average recoveries and chemistry from Zones 1,2 and 3. The factors were similar to the co-located destructive and core holes drilled in 2001.

The results of the saprolite screen analysis study are shown on Table A-1 in Appendix A are referenced from the Goro Deposits mineral resources block model. The transition layer upgrading results are also shown on Table A-2 in Appendix A while Tables A-3 and A-4 in Appendix A show the average grade for Zones 1, 2 and 3.

Earlier studies based on fewer holes and similar results from the most-recent core drilling support the conclusion that the -50mm size fraction is the optimum size fraction of the saprolite to feed to the process plant to meet the grade and chemistry constraints for optimal production requirements and to efficiently manage the available mineral resources.

### 14.2.2 Layer Chemistry Bias in Destructive Drilling

The chemical analyses of samples collected during the destructive drilling was compared with the core drilling chemistry and was found to be biased in the saprolite layer but not in other layers. Correction factors were estimated between destructive and core drilling for chemistry and screen recovery in the saprolite. Checks were done to verify the robustness of the factors and it was concluded that they consistently fell within a narrow range and could be applied globally but not to individual samples. It is to be noted that the corrected saprolite chemistry from the destructive drilling was not used in the mineral resources estimates in Zones 1, 2 and 3 but were used in Zone 4 and in the nearest neighbour model. The correction or adjustment factors applied to the destructive drilling results for Zone 4 are shown in Table 14.2.2 below.

**Table 14.2.2: Adjustment Factors for Saprolite Chemistry in Zone 4**

| | Destructive adjustment factor |
|---|---|
| %Ni | 1.282 |
| %Co | 0.805 |
| %Fe | 0.931 |
| %$SiO_2$ | 1.146 |
| %MgO | 1.087 |
| %$Al_2O_3$ | 0.808 |
| %$Cr_2O_3$ | 0.715 |
| %MnO | 0.554 |

### 14.2.3 Drill Recovery

Drill recovery is a measure of how much of the laterite in samples is recovered while drilling. It is an important factor in laterite evaluation because poor recovery may be caused by loss of fine material, and consequently may cause unrepresentative lower nickel and cobalt grades may result in the sample or may result in erroneous material densities.

Goro Nickel has implemented strict monitoring procedures whereby a Goro Nickel technologist is present at all times during the drilling operation to record metre by metre the drilled length versus the drill recovered length. Furthermore, contractors are required to redrill any hole when more than three consecutive metres of drilled length yield <70% recovery in limonite and <85% recovery in saprolite, excluding voids.

At the Goro Deposits, core recoveries have been measured systematically since the introduction of core drilling in 1994 and have been closely monitored during drilling operations. Anomalous samples in all layers have been found with recoveries above 100 percent. This phenomenon is recognized in New Caledonian and most other laterites. The most likely explanation for this excess material is a process called extrusion where material with a constituency of paste is partly cut by the core bit and partly flows inside the core tube. This occurs when drilling advances faster than the bit cutting action in soft formations. Therefore, in a hole of HQ size, where the outside hole diameter is 96 mm and the core diameter is 63.5mm, as much as 151% core recovery can be expected if none of the material is cut by the bit face but rather is pushed into the core barrel during drilling. Extrusion is not observed as frequently in the saprolite where there are boulders nor in the 20 cm core holes.

Theoretical adjustment factors may be required even when core recovery is reported at 100% or less. Recoveries of greater than 100% may be due to another factor. When core is lost down the hole, it may be picked up in the next drilling run and therefore be missing from the first run or core which would show a recovery of less than 100%, and added to the subsequent run that could report more than 100%. Using a global recovery average for all sampling intervals in the laterite tends to compensate the local variability in core recovery by factoring the entire drilled length versus the entire length of the recovered core for specific formations.

The core recovery by layer is stated in the Table 14.2.3 below:

**Table 14.2.3: Core Recovery by Layer**

| Layer | Recovery % | |
|---|---|---|
| | Average Uncorrected | Average Corrected |
| Yellow laterite | 132.4 | 87.7 |
| Transition | 119.9 | 79.4 |
| Saprolite | 92.4 | 61.2 |

In Table 14.2.3 above, the recovery was corrected by applying a constant 151% extrusion on all samples. This is an extreme scenario, because extrusion does not occur at all times. In fact, the true recovery is unknown and the core recovery is much higher than the average corrected number but lower than the average uncorrected stated above.

### 14.2.4 Density (Tonnage Factors)

The density of the different layers of the saprolite cannot be measured by known conventional techniques using Shelby tubes. The physical parameters of *in situ* density, moisture and ore size characterization are therefore derived from core drilling and are, accordingly, affected by the quality of core recovery. Global averages can be derived from the information collected during the systematic sampling of the core, for dry tonnage factors and wet tonnage factors, and moisture for an entire layer based on dry weights and moisture contents recorded while treating the core samples. This sampling method assumes that there is no moisture loss or gain between the time the sample is taken out of the core tube and the time it is weighed. A second assumption is that the volume sampled remains constant. Half of the core is weighed and the other half is kept for reference.

For the purpose of these calculations, both assumptions are believed to have a negligible impact on the average value calculated. This has been demonstrated by one series of 25 ERTK 20cm diameter holes that were fully sampled and compared to a global average of HQ core holes located in the same general area.

The estimates of the tonnage factors, densities or and moisture contents used in the post-BFS mineral resources model are shown in Table A-5 in Appendix A.

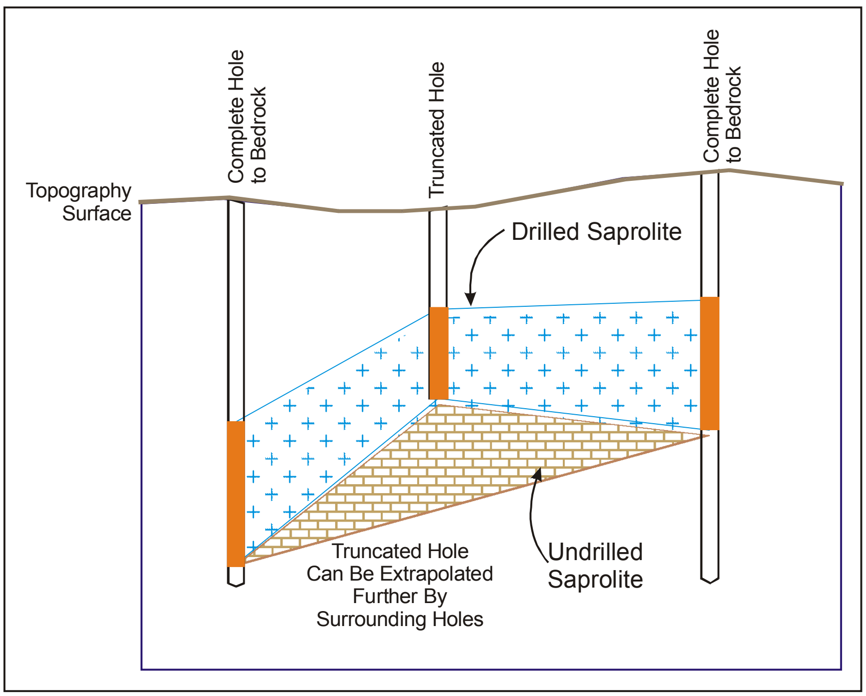
### 14.2.5 Truncated Holes

Drilling has not consistently penetrated the entire laterite profile due, in part, to the focus of the 1968-70 exploration drilling on the limonite. From the database, 637 holes located in the seam model and core holes did not fully penetrate the saprolite layer and 67 holes did not penetrate the entire transition layer. While the effect of this constraint on the sampling of the transition layer is insignificant, there was a need to assess if it added a significant influence on the saprolite layer mineral resources estimates.

The approach taken to assess the potential bias was to create a 3 dimensional surface from the holes that penetrated the entire profile and to compare it with a 3 dimensional surface from all the holes. The following two cases were studied:

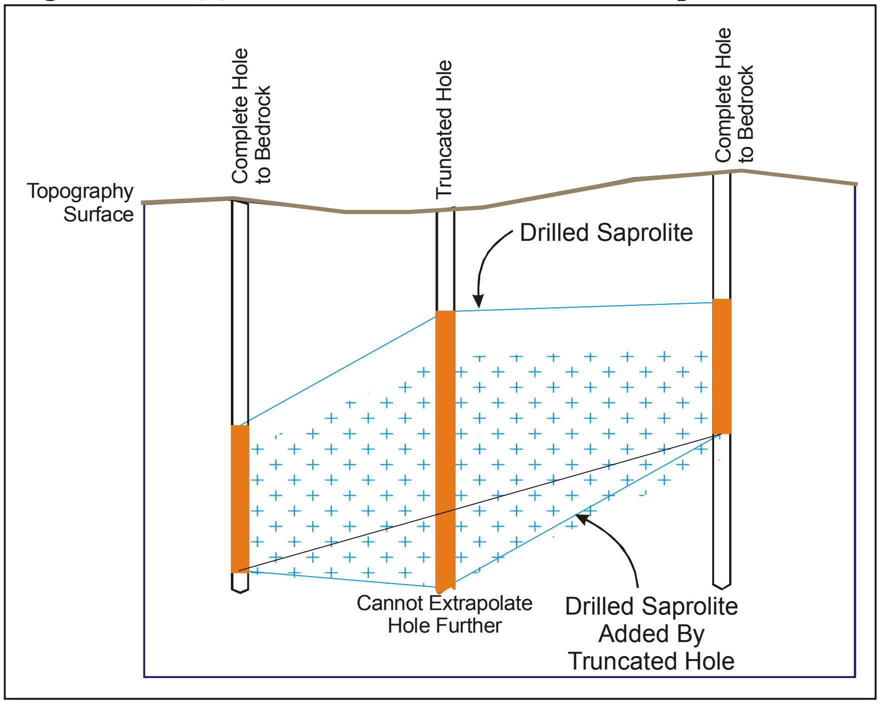
Case 1 (Figure 14.2.5(a))**:** The profile generated by all holes clearly showed that the truncated holes did not penetrate the full profile and accordingly, estimates of the volume of laterite would be understated using this data. The true volume of the laterite can be estimated by adding the material between the bottom of the truncated holes and the bedrock contact that is based on the projections from the surrounding non-truncated holes.

**Figure 14.2.5(a): Truncated Hole Case 1 – Saprolite Added**



Case 2 (Figure 14.2.5(b))**:** The truncated hole had intersected deeper saprolite than the surrounding non – truncated holes. The truncated hole is left in the database and no extension is added.

**Figure 14.2.5(b): Truncated Hole Case 2 – No Saprolite Added**




The method described above in Figure 14.2.5(b) was adopted for the volume calculation of the saprolite layer in the December 2002 resource model in Zone 4 only. Where the saprolite has been extended the mineral resource classification is classified as Inferred.

### 14.2.6 Core Size

A drilling campaign of twenty-five, 20cm core holes was conducted to study intermediate size fractions in large diameter sampling compared to the sampling data from HQ core size drilling. The holes were distributed on a 400m grid covering the first 10 years of the proposed mining footprint. The entire sample interval from these holes, if it contained heterogeneous size fractions, was screened at 150, 100, 50.8, 25.4, 19.05, 12.7, and 6.35mm.

The chemistry and recovery numbers in the 20cm diameter hole campaign are generally similar to the HQ core size results confirming the results of the HQ core size that the -50mm screen size fraction remains the most appropriate size for process plant feed.

### 14.3 Sampling Interval and Quality

Several factors have influenced the selection of the sampling interval and the quality of sampling in the exploration database. This is because several different companies conducted the sampling and each company had somewhat different sampling methods and objectives.

Inco chose a sampling interval that usually did not exceed 1 meter in length in the Becker RC destructive drilling but without a break of the sample at the geological contacts. For core drilling the sampling interval was usually one meter in length but was broken at geological contacts. The sampling interval was reduced in the boulder section of the saprolite to as small as 10cm length in order to assess a particular mineralogy or aspect of the core. A one meter sampling interval was thought to be appropriate for the estimated degree of homogeneity of the red and yellow laterite. The excavation equipment chosen for the mine cannot extract ore below a limit of less than one metre. In the 1971-1998 BRGM core drilling campaign, there was no clear consistency in the chosen sampling length but the samples were always broken at recognized lithology breaks or relevant geological features in the core.

A bulk mining method is planned for the extraction of the laterite as plant feed to the Goro Project feed preparation plant (FPP). It is expected that all material encountered below the first occurrence of two consecutive meters above cut-off will be mined to the assumed bedrock contact and delivered to the FPP. There will be no attempt to selectively mine different layers. There will be upgrading of the saprolite where it contains boulders to –50.8mm at the FPP. As a result, the sampling interval is only critical in that it takes into account the seven main layers identified in the profile discussed previously in Item 11 of this Report.

All holes in the database have been geologically coded. Codes for each hole have been entered into a “minor table” of each database as discussed in Item 16 of this Report.

Geological coding is based on the chemistry of the sampling intervals of both the Becker holes and the core holes. Minor discrepancies have been discovered between yellow laterite and transition in the coding in the geological logs of the core holes. These are considered to have an insignificant influence on the mineral resource estimation, because it is planned that the two layers will be mined together.

Table 14.3 illustrates the geological layer statistics based on combined core drilling and non-truncated Becker drilling intercepts.

**Table 14.3: Thickness of Five Geological Layers based on Drill Hole Information**

| THICKNESS (m) | ICP + ISH | LATR | LATJ | TRN | SAP |
|---|---|---|---|---|---|
| Min | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 |
| Max | 36.0 | 34.5 | 53.0 | 34.4 | 53.2 |
| Mean | 3.7 | 6.2 | 19.2 | 2.5 | 9.3 |
| Variance | 10.1 | 21.9 | 75.2 | 10.4 | 69.5 |
| Skewness | 2.4 | 1.1 | 0.2 | 2.5 | 1.4 |

In Table 14.3, the iron cap and iron shot layers (ICP and ISH) have been combined.

There were about 87,000 sample intervals as of December 2002 that are essentially true thickness estimates of the laterite because the vertical drilling intersects mostly a horizontal weathering profile. Intervals in saprolite may contain up to eight different size fractions that have been analyzed for all elements, these individual size fractions have been combined into cumulative fractions based on their dry weight and chemistry.

# 15. SAMPLE PREPARATION, ANALYSES AND SECURITY

During the COFIMPAC campaigns, Inco had a sample house and laboratory on site. Since 1992, sample preparation facilities have been used at Goro Nickel's Grand Lac base camp to produce the majority of sample pulps that have been sent to various analytical laboratories.

In all of the Inco exploration campaigns, Inco analytical experts periodically visited the field operations to maintain Inco standards of sample preparation and analytical procedures. The most recent technical audits were conducted in 2001 and 2002.

## 15.1 COFIMPAC Campaigns

During the COFIMPAC Project (1968-1971), the volume of the material excavated by the Becker drill was approximately 12,000 cubic centimetres per meter of advance. The average weight of sample was 30kg. The samples were systematically reduced at the field sample preparation facility to samples of about 2kg weight, which were then sent to the sample house and assay laboratory constructed to Inco's specifications in Nouméa. The adequacy of the sampling procedures was verified by extensive tests as documented in the COFIMPAC feasibility study report.

The drill cuttings for each meter were quartered to less than 10% of the original volume, and the quartered sample was sent to the laboratory for drying and preparation of the assay pulp. Finally, 0.2 grams of pulp were used for analysis. The dried sample was dissolved in perchloric acid, and the concentrations of nickel, iron and cobalt were determined by atomic absorption spectroscopy, a chemical analytical method.

The analytical work was monitored against standard samples of known chemistry that had been carefully analyzed at Inco's research and development facility located in Mississauga, Canada. Every twentieth sample was routinely sent to a commercial laboratory in Australia for reanalysis, and the two sets of analyses were compared in New Caledonia and at Inco's facility in Mississauga, Canada. The standard deviation of nickel and iron analyses in the Nouméa laboratory was three to four per cent.

Inco exploration staff carried out an experiment during the COFIMPAC Project to determine the reproducibility of chemical analyses of different samples taken using the method of cutting the laboratory sample out of the one-meter or approximately 30 kilograms of Becker drill returns. It was concluded that the sampling method was satisfactory and that the standard deviation of an analysis reported on a drill log was approximately 0.1 for nickel, and 1.9 for iron including the analytical error.

## 15.2 BRGM Drilling Campaigns

BRGM core drilled the Goro plateau after termination of the COFIMPAC Project in 1971. The Goro Project inherited the electronic databases of BRGM core drilling when CMX was formed and the BRGM concessions were transferred to CMX in 1992. Although there is little documentation available on the sampling method, sample preparation and analytical controls used by BRGM, there are no reports of quality control problems and we have no reason to believe there were significant problems with the drilling, sampling and analytical methods employed. The BRGM samples were analyzed at the Services des Mines et de l'Energie laboratory in Nouméa.

### 15.3 Goro Nickel Exploration Sample Preparation Facility

Since 1992, all samples have been prepared at the Goro Nickel sample preparation facility at Grand Lac, within the Goro Deposits. In 2000 and 2002, some overflow limonite samples were sent for preparation to Geocal, an independent laboratory located in Nouméa. In 2002 Goro Nickel used a third sample preparation facility, Ingélab, also located in Nouméa.

In 1998, systematic wet screening of saprolite was introduced for regular exploration core drilling and large diameter core drilling or Trivelsdonda shaft sampling. Prior to this period, the saprolite half core was crushed and assayed and assumed to be equivalent to Run of Mine material.

In 2001, modern ring pulverizers replaced the former disc pulverizer and a new jaw crusher was introduced.

In 2001, an ITSL audit was conducted of the sampling and analytical procedures at Goro Nickel's internal exploration sample preparation facility. The audit concluded that Goro Nickel sample preparation facility was adequate but needed more rigorous record keeping on equipment maintenance, procedures and employee training. A follow up ITSL audit in 2002 was conducted. Although the conclusions of the follow up ITSL audit were not available as of the effective date of this Report, discussions with the auditors indicated that they would not recommend material changes that would affect the quality of the data gathered in 2002. Reference is made to reference 16 in Item 23 of this Report for information regarding the 2001 ITSL audit.

### 15.4 Goro Nickel HQ Core Size Sample Preparation

Sampling is carried out by either a Goro Nickel geologist or by a sampling assistant. When the samples do not require wet screening, the core is divided in half, either using a putty knife for the soft material or a core saw for the rocky material and put into trays for drying. Care is taken to recover the fines from the core trays during sampling. If the interval is to be wet screened, then half of the core is collected and sent to the wet screening station before drying. In either case the splitting of the core is always perpendicular to the main core features such as (faults and lithology). The remaining half of the core is covered with plastic for archiving in the same shipping containers used for the long-term storage of the pulps.

The limonite samples are weighed and then dried in ovens at 105°C for 12 hours for moisture determination.

The saprolite fractions are classified according to the categories in Table 15.4 below:

**Table 15.4: HQ Core Size Saprolite Screen Fractions**

| Saprolite Fraction | Size |
|---|---|
| P | >150 mm |
| A | <150 mm and >50.8 mm |
| B | <50.8 mm and >6.35 mm |
| C | <6.35 mm |

### 15.5 Goro Nickel Wet Screening

There are four wet screening stations at the Grand Lac facility, three of which are for the HQ core size and one for the large samples that are either collected from 20cm diameter coring or the Trivelsonda one meter diameter shaft sample.

#### 15.5.1 HQ Core Size

The three HQ core size stations were commissioned in 2001. Prior to the commission of these stations, the HQ core size was treated at the preparation station for large samples. At present, the core samples are wet screened at 150mm, 50.8mm and 6.35mm using a vibratory wet screen. The 6.35mm is then pressure filtered to reduce the drying time. Between sample runs at both screening stations all screens are dismantled and washed before another sample is processed.

Each size fraction is identified on aluminium tags by the classification categories referred to in Table 15.4 followed by a number that is identical for the same sample interval. A sampling interval can yield, if all size fractions are present, four samples with the identical numeric identifier but with separate alpha prefixes. All size fractions are sent for drying.

#### 15.5.2 Large Samples

The large diameter core and the Trivelsonda samples are treated on a static screen to first remove the +150mm and +100mm. The 100mm screen product is then wet screened into several size fractions (50.8, 25.4, 19.05, 12.7 and 6.35mm) on 36" diameter screens. Only the 6.35mm has to be riffled (25% of the sample is kept) due to the excessive amount of material to be dried.

Approximately one percent of the material is lost during the handling and screening. This loss is to be expected with the large mass of samples (70 + Kg per sample) that has to be treated. Between sample runs all screens are dismantled and washed before another sample is processed.

Each size fraction is identified on aluminium tags by a prefix followed by a number that is identical for the same sample interval. A sampling interval can yield, if all size fractions are present, eight samples with the identical number but with separate alphabetic prefixes. All size fractions are sent for drying.

### 15.6 Goro Nickel Sample Drying

The samples are dried in an oven at 105°C. Depending upon the sample type, samples were dried for either 5 or 15 hours.

A dry weight is recorded electronically into a spreadsheet. Samples are stored in a warming oven until they are ready for preparation. Dried samples are crushed to 5mm using a jaw crusher. The jaw crusher is cleaned with compressed air between samples.

The crushed samples are returned to the drying oven to remove any moisture pickup and then the entire crushed sample is pulverized.

### 15.7 Sample Pulverization

Pulverization times are dependent on the type of material that has been entered on the sample sheet, which accompanies the samples in the form of a coded short form (in bold below) and a colour-coding scheme. Both pulverization stations have a chart similar to Table 15.7, which includes the colour-coding scheme:

**Table 15.7: Sample Pulverization**

| **Lithology/Material Type** | **Pulverization Time** |
|---|---|
| **ICP** – Iron Cap | 10 minutes |
| **ISH** – Iron Shot | 10 minutes |
| **LOB** – Red Laterite | 4 minutes |
| **LIM** – yellow laterite | 3 ½ minutes |
| **TRN** – Transition | 3 minutes |
| Saprolite – Fractions “**P**” and “**A**” | 12 minutes |
| Saprolite – Fraction “**B**” | 3 minutes |
| Saprolite – Fraction “**C**” | 3 minutes |
| **BRK** – Bedrock | 12 minutes |

At the end of the pulverizing cycle, the pulp material is sampled using a metal spoon to obtain three splits, one for external laboratory analysis, and two for storage.

It is assumed that the pulverized material is homogenous.

### 15.8 Analytical Laboratories

Three laboratories were used between 2000 and 2002 for analytical work. All three laboratories, as discussed below, have internal quality control checks but only two, ALS Chemex and Amdel, are certified to an ISO standard (9002 and 17025, respectively) by an outside regulatory body (NATA - National Association of Testing Authorities, Australia).

#### 15.8.1 ALS Chemex Laboratory, Brisbane, Australia

The bulk of Goro Nickel exploration samples are analyzed at the industrial minerals section of the ALS Chemex laboratory in Brisbane. The laboratory’s quality control and assurance manual and operations were examined in an audit conducted by ITSL which concluded that, overall, the laboratory was acceptable for laterite sample ICP analytical procedures and results.

#### 15.8.2 Service des Mines et de l’Energie Laboratory, Nouméa (SME)

SME has a mineral laboratory in Nouméa that Goro Nickel has used since 1992. An audit of this laboratory by ITSL was conducted most recently in 2001. The laboratory quality control procedures and analytical methods were reviewed and the audit concluded that, while the laboratory is limited for throughput for commercial samples, it provides acceptable results for ICP analyses of laterite samples. The SME mineral laboratory is currently working towards ISO 9002 accreditation through the Comité Français d’Accréditation.

### 15.8.3 Amdel Laboratory, Adelaide, Australia

The Amdel laboratory was the subject of an audit by ITSL in 2001. The laboratory includes both a full analytical laboratory and facilities for metallurgical testing of ores on a mini-plant scale. Laboratory documentation of the methods used for Goro Nickel exploration samples was found to be up to date and the methods used are accredited. All the necessary documentation for accreditation is centrally filed, in order and current. The audit concluded that the laboratory was acceptable for use of Goro Nickel ICP analyses of nickel laterite samples.

## 15.9 COFIMPAC – Investigation of Sampling Errors

As documented in the COFIMPAC feasibility study report of 1970, but not verified by the authors of this Report, an experiment was carried out to determine the reproducibility of Becker primary drill sample chemical analysis using the method of weighing approximately 30kg per meter of hole, and splitting of 2kg of sample to be submitted to the laboratory for preparation of the analytical sample.

In the described sampling procedure, the cuttings from each meter of advance were collected in a tub, dumped on a metal-plate, mixed, and split twice; the reject from each split was discarded, and one quarter of the remaining lot was bagged for shipment to the laboratory. For the sampling experiment, three samples were taken from each primary sample. The latter was quartered down in the usual manner to prepare the first or control sample; a check sample was cut out from the reject of the final quartering step, and a second check sample was prepared from the reject yielded by the first split of the primary sample. The sampling was done as the holes were being drilled to eliminate the possibility of segregation by partial loss of fines from the reject stored at the drill site. The database for the experiment consisted of 59 primary samples, ranging in grade from 0.35 to 3.5% nickel. The control samples were analyzed in duplicate, and the check samples were analyzed in triplicate.

An analysis of variance of the nickel analyses was carried out on the primary sample and three splits to determine if there was a statistically significant difference between results of the splits and the primary sample. The error between primary samples was significant, but the error between splits was not significant and indicated that differences were random and the method of sample reduction was not biased.

## 15.10 Inco Analytical Control Checks on COFIMPAC Drill Pulps

In 1993, Inco re-analyzed most of the Becker drilling sample pulps taken during the COFIMPAC Project (1968-1971) in the mineralized zones to expand the database to eight elements for process modeling (Ni, Co, Fe, $SiO_2$, MgO, $Al_2O_3$, MnO, $Cr_2O_3$) using the SME mineral laboratory. SME used sodium peroxide for digestion with atomic absorption analysis. Old analytical results based on a perchloric acid digestion and concentrations of nickel, cobalt and iron based on atomic absorption were replaced. During that check analysis program, it was determined that a bias existed between the two analytical results of nickel and cobalt for the limonite and saprolite samples. Correction factors, based on the control samples that were re-analyzed, were applied to the Becker samples located within the seam model.

It should be noted that for the mineral reserves estimate only the core data is used for the saprolite grade and chemistry and to this date less than 20% of samples over the entire sample population in databases have been adjusted by a correction factor for chemistry.

## 15.11 Recent Exploration Analytical Control Checks by Goro Nickel

In order to ensure that the analyses from the exploration programs beginning in 2000 were reliable and could be used in the calculation of the Goro Project's mineral resources estimates, the following two check exercises were carried out as part of the 2001 ITSL analytical audit:

1) Pulp Re-analysis: 5% of the pulps were re-analyzed by the same or another laboratory

2) Half Core Test: The remaining half of the core of selected drill holes was analyzed to find potential biases in wet screen sample preparation

### 15.11.1 Pulps Re-analysis

The results of several check exercises done on Goro Nickel samples were closely examined with respect to all of the elements reported. The relevant external check assay groups included:

- ALS Chemex replicate samples
- Check analyses of pulps from the year 2000
- Analyses of half core samples that were prepared by two different methods

Since 2000, splits of the original pulps representing about 5% of the total pulps generated were selected from the Goro Nickel storage based on the type of laterite, grade and the different exploration areas. One set of samples was originally analyzed at ALS Chemex. A second set of samples was analyzed at SME. For both groups of samples, the check exercise included one pulp going to Amdel.

In most cases, as illustrated in Table 15.11.1 below, there was reasonable agreement between the Amdel, ALS Chemex and SME laboratories.

**Table 15.11.1: Check Analysis between Laboratories**

| | | Checks Group 1 (321 samples | | Checks Group 2 (235 samples) | |
|---|---|---|---|---|---|
| | | **Amdel** | **ALS** | **Amdel** | **SME** |
| Nickel | Mean | 1.477 | 1.494 | 1.366 | 1.363 |
| | Variance | 0.638 | 0.626 | 0.585 | 0.478 |
| Cobalt | Mean | 0.095 | 0.099 | 0.111 | 0.111 |
| | Variance | 0.041 | 0.043 | 0.038 | 0.035 |
| Iron | Mean | 25.9 | 26.0 | 31.5 | 32.1 |
| | Variance | 362.6 | 361.9 | 366.0 | 387.0 |
| Silica | Mean | 27.4 | 27.5 | 21.3 | 21.5 |
| | Variance | 418.7 | 418.0 | 445.5 | 446.1 |
| Magnesia | Mean | 15.5 | 15.7 | 11.5 | 11.7 |
| | Variance | 217.8 | 222.2 | 200.4 | 204.8 |
| Alumina | Mean | 3.48 | 3.45 | 4.49 | 4.45 |
| | Variance | 13.34 | 13.49 | 19.95 | 19.14 |

### 15.11.2 Half Core Test

Goro Nickel generally collects samples of half of the core and keeps the remainder as a back-up. However, in 2000, back-up samples were not retained for 939 of the core holes in saprolite. These samples had not been wet-screened but the half core had been crushed and pulverized (representing the "QC" samples in Figures 15.11.2(a) and 15.11.2(b) below). Since 2000, the other half of the core have been wet screened and then analysed in accordance with the set procedure (representing the "REC" samples in Figures 15.11.2(a) and 15.11.2(b). This change provided a further opportunity to assess if any bias existed between the two sample preparation techniques or if a nugget effect existed between both half cores.

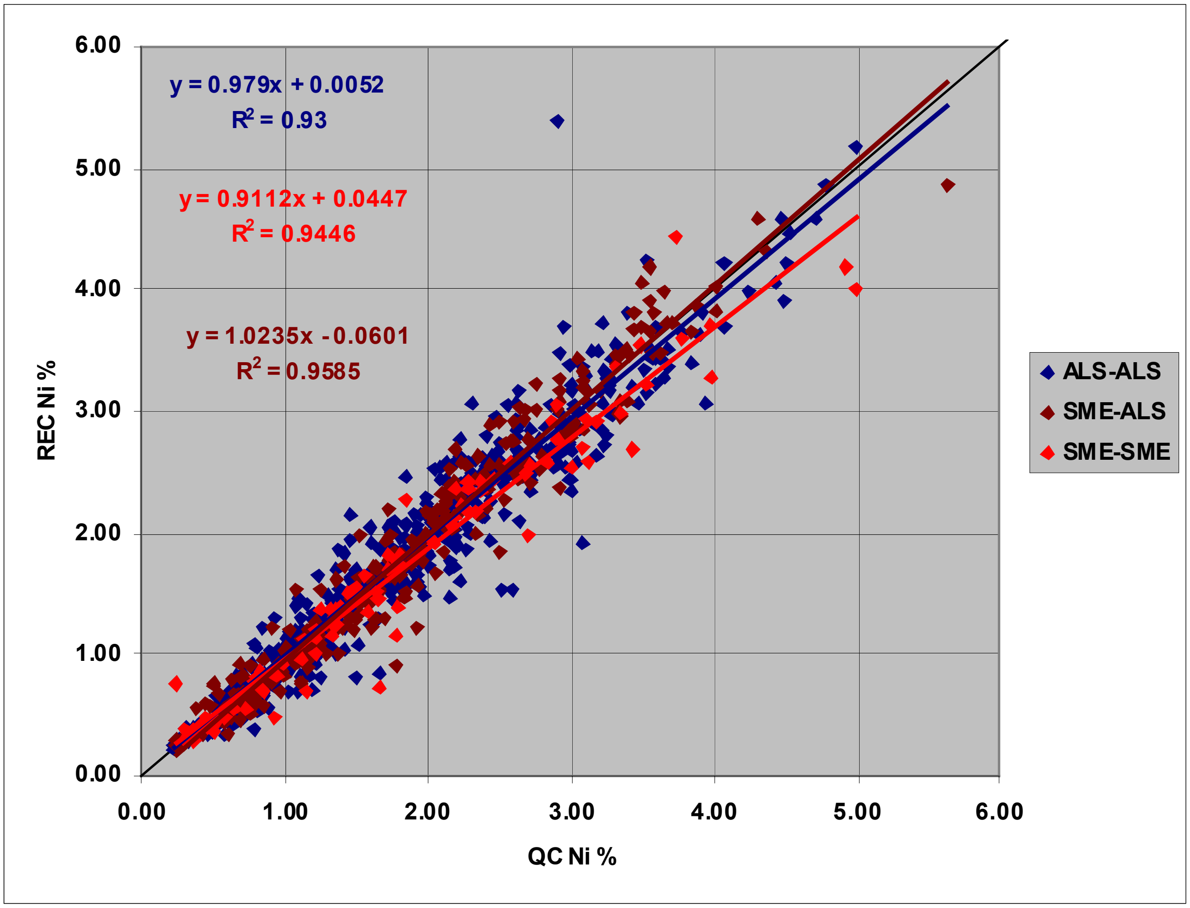
As shown in Table 15.11.2, the precision difference is generally within expectation, especially when dealing with different pulps, sample preparation procedures and laboratories.

**Table 15.11.2: Average Precision between Half Core Samples**

| Precisions for all samples in the Half Core Test | | | | | | | |
|---|---|---|---|---|---|---|---|
| **Ni** | **Co** | **Fe** | **SiO2** | **MgO** | **Al2O3** | **Cr2O3** | **MnO** |
| 22.3% | 32.8% | 16.4% | 7.5% | 17.4% | 39.0% | 23.6% | 20.7% |

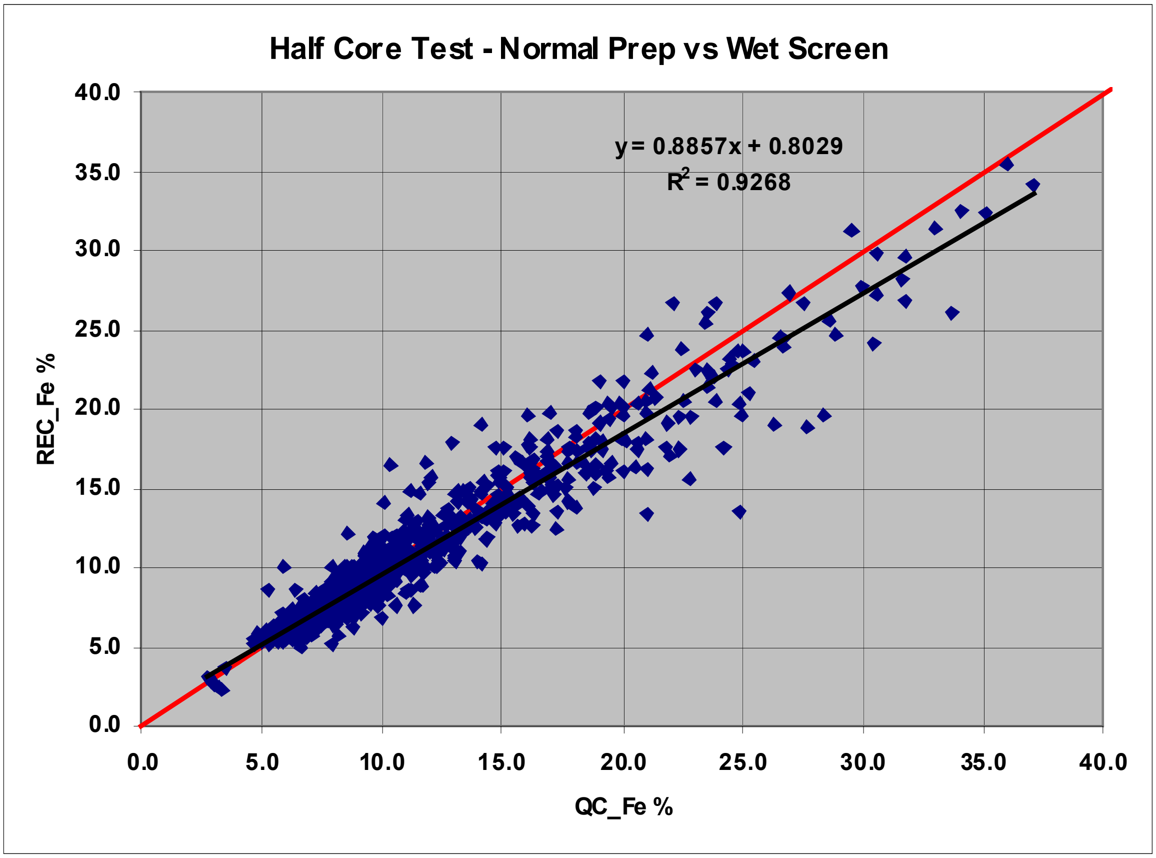
The check exercise suggests that there is a minor bias between the two half core samples. It is presumed that the bias has been created by the loss of small amounts of fine material while screening, resulting in underestimation of the nickel and iron content in the screened core results that are preferentially located in the fine fractions.

**Figure 15.11.2(a): Half-Core Test QC Ni vs % Ni REC**




In the case of iron, as shown in Figure 15.11.2(b) below, it was determined that the bias covered effectively all ranges of iron and that the screened and reconstituted sample had a lower iron content on average compared to the initial sample that was not screened.

**Figure 15.11.2(b): Half-Core Test QC Fe vs % Fe REC**




### 15.12 Conclusions

Based on the ITSL audits of Goro Nickel exploration procedures, Goro Nickel has good storage and documentation procedures for pulp and core that should ensure retrieval of core boxes for due diligence studies or for future technical studies. The core boxes are individually wrapped to prevent deterioration and cross-contamination of the core from other boxes. The borehole number and the box number are marked on the outside of the box. Pulps are stored in sealed metallic pails with identification on the content of the pail. The individual pulps stored in the pails are identified by a printed label and handwritten ID numbers on each bag. Core and pulps are stored in closed steel shipping containers.

While some concerns were raised during technical audits about the use of certain outside laboratories, such concerns do not materially affect the reliability of the mineral resources estimate. Based on site visits and in reference to audits conducted, the general quality of the samples collected and prepared by Goro Nickel is of sufficient quality to meet the necessary standards required for estimation of mineral resources.

The analytical laboratories used by Goro Nickel have reliable procedures for the treatment of laterite samples such that the analyses reported are within accepted tolerances and can be used to estimate mineral resources.

## 16. DATA VERIFICATION

The data is captured in an electronic database that handles multi-size fraction analyses for each sampling interval. Geological technicians validate the data entry.

The chemical analyses are merged into the validated database electronically. Once a hole is completed, all handwritten information about the hole is sent to the office for back-up. A copy of the updated database is sent periodically for backup to the Goro Nickel geology office in Nouméa.

Data related to the Goro Project has been audited several times over the years by independent consultants during the course of feasibility studies and more recently by ITSL. These audits, listed as references 9, 11, 12, 15 and 16 in Item 23 of this Report, were in addition to verification procedures already in place at Goro Nickel.

There are five Goro Project databases used for the mineral resources estimate. Each database is created on the basis of the year that the drilling occurred or the type of holes the database contains. Over the past several years, 10% of the holes contained in the more recent database have been randomly selected and all records on these holes have been verified against the original paper records that have been received from the laboratories, surveyor's office, or Goro Nickel field or paper logs written by Goro Nickel geologists. (Reference is made to Tables B-1 to B-9 in Appendix B for examples of the main tables in the database). For audit purposes, all information about the selected drill holes was verified and checked against written original records.

## 17. ADJACENT PROPERTIES

***Not applicable***

# 18. MINERAL PROCESSING AND METALLURGICAL TESTING

## 18.1 Process Development History

Extensive testing was conducted during 1967 to 1974 on ore samples from the Goro Deposits, including two bulk shipments in 1968 and 1971, at Inco's research laboratory in Mississauga, Ontario and at pilot plant facilities at Inco's Port Colborne, Ontario facility. Major pilot testing of selective reduction flowsheets, which involved drying of the ore followed by high temperature selective reduction of the nickel, was conducted. Nickel was then recovered either by a carbonyl process or by atmospheric pressure acid leaching. Considerable pressure acid leach testwork on limonite ores was also conducted at Inco's research laboratories and pilot plant facilities.

In 1992, a process selection study concluded that the most appropriate process for treating the ores should be based on pressure acid leaching of the limonite and saprolite ores together. This study concluded that recovery of nickel and cobalt from the leach solution by a new solvent extraction process would be economically preferable to the proven sulphide precipitation. The study also concluded that processes that involve drying of the laterites are not economic because they are very energy intensive.

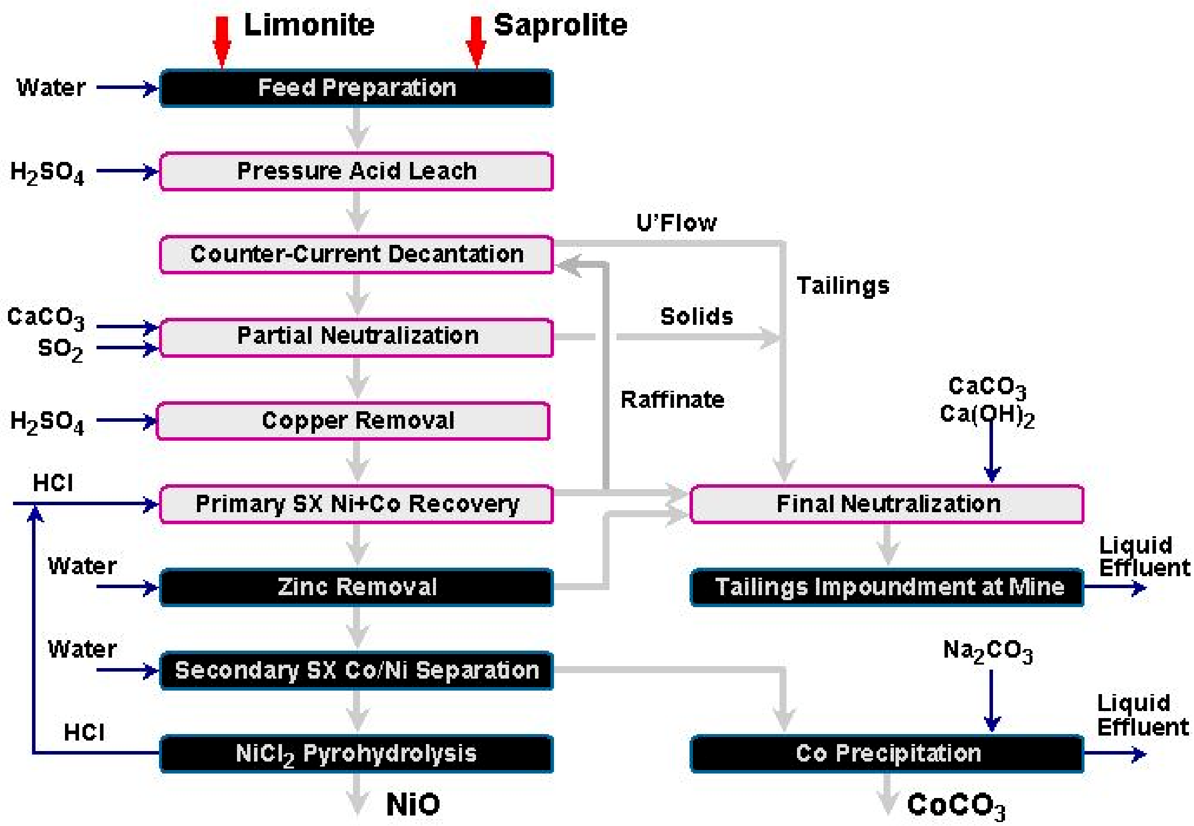
Based on the process selection study, an extensive research and development program was initiated to develop a new proprietary process for treating the laterite. This program has included laboratory scale work, mini-pilot plant and pilot plant continuous testing and, as previously noted in this Report, culminated in the construction and operation of a fully integrated pilot plant at the Goro Project site.

The flowsheet developed during the mini-pilot plant work and subsequently further tested in the New Caledonia pilot plant is shown in Figure 18.1. There are three areas of new technology in this flowsheet, as discussed below.

The design of the pressure leach circuit, which operates at 270$^{o}$C, is based on test work carried out at the Amax-COFREMMI pilot plant in Denver, Colorado (the rights to which Inco purchased in 1992), as well as extensive batch and continuous leaching, including ten integrated mini-pilot plant campaigns carried out at the Inco research facility in Mississauga, over two years of larger scale piloting at the pilot plant Goro Project site in New Caledonia and on relevant external commercial plant pressure leaching experience. The latter included experience gained in the earlier laterite pressure leach plants, as well as experience from similar operations in the alumina, gold and zinc industries. Indirect slurry heating was extensively and successfully tested throughout piloting for the Goro Project.

**Figure 18.1 Block Flow Diagram of the Goro Process**

1




Copper removal is required prior to solvent extraction. Ion exchange was chosen to remove the low levels of copper in the feed solution to solvent extraction.

The primary solvent extraction circuit for the simultaneous recovery of nickel and cobalt from the neutralized and copper-free leach solution is based on a proprietary process, which uses a Cytec Industries Inc. ("Cytec") reagent which was jointly patented in 1998 by Inco and Cytec. The process was developed specifically for Goro Nickel at the Inco laboratories in Mississauga, Canada. Zinc is the only impurity, which is co-extracted and this is subsequently removed by conventional ion exchange. Challenges related to extractant stability were solved, and the process was successfully demonstrated through mini-pilot plant (Mississauga) and pilot plant (Port Colborne) scale. The process was subsequently validated during operation of the integrated pilot plant in New Caledonia, using both the conventional mixer-settler technology and the emerging pulsed column technology. As a result, the decision was made to utilize pulsed columns in the commercial plant. Similar columns have been successfully used by WMC Limited and one of its successors, WMC Resources Limited, for uranium recovery in Australia.

Pyrohydrolysis involves high temperature decomposition of nickel chloride solution to produce nickel oxide, and regenerate hydrochloric acid for recycling to the primary solvent extraction step. Two technologies, spray roasting and fluid bed roasting, are commonly used in the steel industry for treating spent pickle liquors. The fluid-bed pyrohydrolysis technology to be used for producing a granular nickel oxide product has been developed through a joint research project between Andritz-Ruthner of Austria and Inco. It was initially demonstrated in a mini-pilot plant campaign in Vienna, and subsequently validated in a pilot fluid-bed roaster at the integrated pilot plant in New Caledonia. Falconbridge Ltd., a Canadian mining company, had used a similar process for producing nickel oxide in the past.

**Mini-Pilot Plant**

A mini-pilot plant was designed and built at Inco's research and development facility in Mississauga, Ontario to test the proposed flowsheet. The unit operations from pressure acid leach to primary solvent extraction and also the final neutralization were operated in an integrated manner. Feed preparation was performed off-line. The other unit operations such as zinc removal and secondary solvent extraction were too small to incorporate into the miniplant but development work was done separately using solutions produced in the mini-plant. Pyrohydolysis testwork and development was conducted at the Andritz-Ruthner pilot plant in Vienna using synthetic solutions.

The mini-pilot plant was designed to treat approximately 6kg/h of feed (dry basis) and represented approximately 1/80,000 of the commercial plant scale. Campaign duration was generally 5 to 10 days with continuous operation.

During 1994 and 1995 nine campaigns were conducted using mineralized laterite from the Goro Deposits.

Autoclave feed materials for most of the campaigns consisted either of limonite mineralisation or blends of limonite and saprolite mineralizations extracted from a test mine located in Zone 4. Two campaigns were conducted using material from 20 cm drill core extracted from Zones 1, 2, and 4. The drill core samples represented the entire profile of the orebody and detailed screening, particularly for the saprolite, was conducted to determine which size fractions should be processed to maximize revenue.

Feed blends were made to test a range of compositions, with nickel in the range of 1.4% to 1.8% and magnesium (the major acid consumer) in the range of 0.3% to 3.8%.

Operation of the mini-pilot plant confirmed the expected metal extractions and operating conditions of the leach and other operations, allowed development of some of the process steps, and provided preliminary design criteria for the initial feasibility study which was conducted by SNC-Lavalin in 1995-1997. In particular, the mini-pilot plant identified the required operating conditions for the novel primary solvent extraction process and also identified some issues, which required further development.

### 18.2 Primary Solvent Extraction Pilot Plant

The mini-pilot plant work indicated that the jointly patented Cytec reagent was susceptible to oxidation (degradation). Extensive laboratory testwork determined the mechanism of the degradation and ways in which it could be minimized. More importantly, it was shown that the degradation process was reversible and a patented process was developed which allows the reagent to be regenerated. This regeneration process is key to the commercial application of the process.

A solvent extraction pilot plant, using commercial design mixer settlers, was constructed at Inco's Port Colborne refinery to confirm the findings of the laboratory work and to demonstrate the regeneration process on a larger scale. The zinc ion exchange process was also demonstrated in this pilot plant. Two campaigns, each of approximately three months in duration, were conducted during 1997 and 1998 in this pilot plant using synthetic solutions prepared to simulate the expected compositions from leaching typical ore blends. The second campaign demonstrated that the rate of degradation of the extractant could be minimized under the optimum operating conditions and confirmed the requirement to exclude oxidizing species from the system. The regeneration process was shown to be very effective in restoring the capacity of the degraded extractant.

### 18.3 New Caledonia Integrated Pilot Plant

Following the completion of a feasibility study by SNC-Lavalin in 1997, Inco decided that the Goro Project should proceed with a fully integrated pilot plant in New Caledonia. The key objectives of the pilot plant study were as follows:

- Confirm the technical feasibility of the process for extraction of nickel and cobalt from the Goro Deposits;
- Investigate and implement process design changes that improve plant performance;
- Collect data for the design of the commercial plant equipment;
- Train personnel for the commercial plant operation;
- Confirm the volumes and properties of the plant effluents;
- Confirm and determine materials for construction;
- Optimize tailings disposal methods;
- Update process design criteria for the commercial plant; and
- Demonstrate product quality and market acceptability.

The pilot plant was designed by SNC-Lavalin to process 0.5t/h (dry basis) of autoclave feed (1/1000 of commercial scale) and constructed in modular form in Canada during 1998 and 1999. Modules were then shipped to New Caledonia, arriving in April and July 1999, and were installed on a plateau approximately two kilometres east of the eastern shore of the Baie du Prony.

The pilot plant was fully integrated with all unit operations and all recycle streams included. It also included tailings cells for testing tailings disposal methods and stability.

Commissioning of the pilot plant started in August 1999 and eight campaigns of varying length were conducted over an approximately two year period ending in April 2002. The pilot plant was staffed almost exclusively by New Caledonians with a few expatriates in key technical positions and with assistance from Inco personnel as required. Considerable back-up testing was also conducted at the ITSL laboratory in Mississauga and on site in New Caledonia to develop some improvements and changes to the process flowsheet.

Feed materials for the pilot plant for five of the campaigns consisted of previously stockpiled limonite and saprolite. Large diameter (1 m) Trivelsonda shafts were drilled in three areas of the Goro Deposits to provide fresh, more representative feed to the pilot plant. Limonite and saprolite were slurried and screened separately and slurries were blended appropriately to give a range of autoclave feed compositions.

Early campaigns were devoted to tasks such as operator training and equipment modifications. Improvements were made to both equipment and operating conditions and by the end of the pilot plant operation all of the unit operations had demonstrated the required process design requirements and all of the above objectives had been achieved. In some areas, such as solvent extraction and pyrohydrolysis, alternative equipment types were tested which has allowed the design for the commercial plant to be determined.

In addition to meeting the key objectives outlined above, some of the other major achievements resulting from the pilot plant campaigns were as follows:

- The pilot plant operation demonstrated that the new proprietary process for treating laterite developed by Inco is a technically and economically sound process, which has relatively low reagent consumptions (operating costs).

- Excellent nickel and cobalt extractions were demonstrated from all feed blends tested. In fact the cobalt extractions were higher than previously attained in the mini-pilot plant and the design recovery of cobalt was increased.

- All reagent consumptions were confirmed. Acid consumptions in the autoclave were as expected and varied with the ore composition as predicted by the mass balance model.

- Overall design recoveries of nickel and cobalt were demonstrated in the later campaigns, which used samples of fresh ore from several locations in the orebody.

- The production of high quality nickel oxide and cobalt carbonate products, meeting current market specifications, was successfully demonstrated.

- Extensive materials testing, particularly in the pressure acid leach area, has allowed the appropriate materials of construction to be chosen.

- Two types of equipment were successfully tested for the solvent extraction processes – mixer settlers and pulsed columns. The performance of the pulsed columns was superior to the mixer settlers and the columns were shown to be more appropriate and cost effective for the commercial plant design.

- The two available technologies for pyrohydrolysis - spray roasting and fluid bed roasting - were successfully tested. Fluid bed technology gives a granular product, which is preferable for environmental and handling reasons. The fluid bed technology was therefore confirmed for the commercial plant design.

- Thickening of tailings to produce a paste was demonstrated at the pilot plant scale and pumping of the thickened tailings was also demonstrated. This has allowed the changing of the tailings disposal design for the commercial plant to a lower cost, more environmentally benign thickened tailings scheme, which does not require large dams to retain a conventional tailings pond.

- A new effluent treatment process was developed which reduces lime consumption (reduces capital and operating costs) but meets all discharge requirements for the tailings and discharge to the environment.

The design recoveries of metal values from the FPP product (representing autoclave feed which represents the project's "ore") to the product for the commercial plant were determined on the basis of optimized performance of each unit operation of the integrated pilot plant in New Caledonia to be about 92.2% nickel and 90.8% cobalt. The maximum sulphuric acid consumption in the pressure acid leach process is projected to be about 371kg/tonne of autoclave feed treated, on the basis of the treatment of representative ore samples in the integrated pilot plant in New Caledonia.

The operating performance achieved during the final three campaigns in the fully integrated New Caledonia pilot plant are summarized in Table 18.3. Feeds for Campaigns 5 and 6 were the freshly drilled Trivelsonda material, whereas feed for Campaign 7 was from old stockpiles.

**Table 18.3: Results from the last three Goro Pilot Plant Campaigns**

| Pilot Plant Campaign | Feed Composition, % by wt | | | Recovery, % | | Acid |
|---|---|---|---|---|---|---|
| | Ni | Co | Mg | Ni | Co | kg/t ore |
| 5 | 1.46 | 0.14 | 3.25 | 87.9 | 87.7 | 366 |
| 6 | 1.46 | 0.13 | 2.25 | 92.6 | 90.5 | 318 |
| 7 | 1.37 | 0.11 | 2.95 | 91.2 | 77.4 | 380 |
| Commercial Design Basis | 1.5 | 0.13 | 2.75 | 92.2 | 90.8 | 340 |

The commercial plant is projected to be capable of higher recoveries of nickel and cobalt, as is standard practice based on experience in going from pilot plant to full-scale hydrometallurgical plants. In addition, the pilot plant circuit had some differences from the commercial design and there were still some ongoing equipment deficiencies. For example, the pilot plant counter current decantation ("CCD") circuit was operated with only four or five thickeners in series whereas the commercial plant will have six. Therefore, recoveries in the pilot plant CCD circuit were slightly lower than expected. The mixer settler circuit used in most campaigns for secondary solvent extraction ("SX2") was very small and could not be used with the preferred diluent. Consequently, recoveries of nickel and cobalt into the appropriate streams were always less than expected. Campaign 6 used the pulsed columns for SX2 when excellent recoveries of both nickel and cobalt were achieved in SX2 and overall.

The pilot plant results have been used to update the commercial plant design and the process design criteria.

### 18.4 Product Quality

An important aspect of the pilot plant was to demonstrate that marketable products could be produced, and to provide samples for market testing.

The nickel product from the new proprietary process for treating laterite developed by Inco is expected to be nickel oxide, produced by pyrohydrolysis. The nickel oxide will be smelted in other Inco facilities to produce UTILITY® nickel as a feed to the stainless steel industry. As designed and built, the integrated pilot plant incorporated a spray reactor for pyrohydrolysis. The spray reactor was chosen because it is a relatively simple piece of equipment and can be operated with the inexpensive No. 6 fuel oil. A chemically acceptable product was obtained, but the spray roaster product is a fine powder which was considered to be unacceptable from a handling and environmental point of view.

Testwork conducted in conjunction with Andritz-Ruthner had shown the viability of fluid bed pyrohydrolysis for producing a granular, non-dusting nickel oxide. Andritz-Ruthner was commissioned to design and build a 0.8m diameter fluid bed pyrohydrolysis unit for testing at the integrated pilot plant. This unit was installed and operated for two months and produced about 15 tonnes of granular, non-dusting, nickel oxide. Chemically, the fluid bed product was slightly higher purity than the spray reactor product in that the chloride content was lower.

Approximately 13 tonnes of the fluid bed nickel oxide product has been smelted at a facility of one of Inco's affiliates in Taiwan to ensure that the resulting product would represent an acceptable feed to the stainless steel industry.

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The cobalt product from the new proprietary process for treating laterite developed by Inco is planned to be cobalt carbonate. Part of the product will be used in an application which requires a very high cobalt:nickel ratio. The originally installed mixer settler circuit for the secondary solvent extraction (separation of cobalt from nickel) was inadequate to produce the desired product purity. A separate, stand-alone secondary solvent extraction campaign was undertaken, using the pulsed column technology and incorporating additional scrubbing stages. A high purity (with respect to nickel content) cobalt strip solution was produced from which a cobalt chloride product of acceptable chemical purity was produced. The testwork also confirmed that a product with good physical properties could be produced.

## 19. MINERAL RESOURCES AND MINERAL RESERVES ESTIMATES

Jean-Yves Cloutier and Olivier Tavchandjian, both "qualified persons" as defined in National Instrument 43-101 and employed under the direct supervision of R.A. Horn and assisted by Daniel Godillot, compiled the mineral resources estimates contained in this Item 19. The mineral reserves were estimated from mine plans and mine production schedules developed by IMC from data and design parameters provided by Inco.

### 19.1 Mineral Resources Categories

The mineral resources estimates are classified according to the level of confidence associated with the estimate and are exclusive of mineral reserves. The level of confidence is based on the drilling density, the quality of the data, the type of data, the lateral continuity of the layers within the laterite landform and the proposed mining method. In the case of the Goro Deposits, the guidelines applied for the estimation of mineral resource classifications follow, as reflected in Item 8.4 of this Report, the CIM Standards on Mineral Resources and Reserves Definitions and Guidelines adopted by the CIM Council on August 20, 2000.

#### 19.1.1 Measured

The measured mineral resources category is used for the yellow laterite layer and transition layer. For Goro Nickel, the measured mineral resources generally corresponds to a drilling grid of 200 by 200m or denser, consisting of a mixture of destructive and core holes. The measured mineral resources classification may also be supported by local geostatistical tests and bulk samples. An exception has been made to this classification immediately to the east of the proposed pit crest. Although the information gathered in this area supports measured mineral resources classification, the feasibility of excavating a pit without affecting the Wadjana drainage is uncertain at this time. Therefore, this material has been downgraded to a possible mineral deposit until further assessments are made.

#### 19.1.2 Indicated

The indicated mineral resources category has been applied to:

- Yellow laterite and transition layers on the same laterite plateau and in continuity with the measured mineral resources of yellow laterite and transition layers. The drilling grid varies from 200m to 400m.

- Saprolite with a drill density, generally, of 200 by 200m, or denser, composed of a mixture of destructive and core holes. The indicated mineral resources classification may be also supported by local geostatistical tests and bulk samples.

- An exception has been made to this classification immediately to the east of the proposed pit crest. Although the information gathered in this area supports an indicated mineral resources classification, the feasibility of excavating a pit without affecting the Wadjana drainage is uncertain at this time. Therefore, this material has been downgraded to a possible mineral deposit until further assessments are made.

### 19.1.3 Inferred

The inferred mineral resources classification has been applied to:

- Saprolite beneath truncated drill holes, where limited sampling causes a lower degree of confidence than fully sampled profiles.

- Transition and saprolite layers on the same laterite plateau and in continuity with the measured mineral resources of the transition layer or the indicated mineral resources of the saprolite layer where the drilling grid density is from 200m to 800m.

- Yellow laterite, transition, and saprolite mineral resources located outside the seam model. These mineral resources have been estimated using a nearest neighbour model. The mineral resources are located within an open rectangle formed by three ridges where, the south-west corner of the open rectangle is defined by the Col de l'Antenne. The drilling grid density varies from 200m to 800m with some local clusters of higher density. No screening information and only partial chemistry is available within these areas.

### 19.1.4 Possible Mineral Deposits

Located within the seven concessions of the Goro Deposits, there are possible mineral deposits that are expected to add significantly to the mineral resources with further exploration work. The work required depends on the data already available and the location of the deposit. In general, a desktop mining exercise should be completed for these deposits to define appropriate economic factors to be applied to the mineral resources.

## 19.2 Goro Mineral Resources Estimates

The mineral resources within the seven concessions of the Goro Deposits have been estimated using two block models. A kriged block model has been built for the area that has been delineated by core drilling and destructive drilling, which includes the location of the proposed mine in Zones 1, 2, 3 and 4. A "nearest neighbour model" has been developed for the remaining areas, which have been investigated by destructive drilling on grids varying from 200m to 800m.

The following tables contain the mineral resource estimates at 50.8mm screen size fraction derived from Krige model and the nearest neighbour model created in December 2001 and December 2002, respectively. As the nickel grades normally increase with depth, under the present Mine Production Schedule (reference is made to Table 20.1.6) excavate down to a grade cut-off of 1.2% nickel surface, which lies within the yellow laterite layer. Material above this horizon will not be used for feed while all material below, down to bedrock, will be sent to the FPP for upgrading and blending prior to being fed to the process plant. For blending purposes and production targets, the cut-off surface, lithological surfaces, the blend chemistry and the bedrock surface are key elements to model and incorporate into the daily mine operation. This cut-off plane is defined at the top of the first occurrence of two consecutive vertical blocks of 1 meter thickness above the 1.2% nickel cut-off. This same cut-off has been used for both mineral resources and mineral reserves estimates. Sterilized mineral resources under waste dump and tailing disposals have been removed from these estimates.

Consistent with Inco policies and practices, the mineral reserves estimates are in addition to the mineral resources estimates. The following tables reflect data on the mineral resources estimates for the Goro Deposits:

**Table 19.2(a): Goro Deposits Mineral Resources Estimates**

| Mineral Resources @ 1.20 % nickel cut-off, -50.8mm screening size | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| **Classification** | **Tonnes Million** | **Ni %** | **Co %** | **Fe %** | **$SiO_2$ %** | **MgO %** | **$Al_2O_3$ %** | **$Cr_2O_3$ %** | **MnO %** |
| Measured | 56.0 | 1.40 | 0.14 | 47.1 | 5.76 | 1.33 | 5.68 | 3.38 | 1.24 |
| Indicated | 39.1 | 1.84 | 0.12 | 31.4 | 22.53 | 12.00 | 3.40 | 2.24 | 0.69 |
| **Total** | **95.1** | **1.58** | **0.13** | **40.7** | **12.65** | **5.72** | **4.74** | **2.91** | **1.02** |
| **Inferred Mineral @ 1.20 % nickel cut-off, -50.8mm screening size** | | | | | | | | | |
| Inferred (Krige) | 21.0 | 2.1 | 0.10 | 21.38 | | | | | |
| Inferred (Nearest) | 122.8 | 1.6 | 0.13 | 39.70 | | | | | |
| **Total** | **143.8** | **1.7** | **0.12** | **37.0** | | | | | |

**Table 19.2(b): Goro Deposits Mineral Resources Estimates by Layers**

| Mineral Resources @ 1.20 % nickel cut-off, -50.8mm screening size | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| **Classification** | | **Tonnes Million** | **Ni %** | **Co %** | **Fe %** | **$SiO_2$ %** | **MgO %** | **$Al_2O_3$ %** | **$Cr_2O_3$ %** | **MnO %** |
| **MEASURED** | LATJ | 49.4 | 1.37 | 0.13 | 48.3 | 4.12 | 0.93 | 5.80 | 3.45 | 1.22 |
| | TRN | 6.6 | 1.68 | 0.19 | 38.1 | 18.14 | 4.36 | 4.84 | 2.86 | 1.41 |
| | **TOTAL** | **56.0** | **1.40** | **0.14** | **47.1** | **5.76** | **1.33** | **5.68** | **3.38** | **1.24** |
| **INDICATED** | LATJ | 18.0 | 1.35 | 0.16 | 48.2 | 3.92 | 1.16 | 5.03 | 3.52 | 0.88 |
| | SAP | 21.1 | 2.26 | 0.08 | 17.1 | 38.36 | 21.22 | 2.00 | 1.15 | 0.54 |
| | **TOTAL** | **39.1** | **1.84** | **0.12** | **31.4** | **21.53** | **12.00** | **3.40** | **2.24** | **0.69** |
| **INFERRED** | LATJ Near. | 78.6 | 1.4 | 0.14 | 47.5 | | | | | |
| | TRN Near | 11.4 | 1.5 | 0.20 | 39.1 | | | | | |
| | SAP Near | 32.8 | 2.2 | 0.07 | 21.1 | | | | | |
| | SAP Krige | 18.9 | 2.1 | 0.09 | 19.4 | 37.64 | 21.96 | 1.80 | 1.05 | 0.54 |
| | TRN Krige | 2.1 | 1.64 | 0.23 | 39.5 | 13.62 | 4.49 | 5.82 | 3.91 | 1.29 |
| | **TOTAL** | **143.8** | **1.7** | **0.12** | **37.0** | | | | | |

### 19.3 Block Modeling Methodology

The steps listed below have been followed by ITSL to derive the mineral resources and mineral reserves estimates since 2001:

- Study the possible bias between the Becker RC hammer destructive, and the HQ core size, hole data.
- Review, by an external consultant, of the preliminary assessment of the destructive hole bias.

- Extend the geological coding of drill holes to all holes drilled within the seven concessions of the Goro Deposits.
- Assess the short range variability of the layers by analyzing the core hole data available from a small block located in the area proposed for the first few years of mining.
- Run stochastic simulation to estimate dilution and recovery.
- Review grade cut-off strategies and mining selectivity based on short-range variability and simulation.
- Validate the new data.
- Create an intermediate resource model based on the nearest neighbour approach.
- Create intermediate 2D and 3D seam models using the unfolding algorithm.
- Validate the unfolded model against a model that did not use the unfolding algorithm.
- Review by an external consultant of the methodology used for composite selection and block modeling technique.
- Review the impact of truncated holes on saprolite modeling.
- Construct a first pass mine production schedule and review any shortfalls in the block model and mine schedule.
- Conduct internal audits of databases and analytical labs used for the new block model.
- Construct final block models for resource reporting and mine scheduling.

## 19.4 Krige Model

In the evaluation the Goro Deposits, special care has been taken in the selection of the composites that are used to estimate the grades and tonnes of a particular block within the block model. An estimation method has been selected that will best reflect reality. In the evaluation of nickel laterites, the bedrock type and structures, which are important factors in the formation of favourable laterite landforms and the overall topography are used in the estimate. Laterites are best represented by what are called 'seam models' that recognise their layered nature.

In discussions between Goro Nickel and ITSL geologists, it was decided that the reference horizon for modeling should be the base of the yellow laterite layer. From this horizon, all layers above are modeled using the base of the layer as a reference while the layers below the reference horizon are modeled using the top of the layer as reference.

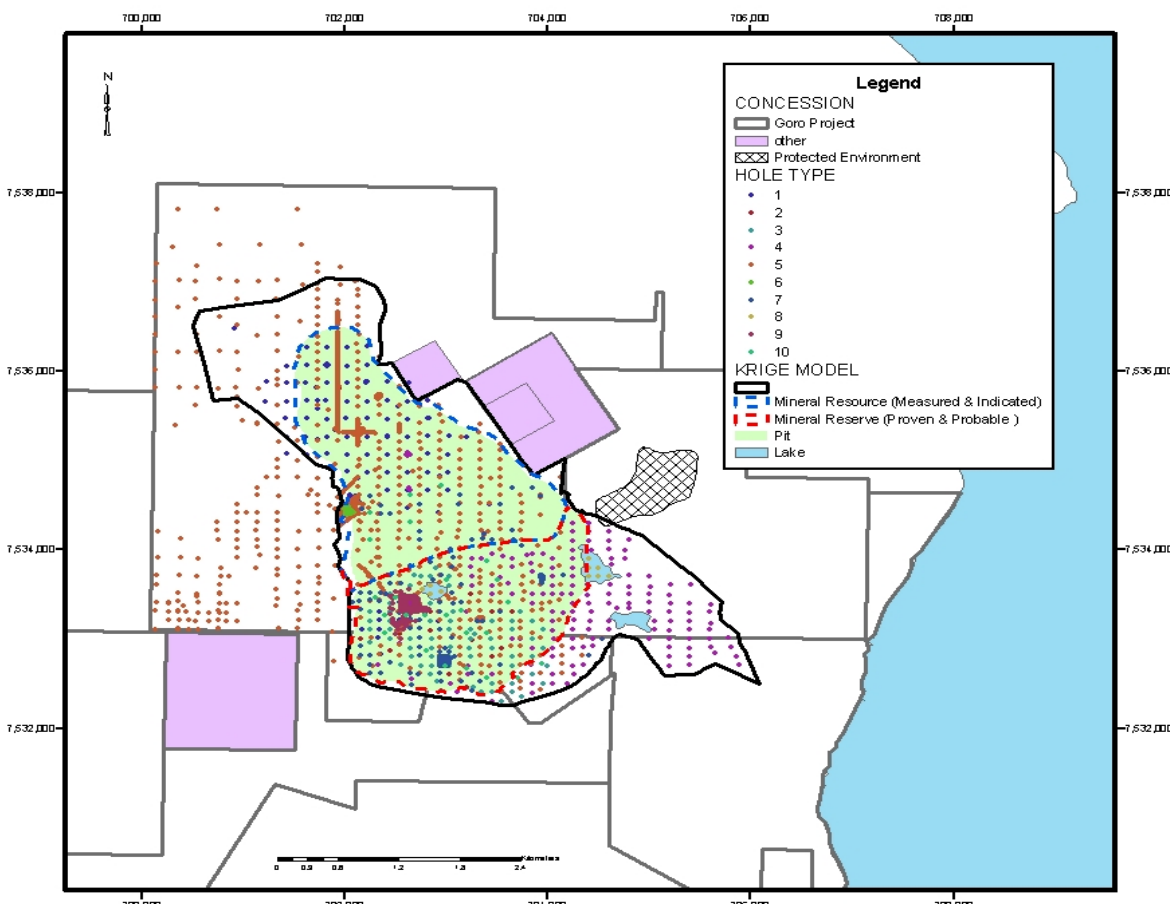
### 19.4.1 Krige Model Database

For the resource block modeling, the databases were combined in such a way that maintained consistency with the original databases. Holes were eliminated for several reasons, such as the absence of chemical analysis where assays were not yet available, samples collected in a unique fashion such as those of the Benoto shafts and samples that were considered for other reasons to be inadequate, or there were missing geological codes. The resulting database shown in Table 19.4.1 contained 1,869 holes.

**Table 19.4.1: Drill Hole Database for Krige Model**

| Hole Type | Drilling | Screen Aperture | Number of Holes | Total Length (m) |
|---|---|---|---|---|
| prior to 1998 | Core drilling | No screening | 176 | 7,839.0 |
| 1998 | Core drilling | No screening | 16 | 621.8 |
| 1998 | Core drilling | 50.8, 6.5mm | 69 | 3,196.7 |
| 2000 | Core drilling | 150, 50.8, 6.5mm | 212 | 11,007.5 |
| prior to 1998 | Becker RC Destructive drilling | No screening | 995 | 38,163.8 |
| prior to 1998 | RBCD Core drilling | No screening | 60 | 1,052.9 |
| 2001 | Core drilling | 150, 50.8, 6.5mm | 132 | 5,887.5 |
| 2001 | Core drilling resource under lakes | 150, 50.8, 6.5mm | 10 | 341.6 |
| 12.5m grid | Core drilling | 150, 50.8, 6.5mm | 174 | 10,398.7 |
| ERTK 2002 | 20cm core drilling | 150, 100, 50.8, 25.4, 19, 12.7, 6.35mm | 25 | 1,143.3 |
| **TOTAL** | | | **1,869** | **79,652.8** |

**Figure 19.4.1: Hole Type Location Map used in Seam Model**



### 19.4.2 Krige Model Location and Specifications

The Krige model is located between Easting 700,515 and 706,035 and between Northing 7,532,205 to 7,537,035 in the IGN72 UTM coordinate system and covers portions of the Robert, Cascade, Fer and Fer Ext. 2 Pte concessions. The block model is based on a 30m by 30m by 1m (x,y,z) block size. Each of the blocks was subdivided in a matrix of 5 by 5 by 2 points (x, y, z). Each point was interpolated using kriging and then averaged to yield a tonnage and grade value for each block.

Ni, Co, Fe, $SiO_2$, MgO, $Al_2O_3$, $Cr_2O_3$, MnO values and screen recoveries were interpolated for each block for each of the yellow laterite, transition and saprolite layer and for ROM and the -50.8mm fractions. For the remaining layers (iron cap, iron shot, red laterite), since they are not considered to be mineral resources, the average chemical composition of the layer was loaded in each block within the block model. Bedrock values were not loaded into the block model. Other chemical analyses for Cu, Zn, $K_2O$, $TiO_2$, CaO and LOI were not included in the block model because in the quantities present they do not have a significant impact on the process or are not required for the evaluation of the project viability.

Composites of one meter in length for individual layers were studied using variograms. No cut-off grades were used to discriminate between composites to be loaded in the model.

The saprolite layer data was loaded into the model based on the core size fraction studies and specific adjustment factors for the Becker drilling for specific zones of the deposit. The zones were previously discussed in Item 14 of this Report (reference is made to Figure 6.2).

In Zones 1, 2 and 3 only screened core data were used to interpolate screen recoveries and chemistries in saprolite and are the locations of the mineral resources that have been converted into mineral reserves. The saprolite layer thickness was modeled using the methodology described in Item 14 to reflect the presence of truncated holes in the database.

In Zone 4 there is limited core screening information and the majority of the information is derived from Becker holes. For Zone 4, the saprolite ROM tonnage and chemistry were interpolated globally from all the available screened core holes and then adjusted for the equivalent size fractions in the destructive holes by applying the appropriate global factors for each size fraction in each blocks located within Zone 4 and the saprolite layer. For the –50mm fraction, the factors presented in Table 14.2.2 were applied to individual blocks and not to individual holes. The saprolite layer thickness was also interpolated using the methodology described in Item 14 of this Report. However, a flag was added to the block model to identify the additional blocks created by the removal of the truncated holes and classified as inferred mineral resources.

The three major lakes within Zones 1, 2, and 3 were excluded from the mineral resources estimates. Recent drilling has indicated that the laterite profile has collapsed under these shallow lakes and that no mineralization could be recovered at these locations. The average chemistry of the laterite under the lakes is derived from the 10 holes drilled from platforms on the lakes.

### 19.4.3 Krige Model Statistics

The 1869 holes used in the geostatistical analysis in the December 2002 Krige block model indicate that for most of the elements and oxides, about 60% of the variance, with the exception of the yellow laterite layer, occurs within the first 30 metres of the sample location. The observed short range variance is not uncommon for this type of deposit when the factors that contributed to the formation of a lateritic profile are considered. The high variance between closely spaced sample points still permits an estimate of grade that is reasonably accurate globally but not for local variations.

The contribution of the short range variance over the total variance per layer is shown in Tables C-1, C-2 and C-3 in Appendix C as CSRV / TV (Short Range Variance/Total Variance).

The Krige model was used to develop mine plans and production schedules and reporting mineral reserves and mineral resources estimates. Statistics from the Krige model for the various layers are presented in Tables 19.4.3(a) to 19.4.3(d) below with no cut-off applied.

**Table 19.4.3(a): Krige Model Global Statistics for ICP+ISH and LATR layers**

| | ICP + ISH | LATR |
|---|---|---|
| %Ni | 0.41 | 0.79 |
| %Co | 0.022 | 0.029 |
| %Fe | 51.0 | 50.7 |
| %$SiO_2$ | 0.88 | 1.77 |
| %MgO | 0.31 | 0.41 |
| %$Al_2O_3$ | 5.12 | 5.38 |
| %$Cr_2O_3$ | 3.70 | 3.34 |
| %MnO | 0.31 | 0.47 |

**Table 19.4.3(b): Krige Model, Yellow Laterite Layer Statistics**

| | %Ni | %Co | %Fe | %$SiO_2$ | %MgO | %$Al_2O_3$ | %$Cr_2O_3$ | %MnO |
|---|---|---|---|---|---|---|---|---|
| Min | 0.69 | 0.006 | 32.1 | 1.3 | 0.2 | 2.80 | 0.42 | 0.00 |
| Max | 2.13 | 0.908 | 53.5 | 20.7 | 7.6 | 14.10 | 6.48 | 5.67 |
| Mean | 1.30 | 0.116 | 49.1 | 3.5 | 0.8 | 5.57 | 3.54 | 1.03 |
| Variance | 0.02 | 0.005 | 4.3 | 4.3 | 0.2 | 0.94 | 0.15 | 0.32 |
| Skewness | 0.4 | 0.8 | -1.0 | 2.2 | 2.0 | 1.0 | 0.2 | 0.9 |

**Table 19.4.3(c): Krige Model, Transition Layer Statistics**

| | %Ni | %Co | %Fe | %$SiO_2$ | %MgO | %$Al_2O_3$ | %$Cr_2O_3$ | %MnO |
|---|---|---|---|---|---|---|---|---|
| Min | 0.40 | 0.030 | 26.4 | 3.1 | 0.6 | 2.30 | 1.47 | 0.15 |
| Max | 2.62 | 0.703 | 48.9 | 39.2 | 17.1 | 14.85 | 6.94 | 4.30 |
| Mean | 1.65 | 0.190 | 38.7 | 17.5 | 3.9 | 5.05 | 3.11 | 1.31 |
| Variance | 0.04 | 0.046 | 9.1 | 23.3 | 1.9 | 1.95 | 0.48 | 0.24 |
| Skewness | 0.1 | 1.1 | 0.0 | 0.0 | 1.2 | 1.2 | 3.8 | 0.5 |

**Table 19.4.3(d): Krige Model, -50mm Saprolite Layer Statistics in Zones 1, 2 and 3**

| | %Ni | %Co | %Fe | %$SiO_2$ | %MgO | %$Al_2O_3$ | %$Cr_2O_3$ | %MnO |
|---|---|---|---|---|---|---|---|---|
| Min | 0.61 | 0.011 | 6.0 | 8.5 | 2.0 | 0.40 | 0.42 | 0.13 |
| Max | 3.75 | 0.643 | 48.1 | 61.8 | 38.2 | 8.23 | 6.63 | 3.85 |
| Mean | 2.08 | 0.105 | 18.6 | 38.1 | 17.9 | 2.24 | 1.45 | 0.68 |
| Variance | 0.14 | 0.004 | 25.7 | 33.6 | 13.0 | 0.70 | 0.15 | 0.12 |
| Skewness | 0.50 | 1.8 | 0.7 | -0.4 | -0.1 | 1.3 | 1.2 | 2.4 |

### 19.4.4 Krige model - Stochastic Simulations

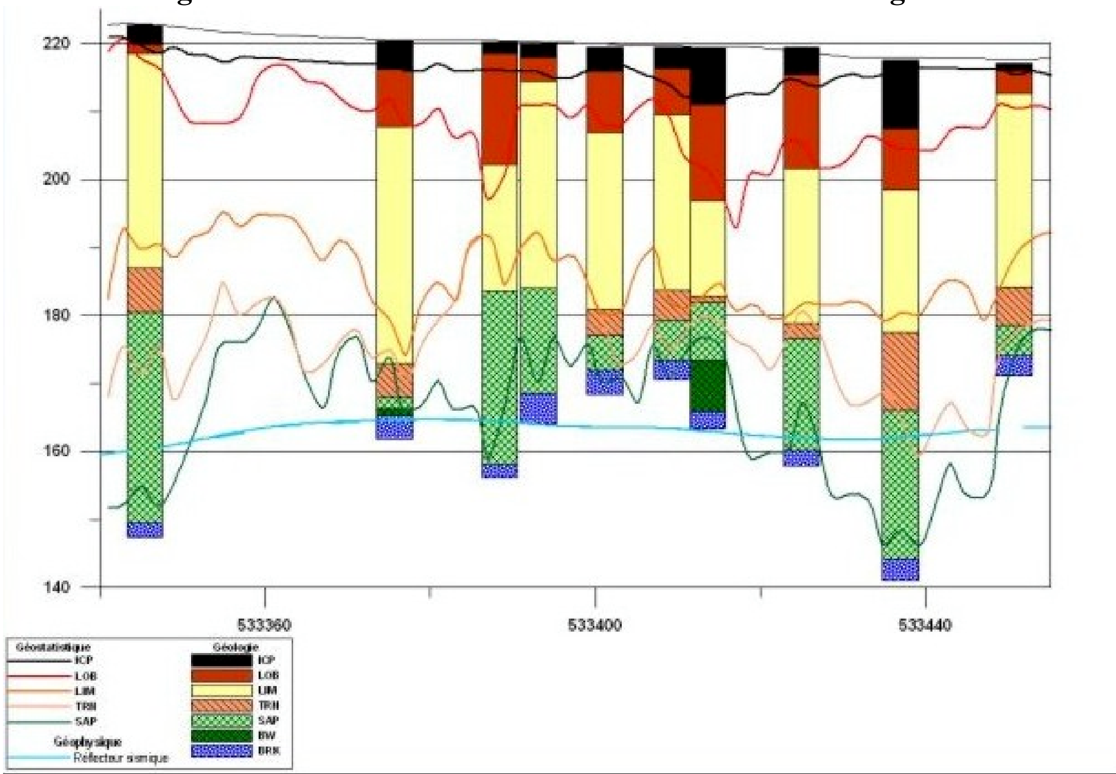
The majority of the December 2002 Krige block model is based on approximately a 100-200 m drilling grid. The Krige block model may show less variability than reality and the use of stochastic simulations may be used to simulate variability in the layers' thickness and chemistry that will be encountered during mining. Reference is made to reference 13 in Item 23 of this Report for information regarding the Krige model.

The results of these studies, shown below in Figure 19.4.4, were incorporated into the mine design parameters and mineral reserves estimate and reflected the following:

- The use of a cut-off grade inferior to 1.3% was reproducible and achievable.
- Due to the variation in layer positions and thickness there will be periods during mining that limonite and saprolite could not be mined separately at the mine face.
- On average, only 79% of the saprolite resources can be recovered based on the actual mining method and mine equipment.
- The highest probability to intercept high grade is located within the first three years of the mine production schedule.

Figure 19.4.4 shows one of 40 simulations of layer variations generated by the stochastic simulation process. The lines between drill holes on the section shown in Figure 19.4.4 represent the layer contacts (no cut-off applied). The line at the bottom of Figure 19.4.4 is determined from a geophysical survey conducted in 2000. The geophysical survey does not match the detailed borehole data because the influence of each survey is different. The geophysical footprint represents more or less an average of the bedrock height over a 10 to 20m circular area whereas the drill hole footprint represents a 63.5mm point.

**Figure 19.4.4: South to North Section 702650E Looking West**




### 19.5 Nearest Neighbour Model

A nearest neighbour model was built to assess the mineral resources estimates immediately outside the primary mine development area as shown on Figures 6.2 and 13.1. This model was used in preference to a Krige model due to the sparse drill information. All of the estimated mineral resources in these areas have been categorized as inferred mineral resources. No screen fractions are available and only global mineral resources estimates can be derived.

The nearest neighbour model is located between Easting 695,000 and 717,600 and between Northing 7,527,500 to 7,540,100 in the IGN72 UTM coordinate system and covers portions of Kwe, AS1, AS7 and Fer concessions. The block model is formed by 50 by 50 meter 2D (x,y) block size.

Most of the holes involved in this estimate are Becker holes with only partial chemistry information (nickel, cobalt and iron). Two dimensional composites of ROM thickness, nickel, cobalt and iron were interpolated for yellow laterite, transition and saprolite layers in each block using a polygonal approach. A 1.2% nickel cut-off in the holes was applied in the yellow laterite layer to discriminate between mineralized and low grade laterite. Once the first two consecutive intervals grading above the cut-off grade are found in a hole then all material located below, including the first two intervals above the 1.2% nickel cut-off (yellow laterite, transition, and saprolite), are considered mineral resources regardless of the grade.

The estimates from the model were then corrected using the same factors used for Zone 4. Dry tonnage factors were assumed to be the same as those derived for the seam model.

Mineral resources estimates within the nearest neighbour block model were constrained by concession boundaries, laterite landforms, forest reserves, and drilling coverage. Drilling data on the nearest neighbour block model is shown in Table 19.5.

**Table 19.5: Nearest Neighbour Drill Hole Statistics**

| Hole type | No.# of holes | Metres drilled |
|---|---|---|
| 20 cm | 5 | 164.0 |
| Core (BRGM) | 226 | 6,271.6 |
| Becker | 4 | 171.0 |
| **Total** | **235** | **6,606.6** |

## 19.6 Mineral Reserves Categories

Mineral reserves have been estimated in those areas of the Goro Deposits that contain measured or indicated mineral resources.

There are additional measured and indicated mineral resources immediately to the north of the mineral reserves, where there is little screen information, that have not been converted into mineral reserves. These additional mineral resources are located within Zone 4 (see Figure 6.2 and Figure 19.4.1). Mining in Zone 4 does not occur before year 14 in the mine life. As core drilling progresses to the north it is anticipated that these mineral resources will be converted into mineral reserves. Inferred mineral resources are not included in the production schedules nor in the mineral reserves estimate.

### 19.6.1 Proven Mineral Reserves

Proven mineral reserve estimates is represented by the portion of yellow laterite and transition that lies within the pit limit and that is equivalent to the measured mineral resources category after applying all mining and financial parameters.

### 19.6.2 Probable Mineral Reserves

Probable mineral reserves estimates is represented by the portion of the saprolite layer that lies within the pit limit and that is equivalent to the indicated mineral resources category after applying all mining and financial parameters.

## 19.7 Pit Optimization and Pit Design

A pit optimization analysis performed by IMC in 2001 and listed as reference 14 in Item 23 of this Report was based on a floating cone economic base study. The results of the study showed that:

- The mine development should be started in the northwest corner of Zone 1 and move toward the east-northeast; and
- Constraints on the mine area due to tailings impoundments and concessions constraints place greater limitations than the theoretical economic limits.

The pit design took into consideration the results of the floating cone analysis as well as mining constraints such as property boundaries, active drainage areas, out of pit and in pit tailing disposal, waste dumps, impoundments, mineralization limits, minimum mining widths, bench heights, and slope angles.

A number of mining schedules and mining scenarios for excavating the contained material have been completed. These studies tried various cut-offs, production rates and blend mixtures. In order to sequence the material, the pit was subdivided into working pit phases. Material in each schedule was then selected from adjoining phases as per the criteria of the scenario under study.

### 19.8 Constraints on Mine Production Schedule

There are three main constraints imposed on the proposed mine production schedule as follows:

- *Tonnage throughput restriction*: There is a limited amount of material that can be fed through the autoclave. Based on the size of the autoclave, the maximum pulp density in the autoclave and the plant utilization. Therefore, the yearly autoclave feed should not exceed 3.973M dry metric tonnes per year of minus 50mm material.

- *Nickel production*: There is a limited amount of nickel that can be produced based on the actual plant design criterion This has been estimated as 123Mlbs of nickel in nickel oxide.

- *Acid to Ore ratio*: Based on the size of the sulphuric acid plant and the amount of acid required to leach the metals it can be calculated that the maximum acid to ore ratio should not exceed 0.371 for yearly autoclave feed of 3,973M dry tonnes.

Based on the tonnage constraints for autoclave feed, the acid to ore ratio and the nickel metal constraints, there is a single theoretical head grade feed to the autoclave that will maximize plant nickel production but would not necessarily maximize revenues during any given planning period.

### 19.9 Application of Cut-Off Grade

Due to the short-range variability of chemistry and layers, the cut-off grade at the top of ore elevation defines the limit of ore and is followed by extraction of all material below that cut-off elevation to the bedrock surface. This approach is based on the following observations about the Goro Deposits:

- The vast majority (92.5%) of the transition and saprolite tonnage has higher grade than the average target autoclave head grade of 1.51% nickel after screening at 50mm. Over 99% of the saprolite and transition material at the 50mm screen size has a grade higher than the 1.20% nickel cut-off that is the maximum considered in the mine production schedules.

- The yellow laterite nickel grades generally increase with depth. There is a positive correlation between nickel grade and depth in yellow laterite.

- Full three dimensional ore control as applied at other open pit operations would be problematic due to the effects of short-range variability. Efforts to select high-grade components of the deposit by conventional bench ore control would probably result in significant misallocation errors.

The application of cut-off grade to the Goro Deposits is called a two dimensional "2D" or "top down" approach. The treatment of the block model for mine planning paralleled this 2D concept.

The procedure for marking blocks that met the 2D cut-off was as follows:

1. Each 30m x 30m column of model blocks was scanned vertically from the top down.

2. For a given cut-off, two vertically adjacent model blocks (1m blocks) above that cut-off value were required to establish the top of ore. This boundary occurred within the yellow laterite horizon.

3. The top of ore was marked at the top of the two adjacent blocks and all blocks from that point down to the bedrock contact were also marked as ore at that cut-off.

4. The procedure was repeated at a number of cut-offs from low to high.

5. The variable within the IMC installation was called "dilute" and was marked with nickel cut-offs of 0.80 0.85, 0.90, 0.95, 1.00,1.05, 1.10, 1.15, 1.20, 1.25, 1.27, 1.30, 1.32, 1.35, 1.40 and 1.45.

Table 19.9 summarizes the mineral reserves estimated based on a fixed cut-off of 1.20% within Zones 1, 2 and 3 of the Krige model. The Krige model has not been applied to Zone 4.

**Table 19.9: Goro Deposits Estimated Mineral Reserves**

| **Mineral Reserves @ 1.20 % nickel cut-off, -50.8mm.** | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| Classification | Tonnes Million | Ni % | Co % | Fe % | Si % | Mg % | Al % | Cr % | Mn % |
| Proven | 43.9 | 1.41 | 0.13 | 46.4 | 3.37 | 0.91 | 2.98 | 2.39 | 0.84 |
| Probable | 13.0 | 1.92 | 0.08 | 17.0 | 18.41 | 11.8 | 1.21 | 0.88 | 0.45 |
| **Total** | **56.9** | **1.52** | **0.12** | **39.6** | **6.81** | **3.40** | **2.57** | **2.05** | **0.75** |

As of the effective date of this Report, the authors' do not have knowledge of any environmental, permitting, legal, ownership, taxation, political or other relevant issues that would materially affect the mineral reserves and mineral resources estimates for the Goro Deposits not otherwise disclosed in Item 3 of this Report.

### 19.10 Factors that may have an Impact on Mineral Resources and Mineral Reserves Estimates

The following factors could affect the mineral resources and mineral reserves estimates for the Goro Project:

1. The Goro Project area is comprised primarily of ultramafic rocks that do not support agriculture and pastoral grasses, thus reducing human economic activity and human occupation on the site. Archaeological surveys of the site have found little evidence of past human settlement. However, while Goro Nickel is not aware of any such claim, it is possible that local residents within the vicinity of the Goro Project area could seek to assert rights to certain portions of the Goro plateau based upon principles of previous occupation or use of land as the Goro Project proceeds.

2. The flora found in the Goro Project areas is categorized into nine different vegetation types. It has been estimated that within the Goro Project area there are 600 species belonging to 289 genera and 90 families. Of these species, 75% are endemic to ultramafic rocks in New Caledonia. Goro Nickel has developed a plan to enhance sensitive species found within the Goro Project site. A number of mitigation measures are, however, expected to be in place, including transportation of rare and endangered species, developing buffer zones and, where possible, avoiding critical habitat. However, it is possible that future laws and regulations could be enacted which would create protective requirements and exclusion zones in certain areas within the Goro Deposits, but Inco and Goro Nickel believe that the areas of these potential sites is expected to be relatively small and on the periphery of the delineated mineralized zones.

3. The laterite layers for the most part erode easily and are subject to changing groundwater flows. The original bedrock structure controls the groundwater systems that changes both vertically and horizontally. A combination of interconnected groundwater systems, inter-watershed flows and the creation of sinkholes (dolines) complicates the regional hydrogeology. The estimate of

mineral resources could be adversely affected by any such developments. Goro Nickel has developed mitigation procedures and actions to address these potential issues.

4. No mine test has been carried out to date within the Goro Deposits, which has an effect on the degree of confidence of the mineral reserves and mineral resources estimates stated in this Report.

5. The ability of the FPP to screen to 50mm without significant fine losses or boulder attrition has not yet been thoroughly tested. A boulder attrition factor has been included in the mineral reserves estimates but no factors were estimated for lost of fine material.

6. Bulk samples for pilot plant testing were conducted using material taken from several different locations within the first 20 years of mine life for the Goro Project. While there have been no known problems pertaining to the processing of the mineral reserves and mineral resources estimates in this Report, problems could occur once commercial production commences, given the limited processing of material to date.

7. The principal taxation factors that could affect the mineral resources and mineral reserves estimates are related to New Caledonia income taxes, the French tax regime and certain Canadian taxes. Any increased taxation would affect the quality and amount of material that would need to be exploited to provide the projected profitability and returns envisioned.

# 20. OTHER RELEVANT DATA AND INFORMATION, INCLUDING ADDITIONAL REQUIREMENTS FOR TECHNICAL REPORTS ON DEVELOPMENT PROPERTIES AND PRODUCTION PROPERTIES

## 20.1. Mine

### 20.1.1. Mining Operation and Method

The Goro Deposits will be mined by conventional open pit methods using hydraulic excavators and rigid frame trucks. The Goro Deposits are generally flat, lying with an overall geologic profile not exceeding 90m in depth, a depth which is favourable to open pit operation. The mining method is based on the operating experience of Inco's operations in Indonesia as well as the operations that Inco had conducted at its Guatemala operations prior to those operations having been mothballed in the early 1980s. These operations share some similarities with the Goro Deposits.

The mining rate will be approximately 4.8 million dry tonnes of ROM with an average waste to ore stripping ratio of 1:1. The ROM will be delivered to the FPP located at the north-western edge of the mining area where the oversize material is wet screened. The limonite and saprolite ROM will be either directly dumped into the FPP or placed on separate stockpiles ahead of FPP. It is anticipated that a certain amount of short term and long term blending could be required as the mined out material available at the face may not meet the process plant requirement. The short term blending stockpile is planned to accommodate the local chemistry variations encountered in the pit over a few days period, while the long term stockpile is planned to accommodate for the lack or excess of limonite or saprolite encountered in successive pushbacks. The stockpiles will also accommodate the FPP during mine's shift changes and mine downtimes due to inclement weather. The actual size of the stockpiles are planned to be large enough to feed the FPP over a 14 days period.

Where possible, vegetation will be commercially recovered; vegetation not recovered will be chipped and mulched for later use in rehabilitation. While topsoil varies from sparsely distributed to non-existent, to the extent possible, it will be recovered and stockpiled for later rehabilitation.

The topsoil removal will be followed by recovery of iron cap and iron shot, both of which are potentially valuable materials for road building, bench sheeting, dam/impoundment construction and for engineered backfill in civil construction. Generally, bulldozer ripping of the iron cap layer will be sufficient to loosen the material ahead of loading by a backhoe excavator into haul trucks, although a limited small diameter drilling and blasting may be necessary to break the hard Iron Cap material into a manageable size.

The limonite overburden, comprised of yellow and red limonite, will be loaded by hydraulic excavators and hauled by truck to the overburden dump. Approximately 50% of overburden volume will be used as construction material for building the tailings containment dykes and berms.

All ore zones will be mined by using hydraulic excavators. While the limonite ore has a homogenous and generally fine-grained texture, the saprolite ore is characterized by the boulder content. Some drilling and blasting of oversize boulders and bedrock pinnacles could be required to facilitate mining and provide excavator access to saprolite that would otherwise be inaccessible.

Mine layout and the direction and sequence of mining are largely dictated by the requirement to focus initial mining on Zone 1, to maximize in-pit waste disposal and to minimize resource sterilization. Other factors influencing mine layout will be ore quality distribution, drainage and water management.

The top of ore will be defined by in fill core drilling. From this horizon, the limonite ore will be mined through the transition zone and into the saprolite until bedrock is reached. Within the limits of the bulk mining method and equipment, the limonite and saprolite will be mined and handled separately. Overburden removal and limonite and saprolite mining will occur simultaneously at various parts of the pit to maintain an approximately constant limonite to saprolite ratio to the autoclave feed.

The planned mining method will utilize hydraulic excavators in the 13.6 cubic meter range for loading 100 tonne haul trucks. A fleet of two 4.4 cubic meters backhoes loading 50 tonne haul trucks will excavate some of the pit material as well as excavate and maintain pit drainage structures. Both overhand and underhand mining methods will be planned as necessary for the local conditions prevailing in the pit. Overhand mining by hydraulic excavators will face up to benches not exceeding 10 metres in height, while underhand mining is not planned to exceed 5 metres in depth.

Estimates of sheeting use and dilution were based on the mining method described above, with sheeting required at every 15m vertically. There is some flexibility in those assumptions depending on the amount of bench sheeting on each bench and the number of working bench elevations necessary in each mine area. Bulldozer separation of the top of saprolite as well as the overburden (non-economic) material within the yellow laterite to the nearest 1 meter vertical is contemplated. The smaller equipment fleet will also be used for bottom saprolite recovery.

Pushback or phase designs are a series of practical expansions of the pit from the initial mine opening to the final pit geometry. The relative extraction sequence of the phases is established during their design, but the precise timing of phase mining is determined during the development of the mine production schedule. At any point in time, between 2 and as many as 5 pushbacks will be mined at a time at the Goro Deposits for blending and mining versatility. In general, the pushbacks are incremental 100m wide expansions of the pit. Each pushback is mined in sequence from the top down so that a 100m wide working flat exists on each pushback for the mine equipment. The pushback designs incorporate all of the practical constraints into the design of operable mine geometries. The basic design criteria for the pushbacks is presented in Table 20.1.

**Table 20.1: Key Factors and Pit Design Parameters**

| | Tonnage factors | | | Dilution | | | |
|---|---|---|---|---|---|---|---|
| | In Situ Dry Density | Moisture % of Total | Mining Recovery | Overburden % by | Sheeting % by | Bedrock % by | Boulders % by |
| | $t/m^3$ | % | % | Volume | Weight | Weight | Weight |
| Material | | | | | | | |
| Fill | 1.75 | 16 | | | | | |
| Iron Cap | 2.00 | 12 | | | | | |
| Iron Shot | 1.50 | 20 | | | | | |
| Red Laterite | 1.00 | 38 | | | | | |
| Yellow Laterite (-150 mm) | 0.90 | 45 | 97 | 3 | 0.75 | | |
| Transition (-150 mm) | 0.90 | 49 | 100 | | | | |
| Saprolite ROM (Ave.) | 1.35 | 26 | 79 | | 0.5 | 0.5 | 5.00 |
| Boulders | 2.24 | 8 | | | | | |
| Bedrock | 2.24 | 4 | | | | | |
| Sheeting | 1.60 | 12 | | | | | |
| Dilution Grades: | | | | | | | |
| Ni % | | | | Variable | 0.5 | 0.6 | 0.6 |
| Co % | | | | Variable | 0.02 | 0.03 | 0.02 |
| Mg % | | | | Variable | 11.7 | 21.4 | 23.1 |
| **Pit Design Parameters** | | | | | | | |
| Interramp Slope Angles | 20 Degrees | | | | | | |
| Haul Road Width | 27m | | | | | | |
| Maximum Road Grade | 8% | | | | | | |
| Max. Bench Height Overhand | 10m | | | | | | |
| Max. Bench Height Underhand | 5m | | | | | | |
| Min. Pushback Width | 100m | | | | | | |
| Bench Sheeting Thickness | 1.5m | | | | | | |
| Main Haul Roads Sheeting | 1.75m | | | | | | |

### 20.1.2 Dilution and Recovery

This section summarizes the mining recovery, dilution, and attrition factors used by IMC in the development of mine and autoclave feed production schedules. Some of these factors are the result of simulation work completed by Inco personnel during the 2001-2002 period. Others are assumptions made by both Inco and IMC personnel working together. Reference is made to reference 10 in Item 23 of this Report for information regarding the source of dilution and recovery factors.

Mining recovery addresses the volume and corresponding tonnage of material that can be delivered to the FPP compared with the volume and tonnage that the model estimates is in the ground. The recovery rate assumptions are based on recovery of sample. The following mining recovery rates are applied by material type:

- yellow laterite ore 97%
- transition ore 100%
- saprolite ore 79%

Transition ore is expected to be 100% recovery because of its location between the yellow laterite and saprolite layers and combined with yellow laterite makes up the limonite component of the feed.

The mining recovery factors were determined based on the results of the stochastic simulation of the geologic contacts as discussed previously. These recovery factors are applied to determine the undiluted component of FPP feed and do not reflect recoveries due to screening or processing. The above rates only represent the inability of mining equipment to excavate all of the ore in the ground.

Four sources of dilution have been identified and incorporated into the production schedules:

1. Dilution from laterite overburden (sub-cut-off grade yellow laterite)
2. Dilution from bench sheeting
3. Dilution from bedrock
4. Dilution (attrition) from boulders contained within the saprolite

The following represents an explanation of each of these four sources of dilution:

1. *Dilution from laterite overburden*:

During the stripping operation a certain amount of waste will be removed with the ore due to local variations in the ore contact. The overburden dilution was estimated using the geological layers contact between the red and yellow laterite at about 3% or 0.75m on average of the yellow laterite recovered. The average amount of dilution was calculated based on applying reasonable mine geometry constraints to a series of simulated cross sections through the geologic profile.

The grade of the laterite dilution is based on the average grade of a one-meter thick layer that overlies the entire deposit immediately above the cut-off grade surface. It is assumed that laterite dilution will pass completely through the FPP process and report in its entirety to the autoclave feed.

2. *Dilution from bench sheeting:*

Bench sheeting or “plating” is an aggregate that is placed on the bench surfaces so that mine equipment can operate over the ground with poor traffic ability. Sheeting is planned to cover 50% of each 15m vertical in the mine (one 10m overhand and one 5m underhand bench) with a thickness of 1.5m. It is assumed that about 1.25m of sheeting will be recovered and an equivalent of 0.25m of the sheeting will not be segregated and will be included within the feed to the FPP as dilution. Approximately 50% of the bench sheeting fed to the FPP is assumed to be less than 50mm in size which will not be rejected at the FPP and will become part of the autoclave feed as dilution.

The sheeting dilution is estimated at about 0.8% by volume. It is assumed that the sheeting material will be composed of boulder reject from the FPP and of iron cap material in about the same proportions and therefore the grade of this dilution is derived from the grade of boulders and the iron cap. The estimated grade of bench sheeting dilution to the autoclave is estimated to be about 0.5% nickel, 0.02% cobalt and 11.7% magnesia.

3. *Dilution from bedrock:*

Based on a desktop mining exercise done in 2001, it was concluded that to achieve a higher saprolite recovery a certain amount of bedrock pinnacles may require blasting. By selectively blasting the bedrock, it is planned that about 79% of saprolite recovery can be realized.

The amount of bedrock that will have to be mined and blasted is about 11% of the saprolite volume recovered (20% by weight) or about 0.65m bedrock thickness on average. It is assumed that about fifty percent of the blasted rock will be discarded as waste at the mine. The remaining fifty percent of the blasted rock will be hauled to the FPP. About five percent of this material will be composed of fines generated from the blasted rock and will be smaller than 50mm in size. This will result in about 0.5% dilution by weight (20.33% x 50% x 5%) or 0.016m (0.65m x 50% x 5%) of bedrock will be fed to the autoclave. The grade of the bedrock dilution is at the average of bedrock material and the dilution grade are estimated to be about 0.6% nickel, 0.03% cobalt and 21.4% magnesia.

4. *Dilution from boulders:*
The saprolite feed to the FPP will contain boulders and there is expected to be some size reduction of those boulders that will contribute to dilution. These factors are applied on a global basis to estimate the overall boulder attrition, which includes both (i) mining attrition within the FPP feed, and (ii) screening attrition to the autoclave feed.

It is assumed that 5% of the mine recovered +50mm sized material will be sent to the FPP and through to the autoclave as dilution. The attrition within the FPP feed is already contained within the estimated saprolite feed tonnage. Boulder attrition grades are estimated at about 0.6% nickel, 0.02% cobalt and 23.1% magnesia, which is the average of boulder composition grades.

### 20.1.3 Tailings and Overburden Disposal

The process residues are produced during the removal nickel and cobalt from the lateritic ore andmine overburden is removed to access the nickel containing lateritic ores. The placement of the process residues and mine overburden are are interdependent on mining and integrated in the overall mining concept to maximize disposal within the mined out open pit. Initially, the process residues will be disposed of out-of pit for the first five years in the upper part of the East Kwe valley and the mine overburden that is not suitable, or is not required, for the construction of the containment structures or berms will be disposed of for some years in the out-of-pit east overburden dump.

The residues coming from the process plant will be thickened to about 50% solids at the effluent treatment plant, to be located on the ridge to the south of the mine. Ongoing berm construction will allow for in-pit disposal of residues commencing around year 6 of the mine life. From this point, additional cells will be constructed behind the mining front to allow for continual residues disposal until the end of the mine life. All berms will be constructed from a suitable overburden material. It is estimated that about 50% of the mine overburden will be used to construct the tailings containment berms and as haul road sheeting.

Containment berms constructed from the mine waste will have a minimum crest width of 30 m. Overall berm slopes will not be steeper than 2:1 (horizontal to vertical). The residue deposition beach slope is assumed at 5%. Supernatant and runoff water will be collected and pumped to the effluent treatment plant and then discharged offshore through the ocean outfall pipe. The remainder of the natural runoff from the mine and waste dump will be collected in the mine and discharged to the North Kwe and ultimately into the planned water reservoir. The reservoir will be used to supply water for the process plant and other operational needs. Reference is made to reference 18 in Item 23 of this Report for information regarding the citation of tailings and overburden disposal plans.

#### 20.1.4 In-Pit Drainage Management

Appropriate in-pit drainage and water management will be essential to sustain operational efficiency, which will rely on maintaining the main haul road system and pit benches. Therefore, an essential component of the drainage management strategy is to capture, as much as possible, groundwater flow before it can enter the mining excavations. To accomplish this, a trench will be excavated ahead of the advancing mining face through the iron cap/iron shot layer to the top of limonite to intersect and divert this ground water flow into large volume sumps located outside the mining area.

In-pit water management will rely on a system of in-pit trenches and sumps to control seepage from the overburden, flows from local bedrock fracture zones, and in pit precipitation. Sloping benches with culverts at strategic locations, drainage ditches and pumps powered by diesel generator sets will be used to transfer collected water to the main collector sumps. In addition, large capacity drains will be excavated and maintained beside haul roads to prevent the road base from becoming saturated with standing water.

#### 20.1.5 Mine Equipment

The main considerations in the mine design and equipment selection are climate and its impact on ground conditions and use of conventional wheeled vehicles. Equipment sizing is based on the characteristics of the materials to be mined and equipment quantities are based on an assumed two shifts, 20 hour/day, seven day per week schedule, with an allowance for rain delay of 21 days per year. The primary mine production fleet and sizes are planned as follows:

| *Description* | *Nominal Size* | *No.# of Units* |
|---|---|---|
| Hydraulic Excavator Front Shovel | 13.6m$^3$ | 3 |
| Hydraulic Excavator Backhoe | 4.4m$^3$ | 2 |
| Off Highway Dump Truck | 100 | 13 |
| Off Highway Dump Truck | 50 | 5 |
| Track Dozer | 302kW | 3 |
| Track Dozer LGP | 228kW | 4 |

In addition, auxiliary equipment and light duty vehicles will provide support for the production operation.

#### 20.1.6 Mine Production Schedule

The mine production schedule has been developed on a yearly basis taking into account the footprint of utilizing the mineral reserves estimate.

The mine production schedule is based on a fixed cut-off of 1.20% nickel and the use of an intermediate stockpile between the FPP and the mine face for blending purposes. Table 20.1.6 shows the following:

- that the process plant production after year 7 is below 123 Mlbs of nickel per year due to high acid consumption. This limitation is imposed by the capacity of the actual sulphur burning plant to produce acid for leaching minerals in the autoclave;

- that at the projected rate of production the planned pit design reaches the limit of the mineral reserves estimates at year 14 of the Goro Project; and

- the mineral reserves estimates are totally mined out at year 14. This will represent 52.3 million tonnes out of the 56.9 million tones currently estimated. The difference is due to the requirement that the mining of the remaining mineral reserves would involve mining mineral resources simultaneously with the mineral reserves for blending purposes. Mineral resources were not included in the yearly production schedule nor in the economic analysis set forth in Item 20.9 of this Report.

# Table 20.1.6: Mine Production Schedule

**Feb43-101Fix120 Schedule, Stockpile Utilization**

**December 2002 Block Model**
**Grades are Based on 50mm Screen, Cut-offs Reflect Top of Laterite Selection**

| Year | Nickel Cutoff% | Autoclave Feed Dry Ktonnes | Element Grades Ni % | Co% | Mg% | Al% | Mn% | Fe% | Si% | Cr% | Screen Reject Ktonnes +50mm | Calculated Total Ktonnes | Recov Ni Mlbs 0.922 | Recov Co Mlbs 0.908 | Acid Factor Average | Acid Product Check |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 0 | 1.20% | 633 | 1.266 | 0.071 | 0.360 | 2.892 | 0.638 | 50.312 | 1.042 | 2.445 | 4 | 7975 | 16.3 | 0.9 | 0.180 | 113.9 |
| 1 | 1.20% | 1687 | 1.401 | 0.137 | 1.230 | 3.243 | 0.922 | 45.913 | 2.836 | 2.383 | 88 | 6999 | 48.0 | 4.6 | 0.271 | 456.7 |
| 2 | 1.20% | 3776 | 1.532 | 0.125 | 2.746 | 2.498 | 0.806 | 41.598 | 5.888 | 2.202 | 698 | 7001 | 117.6 | 9.4 | 0.339 | 1281.1 |
| 3 | 1.20% | 3947 | 1.533 | 0.135 | 1.824 | 2.544 | 0.815 | 44.514 | 4.433 | 2.426 | 430 | 7000 | 123.0 | 10.7 | 0.292 | 1151.1 |
| 4 | 1.20% | 3793 | 1.595 | 0.121 | 3.056 | 2.406 | 0.712 | 40.751 | 6.470 | 2.155 | 1019 | 7000 | 123.0 | 9.2 | 0.350 | 1327.4 |
| 5 | 1.20% | 3730 | 1.622 | 0.139 | 3.238 | 2.657 | 0.807 | 40.091 | 6.359 | 2.218 | 882 | 7000 | 123.0 | 10.4 | 0.368 | 1371.9 |
| 6 | 1.20% | 3804 | 1.591 | 0.155 | 2.725 | 2.996 | 0.908 | 41.014 | 5.681 | 2.172 | 821 | 8849 | 123.0 | 11.8 | 0.357 | 1356.5 |
| 7 | 1.20% | 3832 | 1.579 | 0.133 | 3.461 | 2.911 | 0.773 | 39.174 | 6.742 | 1.962 | 1106 | 8850 | 123.0 | 10.2 | 0.381 | 1459.0 |
| 8 | 1.20% | 3798 | 1.493 | 0.117 | 4.031 | 2.680 | 0.714 | 37.324 | 7.896 | 1.936 | 1224 | 5503 | 115.3 | 8.9 | 0.388 | 1473.7 |
| 9 | 1.20% | 3890 | 1.441 | 0.112 | 3.635 | 2.853 | 0.719 | 38.354 | 7.291 | 1.928 | 1159 | 8669 | 113.9 | 8.7 | 0.379 | 1474.3 |
| 10 | 1.20% | 3954 | 1.425 | 0.111 | 3.422 | 2.874 | 0.746 | 38.504 | 7.338 | 1.945 | 1188 | 8479 | 114.5 | 8.7 | 0.373 | 1474.3 |
| 11 | 1.20% | 3824 | 1.454 | 0.102 | 4.028 | 2.635 | 0.692 | 36.990 | 8.242 | 1.866 | 1412 | 8086 | 113.0 | 7.8 | 0.386 | 1474.3 |
| 12 | 1.20% | 3794 | 1.490 | 0.100 | 4.298 | 2.400 | 0.685 | 37.476 | 7.897 | 1.884 | 1498 | 8023 | 114.9 | 7.6 | 0.389 | 1474.3 |
| 13 | 1.20% | 3931 | 1.495 | 0.101 | 3.898 | 2.305 | 0.721 | 39.240 | 7.056 | 1.982 | 1339 | 8318 | 119.5 | 8.0 | 0.375 | 1474.4 |
| 14 | 1.20% | 3974 | 1.450 | 0.107 | 2.327 | 2.365 | 0.730 | 43.742 | 5.005 | 2.172 | 680 | 8404 | 117.1 | 8.5 | 0.303 | 1202.7 |
| TOTAL | | 52367 | 1.508 | 0.120 | 3.178 | 2.648 | 0.760 | 40.240 | 6.442 | 2.080 | 13549 | 116157 | 1605.1 | 125.4 | 0.355 | |

### 20.2. Process

The tonnage throughput of the process plant is determined by the autoclave size and residence time requirements to achieve the required leaching of nickel and cobalt at the planned operating temperature of 270°C. The annual nickel and cobalt annual production is then calculated taking into account the following parameters:

- Annual tonnage of autoclave feed processed
- Average annual nickel and cobalt grade of the autoclave feed
- Overall recoveries from the process plant
- Utilization of the process plant
- Maximum acid production

A detailed mass balance has been constructed to calculate the overall recoveries of nickel and cobalt. The nominal feed rate to the autoclaves is 536.5 t/h. Recoveries from the individual unit operations are based on results obtained in the pilot plant. The overall recoveries of nickel and cobalt from autoclave feed used in the design are 92.2% and 90.8%, respectively.

The utilization of the process plant will be dictated primarily by the time that the autoclaves are on line. A detailed evaluation of the utilization of the autoclaves was conducted during the BFS and has been updated during the basic engineering for the commercial plant design. The utilization takes into account planned shutdowns of the autoclaves for such occurrences as descaling, and an estimate of unplanned shutdowns due to potential mechanical issues. The impact of other unit operations on the utilization of the autoclaves is also taken into account, but this is minimized by the presence of adequate surge capacity in front of (autoclave feed storage tanks) and after (large surge pond) the leaching circuits. The overall plant utilization is estimated to be 84.5%.

An engineering design margin of 15% has been applied to all nominal process flows to allow for factors that may impact negatively on the overall plant capacity and to allow for limited variation in operating rates.

The design basis is considered to be sufficient to ensure that the required production rate for nickel and cobalt can be met.

### 20.3. Markets

It is currently planned that the Goro Project will produce a single nickel product and a single cobalt product. The nickel end product will be a granular nickel oxide, analyzing about 78% nickel, produced by fluid bed pyrohydrolysis. The nickel end product is a so-called Class 2 primary nickel form that is expected to be consumed as an intermediate product for further refining and, to a lesser extent, as a charge product in the production of stainless steel. The granular nature of the product makes for a non-dusting, free flowing material, which is expected to be easy to handle in the end-users' existing equipment. The expected cobalt end product is a cobalt carbonate filter cake, containing about 47% cobalt, to be used either as end product or an intermediate product to produce other cobalt chemicals or metals.

### 20.3.1. Use of the Products

**Nickel Oxide**

Nickel oxide is currently utilized by Inco as feed to several of its joint venture refineries. These joint venture refineries include Taiwan Nickel Refining Corporation (TNRC) in Taiwan, and Korea Nickel Corporation (KNC) in South Korea.

The TNRC and KNC joint ventures produce UTILITY® nickel for the stainless steel mills in Korea and Taiwan. UTILITY® nickel is an easily handled and well-accepted product by the stainless steel industry.

The nickel oxide to be produced in connection with the Goro Project is expected to be used by Inco, primarily as feed for the existing TNRC and KNC joint venture refineries and a new refinery to be constructed in the Asia Pacific region. The Goro Project focus on the production of this intermediate product provides the flexibility to ship product to other existing Inco refineries worldwide or to new refineries that may emerge.

**Cobalt Carbonate**

Inco intends to maximize the revenue to be generated from the cobalt carbonate sales by supplying expected customers for this product directly and by utilizing current technology and expanded production facilities to produce high grade electrocobalt. The purity of Inco's electrocobalt is expected to enable direct consumption into the demanding industry sector of superalloys. It would also be possible to use this product in the cobalt chemical industry.

With significant increases in the global supply of cobalt and changes in demand, the price of cobalt has fluctuated significantly over the past several years. The financial analysis undertaken in support of the substantial investment to be made with respect to the Goro Project has been based upon a long-term price of cobalt of \$15,400 per tonne (\$7.00 per pound). If realized cobalt prices, as well as realized prices for nickel to be produced by the Goro Project, were to be below the long-term prices assumed for the Goro Project, the expected financial returns from, and the expected cash and other unit costs of production for the Goro Project, would be adversely affected.

### 20.3.2. Demand for the Products

**Nickel Oxide**

Historically, nickel demand has grown on average 4 percent per annum. This growth in demand has been driven primarily by growth within the stainless steel market. Since 1950, stainless steel production has shown a steady growth pattern, with a compound annual growth rate of approximately 6% per annum.

Over the last 10 years, however, Asia has dominated the market, with demand for nickel in Asia (excluding Japan but including China) growing at an average annual rate of 11% per annum. In 2001, stainless steel demand for primary nickel in Asia (excluding Japan) accounted for almost

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64% of total nickel demand within the region. Production of stainless steel within the region has grown almost five-fold within the last decade. Growth in particular has been concentrated in Korea and Taiwan. Asian production of stainless steel in 2002 accounted for 18% of world stainless steel output, compared with only 7% in 1992. Demand for primary nickel in stainless steel has grown on average 12 percent per annum since 1992.

Since the nickel oxide to be produced by the Goro Project would mainly be used for producing UTILITY® nickel, the growth in demand for the product will be tied to the growth in demand for stainless steel. From the above information, it should be understood that the Goro Project will play an important role in meeting the future nickel requirements of the stainless steel industry within the Asian market.

Overall, global stainless steel demand is expected to continue at least at long-term trend growth rates of 5 to 6 per cent as strong growth in Asia and China is expected to be balanced with more modest growth in other regions of the world. Nickel containing stainless steel scrap will remain a competitive factor in the market and a key determinant of primary nickel demand in stainless steel production. In the short term, scrap supply will depend upon the pricing and investment activity, which results in the dismantling of plant and equipment and a consequent supply of obsolete scrap into the market. In the longer term, scrap supplies depend upon both the current and past consumption of stainless steel. Looking forward, supplies of obsolete scrap are expected to continue to dominate the growth in scrap usage. Overall, the proportion of nickel units in stainless steel production obtained from stainless steel scrap will likely continue to range between 44% and 48%. Stainless steel scrap is therefore not expected to negatively impact primary nickel demand over the longer term.

**Cobalt Carbonate**

Unlike nickel usage in stainless steel, cobalt demand is not dominated by a single market segment. The principal current uses of cobalt by consumption are shown in Table 20.3 below.

The largest sector, superalloys, is very sensitive to the capital expenditure cycle since the major markets are aircraft engines/turbines and land-based turbines. While global expenditure growth has not been significant over the recent year, vehicle aircraft and energy generation are two end-use markets for cobalt that are expected to have a bright future.

The most rapidly growing area of cobalt usage has been in the communications sector. The use of cobalt in lithium batteries for portable devices, such as cellular phones, computers and pagers has been increasing at double-digit growth. Most forecasters expect at least a 300 per cent growth in cobalt usage in this area over the next 10 years. There may be additional opportunities for uses in batteries in the automotive industry.

Cobalt is also used for many special salts and catalysts. New mandates for cleaner fuels represent new opportunities for cobalt catalysts.

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**Table 20.3: Distribution of Cobalt in Products**

| Product | Consumption |
|---|---|
| Super Alloys | 23 |
| Hard Metals | 22 |
| Electronics/Power | 10 |
| Paint/Colours | 12 |
| Magnets | 9 |
| All Other | 24 |
| **Total** | **100** |

### 20.4. Contracts

As previously stated, the nickel oxide to be produced by the Goro Project is currently planned to be used primarily as feed for Inco's joint venture refineries that perform the final stage processing, packing and delivery of UTILITY® nickel. At present, the feed to the joint ventures is sourced from nickel oxide produced at Inco's existing operations and through third party purchase contracts. The price that Inco receives for its nickel oxide relates to the London Metal Exchange (LME) cash price for nickel, normally subject to a discount to reflect additional costs such as processing, freight and duty, to convert to a product sold to the stainless steel industry.

The commercial agreements with Goro Nickel will be between Inco or one of its other affiliates in a manner that is consistent with Inco's existing agreements for nickel sales between its joint venture partners. Inco's marketing network is worldwide with office locations in each major region around the world.

The commercial agreements that are envisioned with Goro Nickel will be based upon Inco's knowledge of the relevant market and from the current dealings it has in this business. The commercial terms expected with Goro Nickel have been modeled from this experience and reflect the commercial risks that would exist in these arrangements, including the product quality, quantity and duration of the agreements. It should be noted that the market for nickel oxide beyond Inco and its affiliates and joint ventures partners is currently limited.

**Hedging**

The objective of the hedge program for the Goro Project will be to minimize the risk of foreign exchange rate fluctuations for a portion of its cash flows, including payables. A formal hedging strategy for Goro Nickel has been developed by the corporate treasurer and comptroller departments and approved by Inco's Risk Management Committee. The hedge program is expected, when implemented, to cover a portion of the forecasted cash requirements associated with the construction of Goro Nickel's production facilities in New Caledonia and manage exposures to changes in currency exchange rates other that Inco's functional currency. Inco's intention is to ensure that, for the amounts being hedged, Inco is not at risk for changes in foreign exchange rate.

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Goro Nickel has not entered in to any hedging transactions. As at December 31, 2002, Inco had outstanding forward currency contracts to purchase Aus$274 million in 2003 at an average price of $0.518 and Aus$116 million in 2004 at an average price of $0.522 and Euro 213 million in 2003 at an average price of 0.886 and Euro 60 million in 2004 at an average price of $0.873. The purpose of Inco's Australian dollar and Euro forward currency contracts is to hedge a portion of the forecast capital costs of the Goro Project in those currencies. As discussed above, some of these forward currency contracts maturing in 2003 were closed out in January 2003 since they no longer matched the timing of the planned expenditures due to the expected deferral of those expenditures as a result of the review of the Goro Project referred to in Item 3 of this Report.

### 20.5. Environmental Considerations: Closure Plan

A conceptual closure and reclamation plan has been prepared for the Goro Project. The reclamation plan is intended to be part of the overall Goro Project environmental management system and is expected to be revised and updated on a regular basis during all phases of the Goro Project. The closure and reclamation plan has been designed to meet local legislative requirements, Inco environmental health and safety policies and guidelines, industry standards and the concerns of local residents. The reclamation plan was prepared considering nine objectives and includes the criteria used for closure as well as the staged reclamation of disturbed areas. The nine objectives were to:

- restore affected landscapes to a stable and safe condition that will protect public health and safety;
- alleviate or eliminate environmental damage and liabilities;
- re-establish conditions that permit productive use of the land and the natural resources of the area in a manner similar to their original use, or an acceptable alternative;
- work toward an eventual passive closure;
- implement concurrent or progressive reclamation measures during the operation of the mine, where appropriate;
- minimize significant adverse effects on adjacent surface and groundwater resources;
- minimize erosion and contaminant dissolution in runoff;
- minimize long-term maintenance requirements; and
- provide for post-closure monitoring of the site to assess the effectiveness of the closure measures.

Reclamation will form an integral part of the mine plan and will be undertaken progressively where applicable over the life of the mine. The plan considers the long-term physical and chemical stability of the site including reclamation of the surfaces disturbances. The plan proposes management and monitoring systems that would be implemented during decommissioning and post-closure.

### 20.6. Taxes

The New Caledonian authorities enacted a fiscal regime in 2001 which provides a nominal 15-year tax holiday plus an additional five years at tax rates that are 50% of the prevailing tax rates for qualifying metallurgical companies such as Goro Nickel. If the Goro Project achieves an internal rate of return in excess of a cumulative threshold rate during this 20-year period, the applicable tax rates or levels for the Goro Project would then be adjusted prospectively to be equivalent to the general rates or levels then in effect for mining and processing companies.

Pursuant to current French taxes, except for a 5% withholding tax between France and Canada, France does not effectively tax any distributions from New Caledonia.

Under the Canadian tax laws, dividends and other amounts received from a source to which Canada has a Double Tax Treaty are exempt from further taxation.

There are no royalties payable in connection with the Goro Project.

### 20.7. Operating Costs Estimates

While the comprehensive review of the Goro Project referred to under "Review of the Goro Project" in Item 3 of this Report will include a review of the Goro Project's capital cost estimate, it is expected that this review will also include an evaluation of operating costs. While Goro Nickel cannot predict at this time what, if any, effect such review will have on its operating costs estimates, it is not aware of any significant changes in the estimates of operating costs which were included in the BFS.

The principal operating costs for the Goro Project are expected to be labour, reagents, including sulfur, power/energy and other outsourced services.

The following table sets forth the operating costs estimates for the first 14 years of the Goro Project, the same period covered by the economic analysis referred to in Item 20.9 below, per pound of nickel produced by the Goro Project, broken down as follows:

1. On-site estimated operating costs (before and after by-product credits for cobalt): these estimated operating costs include all such costs incurred by Goro Nickel for the Goro Project at the site of the Goro Project; and

2. Total estimated operating costs (before and after by-product credits for cobalt): these estimated operating costs include the costs referred to in (1) above together with the estimated technical and marketing fees, operating cost taxes and costs associated with certain administrative facilities in Paris, France.

The table below also includes a weighted average breakdown (in percentages) of the principal estimated operating costs over the 14 years referred to above.

# Table 20.7: Goro Project Operating Costs Estimates

| GORO PROJECT BASE CASE | | TOTAL | 2001 | 2002 | 2003 | 2004 | 2005 | 2006 | 2007 | 2008 | 2009 | 2010 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Base Case key assumptions: Capex US$1.675 B; Aug 2005 start-up; LME US$3.20/lb Ni, MB US$7.00/lb Co (pre discount); mine to 2019; no terminal value | | | | | | | | | | | | |
| Includes: 2.75% marketing fee. | | | | | | | | | | | | |
| **Operating Cost Estimates** | | | | | | | | | | | | |
| **On-Site Estimated Operating Costs** | | | | | | | | | | | | |
| Before By-Product Credit | US$/lb Ni | $1.39 | $0.00 | $0.00 | $0.00 | $0.00 | $3.78 | $1.59 | $1.33 | $1.33 | $1.32 | $1.32 |
| After By-Product Credit | US$/lb Ni | $0.95 | $0.00 | $0.00 | $0.00 | $0.00 | $3.30 | $1.12 | $0.87 | $0.87 | $0.88 | $0.82 |
| **Total Estimated Operating Costs** | | | | | | | | | | | | |
| Before By-Product Credit | US$/lb Ni | $1.51 | $0.00 | $0.00 | $0.00 | $0.00 | $4.10 | $1.71 | $1.45 | $1.45 | $1.44 | $1.44 |
| After By-Product Credit | US$/lb Ni | $1.07 | $0.00 | $0.00 | $0.00 | $0.00 | $3.62 | $1.25 | $0.99 | $0.99 | $1.00 | $0.94 |
| **Weighted Average Breakdown** | **On-Site** | **Total** | | | | | | | | | | |
| Labour | 20.5% | 18.9% | | | | | | | | | | |
| Operating Supplies | 41.1% | 37.7% | | | | | | | | | | |
| Power | 13.6% | 12.5% | | | | | | | | | | |
| Maintenance Materials | 15.9% | 14.6% | | | | | | | | | | |
| Contracted Services | 6.9% | 6.3% | | | | | | | | | | |
| Sundry Overhead Costs | 2.1% | 1.9% | | | | | | | | | | |
| On-Site Operating Costs | 100.0% | 91.9% | | | | | | | | | | |
| G&A (including Paris) | | 0.3% | | | | | | | | | | |
| Technical Fees | | 1.5% | | | | | | | | | | |
| Marketing Fees - Nickel | | 5.5% | | | | | | | | | | |
| Marketing Fees - Cobalt | | 0.8% | | | | | | | | | | |
| Total Operating Costs | | 100.0% | | | | | | | | | | |

| GORO PROJECT BASE CASE | | TOTAL | 2011 | 2012 | 2013 | 2014 | 2015 | 2016 | 2017 | 2018 | 2019 | 2020 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Base Case key assumptions: Capex US$1.675 B; Aug 2005 start-up; LME US | | | | | | | | | | | | |
| Includes: 2.75% marketing fee. | | | | | | | | | | | | |
| **Operating Cost Estimates** | | | | | | | | | | | | |
| **On-Site Estimated Operating Costs** | | | | | | | | | | | | |
| Before By-Product Credit | US$/lb Ni | $1.39 | $1.33 | $1.35 | $1.38 | $1.39 | $1.39 | $1.39 | $1.37 | $1.36 | $1.37 | $0.00 |
| After By-Product Credit | US$/lb Ni | $0.95 | $0.82 | $0.90 | $0.95 | $0.96 | $0.98 | $1.01 | $1.00 | $0.97 | $0.96 | $0.00 |
| **Total Estimated Operating Costs** | | | | | | | | | | | | |
| Before By-Product Credit | US$/lb Ni | $1.51 | $1.45 | $1.47 | $1.50 | $1.51 | $1.51 | $1.51 | $1.49 | $1.48 | $1.50 | $0.00 |
| After By-Product Credit | US$/lb Ni | $1.07 | $0.94 | $1.02 | $1.07 | $1.08 | $1.10 | $1.13 | $1.12 | $1.09 | $1.10 | $0.00 |
| **Weighted Average Breakdown** | **On-Site** | **Total** | | | | | | | | | | |
| Labour | 20.5% | 18.9% | | | | | | | | | | |
| Operating Supplies | 41.1% | 37.7% | | | | | | | | | | |
| Power | 13.6% | 12.5% | | | | | | | | | | |
| Maintenance Materials | 15.9% | 14.6% | | | | | | | | | | |
| Contracted Services | 6.9% | 6.3% | | | | | | | | | | |
| Sundry Overhead Costs | 2.1% | 1.9% | | | | | | | | | | |
| On-Site Operating Costs | 100.0% | 91.9% | | | | | | | | | | |
| G&A (including Paris) | | 0.3% | | | | | | | | | | |
| Technical Fees | | 1.5% | | | | | | | | | | |
| Marketing Fees - Nickel | | 5.5% | | | | | | | | | | |
| Marketing Fees - Cobalt | | 0.8% | | | | | | | | | | |
| Total Operating Costs | | 100.0% | | | | | | | | | | |

### 20.8. Capital Costs Estimates

The following reflects, given the current status of the "Review of the Goro Project " discussed in Item 3 of the Report, a capital cost estimate for the Goro Project representing approximately a 15% increase in the $1.45 billion estimate based upon the BFS. This estimate allocates the approximately 15% increase among a number of specific areas based upon the current best judgment of Inco and Goro Nickel. This total figure represents the capital cost estimate that has been used for the "Economic Analysis" referred to in Item 20.9 below and for the mineral reserve estimate for the Goro Deposits as of year-end 2002.

**Table 20.8:Capital Cost Estimates[1]**

| **Principal Areas** | |
|---|---|
| Mine | 38,127,000 |
| Process Plant | 512,073,000 |
| Ancillary Facilities | 186,604,000 |
| General Facilities | 119,005,000 |
| Infrastructure | 57,273,000 |
| Capital Spares | 4,119,000 |
| Construction Indirects | 209,905,000 |
| Field Office Work | 58,391,000 |
| Project Office Work (Project Engineering And Management Work Consortium) | 170,819,000 |
| Freight | 50,138,000 |
| Project Engineering And Management Consortium Fee | 23,000,000 |
| Contingency | 103,862,000 |
| Owner's Cost | 141,718,000 |
| TOTAL | $1,675,034,000 |

[1] Reflecting approximately a 15% increase in the $1.45 billion estimate based upon the BFS.

### 20.9. Economic Analysis

The following tables set forth an economic analysis for the Goro Project, referred to as the Goro Project Base Case, reflecting a cash flow forecast and certain sensitivity analyses described below. This cash flow forecast has been based upon only the mineral reserve estimate for the Goro Deposits as of year-end 2002, representing approximately a 14 year life for the Goro Project. Goro Nickel and Inco believe that, given the significant estimated mineral resources already identified within the Goro Deposits and the reasonable expectation that significant additional mineral resources will be identified based on work completed to date, the life of the Goro Project will be significantly longer than 14 years.

Based upon a 14 year life using (1) mineral reserve estimates only, and (2) a discount rate of slightly below Inco's current weighted average cost of capital, the net present value of the cash flow forecast set forth below in Table 20.9(a) would be slightly positive. Goro Nickel and Inco do not believe, for the reasons noted above with respect to the Goro Deposits mineral resources, that the Goro Project Base Model represents a realistic life for the Goro Project. Accordingly, a significantly longer life for the Goro Project, as would be expected, would result in a significant positive net present value of the cash flow forecast using a discount rate marginally above Inco's current weighted average cost of capital.

The cash flow forecast in Table 20.9(a) reflects the following key assumptions:

- the capital cost estimate used represents the approximately 15% increase in the $1.45 billion estimate based upon the BFS;

- the nickel grades and other data used is the same as shown in Table 20.1.6 except that the annual cash flow forecasts are based upon a calendar year whereas Table 20.1.6 was compiled based upon collating data for project years and not calendar years;

- the nickel and other price assumptions reflect discounts from certain benchmarks given the projected product forms to be produced (for nickel oxide, a discount of $0.20 and for cobalt carbonate, a discount of $1.40 per pound) and a projected marketing fee to be paid by Goro Nickel of 2.75% on each pound of nickel and cobalt sold; and

- all prices and costs are unescalated.

Tables 20.9(b) - 20.9(e) reflects sensitivity analyses, using the same range of percentage changes, based upon the Goro Project Base Case for (1) the two metal products to be produced, nickel and cobalt; (2) operating costs; and (3) capital costs. These sensitivity analyses reflect that the Goro Project Base Case is particularly sensitive to changes in nickel prices and capital costs, and not very sensitive to the corresponding percentage changes in cobalt prices and operating costs.

# Table 20.9(a): Goro Project Base Case

| GORO PROJECT BASE CASE | | | TOTAL | 2001 | 2002 | 2003 | 2004 | 2005 | 2006 | 2007 | 2008 | 2009 | 2010 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Base Case key assumptions: Capex US$1.675 B; Aug 2005 start-up; LME US$3.20/lb Ni, MB US$7.00/lb Co (pre discount); mine to 2019; no terminal value** | | | | | | | | | | | | | |
| **Includes: 2.75% marketing fee.** | | | | | | | | | | | | | |
| **Production & Revenue** | | | **TOTAL** | | | | | | | | | | |
| **Autoclave Feed** | Average | | | 0 | 0 | 0 | 0 | 538,808 | 3,810,735 | 7,740,369 | 11,623,203 | 15,389,953 | 19,150,787 |
| Tonnage | 3,702,014 | dmt | 52,367,000 | 0 | 0 | 0 | 0 | 538,808 | 3,271,927 | 3,929,634 | 3,882,834 | 3,766,750 | 3,760,834 |
| Nickel Grade | 1.508% | % | 1.508% | 0.000% | 0.000% | 0.000% | 0.000% | 1.364% | 1.441% | 1.532% | 1.558% | 1.606% | 1.609% |
| Cobalt Grade | 0.120% | % | 0.120% | 0.000% | 0.000% | 0.000% | 0.000% | 0.119% | 0.122% | 0.129% | 0.129% | 0.128% | 0.146% |
| **Nickel** | | | | | | | | | | | | | |
| Final Product Sales | 113,585 | 1000 lb | 1,603,192 | 0 | 0 | 0 | 0 | 13,004 | 95,806 | 122,406 | 122,986 | 122,977 | 122,997 |
| Realized Price | $3.00 | $/lb | $3.00 | $3.000 | $3.000 | $3.000 | $3.000 | $3.000 | $3.000 | $3.000 | $3.000 | $3.000 | $3.000 |
| Revenue | $340,755 | US$1000 | $4,809,577 | $0 | $0 | $0 | $0 | $39,013 | $287,419 | $367,217 | $368,957 | $368,930 | $368,992 |
| **Cobalt** | | | | | | | | | | | | | |
| Final Product Sales | 8,871 | 1000 lb | 125,314 | 0 | 0 | 0 | 0 | 1,117 | 7,973 | 10,162 | 10,050 | 9,684 | 10,972 |
| Realized Price | $5.60 | $/lb | $5.60 | $6 | $6 | $6 | $6 | $6 | $6 | $6 | $6 | $6 | $6 |
| Revenue | $49,679 | US$1000 | $701,756 | $0 | $0 | $0 | $0 | $6,257 | $44,651 | $56,909 | $56,282 | $54,230 | $61,445 |
| **Base Case Cash Flow** | | | **TOTAL** | | | | | | | | | | |
| Total Revenue | $390,433 | | $5,511,334 | $0 | $0 | $0 | $0 | $45,270 | $332,070 | $424,126 | $425,239 | $423,160 | $430,437 |
| Total Operating Costs | ($169,080) | | ($2,422,079) | ($400) | ($400) | ($400) | ($400) | ($53,361) | ($164,110) | ($177,761) | ($177,875) | ($177,075) | ($177,206) |
| Working Capital Changes (Cash) | | | ($8,471) | $66 | $0 | $0 | ($29,060) | $1,704 | ($12,224) | ($6,063) | ($78) | $89 | ($572) |
| Net Cash Flow From Operations | | | $3,080,784 | ($334) | ($400) | ($400) | ($29,460) | ($6,386) | $155,737 | $240,302 | $247,285 | $246,173 | $252,659 |
| Initial Investment | | | ($1,675,034) | ($23,851) | ($386,222) | ($109,720) | ($723,252) | ($410,909) | ($21,080) | $0 | $0 | $0 | $0 |
| Timing adjustment for 2002 actuals | | | $0 | $0 | $19,464 | ($9,732) | ($9,732) | $0 | $0 | $0 | $0 | $0 | $0 |
| Sustaining Capital | | | ($204,046) | $0 | $0 | $0 | $0 | $0 | ($16,709) | ($10,198) | ($7,022) | ($8,163) | ($17,634) |
| Pons Act Net Contributions | | | $110,000 | $0 | $0 | $100,000 | $10,000 | $0 | $0 | $0 | $0 | $0 | $0 |
| Cash Flow Before Taxes | | | $1,311,704 | ($24,185) | ($367,158) | ($19,852) | ($752,444) | ($417,295) | $117,947 | $230,104 | $240,264 | $238,010 | $235,025 |
| Export Taxes | | | ($19,841) | $0 | $0 | $0 | $0 | ($163) | ($1,195) | ($1,527) | ($1,531) | ($1,523) | ($1,550) |
| Taxes on Opex | | | ($52,418) | $0 | $0 | $0 | $0 | ($1,808) | ($3,615) | ($3,615) | ($3,615) | ($3,615) | ($3,615) |
| Repatriation Tax | | | ($87,662) | $0 | $0 | $0 | $0 | $0 | $0 | ($6,426) | ($7,634) | ($7,547) | ($8,472) |
| Income Taxes | | | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 |
| Dividend W/H tax | | | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 |
| **Total Cash Flow** | | | $1,151,783 | ($24,185) | ($367,158) | ($19,852) | ($752,444) | ($419,266) | $113,137 | $218,536 | $227,484 | $225,324 | $221,388 |
| | | Cumulative Cash Flow | | ($24,185) | ($391,343) | ($411,195) | ($1,163,639) | ($1,582,905) | ($1,469,768) | ($1,251,233) | ($1,023,749) | ($798,425) | ($577,036) |
| | | Payback Period | 8.7 yrs | | | | | | | | | | |
| **Prices & Discounts** | | | | | | | | | | | | | |
| **Nickel** | | | | | | | | | | | | | |
| Reference price (average for year) | | | $3.20 | $3.20 | $3.20 | $3.20 | $3.20 | $3.20 | $3.20 | $3.20 | $3.20 | $3.20 | $3.20 |
| Discount to LME for nickel oxide | | Discount $/lb | **$0.20** | ($0.20) | ($0.20) | ($0.20) | ($0.20) | ($0.20) | ($0.20) | ($0.20) | ($0.20) | ($0.20) | ($0.20) |
| Realized Price | | | $3.00 | $3.00 | $3.00 | $3.00 | $3.00 | $3.00 | $3.00 | $3.00 | $3.00 | $3.00 | $3.00 |
| **Cobalt** | | | | | | | | | | | | | |
| Reference price (average for year) | | | $7.00 | $7.00 | $7.00 | $7.00 | $7.00 | $7.00 | $7.00 | $7.00 | $7.00 | $7.00 | $7.00 |
| Discount for cobalt carbonate | | Discount | **20.0%** | ($1.40) | ($1.40) | ($1.40) | ($1.40) | ($1.40) | ($1.40) | ($1.40) | ($1.40) | ($1.40) | ($1.40) |
| Realized Price | | | $5.60 | $5.60 | $5.60 | $5.60 | $5.60 | $5.60 | $5.60 | $5.60 | $5.60 | $5.60 | $5.60 |
| **Inflation Rates & Factors** | | | | | | | | | | | | | |
| **Inflation Rates** | Revenues | | | **0.00%** | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% |
| | Operating costs | | | **0.00%** | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% |
| | Capital costs | | | **0.00%** | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% |
| **Inflation Factors** | Revenues | | | 1.0000 | 1.0000 | 1.0000 | 1.0000 | 1.0000 | 1.0000 | 1.0000 | 1.0000 | 1.0000 | 1.0000 |
| | Operating costs | | | 1.0000 | 1.0000 | 1.0000 | 1.0000 | 1.0000 | 1.0000 | 1.0000 | 1.0000 | 1.0000 | 1.0000 |
| | Capital costs | | | 1.0000 | 1.0000 | 1.0000 | 1.0000 | 1.0000 | 1.0000 | 1.0000 | 1.0000 | 1.0000 | 1.0000 |

| GORO PROJECT BASE CASE | | | TOTAL | 2011 | 2012 | 2013 | 2014 | 2015 | 2016 | 2017 | 2018 | 2019 | 2020 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Base Case key assumptions: Capex US$1.675 B; Aug 2005 start-up; LME US** | | | | | | | | | | | | | |
| **Includes: 2.75% marketing fee.** | | | | | | | | | | | | | |
| **Production & Revenue** | | | **TOTAL** | | | | | | | | | | |
| **Autoclave Feed** | Average | | | 22,966,454 | 26,784,287 | 30,620,621 | 34,537,288 | 38,437,122 | 42,248,622 | 46,099,706 | 50,048,710 | 52,367,000 | |
| Tonnage | 3,702,014 | dmt | 52,367,000 | 3,815,667 | 3,817,834 | 3,836,334 | 3,916,667 | 3,899,834 | 3,811,500 | 3,851,084 | 3,949,005 | 2,318,290 | |
| Nickel Grade | 1.508% | % | 1.508% | 1.586% | 1.543% | 1.471% | 1.434% | 1.437% | 1.469% | 1.492% | 1.476% | 1.450% | |
| Cobalt Grade | 0.120% | % | 0.120% | 0.146% | 0.126% | 0.115% | 0.112% | 0.107% | 0.101% | 0.100% | 0.104% | 0.107% | |
| **Nickel** | | | | | | | | | | | | | |
| Final Product Sales | 113,585 | 1000 lb | 1,603,192 | 123,010 | 119,772 | 114,712 | 114,188 | 113,902 | 113,807 | 116,805 | 118,491 | 68,330 | |
| Realized Price | $3.00 | $/lb | $3.00 | $3.000 | $3.000 | $3.000 | $3.000 | $3.000 | $3.000 | $3.000 | $3.000 | $3.000 | |
| Revenue | $340,755 | US$1000 | $4,809,577 | $369,030 | $359,316 | $344,137 | $342,564 | $341,705 | $341,422 | $350,415 | $355,473 | $204,989 | |
| **Cobalt** | | | | | | | | | | | | | |
| Final Product Sales | 8,871 | 1000 lb | 125,314 | 11,136 | 9,658 | 8,823 | 8,748 | 8,378 | 7,719 | 7,742 | 8,183 | 4,966 | |
| Realized Price | $5.60 | $/lb | $5.60 | $6 | $6 | $6 | $6 | $6 | $6 | $6 | $6 | $6 | |
| Revenue | $49,679 | US$1000 | $701,756 | $62,363 | $54,084 | $49,409 | $48,991 | $46,920 | $43,228 | $43,355 | $45,826 | $27,808 | |
| **Base Case Cash Flow** | | | **TOTAL** | | | | | | | | | | |
| Total Revenue | $390,433 | | $5,511,334 | $431,393 | $413,400 | $393,546 | $391,555 | $388,624 | $384,650 | $393,770 | $401,298 | $232,796 | |
| Total Operating Costs | ($169,080) | | ($2,422,079) | ($177,982) | ($175,685) | ($172,324) | ($171,977) | ($171,836) | ($171,820) | ($173,768) | ($174,930) | ($102,768) | |
| Working Capital Changes (Cash) | | | ($8,471) | ($1) | $1,221 | $1,267 | $126 | $222 | $318 | ($543) | ($491) | $35,548 | |
| Net Cash Flow From Operations | | | $3,080,784 | $253,409 | $238,936 | $222,489 | $219,704 | $217,010 | $213,147 | $219,459 | $225,877 | $165,577 | |
| Initial Investment | | | ($1,675,034) | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | |
| Timing adjustment for 2002 actuals | | | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | |
| Sustaining Capital | | | ($204,046) | ($15,532) | ($15,532) | ($15,532) | ($15,532) | ($15,532) | ($16,665) | ($16,665) | ($16,665) | ($16,665) | |
| Pons Act Net Contributions | | | $110,000 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | |
| Cash Flow Before Taxes | | | $1,311,704 | $237,878 | $223,404 | $206,957 | $204,172 | $201,478 | $196,483 | $202,794 | $209,212 | $148,912 | |
| Export Taxes | | | ($19,841) | ($1,553) | ($1,488) | ($1,417) | ($1,410) | ($1,399) | ($1,385) | ($1,418) | ($1,445) | ($838) | |
| Taxes on Opex | | | ($52,418) | ($3,615) | ($3,615) | ($3,615) | ($3,615) | ($3,615) | ($3,615) | ($3,615) | ($3,615) | ($3,615) | |
| Repatriation Tax | | | ($87,662) | ($8,452) | ($7,631) | ($6,769) | ($6,647) | ($6,472) | ($6,234) | ($6,551) | ($6,827) | ($2,000) | |
| Income Taxes | | | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | |
| Dividend W/H tax | | | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | |
| **Total Cash Flow** | | | $1,151,783 | $224,258 | $210,669 | $195,156 | $192,500 | $189,992 | $185,249 | $191,211 | $197,325 | $142,459 | |
| | Cumulative Cash Flow | | | ($352,779) | ($142,109) | $53,047 | $245,547 | $435,539 | $620,787 | $811,999 | $1,009,324 | $1,151,783 | |
| | Payback Period | | 8.7 yrs | | | | | | | | | | |
| **Prices & Discounts** | | | | | | | | | | | | | |
| **Nickel** | | | | | | | | | | | | | |
| Reference price (average for year) | | | $3.20 | $3.20 | $3.20 | $3.20 | $3.20 | $3.20 | $3.20 | $3.20 | $3.20 | $3.20 | |
| Discount to LME for nickel oxide | | Discount $/lb | **$0.20** | ($0.20) | ($0.20) | ($0.20) | ($0.20) | ($0.20) | ($0.20) | ($0.20) | ($0.20) | ($0.20) | |
| Realized Price | | | $3.00 | $3.00 | $3.00 | $3.00 | $3.00 | $3.00 | $3.00 | $3.00 | $3.00 | $3.00 | |
| **Cobalt** | | | | | | | | | | | | | |
| Reference price (average for year) | | | $7.00 | $7.00 | $7.00 | $7.00 | $7.00 | $7.00 | $7.00 | $7.00 | $7.00 | $7.00 | |
| Discount for cobalt carbonate | | Discount | **20.0%** | ($1.40) | ($1.40) | ($1.40) | ($1.40) | ($1.40) | ($1.40) | ($1.40) | ($1.40) | ($1.40) | |
| Realized Price | | | $5.60 | $5.60 | $5.60 | $5.60 | $5.60 | $5.60 | $5.60 | $5.60 | $5.60 | $5.60 | |
| **Inflation Rates & Factors** | | | | | | | | | | | | | |
| **Inflation Rates** | Revenues | | | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% |
| | Operating costs | | | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% |
| | Capital costs | | | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% |
| **Inflation Factors** | Revenues | | | 1.0000 | 1.0000 | 1.0000 | 1.0000 | 1.0000 | 1.0000 | 1.0000 | 1.0000 | 1.0000 | 1.0000 |
| | Operating costs | | | 1.0000 | 1.0000 | 1.0000 | 1.0000 | 1.0000 | 1.0000 | 1.0000 | 1.0000 | 1.0000 | 1.0000 |
| | Capital costs | | | 1.0000 | 1.0000 | 1.0000 | 1.0000 | 1.0000 | 1.0000 | 1.0000 | 1.0000 | 1.0000 | 1.0000 |

**Table 20.9(b): Nickel Price Variance**

| Nickel Price Sensitivity | Nickel Price $/lb | Cash Flow Cumulative US$million |
|---|---|---|
| | $3.20 | $1,152 |
| 80.0% | $2.56 | $211 |
| 90.0% | $2.88 | $681 |
| 100.0% | $3.20 | $1,152 |
| 110.0% | $3.52 | $1,622 |
| 120.0% | $3.84 | $2,093 |

**Table 20.9(c): Cobalt Price Variance**

| Cobalt Price Sensitivity | Cobalt Price $/lb | Cash Flow Cumulative US$million |
|---|---|---|
| | $7.00 | $1,152 |
| 80.0% | $5.60 | $1,023 |
| 90.0% | $6.30 | $1,087 |
| 100.0% | $7.00 | $1,152 |
| 110.0% | $7.70 | $1,216 |
| 120.0% | $8.40 | $1,281 |

**Table 20.9(d): Operating Cost Variance**

| Operating Cost Sensitivity | Opex On-Site $/lb Ni | Cash Flow Cumulative US$million |
|---|---|---|
| | $1.39 | $1,152 |
| 80% | $1.11 | $1,573 |
| 90% | $1.25 | $1,362 |
| 100% | $1.39 | $1,152 |
| 110% | $1.53 | $941 |
| 120% | $1.67 | $731 |

**Table 20.9(e): Capital Cost Variance**

| Capital Cost Sensitivity | Capex US$million | Cash Flow Cumulative US$million |
|---|---|---|
| | $1,675 | $1,152 |
| 80% | $1,341 | $1,471 |
| 90% | $1,508 | $1,312 |
| 100% | $1,675 | $1,152 |
| 110% | $1,842 | $991 |
| 120% | $2,009 | $830 |

### 20.10. Payback

Based upon the Goro Project Base Case referred to in Item 20.9, the payback period would be approximately 8.7 years for the $1.675 billion capital cost estimates used. Since the economic analysis for the Goro Project reflected in the Base Case was on an unleveraged basis, no inputed or actual interest factor was utilized.

### 20.11. Mine Life

As noted in Item 20.9, Goro Nickel and Inco believe that for a number of reasons, including the extensive nature and characteristics of the Goro Deposits, and the expectation that significant additional mineral resources will be identified as and when Goro Nickel and Inco believe it to be appropriate to spend the additional funds on exploration, that the expected mine life for the Goro Project will be significantly longer than the 14 year life based upon only the mineral reserve estimate as of year end 2002.

## 21. INTERPRETATION AND CONCLUSIONS

The Goro Deposits represent deposits with very substantial resources. Additional exploration work will be required to significantly expand the mineral resource estimate, and ultimately, the mineral reserve estimate. The associated accuracy of the work done in support of the mineral reserve and mineral resource estimate meets or exceeds the accuracy expected at the bankable feasibility study level.

# 22. RECOMMENDATIONS

The comprehensive review of the Goro Project as discussed in Item 3 of this Report will determine if the mineral resource and mineral reserve estimates of the Goro Project are sufficient to justify completion of construction and commercial operation. Based on the assumptions set forth in the BFS, the authors do not recommend that any additional exploration work would be necessary to support the mineral resource and mineral reserve estimates stated in this Report.

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## 24. DATE AND SIGNATURE PAGE

This Report is effective as of December 31, 2002.

***"W.G. Bacon"***
W. Gordon Bacon

March 19, 2003
Date

***"R.A. Horn"***
R. A. Horn

March 20, 2003
Date

# APPENDICES

# APPENDIX A

## Size Fractions

**Table A-1: Statistics for Saprolite by Zones of Screen Half-core Data**

| | ZONE 1 Declustered SAP Means | ZONE 2 Declustered SAP Means | ZONE 3 Declustered SAP Means | | ZONE 1 Declustered SAP Means | ZONE 2 Declustered SAP Means | ZONE 3 Declustered SAP Means |
|---|---|---|---|---|---|---|---|
| **Run of Mine** | | | | **-50 mm** | | | |
| Ni | 1.59 | 1.20 | 1.47 | Ni | 2.21 | 1.91 | 2.10 |
| Co | 0.071 | 0.051 | 0.071 | Co | 0.104 | 0.079 | 0.117 |
| Fe | 13.02 | 11.94 | 13.73 | Fe | 17.54 | 17.71 | 19.75 |
| $SiO_2$ | 40.89 | 40.58 | 39.04 | $SiO_2$ | 39.92 | 39.14 | 36.63 |
| MgO | 25.74 | 28.66 | 26.33 | MgO | 17.59 | 18.43 | 17.58 |
| $Al_2O_3$ | 1.86 | 1.44 | 1.24 | $Al_2O_3$ | 2.50 | 2.29 | 1.87 |
| $Cr_2O_3$ | 1.03 | 0.88 | 1.01 | $Cr_2O_3$ | 1.40 | 1.32 | 1.49 |
| MnO | 0.52 | 0.40 | 0.50 | MnO | 0.69 | 0.57 | 0.74 |
| Recovery | 100 | 100 | 100 | Recovery | 52.2 | 42.9 | 50.8 |
| Ni content | 100 | 100 | 100 | Ni content | 72.6 | 68.3 | 72.6 |
| Co content | 100 | 100 | 100 | Co content | 76.5 | 66.5 | 83.7 |
| MgO content | 100 | 100 | 100 | MgO content | 35.7 | 27.6 | 33.9 |
| | | | | | | | |
| **-150 mm** | | | | **-6.35 mm** | | | |
| Ni | 2.00 | 1.58 | 1.83 | Ni | 2.44 | 2.12 | 2.28 |
| Co | 0.095 | 0.067 | 0.099 | Co | 0.117 | 0.089 | 0.123 |
| Fe | 15.88 | 15.09 | 17.40 | Fe | 20.09 | 20.52 | 22.61 |
| $SiO_2$ | 40.40 | 39.88 | 37.64 | $SiO_2$ | 37.94 | 36.42 | 33.88 |
| MgO | 20.40 | 22.86 | 20.96 | MgO | 15.10 | 26.20 | 15.44 |
| $Al_2O_3$ | 2.28 | 1.92 | 1.62 | $Al_2O_3$ | 2.58 | 2.50 | 2.07 |
| $Cr_2O_3$ | 1.28 | 1.12 | 1.30 | $Cr_2O_3$ | 1.65 | 1.60 | 1.75 |
| MnO | 0.66 | 0.51 | 0.65 | MnO | 0.80 | 0.67 | 0.79 |
| Recovery | 64.6 | 61.5 | 67.4 | Recovery | 33.4 | 26.7 | 34.6 |
| Ni content | 81.3 | 81.0 | 83.9 | Ni content | 51.2 | 47.2 | 53.7 |
| Co content | 86.4 | 80.8 | 94.0 | Co content | 55.0 | 46.6 | 59.9 |
| MgO content | 51.2 | 49.1 | 53.7 | MgO content | 19.6 | 24.4 | 20.3 |

## Table A-2: Statistics for Transition by Zones of Screen Half-core Data

| | ZONE 1 Declustered TRN Means | ZONE 2 Declustered TRN Means | ZONE 3 Declustered TRN Means | | ZONE 1 Declustered TRN Means | ZONE 2 Declustered TRN Means | ZONE 3 Declustered TRN Means |
|---|---|---|---|---|---|---|---|
| **Run of Mine** | | | | **-50 mm** | | | |
| Ni | 1.49 | 1.53 | 1.73 | Ni | 1.54 | 1.56 | 1.73 |
| Co | 0.161 | 0.128 | 0.194 | Co | 0.168 | 0.132 | 0.193 |
| Fe | 39.37 | 40.36 | 41.72 | Fe | 41.08 | 41.37 | 41.72 |
| $SiO_2$ | 15.54 | 16.37 | 15.32 | $SiO_2$ | 14.20 | 15.62 | 15.33 |
| MgO | 4.27 | 3.79 | 2.59 | MgO | 2.39 | 2.69 | 2.59 |
| $Al_2O_3$ | 5.96 | 4.91 | 3.98 | $Al_2O_3$ | 6.24 | 5.05 | 3.98 |
| $Cr_2O_3$ | 3.09 | 3.08 | 3.18 | $Cr_2O_3$ | 3.22 | 3.16 | 3.18 |
| MnO | 1.16 | 0.98 | 1.39 | MnO | 1.20 | 1.00 | 1.39 |
| Recovery | 100 | 100 | 100 | Recovery | 95.1 | 97.0 | 99.9 |
| Ni content | 100 | 100 | 100 | Ni content | 98.7 | 99.3 | 99.9 |
| Co content | 100 | 100 | 100 | Co content | 99.3 | 99.7 | 99.9 |
| MgO content | 100 | 100 | 100 | MgO content | 53.2 | 68.9 | 99.9 |
| | | | | | | | |
| **-150 mm** | | | | **-6.35 mm** | | | |
| Ni | 1.54 | 1.55 | 1.73 | Ni | 1.54 | 1.57 | 1.74 |
| Co | 0.168 | 0.130 | 0.194 | Co | 0.169 | 0.132 | 0.194 |
| Fe | 40.99 | 41.04 | 41.72 | Fe | 41.18 | 41.50 | 41.85 |
| $SiO_2$ | 14.28 | 15.87 | 15.32 | $SiO_2$ | 14.14 | 15.36 | 15.12 |
| MgO | 2.49 | 3.07 | 2.59 | MgO | 2.32 | 2.68 | 2.59 |
| $Al_2O_3$ | 6.22 | 5.00 | 3.98 | $Al_2O_3$ | 6.24 | 5.08 | 4.00 |
| $Cr_2O_3$ | 3.22 | 3.13 | 3.18 | $Cr_2O_3$ | 3.24 | 3.17 | 3.20 |
| MnO | 1.20 | 0.99 | 1.39 | MnO | 1.20 | 1.00 | 1.39 |
| Recovery | 95.4 | 98.0 | 100.0 | Recovery | 94.4 | 96.4 | 99.4 |
| Ni content | 98.8 | 99.5 | 100.0 | Ni content | 97.9 | 99.0 | 99.8 |
| Co content | 99.4 | 99.8 | 100.0 | Co content | 98.8 | 99.2 | 99.7 |
| MgO content | 55.7 | 79.5 | 100.0 | MgO content | 51.2 | 68.3 | 99.4 |

**Table A-3: Global Statistics of Zones 1, 2 and 3 for the Transition Layer for Zone 4 Resources Estimation**

| | Run of Mine | -150 mm | -50 mm | -6.35 mm |
|---|---|---|---|---|
| Ni | 1.55 | 1.58 | 1.59 | 1.59 |
| Co | 0.153 | 0.157 | 0.158 | 0.158 |
| Fe | 40.3 | 41.2 | 41.3 | 41.5 |
| $SiO_2$ | 15.9 | 15.2 | 15.0 | 14.9 |
| MgO | 3.73 | 2.76 | 2.56 | 2.53 |
| $Al_2O_3$ | 5.12 | 5.24 | 5.26 | 5.28 |
| $Cr_2O_3$ | 3.10 | 3.17 | 3.19 | 3.20 |
| MnO | 1.12 | 1.15 | 1.15 | 1.16 |
| Recovery | 100.0 | 97.4 | 96.9 | 96.2 |
| Ni% of total Content | 100.0 | 99.4 | 99.2 | 98.8 |
| Co% of total Content | 100.0 | 99.7 | 99.6 | 99.2 |
| MgO% of total Content | 100.0 | 72.2 | 66.5 | 65.3 |

**Table A-4: Global Statistics of Zones 1, 2 and 3 for the Saprolite Layer for Zone 4 Resources Estimation**

| | Run of Mine | -150 mm | -50 mm | -6.35 mm |
|---|---|---|---|---|
| Ni | 1.40 | 1.78 | 2.06 | 2.27 |
| Co | 0.063 | 0.086 | 0.100 | 0.110 |
| Fe | 12.9 | 16.2 | 18.5 | 21.2 |
| $SiO_2$ | 40.1 | 39.2 | 38.4 | 35.8 |
| MgO | 27.1 | 21.5 | 17.9 | 15.6 |
| $Al_2O_3$ | 1.47 | 1.90 | 2.18 | 2.35 |
| $Cr_2O_3$ | 0.97 | 1.23 | 1.41 | 1.68 |
| MnO | 0.47 | 0.60 | 0.67 | 0.75 |
| Recovery | 100.0 | 64.4 | 48.1 | 31.2 |
| Ni% of total Content | 100.0 | 81.9 | 70.8 | 50.6 |
| Co% of total Content | 100.0 | 87.9 | 76.3 | 54.5 |
| MgO% of total Content | 100.0 | 51.2 | 31.7 | 18.0 |

**Table A-5: Tonnage and Moisture Factors Used in the Resources Models**

| Layer | Layer Code | Screen fraction | No. Samples | Density | Moisture (%) | Dry Tonnage Factor | Wet Tonnage Factor |
|---|---|---|---|---|---|---|---|
| Surface fill | FILL | Unscreened | N.A. | 2.43 | 16 | 1.75 | 2.08 |
| Iron Cap | IC | Unscreened | N.A. | 2.63 | 12 | 2 | 2.27 |
| Iron Shot | ISH | Unscreened | N.A. | 2.14 | 20 | 1.5 | 1.88 |
| Red Laterite | LATR | Unscreened | N.A. | 1.61 | 38 | 1 | 1.61 |
| Yellow Laterite | LATJ | Unscreened | | 1.51 | 45 | 0.9 | 1.64 |
| Transition | TRN | ROM | | 1.61 | 49 | 0.9 | 1.76 |
| Saprolite[1] | SAP | ROM | 4612 | | 21.3 | 1.37 | 1.74 |
| | SAP | +150mm | 444 | | 4 | 2.28 | 2.36 |
| | SAP | +50 to _150 | 141 | | 8 | 1.65 | 1.75 |
| | SAP | +6.35 to 50 | 18 | | 13 | 1.26 | 1.65 |
| | SAP | - 6.35mm | 818 | | 42 | 0.94 | 1.63 |
| Bedrock | BRK | unscreened | | 2.65 | 4 | 2.4 | 2.5 |

Note: The saprolite does not include core recovery.

# APPENDIX B

## Databases

## 1. DRILL TABLE

The following Drill table contains general information about the drill hole.

**Table B-1: Drill table**

| | |
|---|---|
| hole_id | Borehole identification number |
| drill_rig | Type of drill used |
| Driller | The name of the driller |
| hole_diameter | Core diameter |
| hole_type | 4 letter codes used to identify type of drilling done |
| Logger | Name of the person who logged the core |
| Relogger | The name of the person that has relogged the core |
| Sampler | The person who took the core samples out of the core box for analysis |
| end_date | The date that the borehole drilling was completed |
| log_date | The date that log was completed |
| relog_date | The date that the relog (if there is) was done. |
| start_date | The date that the borehole drilling started |
| nbcorebox | Total number of core boxes used for storing the borehole |
| storingloc | The container number in which the borehole core is kept |
| gear_left | Is there was any drilling equipment lost or left in the hole |
| piezo_qty | Number of piezometer in the hole |
| Status | The status of the information in the database (complete or in progress) |

During the course of the audits, the documentation of the location of the core and the pulp storage is reviewed and physically check.

## 2. COLLAR TABLE

The following Collar table contains information about the hole location, depth and survey status.

**Table B-2: Collar table**

| | |
|---|---|
| hole_id | Borehole identification number |
| X | UTM easting collar coordinate in IGN72 system |
| Y | UTM Northing collar coordinate in IGN72 system |
| Z | collar elevation coordinate (sea elevation is zero) |
| hole_path | Describes the hole curvature valid options are linear or curved |
| max_depth | The length of the hole at its completion |
| Location | Concession in which the hole is located |
| Section | Nearest Easting |
| Purpose | Drill hole justifications (geologic, geostatistic, geotechnic etc.). |
| hole_type | 4 letter codes used to identify type of drilling done |
| Top_rock | Depth to bedrock |
| collar_position | Survey status (final or preliminary) |
| downhole_survey | Not used |
| Precollar_depth | Not used |
| Modifications | Comments if further work was done on the hole |

One important control in this table is the agreement between the surveyor's report and the electronic file. To date, Goro Nickel has used the services of Société Photogrammétrique Du Pacifique (SPP) for spotting and final survey of the holes. The survey location of the holes is verified against stamped files from the surveyor's office.

## 3. MINOR TABLE

The following Minor table contains relevant information about the geological and physical aspects of the core. It contains the following fields:

**Table B-3: Minor table**

| | |
|---|---|
| hole_id | Borehole identification number |
| depth_from | Beginning of geological interval |
| depth_to | End of geological interval |
| Layer* | 4 letter code used to describe the main geological layer horizon |
| code1* | 4 letter code used to describe the main geological constituent that is in excess of 60 % |
| code2* | 4 letter code used to describe the minor geological constituent that is less than 40 % |
| code1gt60 | Proportion of code 1 express in %. Cannot be less than 60% |
| code2lt40 | Proportion of code 2 express in %. Cannot be more than 40% |
| Couleur* | Core colour |
| Texture* | Core texture |
| Aspect* | Core aspect |
| Structure* | Structure seen in core |
| Alteration* | Core alteration |
| Mineralisation | Mineral identified visually in the core |
| Desc | Geological description |
| thin_section | If a thin section was requested |
| whole_rock | If a whole rock analysis was requested |
| samp_id | Not used |

The fields marked with an asterisk are linked to a dictionary that ensures that there are no typographic errors and that only codes present in the dictionary can be entered.

Of importance for resource modeling in this table are the Depth-from and Depth to fields as well as the layer codes because they are used for composite selection and wire framing of the mineral resources model layers. During the wire framing a part of the resource estimation process for the model all codes were reviewed for consistency.

## 4. PROP_PHYS TABLE

The following Prop_Phys table, short for Physical Properties of the core, contains all information about core recovery, wet weight, dry weight, sampling interval and screen fractions analyzed in the core. The fields are:

**Table B-4: Physical Property table**

| | |
|---|---|
| Hole_id | Borehole identification number |
| depth_from | Beginning of sampling interval |
| depth_to | End of sampling interval |
| Thick | The difference between Detph_to and Depth_From |
| core_recovery | Recovered core length expressed in length |
| samp_id | The numerical root of the sample ID for the interval |
| Pnh | Wet weight of the core interval |
| Pns | Dry weight of the core interval. This may be a calculated value if more than one fraction is present in the interval |
| Moisture | Moisture is expressed as (1-(PNS / PNH))* 100 |
| Desc | Not used (description field for comments) |
| ore_waste | Not used (ore waste flag) |
| Etf | Not used (effective tonnage factor) |
| Wtf | Not used (Wet tonnage factor) |
| Modifications | Not used (explain type of modification) |
| Headgrade | Flag to create an interval in the Smpheadgrade table |
| p150 | Flag to create an interval in the Smpp150 table |
| A | Flag to create an interval in the Smpp2inch table |
| B | Flag to create an interval in the Smppquarterm2inch table |
| C | Flag to create an interval in the Smpmquarter table |
| Moist calc | Not used (calculated moisture) |

All the fields in this table with the exception of those that are “not used” and the moisture field are important because discrepancies will result in an incorrect sampling interval, sample ID or miscomputation of core recovery or screen recoveries.

## 5. SMPHEADGRADE TABLE

This table contains chemical analyses of the –150mm size fraction (reconstituted analysis on dry weight basis from individual size fraction for transition and saprolite layers). When there is no –150mm fraction then the +150mm is entered. The analytical results for laterite, iron cap and iron shot are also stored in this table. The fields in this table are:

**Table B-5: Sample Head Grade Table**

| | |
|---|---|
| Hole_id | Borehole identification number |
| depth_from | Beginning of sampling interval |
| depth_to | End of sampling interval |
| Core_recovery | Recovered core length expressed in length |
| Thick | The difference between Detph_to and Depth_From |
| samp_id | The root of the sample number for the interval |
| Pnh | Wet weight of the core interval |
| Pns | Dry weight of the core interval. This may be a calculated value if more than one fraction is present in the interval |
| moisture | Moisture is expressed as (1-(PNS / PNH))* 100 |
| Nickel | Nickel analysis in % Ni |
| Cobalt | Cobalt analysis in % Co |
| Iron | Iron analysis in % Fe |
| Silica | Silicon analysis in % $SiO_2$ |
| magnesia | Magnesium analysis in % MgO |
| Alumina | Aluminum analysis in % $Al_2O_3$ |
| Chrome | Chromium analysis in % $Cr_2O_3$ |
| manganese | Manganese analysis in % MnO |
| Copper | Copper analysis in PPM of Cu |
| Zinc | Zinc analysis in PPM of Zn |
| $K_2O$ | Potassium analysis in % $K_2O$ |
| $TiO_2$ | Titanium analysis in % $TiO_2$ |
| CaO | Calcium analysis in % CaO |
| Loi | Loss of ignition in % |
| Desc | Not used (description field for comments) |
| Ore_waste | Not used (ore waste flag) |
| Etf | Not used (effective tonnage factor) |
| Wtf | Not used (Wet tonnage factor) |
| assay_date | Received Date for electronic file |
| ref_code | Lab sample number |
| modifications | Explains modifications |

All records of the holes randomly selected are checked against stamped paper copies issued by the laboratories.

## 6. SMPP150 TABLE

This table contains records of the +150mm size fraction exclusive of laterite, iron cap and iron shot. The fields present in this table are:

**Table B-6: Sample +150 mm Size Fraction Table**

| | |
|---|---|
| hole_id | Borehole identification number |
| samp_id | The root of the sample number for the interval with the prefix "P" added |
| depth_from | Beginning of sampling interval |
| depth_to | End of sampling interval |
| Nickel | Nickel analysis in % Ni |
| Cobalt | Cobalt analysis in % Co |
| Iron | Iron analysis in % Fe |
| Silica | Silicon analysis in % $SiO_2$ |
| magnesia | Magnesium analysis in % MgO |
| alumina | Aluminum analysis in % $Al_2O_3$ |
| Chrome | Chromium analysis in % $Cr_2O_3$ |
| manganese | Manganese analysis in % MnO |
| Copper | Copper analysis in PPM of Cu |
| Zinc | Zinc analysis in PPM of Zn |
| $K_2O$ | Potassium analysis in % $K_2O$ |
| $TiO_2$ | Titanium analysis in % $TiO_2$ |
| CaO | Calcium analysis in % CaO |
| Loi | Loss of ignition in % |
| Pns | Dry weight of the +150mm size fraction within the core interval. |
| ore_waste | Not used (ore waste flag) |
| assay_date | Received Date for electronic file |
| ref_code | Lab sample number |
| modifications | Explains modifications |

The Smpp150 table contains all information on the +150mm size fraction.

## 7. SMPP2INCH TABLE

This table contains all information about the +50.8mm to –150mm size fraction exclusive of laterite, iron cap and iron shot. The fields are:

**Table B-7: Sample +50.8 mm to –150 mm Size Fraction Table**

| | |
|---|---|
| hole_id | Borehole identification number |
| samp_id | The root of the sample number for the interval with the prefix "A" added |
| depth_from | Beginning of sampling interval |
| depth_to | End of sampling interval |
| Nickel | Nickel analysis in % Ni |
| Cobalt | Cobalt analysis in % Co |
| Iron | Iron analysis in % Fe |
| Silica | Silicon analysis in % $SiO_2$ |
| magnesia | Magnesium analysis in % MgO |
| Alumina | Aluminum analysis in % $Al_2O_3$ |
| Chrome | Chromium analysis in % $Cr_2O_3$ |
| manganese | Manganese analysis in % MnO |
| Copper | Copper analysis in PPM of Cu |
| Zinc | Zinc analysis in PPM of Zn |
| $K_2O$ | Potassium analysis in % $K_2O$ |
| $TiO_2$ | Titanium analysis in % $TiO_2$ |
| CaO | Calcium analysis in % CaO |
| Loi | Loss of ignition in % |
| Pns | Dry weight of the +50.8mm to –150mm size fraction within the core interval. |
| ore_waste | Not used (ore waste flag) |
| assay_date | Received Date for electronic file |
| ref_code | Lab sample number |
| modifications | Explains modifications |

## 8. SMPPQUARTERM2INCH TABLE

The Smmppquarterm2inch table contains all information on the +6.35mm to –50mm size fraction exclusive of laterite, iron cap and iron shot. The fields present in this table are:

**Table B-8: Sample +6.35 mm to –50.8 mm Size Fraction Table**

| | |
|---|---|
| hole_id | Borehole identification number |
| samp_id | The root of the sample number for the interval with the prefix "B" added |
| depth_from | Beginning of sampling interval |
| depth_to | End of sampling interval |
| Nickel | Nickel analysis in % Ni |
| Cobalt | Cobalt analysis in % Co |
| Iron | Iron analysis in % Fe |
| Silica | Silicon analysis in % $SiO_2$ |
| magnesia | Magnesium analysis in % MgO |
| Alumina | Aluminum analysis in % $Al_2O_3$ |
| Chrome | Chromium analysis in % $Cr_2O_3$ |
| manganese | Manganese analysis in % MnO |
| Copper | Copper analysis in PPM of Cu |
| Zinc | Zinc analysis in PPM of Zn |
| $K_2O$ | Potassium analysis in % $K_2O$ |
| $TiO_2$ | Titanium analysis in % $TiO_2$ |
| CaO | Calcium analysis in % CaO |
| Loi | Loss of ignition in % |
| Pns | Dry weight of the +6.35mm to –50.8mm size fraction within the core interval. |
| ore_waste | Not used (ore waste flag) |
| assay_date | Received Date for electronic file |
| ref_code | Lab sample number |
| modifications | Explains modifications |

## 9. SMPMQUARTER TABLE

Table Smpmquarter contains all relevant information on the –6.35mm size fraction exclusive of laterite, iron cap and iron shot.

**Table B-9: Sample -6.35 mm Size Fraction Table**

| | |
|---|---|
| hole_id | Borehole identification number |
| samp_id | The root of the sample number for the interval with the prefix "C" added |
| depth_from | Beginning of sampling interval |
| depth_to | End of sampling interval |
| Nickel | Nickel analysis in % Ni |
| Cobalt | Cobalt analysis in % Co |
| Iron | Iron analysis in % Fe |
| Silica | Silicon analysis in % $SiO_2$ |
| magnesia | Magnesium analysis in % MgO |
| Alumina | Aluminum analysis in % $Al_2O_3$ |
| Chrome | Chromium analysis in % $Cr_2O_3$ |
| manganese | Manganese analysis in % MnO |
| Copper | Copper analysis in PPM of Cu |
| Zinc | Zinc analysis in PPM of Zn |
| $K_2O$ | Potassium analysis in % $K_2O$ |
| $TiO_2$ | Titanium analysis in % $TiO_2$ |
| CaO | Calcium analysis in % CaO |
| Loi | Loss of ignition in % |
| Pns | Dry weight of the -6.35mm size fraction within the core interval. |
| Ore_waste | Not used (ore waste flag) |
| assay_date | Received Date for electronic file |
| ref_code | Lab sample number |
| modifications | Explains modifications |

In addition to the tables identified above, there are other tables in the databases including: Survey, Geotech, Photos, Quality Control, Major, Sys_links, Sys_validation and Translation.

The Survey table has not been verified because the information in this table is filled in automatically and no survey is done in the hole. All holes are, by default, vertical with an azimuth of (360°).

The Geotech table contains information on the iron cap and iron shot size fraction. This information is qualitative and serves only as a guide for borrowing material characteristics.

The Photos table was not validated because at the time of the audit the table was not populated. This table will contain the hyperlinks to the photograph taken of each core box. Once the table is filled it will be used as a quick reference tool for visualizing the core.

The Quality Control table contains all duplicated assays obtained to ensure there are no biases in the sampling and analysis of the core.

The Sys_links and Sys_validation tables are tables created by the core logging software and used to link and validate tables.

The Translation table is the dictionary table. It contains a list of all the authorized short codes and their description that can be used throughout the database. Although this table was not revised as such it should be noted that an effort should be made to restrict the codes to certain tables and fields to ensure data integrity.

The authors have relied on personnel under their responsibility to carry out audits on the Goro Deposits, sampling, logging, sample preparation, laboratories and database information. Personnel involved in the review, including the authors, have visited the site.

There are no known limitations to the audits performed in connection with the Goro Project.

# APPENDIX C

## Variography

**Table C-1: Chemistry Variogram Models for the Yellow Laterite Layer**

| Range (m) | Ni | Co | Fe | $SiO_2$ | MgO | $Al_2O_3$ | $Cr_2O_3$ | MnO |
|---|---|---|---|---|---|---|---|---|
| Nugget | 0.02 | 0.003 | 4.0 | 3.0 | 0.05 | 1.0 | 0.20 | 0.25 |
| 12x12x25 | 0.04 | | | | | | | |
| 25x12x15 | | 0.007 | | | | | | |
| 12x12x20 | | | 6.0 | | | | | |
| 20x20x15 | | | | 4.0 | | | | |
| 20x20x3 | | | | | 0.20 | | | |
| 20x20x10 | | | | | | | | 0.60 |
| 15x15x6 | | | | | | | 0.50 | |
| 12x12x6 | | | | | | 5.0 | | |
| 150x150x25 | 0.03 | | | | | | | |
| 150x100x15 | | 0.009 | | | | | | |
| 200x200x30 | | | 4.0 | | | | 0.40 | |
| 250x250x25 | | | | 4.0 | | | | |
| 250x250x35 | | | | | 0.20 | | | |
| 150x150x50 | | | | | | | | 0.45 |
| 80x80x60 | | | | | | 3.0 | | |
| 10000x10000x25 | 0.12 | | | | | | | |
| 10000x10000x30 | | 0.03 | 20.0 | | | | | |
| 10000x10000x25 | | | | 20.0 | | | | |
| 10000x10000x35 | | | | | 1.20 | | | |
| 10000x10000x50 | | | | | | | | 3.30 |
| 10000x10000x35 | | | | | | | 0.20 | |
| CSRV/TV % | 28.5 | 20.4 | 29.4 | 22.6 | 15.15 | 66.7 | 53.8 | 18.5 |

Rotation: $60^o$ clockwise around Z axis

**Table C-2: Chemistry Variogram Models for the Transition Layer**

| Range (m) | Ni | Co | Fe | $SiO_2$ | MgO | $Al_2O_3$ | $Cr_2O_3$ | MnO |
|---|---|---|---|---|---|---|---|---|
| Nugget | 0.05 | 0.005 | 20.0 | 20.0 | 2.0 | 2.0 | 0.30 | 1.00 |
| 10x10x12 | 0.18 | | | | | | | |
| 20x20x10 | | 0.015 | | | | | | |
| 15x15x10 | | | 50.0 | 140.0 | 10.0 | | | 0.70 |
| 30x15x10 | | | | | | | 1.20 | |
| 10x30x10 | | | | | | 15.0 | | |
| 110x60x20 | 0.06 | | | | | | | |
| 250x250x30 | | 0.010 | | | | | | |
| 150x150x10 | | | 20.0 | | 8.0 | | | |
| 150x150x15 | | | | 80.0 | | | | |
| 150x150x20 | | | | | | | | 0.30 |
| 120x200x10 | | | | | | | 0.40 | |
| 180x80x10 | | | | | | 5.0 | | |
| CSRV/TV % | 79.3 | 66.7 | 77.8 | 66.7 | 60.0 | 77.3 | 78.9 | 85.0 |

Rotation: $60^o$ clockwise around Z axis

**Table C-3: Chemistry Variogram Models for the Saprolite Layer**

| **Range (m)** | **Ni** | **Co** | **Fe** | **$SiO_2$** | **MgO** | **$Al_2O_3$** | **$Cr_2O_3$** | **MnO** |
|---|---|---|---|---|---|---|---|---|
| Nugget | 0.20 | 0.002 | 20.0 | 10.0 | 10.0 | 1.0 | 0.10 | 0.10 |
| 15x15x15 | 0.40 | | | | | | | |
| 15x15x10 | | 0.005 | | | | | | |
| 10x10x5 | | | 15.0 | | | | | |
| 15x15x12 | | | | | | | | 0.25 |
| 15x15x8 | | | | | 55.0 | | | |
| 30x30x5 | | | | 25.0 | | | 0.40 | |
| 20x20x12 | | | | | | 5.0 | | |
| 150x150x80 | 0.20 | | | | | | | |
| 150x150x10 | | 0.002 | | | | | | |
| 200x200x20 | | | 40.0 | | | | | |
| 200x200x10 | | | | 40.0 | | | | |
| 150x150x25 | | | | | 15.0 | | | |
| 150x150x30 | | | | | | | 0.10 | |
| 150x150x20 | | | | | | | | 0.10 |
| 180x180x30 | | | | | | 2.5 | | |
| CSRV/TV % | 75.0 | 77.8 | 46.7 | 46.7 | 81.3 | 70.6 | 83.3 | 0.4 |

Rotation: $60^{o}$ clockwise around Z axis