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REPORT

on

EVALUATION OF THE BEACONSFIELD NICKEL PROSPECT

to

KING ISLAND SCHEELITE (1947) LIMITED

by

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from

INTERNATIONAL TECHNICAL SERVICES LIMITED
in conjunction with
THE AUSTRALIAN MINERAL DEVELOPMENT LABORATORIES

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SUMMARY

Background

King Island Scheelite (1947) Limited has been investigating a lateritic nickel deposit located near Beaconsfield in northern Tasmania. Consequent on their finding that the area had some potential for mining nickel, they have requested International Technical Services Limited to conduct a preliminary paper study, without experimentation, to determine whether a mining and processing operation would be a feasible proposition and, if so, to also recommend what additional research might be required.

Exploratory drilling and other geological investigations carried out by King Island Scheelite (1947) Limited have indicated a possible ore reserve of 7 to 11 million tons, assaying 1.03% nickel. A lateritic deposit of this size and grade would normally be considered uneconomical and not worth further investigation. However, the location and nature of the Beaconsfield deposit affords several advantages which tend to off-set the intrinsic economic value of these small reserves. These advantages, in the main, are:

1. The deposit area is 7 miles from an established sea-port.
2. This area occupies a topographic low between two ranges of hills, and is formed of gently rolling terrain that could be inexpensively cleared and prepared. The area is easily accessible by means of established, all-weather roads.
3. The ore-bearing horizon is close to the surface and readily adapted to open-cut mining operations.
4. Electric-power can be transmitted to the site from an existing installation, at cheap industrial rates.
5. Water is readily available, either from a river that flows through the site, or from the reticulation system serving Beaconsfield.
6. The deposits' close proximity to Beaconsfield and Launceston ensures the availability of unskilled labour and minimizes the need for company housing.
7. Bell Bay - 19 road miles from the deposit - is a major storage terminal for petroleum products.

Objectives

The objectives of the investigation are to:

1. determine whether this deposit is capable of supporting a profitable mining and process operation;

2. recommend what research, exploration and development is required if the results are favourable.

Summary of Work Done

A comprehensive literature survey on the Beaconsfield area has been carried out.

A complete mineralogical study on samples collected from a series of exploratory diamond drill holes was completed and the nickel-bearing phases identified. In order to obtain information on the chemical variability of the deposit all of the available assay data was collated and interpreted. These data were used to restrict the number of possible treatment techniques that could be applied to the Beaconsfield ore.

Published information on various nickel processing methods was reviewed and data collected to determine the technical and economic implications of processing the Beaconsfield nickel ore to a saleable product. Only two processes, Nicaro and Pyrosulphidizing, were technically and economically promising. Discounted cash flow analyses on each of these two processes were conducted to determine their economic worth in the present context.

Conclusions

1. Based on preliminary estimates the Beaconsfield deposit appears to have a definite economic potential and warrants further investigation.

2. The minimum level of reserves which could be considered economical is approximately 9.5 million tons of 1.03% nickel ore.

3. The nickel is disseminated through a variety of mineral phases and therefore physical beneficiation is not practical. This eliminates the possibility of developing the deposit with a minimal outlay of capital.

4. Investigations show that the Nicaro process is the most suitable for treatment of the ore. This process is a recognised method for the treatment of nickel ore of the type found at Beaconsfield.

   The capitalization of the venture, including a Nicrao processing plant, mining equipment, housing, loading facilities and working capital, is estimated at 19.3 million dollars. The discounted cash-flow rate of return at this level of investment is 20% assuming a deposit of 11 million tons, and 15% assuming a deposit of 9.5 million tons.

5. The Pyro sulphidizing process appears to be economically feasible but this processing concept is relatively new and untried on a plant scale.

6. Technically the Pug Roast Acid Leach process is capable of treating the Beaconsfield ore, but the requirement of large quantities of sulphuric acid to treat a low-grade nickel ore renders the process uneconomical.

7. Direct smelting by the Ugine process is both technically and economically unsatisfactory.

8. The Beaconsfield nickel prospect appears to be formed of two distinct mineralogical provinces. One centered on Scotts Hill, in which the minerals talc and magnesite are conspicuous, and a second extending over the remainder of the deposit in which serpentine and chlorite are the principal primary minerals.

9. There is a significant lateral and vertical chemical variation in the depth and thickness of the ore-bearing horizon. These factors will necessitate the blending of the ore, in order to obtain a uniform feed for the treatment process, and require exploratory drilling ahead of mining to control the blending.

Recommendations

1. Retain Exploration Licence 7/67 pending further investigations.

2. Consider further the development of the deposit based on a Nicaro ore-processing plant.

3. Institute a research programme, as outlined in the final section of this report, to determine more accurately:
   a. value of finished product,
   b. operating and capital costs,
   c. amount and grade of recoverable ore.

1. INTRODUCTION

A study to evaluate the potential for mining nickel at Beaconsfield in northern Tasmania was initiated with International Technical Services Limited by King Island Scheelite (1947) Limited in March 1968. It was known then that the nickel reserves in this area were low, but many advantageous factors obtaining at the site, primarily due to its location from the mining and processing point of view, indicated that the whole proposition need not be dispensed with as an unremunerative venture. This has now been borne out to an appreciable degree as a result of preliminary findings.

The considerations that have gone into arriving at such a conclusion are based on analyses which have covered the following investigations:

a. Review of previous investigations of the Beaconsfield deposit.
b. Nickel mineralogy.
c. Limitations of physical beneficiation.
d. Aspects of the chemistry of the Beaconsfield deposit.
e. Technical and economic appraisal of alternative processing methods.
f. Required developmental work.

Details of these investigations are outlined in the text of this report.

2. REVIEW OF PREVIOUS INVESTIGATIONS OF THE BEACONSFIELD NICKEL DEPOSIT

In the north of Tasmania, immediately west of the township of Beaconsfield, lies an intrusion of ultrabasic rocks. The Beaconsfield Nickel Deposit has formed by chemical weathering, with consequent supergene enrichment, of this ultrabasic intrusion.

The township of Beaconsfield lies 27 road miles north-northwest of Launceston and 3 miles southwest of Beauty Point on the Tamar River. The ultrabasic intrusion lies a further 4 road miles (2 statute miles) west of Beaconsfield.

The intrusion outcrops over an area approximately 4 miles long and up to 1.5 miles wide. In the past the area has been extensively prospected for asbestos, chromiferous iron-ore, nickel and chromite. Small quantities of asbestos and iron-ore have been won. A brief history of exploration in the area is contained in Anthony (1967).

The area has four distinctive topographic features, the names for which will be used throughout this report. The topographic features are:

1. Barnes Hill:

   A low hill that forms the southerly extension of the intrusion.

2. Mt Vulcan:

   A low hill that forms a part of the northerly extension of the intrusion.

3. Scotts Hill

   A low hill that demarks the extreme northerly extension of the deposit.

4. The "Flat" Area

   A depression in the intrusion lying between the northerly flank of Barnes Hill and the southerly flank of Mt Vulcan.

2.1 Regional Geology

The ultrabasic intrusion has been altered almost completely to serpentinite prior to being subjected to chemical weathering. The weathered serpentinites have altered to clays and a ferruginous laterite to varying degrees.

Anthony (1967) describes the regional setting of the deposit as follows:

"Serpentine outcrops intermittently over an area approximately 4 miles long and up to 1.5 miles in width. The area is bounded by quartzites to the east and claystones and slates to the west. Permian conglomerates overlie the serpentinite to the north and south. The quartzites are of Cambrian age and were intruded by the ultrabasic rocks in Cambrian times. The ultrabasics were subsequently altered to serpentinites. In turn the serpentinites were intruded by granitic rocks in the Devonian Period. The belt of serpentine occupies a topographic low and is surrounded to the west and north by rugged hills."

2.2 The Weathering Profile

From the period 1st August to 31st October, 1967, King Island Scheelite (1947) Limited completed field investigations comprising geological mapping, surveying, diamond drilling and allied duties.

The Senior Geologist (1967) subdivided the products of weathering, derived from the underlying serpentinites, into two broad categories on the basis of their nickeliferous potential. These are:

1. "A more favourable variety, occupying topographic high areas and frequently covered by hard pisolitic laterite" and

2. "A less favourable variety developed in topographic low or flat areas which showed a poorly developed lateritic profile, an absence of hard pisolitic laterite and frequent outcrops of fresh serpentinite".

A definite and recognisable lateritic profile is developed in the more favourable areas and has been described by the Senior Geologist (op cit) as follows.

2.2.1 Laterite

Pisolitic Zone. Hard pisolitic grains of ironstone cemented with a matrix of ferruginous red clay.

Ferruginous Red Zone. Soft chocolate to dark red clay, with occasional hard pisolitic grains and blebs of black ironstone.

Limonitic Yellow Zone. Soft yellow to orange coloured clay with occasional blebs of red clay.

Mottled Zone. Soft bright red or chocolate red, brown, yellow, purple clay with occasional specks of white and black. The colour variations are either in bands * or blebs.

2.2.2 Serpentinite

Transition Zone. Soft, highly decomposed serpentinite, dark to light green in colour, with occasional inclusions of red clay.

Bleached Zone. Soft to hard pale yellow-green serpentinite with occasional specks and stringers of unaltered magnetite.

Fresh Zone. Moderately hard to hard variegated dark green serpentinite, with occasional specks and stringers of magnetite.

Head assays taken from approximately 5-foot intervals or at changes in lithology from each diamond drill core indicate that the transition zone contains the highest nickel values.

All assay data obtained from the diamond drill holes in which economic grades of nickel were intersected are tabulated in Table 1 of this report. Appendix A lists the horizons of chemical weathering, their depth, and their nickel assay for each of the diamond drill holes completed to date.

2.3 Mineralogy

The serpentinite and its weathered horizons at Beaconsfield have been subjected to a number of investigations over a period of time. Most of the work to date has been concerned with the mapping and assessment of potential asbestos, chrome and nickel deposits.

The published work on nickel deposits has concerned the assessment of the grades of nickel in various horizons and the determination of their aerial extent. Little work has been published on the mineralogy of the Beaconsfield deposit.

Ever~~ade~~ard (in Hughes, 1957) appears to have been the first to consider the mineralogy of the nickeliferous horizons. After thin-section examination of some of the Beaconsfield ore he concluded that there were no specific nickel minerals in the serpentinite and considered the serpentinite itself to be nickeliferous. However, in the same paper Hughes (1957) states that laboratory tests suggested that the nickel mineral may be a silicate. He makes no reference to the type of laboratory tests which were undertaken.

Hughes (1962) after further work on the deposit concluded that the form of the nickel was doubtful.

The CSIRO (1956) examined a concentrate that was obtained by washing a sample of the laterite that overlay an area of nickeliferous serpentinite. The concentrate was reported to constitute 10 to 12% of the laterite and consisted predominantly of chromite with a small amount of hematite and occasional grains of

No ilmenite + rutile in our samples *

ilmenite, rutile, leucoxene and translucent minerals. As part of the same investigation a concentrate from a nickeliferous serpentinite was examined and found to be composed dominantly of magnetite with considerable chromite and traces of picotite and hematite.

The CSIRO (1957a) noted that the serpentinite in the vicinity of rodingite dykes assayed as high as 3% nickel. This decreased to 0.9% furthest from the contact. Near the contact the nickel mineral was found to be in the form of a hydrous silicate which occurred along joint planes and slickensided surfaces in the serpentinite. The hydrous nickel silicate (garnierite) formed colloform growth layers, in places interlayered with opal. A sample of garnierite from the sample was hand picked and analysed. The results are tabulated below:

	%	
SiO_2	47.98	
Al_2O_3	0.82	
Fe_2O_3	1.42	1.56 Fe_2O_3
FeO	0.13	= 0.14 % Fe_2O_3
MgO	22.62	
CaO	0.09	
H_2O^+	8.09	16.19
H_2O^-	8.10	
CO_2	nil	
P_2O_5	nil	
NiO	10.68	(8.39 % Ni)
	99.93	

? mn oxide?

graphite on E side of ... ...

The investigators noted that the garnierite is not widespread throughout the serpentinite outcrop. They concluded that the secondary nickel mineral was not due to either supergene or metasomatic alteration of nickel sulphides.

Further samples from the vicinity of a rodingite dyke were examined by the CSIRO (1957b). The rodingite had intruded serpentinite which consisted of serpentinite rock, decomposed serpentinite (brown clay) with, in places, brown and white opal and green chlorite veinlets. The green chlorite was thought to ? be vermiculite and the nickel mineral to be the hydrous silicate, garnierite. In the decomposed serpentinite (brown clay) chromite, magnetite, limonite, opal, chlorite and serpentine were identified. Graphite and hematite were also noted. ?

Stephanski (1959, 1960) has compared the Beaconsfield deposit with that of New Caledonia. He noted that the nickel enrichment takes placed in the highly decomposed serpentinites close to the fresh serpentinite and that the laterites overlying the decomposed serpentinites (clays) become increasingly impoverished in nickel towards the surface.

The hydrous nickel silicate garnierite, although assumed by some writers to be found throughout the deposit, has been observed only in the vicinity of the rodingite dykes. ~~Everade~~ Everard (1962) described a sample of the enriched, decomposed serpentinite as being formed of fibrolamellar antigorite and intergranular isotropic serpophite. "Islands of antigorite occur in veinlets which send off numerous fine branches. The veinlets and branches carry idiomorphic crystals of magnetite coated with brown limonite and limonite stains some of the antigorite". ~~Everade~~ Everard concluded that the nickel probably replaced some of the magnesium in the crystal lattice of the serpentine.

Hughes (1959) analysed two samples of nickel-bearing "clay" from the deposit. He described the first as consisting largely of iron-oxide and called it a red clay. The second sample he described as a black and green clay which appeared to be mostly serpentine weathered in-situ. The results of his analyses are tabulated below:

	red clay Sample 1 %	black + green clay Sample 2 %
SiO_2	14.64	33.19
Al_2O_3	13.69	7.23
Fe_2O_3	52.40	38.95
FeO	2.21	2.46
Cr_2O_3	3.50 - 2.4% Cr.	2.51 - 1.7% Cr.
TiO_2	0.05	0.01
CaO	Trace	Trace
MgO	1.05	5.14
NiO	0.33 = (0.26 Ni)	1.49 = (1.17 Ni)
CoO	0.06	0.07 = 0.05 Co
Ignition loss	11.94	8.39

2.3.1 Conclusions

From the mineralogical work that has been completed it is apparent that the deposit results from the chemical weathering of a serpentinized, ultrabasic body.

Periodic saturation of the fresh rock with percolating meteoric waters has resulted in the leaching of soluble ions from the surface zones and enrichment in nickel at the base of the weathered profile.

INCORRECT; - both 2nd any SiO2 in weath. very.

Over most of the deposit it is evident that the physico-chemical conditions have not been suitable for the co-precipitation of soluble silica and nickel. Thus the hydrous nickel silicate, garnierite, has not formed. Instead, the nickel from the surface zones of the decomposed serpentinite has leached downwards and been precipitated in some other form. In this respect the deposit differs from that found in New Caledonia. The form in which the major part of the nickel now occurs had not been determined prior to the present investigation. Silica has been leached from the fresh rocks and been removed from the system.

In the vicinity of the rodingite dykes it is apparent that the physico-chemical conditions have been suitable for the co-precipitation of the soluble nickel and silicon ions, leached from the surface zones, and hence the hydrous-nickel-silicate garnierite has crystallized.

It is not uncommon for this secondary enrichment to occur at relatively great depths along contacts in this type of environment. The enrichment at depth is probably a function of the relative permeability of these contact zones.

The following minerals have been recorded from the deposit:

Serpentine
Garnierite
Chlorite ('vermiculite') x
—Chromite
—Picotite
Hematite
Limonite
Rutile
oxidic Leucoxene
Graphite

In the absence of other minerals the first three of these undoubtedly contain the bulk of the nickel observed to date, although it is probable that a certain proportion is adsorbed on the surface of the hydrated iron-oxide minerals, hematite and limonite.

2.4 Beneficiation Tests

Manson (1960) carried out a series of leaching tests on a nickel-ore sample from Beaconsfield. The leaching tests were carried out with sulphuric acid at normal temperatures and pressures.

The sample was screened to minus 10 mesh and used in the wet state for the tests, the moisture amounting to 18.9%. The following analysis on a dry basis shows the composition of the ore:

	%
Ni	1.14
Co	0.36
SiO_2	5.31
Al_2O_3	8.48
Fe_2O_3	68.01
FeO	1.71
MnO	3.06
Cr_2O_3	2.95
TiO_2	nil
CaO	nil
MgO	0.87
Ignition loss	8.16

The amount of nickel extracted under the conditions of six leaching tests is shown in Table 2.

In Tests 3, 4 and 5 although greatly increased amounts of sulphuric acid were used, there was only a relatively small increase in nickel extraction. The agitation time for the tests was 44 hours, except Test 6 which was continued for 6 days.

In the final leach solution of Test 1 there was no free sulphur dioxide. In Test 2 there was some free sulphur dioxide while in Tests 3, 4, 5 and 6 there were considerable amounts present.

The weight of other ore constituents dissolved in these tests is shown in Table 3 and apart from silica, is expressed as pounds of metal dissolved from 1 ton of ore.

The extractions of cobalt given in Table 3 amount to 30.6, 38.9, 44.4, 48.6 and 52.8%, respectively, of the cobalt in the ore sample supplied.

Size analyses of the ore and residue from Test 6 are presented in Table 4.

Subsequent to this investigation, Manson (op cit) conducted a series of successive batch leaches with sulphurous acid on a single charge of oxidized nickeliferous ore. The sample was again screened to minus 10 mesh. It then contained 18.9% moisture and was used for the test in that condition. Commercially available sulphurous acid was used and analysis indicated this to contain 4.61 g sulphur dioxide per 100 ml of solution. The tests were performed at atmospheric temperatures and pressures and in closed vessels to prevent loss of sulphur dioxide.

The tests showed that the third leach was ineffective and that under the conditions of leaching total extraction of about 46% of the nickel in the ore was obtained. Nearly all the manganese was dissolved in two series leaches while after the first leach iron was also dissolved.

An attempt to recover the nickel from the leach solutions by caustic soda precipitation was made. Sufficient caustic soda was added to raise the pH of the solution to 4.7 and the hydrates of iron and manganese were filtered off and washed. Further caustic soda was added to bring the pH to 8.2 and the nickel hydrate filtered off and washed. The precipitates were ignited and analysed for nickel. All the nickel was recovered but it was distributed as follows:

	% Distribution
Fe Mn hydrates	34.3
Ni hydrate	65.7

The separation was thus considered to be ineffective.

A sodium bisulphate leach was attempted. This test certainly resulted in greatly lessening the amount of manganese dissolved but the nickel extraction was so low as to render such a process ineffective.

An attempt was made to extract the nickel from the ore by the Nicaro process. Two leaches with intervening filtration resulted in 67.7% of the nickel in the ore being extracted. Separation of manganese and iron was virtually complete. Manson indicates that reduction under carefully controlled conditions could be expected to give a better extraction.

2.4.1 Conclusions

The following beneficiation and extraction techniques have been tested on single samples of the Beaconsfield nickeliferous laterite ore:

1. Sulphuric acid leaching tests.
2. Size analysis of ore.
3. Size analysis of residue of sulphuric acid leach.
4. Sucessive batch leaches with sulphurous acid:
    a. recovery of nickel from leach solutions by caustic soda,
    b. recovery of nickel from leach solutions by ammonia extraction, and
    c. recovery of nickel from leach solutions by sodium bi-sulphate extraction.
5. Extraction of nickel from the ore by the Nicaro process.

The sulphuric acid leaching tests extracted 47 to 50% of the nickel present in the ore in the most successful runs.

A size analysis on the sample used in the sulphuric acid leaching tests did not up-grade the ore. The nickel was distributed in concentrations of from 1% to 1.39% through all size ranges. A size analysis of the leach residue showed the nickel to be distributed through all size ranges at concentrations ranging from 0.12 to 0.75%. The bulk of the nickel was contained in the finest size-fraction (-200 mesh). ??

Successive batch leaches using sulphurous acid extracted 46% of the total nickel in the ore. Caustic soda precipitation of the nickel from the leach solutions was unsuccessful in that 34% of the nickel was precipitated in iron and manganese hydrates. Ammonia extraction gave a recovery of 92% of the nickel in the leach solutions. A sodium bisulphate leach was unsuccessful.

A laboratory test of the Nicaro process resulted in 67.7% of the total nickel in the ore being extracted. It is considered that this result could be improved by carefully controlling the conditions of the reduction.

Of the tests conducted to date it is quite evident that the most suitable process of extracting the nickel from the ore is the Nicaro process which involves a reducing roast of the finely ground ore, followed by leaching with ammonia and ammonium carbonate solution.

2.5 Other Elements and Minerals of Economic Interest

2.5.1 Chromium

After a series of drilling operations in the Mount Scott, Mount Vulcan and Barnes Hill areas, Nye (1930) concluded that "chromiferous iron-ores (at Beaconsfield) are of such a composition that they cannot be used for the manufacture of ferrochrome, refractories or chemicals". The percentage of chromium in the iron-ores varied from 5.3% on Barnes Hill to 3.5% and 3.8% at Mount Scott and Mount Vulcan respectively. These iron-ore deposits are particular cases of laterite development over serpentine.

In a subsequent study of the ~~iron~~ chrome-ores Noldart (1962) concluded that:

1. "Chromiferous deposits of economic grade occur in the basal zone of the Tertiary quartz gravels overlying the ultrabasic complex in the vicinity of Barnes Hill".

2. "Whilst the deposits are of economic, or potentially economic grade, known tonnages available are too small to warrant exploitation".

It is evident therefore that economic grades of chromium do exist in the area and although, in themselves probably inadequate to support a mining industry, warrant further consideration if the area is to be exploited for nickel.

2.5.2 Asbestos

Taylor (1955) reported in a review of the literature on asbestos deposits near Beaconsfield that

"while no spectacular developments could be expected at Beaconsfield there is a reasonable quantity of medium-grade ore available which may serve as the basis of a profitable small industry capable of supplying Tasmania's requirements of asbestos fibre ...".

2.5.3 Iron, Cobalt and Manganese

Iron, cobalt and manganese are all known to occur in appreciable amounts in the weathered profile of the ultrabasic rocks. If the Beaconsfield area is to be exploited, then careful consideration should be given to the possibility of beneficiating these elements together with the major element, nickel.

3. NICKEL MINERALOGY

The diamond drill cores from the diamond drilling programme completed by King Island Scheelite (1947) Limited, on the Beaconsfield Nickel Deposit are stored in the core store of the Tasmanian Department of Mines in Hobart. The cores were examined and ninety-five samples, representative of the various environments of chemical weathering intersected in each hole, were collected and returned to Amdel for mineralogical examination.

Details of the samples collected are contained in Table 5 of this report.

3.1 Experimental Procedure

The mineralogy of selected samples was determined using x-ray diffraction techniques.

The samples were crushed and oriented on porous ceramic plates. Each sample was run on the x-ray diffractometer from 3 to 42 degrees 2θ using cobalt K α radiation. The proportion of serpentine in each sample was determined semi-quantitatively by comparing serpentine (001) peak height with a pure serpentine standard from the area (Sample No.8-60). No correction was made for absorption so the accuracy of the determination was not better than ±5%. Hence, the results have been expressed in terms of the following:

	%
Dominant	>50
Co-dominant	30 - 50
Sub-dominant	20 - 30
Accessory	10 - 20
Trace	<10

$>$ = Greater than.

The proportions of other components were determined by comparison with standards from other areas and again the accuracy cannot be considered high although it will be within the ranges stated above.

In some cases the mineralogy and semi-quantitative abundance determinations were verified using optical techniques.

In order to determine whether or not a single mineral phase contained the bulk of the nickel, samples were selected for electron probe microanalysis. Polished thin sections were prepared and carbon-coated for examination. Spot analyses for nickel were conducted for the various mineral phases. The results are not of high accuracy. Corrections were not made for mass absorption and pure metal standards were used instead of standards with a nickel content comparable to that of the sample being analysed.

3.2 Results

3.2.1 Mineralogy of Nickel-enriched Horizons

Eleven samples were selected from the nickel-enriched horizons of various drill cores for examination using x-ray diffraction techniques. The samples with notes on their hand-specimen characteristics are listed in Appendix G of this report.

The results of the mineralogical analyses are presented in Table 6.

The results indicate that:

1. Serpentine although present in some samples, is not a consistent constituent of the zones enriched in nickel. It is significant, however, that the drill holes that display the highest concentrations of nickel all contain significant amounts of serpentine.

2. The core from DDH 15 has a significantly different mineralogy from the others examined. The presence of dominant talc and a comparatively high nickel assay suggests that this mineral could be nickeliferous.

3. Of the remainder of the samples examined it appears that the clay mineral (smectite), and chromite are the only consistent constituents of the nickeliferous horizons.

Thus it would seem that the potential nickel-bearing minerals in the nickel-enriched horizons are:

Serpentine
Clay mineral
Secondary iron-oxides
Talc

Chlorite was identified from two samples (7B at 20 ft and 10 at 8 ft). This mineral could also contain nickel, although the amount so contained is unlikely to be significant in terms of the deposit as a whole.

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3.2.2 Mineralogy of Barnes Hill Area

Diamond drill hole 8 was selected as being representative of the Barnes Hill area. This hole contains the highest nickel values from this portion of the deposit and all horizons of weathering are represented in the drill core.

Table 7 lists the mineralogy of DDH 8 at the various horizons of weathering.

Significant features of the mineralogy of DDH 8 from the Barnes Hill area are:

1. The presence of large proportions of the clay mineral (smectite) in the nickel-enriched zones.
2. The presence of trace amounts of chromite in all but the surface horizon.
3. The absence of significant amounts of serpentine in the zone with the highest concentrations of nickel.
4. The presence of secondary iron-oxides in the surface horizons of the profile.

It is apparent that this drill hole represents a typical weathering profile of its type. The parent rock for the weathering process has been a serpentinite which has altered to a clay mineral. In the surface horizons all but the iron-rich material has been leached to leave the secondary iron-oxides, hematite and goethite. In this horizon it is probable that the nickel is adsorbed on the surface of amorphous manganese minerals or on hydrogoethite. In the clay-rich zones the nickel must be present in either or both of the magnetite or the clay. In the freshest zone at the base of the profile it is probable that the nickel is contained in the lattice of the serpentine minerals. = POSITIVE CONCLUSION

Thus the nickel is present or associated with secondary iron-oxides in the surface horizons; clay mineral in the clay-rich horizons; and serpentine in the fresh horizons.

3.2.3 Mineralogy of Mount Vulcan Area

Diamond drill hole 16 is the only one from the Mount Vulcan area and was selected for additional work so that the mineralogy from each of the major areas of the deposit could be adequately documented.

The mineralogy of samples from Mount Vulcan is listed in Table 8.

An unusual feature is the presence of significant proportions of serpentine in the surface horizons although this could be a local rather than a general feature. Beneath the surface horizons there is a steady increase in the

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serpentine-to-clay ratio as would be expected. Chromite appears to be more abundant in this area than at Barnes Hill although its distribution within the profile is somewhat sporadic.

In the Mount Vulcan area, then, the nickel could be associated with:

a. Serpentine, hydrogoethite, and/or manganese minerals in the surface layers;

b. Serpentine, and clay minerals in the weathered horizons;

c. Serpentine in the fresh horizons.

It is apparent that the mineralogy of Mount Vulcan does not differ significantly from that in the Barnes Hill area.

3.2.4 <u>Mineralogy of Scotts Hill Area</u>

Diamond drill holes 14 and 15 are the only two drilled in the Scotts Hill area, and were selected for additional mineralogical work because of the apparently anomalous mineralogy found in the initial examination of Sample 15 from a depth of 69 ft.

The mineralogy of DDH 14 and 15 is listed in Table 9.

The mineralogy of the Scotts Hill area appears to differ quite markedly from that of the remainder of the Beaconsfield nickel deposit.

Diamond drill hole 14 displays a similar mineralogy to the rest of the deposit in all but the presence of talc at the 40-ft level. This hole contains the typical sequence of secondary iron-oxides at the surface which grades, in turn, into a clay-rich zone before entering the fresh serpentinite.

Diamond drill hole 15, however, has a mineralogy that contrasts with that of the remainder of the deposit. Talc, carbonate minerals and amphibole are commonly found in association with clay minerals. Towards the bottom of the hole the primary silicate, enstatite, is common. It seems probable that this portion of the deposit was formed predominantly of the rock pyroxenite which has subsequently altered to talc, amphibole and carbonate minerals, while the remainder (Barnes Hill, Mount Vulcan) was composed predominantly of olivine which altered to serpentine and clay minerals. Olivine has not been identified, however, from any area in the deposit and this thesis remains tentative.

The proximity of DDH 14 and 15 and the presence of talc in the assemblage of both holes suggests that these two are intimately related.

3.2.5 Mineralogy of the "Flat" Area

Three samples from DDH 17 (Flat area) were examined to relate this part of the deposit to the remainder.

The mineralogy of samples examined from the "Flat" area is listed in Table 10.

The Flat Area has a poorly developed lateritic profile and does not appear to contain economic grades of nickel. The mineralogy here is similar to that of the Mount Vulcan and Barnes Hill areas and is formed of serpentine in the fresh zones, serpentine with clay minerals in the weathered zones and clay minerals and goethite in the surface horizons.

3.3 Discussion

Two mineralogically distinct provinces can be delineated in the Beaconsfield Nickel Deposit.

The first of these, found in the Barnes Hill, Mount Vulcan and Flat areas, covers the largest portion of the deposit, and contains the bulk of the estimated reserves of nickel. Anthony (1967) estimates the nickel reserves in these areas as follows, (Case A = proven reserves; Case B = projected reserves):

1. Barnes Hill Area:
   Case A: 5,542,000 tons at 1.03% grade.
   Case B: 7,390,000 tons at 1.03% grade.

2. Mount Vulcan:
   Case A: 928,000 tons at 1.14% grade.
   Case B: 2,668,000 tons at 1.14% grade.

3. The Flat Area
   Case A: less than 0.70% grade.
   Case B: less than 0.70% grade.

4. Total:
   Case A: 6,470,000 tons at 1.03% grade.
   Case B: 10,058,000 tons at 1.03% grade.

In these areas a typical weathering profile is formed of a surface horizon composed dominantly of the secondary iron-oxides goethite and hematite with or without lesser amounts of serpentine and clay minerals. This horizon gradually gives way with depth to a clay-rich zone composed dominantly of smectite with or without serpentine and chlorite. Underlying these weathered horizons is the fresh zone of serpentinite. Chromite is found throughout the profile. NB

From the mineralogy it is clear that the following minerals could be potentially nickeliferous:

Serpentine
Clay mineral
Chlorite
Secondary iron-oxides

A second province which is mineralogically distinct from the first is that of Scotts Hill. This area forms the northernmost extremity of the deposit. Anthony (op cit) estimates the nickel reserves in the Scotts Hill area to be:

Case A: 632,000 tons at 1.06% grade.
Case B: 1,655,000 tons at 1.06% grade.

The mineralogy of DDH 15 is taken as representative of the weathering profile developed in the area. This hole and DDH 14 are the only two holes that have been drilled to date and further drilling could modify the generalizations expressed below.

The fresh rock from which the profile developed appears to have been composed predominantly of the silicate enstatite. The mineral has been incipiently replaced by amphibole and subsequently been wholly or partially altered to talc plus the carbonate minerals magnesite and calcite. The weathering process has decomposed these minerals to a clay and small amounts of chlorite. The surface horizons are composed dominantly of secondary iron oxides. Chromite occurs throughout much of the profile. The mineralogy of DDH 14 more closely approximates that of the previously defined province in that amphibole was not identified from the examined samples. The mineralogy is, however, related to DDH 15 in that there is an apparent absence of serpentine at depth and there are significant amounts of talc and magnesite developed.

From the mineralogy it is clear that the following minerals from this province could be potentially nickeliferous:

Talc
Chlorite
Clay mineral
Secondary iron oxides

3.4 Electron Probe Microanalyses

The results of the electron probe microanalysis of the various potentially nickel-bearing phases in the weathered horizons of the Beaconsfield nickel deposit are presented in Table 11.

The results indicate that the following minerals are nickel-bearing:

- Serpentine
- Clay mineral (smectite)
- Goethite
- Chlorite
- Talc
- Enstatite

The average nickel content of the serpentine is approximately 1% or a little lower, although in one sample (16.35) it was somewhat higher at 1.8%. This result could have been affected by the presence of very fine-grained goethite that is disseminated throughout the serpentine fibres. The clay mineral consistently contains nickel although commonly in low concentrations. The highest concentrations of nickel are associated with goethite. It is well known that nickel is readily absorbed in the surface of hydrogoethite. Although a nickeliferous chlorite was identified from only one sample (14.47) it is possible that the chlorites throughout the deposit are potentially nickeliferous. The talc from the Scotts Hill area contains nickel to the extent of approximately 0.8%. The only primary mineral identified from the deposit was the silicate enstatite from DDH 15 at Scotts Hill. This mineral is nickeliferous and contains approximately 0.5% nickel.

It is apparent, therefore, that the bulk of the total nickel is included in the lattice of the clay mineral smectite although the highest concentrations of nickel are in the hydrated iron-oxide goethite.

The results of a series of sulphuric acid leaching tests (Manson, 1960; Section 2.4.1 of this report) indicate that approximately 50% of the total nickel in the test sample is in the form of an exchangeable cation. These exchangeable cations no doubt occur between the silicate layers of the smectite. The 50% of the total nickel that remained after leaching could in part have substituted for aluminium in the crystal lattice of the clay mineral or be combined in the lattice of other silicates.

3.5 Conclusions

The results of a mineralogical examination and electron probe microanalysis on selected samples from the Beaconsfield Nickel Deposit indicate that the nickel is distributed through a variety of minerals.

The deposit may be divided into two distinct mineralogical provinces; one centred on the Barnes Hill, Mount Vulcan and Flat areas to the south, and a second centred on Scotts Hill to the north.

3.5.1 Barnes Hill, Mount Vulcan, "Flat" Area Province

This first province is one in which primary ultrabasic rocks have apparently been wholly serpentinized and subsequently subjected to a prolonged period of chemical weathering. From depth a typical weathering profile extends through a fresh zone (serpentine) to a bleached zone, in which the serpentine has been partially replaced by a clay mineral. This in turn gives way to a transition zone in which the clay mineral forms the bulk of the weathered rock. Overlying the transition zone are a series of zones that become progressively enriched in secondary iron-oxides and deficient in the clay mineral.

The fresh rock contains a variable amount of nickel in the concentration range of from 0.5 to 1.5% nickel approximately. The nickel is wholly contained in the serpentine mineral.

In the bleached zone the nickel is divided between the clay mineral, the serpentine and secondary iron-oxide phases.

In the transition zone the nickel is divided between a clay mineral, secondary iron-oxides and sometimes an iron-rich chlorite phase.

The iron-rich phases undoubtedly contain the highest concentrations of nickel, sometimes up to 5%, but in general, the proportion of these phases is very small. Thus the total nickel combined in the secondary iron oxides will be small. The chlorite, too, is relatively minor and, although nickeliferous, will contain only a small proportion of the total nickel. The clay mineral is by far the most abundant mineral, and although containing relatively low concentrations of nickel (seldom greater than 1%) undoubtedly contains the bulk of the total nickel.

In the surface, iron-rich horizons, the secondary iron-oxides, unlike those at depth, do not contain high concentrations of nickel. In these zones it is apparent that much of the nickel has been leached to greater depths in the weathering profile. Where present the clay mineral contains the bulk of the total nickel although the secondary iron-oxides do contain low concentrations. At or near the surface the secondary iron-oxides contain all of the nickel although in very low concentrations.

In the Barnes Hill, Mount Vulcan, Flat area province then the nickel is found in relatively high concentrations in the transition zone where it is divided between serpentine, clay mineral, chlorite and iron-oxide phases.

3.5.2 Scotts Hill Province

The second mineralogical province is that centred on Scotts Hill. Here the relatively high nickel values are not exclusively confined to the transition zone, but extend above through the "mottled zone" and less frequently to the "limonitic zone". In this province the primary ultrabasic rocks, instead of being serpentinized, have been largely altered to a nickeliferous talc prior to being subjected to a period of prolonged chemical weathering.

The freshest rock intersected by the drill core is a pyroxenite that has been partially replaced by talc and minor amphibole. In this zone the nickel is divided between the talc and the primary silicate, enstatite. The amphibole is devoid of nickel.

In the bleached zone, overlying the fresh zone, the talc alters to carbonate minerals, chlorite and clay. In this zone it appears that the nickel is divided between the talc and the clay mineral. No nickel was recorded from the chlorite.

Samples from the surface horizons overlying the transition zone have not been subjected to electron probe microanalysis but it is probable that as the rock progressively becomes depleted in clay minerals that the nickel is absorbed on the surface of secondary iron oxides. Serpentine was identified from one sample. This mineral is probably nickeliferous and its presence suggests that the primary ultrabasic rock may have been layered with pyroxenites at depth and troctolites or dunites nearer the surface.

The conclusions drawn are necessarily tentative. To confirm the details of the mineralogy and the distribution of the nickel through the profiles a more extensive investigation would be required, involving the detailed mineralogical assessment of a large number of samples. These comments apply particularly to the Scotts Hill area. A distinct mineralogical province has been proposed on the basis of the mineralogy of one diamond drill hole which, however, may not be entirely representative of the region.

Two features that are worthy of note are:

1. In this examination garnierite was not identified. This suggests that this mineral has a restricted occurrence and is found only in the vicinity of rodingite dykes (CSIRO, 1957).
   Knight (1957) reported that outcrops of garnierite bearing weathered serpentinite were restricted largely to a triangular-shaped area 4000 ft by 2500 ft at the south-eastern tip of the ultrabasic mass. Rodingite dykes are quite common in this area.

2. In one sample (12.58) nickel was found in association with manganese. Only one particle was identified that has this association; one that is very common in the ferruginous zones of lateritized ultra-

basic rocks (Turner, 1968).

It is assumed then that the manganese-nickel association is not common in this deposit, possibly because of the incomplete lateritization that has occurred.

4. LIMITATIONS OF PHYSICAL BENEFICATION

The mineralogical examinations indicate quite strongly that physical beneficiation of the ore to up-grade the nickel content is not practical for two reasons:

1. The nickel is distributed through the fine mineral phases of serpentine, clay mineral, goethite, chlorite, and talc which compose the bulk of the nickel-enriched rocks. Hence, the separation of phases barren in nickel, will not significantly up-grade the ore.
2. The mineral phases serpentine, clay mineral (smectite), chlorite and talc have similar physical properties and could not be satisfactorily separated. The goethite, however, could possibly be liberated from the other phases, but its occurrence is so small that physical treatment to isolate this mineral cannot be justified.

In addition a size analysis (Manson, 1960) on an ore sample used in sulphuric-acid leaching tests did not up-grade the ore. The nickel was distributed in concentrations of 1% to 1.39% through all size ranges and the bulk of the total nickel occurred in the minus 200-mesh fraction.

It is thus impracticable to consider physical beneficiation of the Beaconsfield ore.

5. SOME ASPECTS OF THE CHEMISTRY OF THE BEACONSFIELD DEPOSIT

5.1 Discussion

Over a period of time a considerable number of surface and costean samples from the Beaconsfield ultrabasic complex have been analysed for nickel and other elements. The most comprehensive survey has been completed by King Island Scheelite (1947) Limited, as a part of a diamond drilling programme. As the localities of many of the earlier analysed samples are largely unknown, only those results obtained by King Island Scheelite (1947) Limited will be considered here.

All chemical analyses available from samples of drill core that intersected economic grades of nickel plus cobalt are presented in Table 1.

The average content of each of the determined elements from fourteen diamond drill holes are listed in Table 12. The figures obtained for silica, lime, alumina and ignition loss are not of high accuracy because in most cases the analysed samples are not representative of the entire weathering profile. The figures for magnesium, iron, nickel and cobalt are more reliable and represent the complete weathering profile in each hole. Together the results are useful as a guide to geographical variation within the complex. ? not to be expected in a genuine laterite

The most striking feature at once evident from an examination of Tables 1 and 12 is the range of chemical variation within the complex.

Generally the concentrations of silica, lime and magnesia increase with depth through the weathering profile while the concentrations of alumina, iron and, to a lesser extent, chromium decrease with depth. The cobalt concentration remains fairly constant through the weathering profile. The amount of nickel increases with depth and attains a maximum concentration in the transition zone to the fresh serpentinite at which point the concentration falls markedly.

These results are quite characteristic of lateritic profiles developed over serpentinite bodies although at Beaconsfield it is evident that lateritization is not complete.

The average of all analyses for each of the determined elements from the fourteen drill holes in which economic grades of nickel plus cobalt are intersected are tabulated below with the range in average values for each element:

	Average %	Range %
SiO_2	32.30	13.19 - 43.08
CaO	2.02	0.11 - 9.22
Al_2O_3	6.45	1.00 - 11.57
Mg	8.84	2.30 - 15.97
Fe	18.60	7.97 - 27.92
Cr	0.70	0.35 - 1.06
Co	0.03	0.01 - 0.09
Ni	0.47	0.33 - 0.98
Loss on Ignition	13.74	10.50 - 18.40

Apart from the marked chemical variation with depth there is a pronounced lateral variation between adjacent drill holes even in the one horizon of chemical weathering.

King Island Scheelite (1947) Limited assumed in their ore-reserve estimations a cut-off grade of 0.70% nickel plus cobalt. This assumption is retained in the following discussion.

By selecting a cut-off grade of 0.70% nickel plus cobalt particular horizons of ore can be defined throughout the province. Apart from the marked vertical and lateral chemical variation noted above there is also a pronounced chemical variation within the nickel-ore horizon.

The following data obtained from each of fourteen diamond drill cores are presented in Table 13:

a. Depth to ore horizon.

b. Thickness of ore horizon.

c. The average concentration of the elements Ni, Co, Cr, Mg, Fe, SiO_2, CaO and Al_2O_3 in the ore horizon of each hole.

d. The range in concentration of the elements Ni, Co, Cr, Mg, Fe, SiO_2, CaO and Al_2O_3 in the ore horizon of each hole.

These figures are quite reliable as all elements were determined for the complete intersection of the ore-bearing horizon in each hole.

The range and the average concentrations of the determined elements in the ore-horizon are tabulated below:

	Range Beaconsfield %	Average Beaconsfield %	Nicaro Ore %
Ni	0.66 - 1.49	1.06	1.4
Co	0.02 - 0.12	0.06	0.1
Cr	0.11 - 0.75	0.38	-
Mg	1.4 - 11.01	4.89	8.0
Fe	12.4 - 27.4	20.56	38.0
SiO_2	13.19 - 42.97	39.94	14.0
CaO	0.04 - 36.19	4.01	-
Al_2O_3	1.12 - 11.57	6.25	-
LOI[(a)]	10.50 - 20.53	15.19	-

(a) LOI = Loss on ignition.

These figures indicate the significant chemical variation that exists within the ore-bearing horizon at Beaconsfield. In addition the average composition of the Beaconsfield ore differs significantly from that used as feed-ore to the Nicaro

Process in Cuba. By comparison the Beaconsfield ore contains approximately 17% less iron, 25% more silica and 3% less magnesium than the Nicaro ore.

The ore-horizon at Beaconsfield, compared to the deposit as a whole, is relatively enriched in lime, deficient in magnesium and enriched in iron.

Apart from the marked chemical variation through the province, a second point worthy of note is the pronounced variation in the depths of the economically, nickel-enriched horizons.

From place to place throughout the complex the top of the ore-bearing horizon lies beneath an overburden ranging in thickness from 7 ft to 44 ft and in one hole (15) a second ore horizon is found at a depth of 65 ft. The ore-bearing horizon ranges from 3 ft to 39 ft in thickness and varies considerably in thickness between adjacent holes.

5.2 Conclusions

The following conclusions on the chemistry of the Beaconsfield deposit may be drawn on the basis of assay data obtained from samples of a series of diamond drill cores:

1. There is a pronounced variation in the concentrations of magnesium, lime, iron, silica, alumina and nickel with depth. Iron and alumina are relatively enriched in the surface horizons while magnesium, lime, silica and nickel are relatively enriched towards the base of the weathering profile. This pattern of variation is characteristic of a lateritic weathering profile.

2. There is a pronounced lateral, chemical variation throughout the deposit.

3. In the ore-bearing horizon (defined by arbitrarily selected cut-off grades of 0.70% nickel plus cobalt) there is considerable chemical variation. Of the major elements magnesium varies from approximately 1% to 11%, iron from 12% to 27%, silica from 13% to 42% and lime from 0% to 36%.

4. There is a marked variation in depth to and thickness of the nickel-enriched horizon from place to place in the deposit. Depth of overburden ranges from 7 to 44 ft and the ore-bearing horizon varies in thickness from 3 to 39 ft.

The lateral and vertical chemical variation and variation in depth to and thickness of the ore-bearing horizon all have a bearing on the mining methods needed to obtain the ore. In addition if a static feed of ore is required for the selected beneficiation process, blending of the ore will be necessary.

So that the ore bearing horizon can be delineated and blending of the ore can be controlled it will be necessary to pattern-drill the deposit on a closely spaced grid ahead of the mining operation.

6. TECHNICAL AND ECONOMIC APPRAISAL OF ALTERNATIVE PROCESSING METHODS

The inability of the Beaconsfield nickel ore to be physically beneficiated to a saleable concentrate eliminates the possibility of establishing a low-cost operation with minimum capital outlay. This means that the development of the deposit is dependent upon the acceptability of one of four processing alternatives, which are as follows:

1. Pug Roast Acid Leach,
2. Nicaro,
3. Direct smelting (Ugine process), and
4. Pyrosulphidizing.

All four processes are capital intense and have high operating costs. At the same time each process produces a high quality product of considerable value. Acceptance or rejection of any one of the four processes is dependent upon two factors: first, the technical capability of the process to treat the Beaconsfield nickel ore to produce a saleable commodity; and second, the ability of the process to produce the product at a cost which would provide the operators with an adequate return on investment.

The purpose of this section of the report is:

1. to describe the criteria for appraising and comparing each process,
2. to identify the major assumptions that are basic to the study of all four processes,
3. to discuss each process individually in terms of its technical and economic potential,
4. to select, the most promising process, if any, and
5. to evaluate the sensitivity of the selected process to critical variables.

6.1 Appraisal Criteria

The appraisal of long-term, capital-intense projects involves the relating of a project's future cash flow to the investment outlay in such a manner as to permit comparisons with other projects of a similar nature. Several financial measurements may be applied in such an analysis to test a project's profitability. In this study the economic merit of each process is assessed by the use of the discounted cash flow-rate of return method and the net pay-back period, both of which are eminently suited to this type of analysis.

The discounted cash flow, or "DCF" rate of return, as it is described, is a time-adjusted measurement and ranks projects in terms of profitability. Its concept is to discount future after-tax earnings at an interest rate which will equate the present value of these future earnings to the present value of the capital investment. In such an analysis the economic study is concerned with deriving estimates of annual cash flows resulting from all financial transactions associated with the project, but excludes interest and dividend payments. These two items are implicit within the company's cost of capital, which is expressed as a rate of interest.

As advised by the sponsor, the company's cost of capital is to be assessed at 10% and, on this basis, projects yielding returns less than 10% are considered uneconomical. Project returns in excess of 10% but less than 15% should be evaluated as marginally attractive, while projects in excess of 15% could be considered financially attractive, depending on the element of risk associated with the project.

The second financial criterion is the net pay-back period, which measures the speed with which the investment is returned, and hence measures return <u>of</u> investment and not return <u>on</u> investment. It is calculated by summing the annual cash flows to determine the period required to recover the investment outlay. It therefore emphasises the project's liquidity and susceptibility to risk rather than profitability.

6.2 Major Assumptions

The assumptions on which an analysis is based are of great importance in appraising the technical merit of a process and its economic ramifications. Estimates, both technical and economic, are merely a forecast of what is expected to occur, and therefore the accuracy of any appraisal could be greatly affected by the validity of the underlying assumptions. These assumptions, furthermore, may have varying degrees of certainty, depending upon availability of information, experience and, ultimately, upon future economic trends. It is therefore important that the sponsors have a full appreciation of the assumptions on which this analysis is based. For this reason the major assumptions that are basic to the study of all four processes are discussed below.

Size of Deposit and Grade of Ore. Processing alternatives have been compared on the assumption that the deposit contains 11 million tons of ore with an average grade of 1.03% nickel, deviating to 0.7% and 2.37%. The study further assumes that this deposit is depleted over a 15-year period at the rate of 733,000 tons per year.

Processing Operations. Each process is assumed to operate 24 hours a day, 7 days a week, for 330 days a year and have an operating life of 15 years. The volume of ore treated per day is assumed to be 2200 tons. Capital estimates include equipment associated with the processing from ore bin to port, loading facilities, and construction costs of a tailings dam.

Mining Operation. Capital cost estimates for the open-cut mining operation and estimates of operating expenses associated with the mining of the ore have been provided by the sponsor in the report dated July 25th, 1968, and are based on the following assumptions:

1. The treatment plant is located within 2 miles of the mine site.
2. The ratio of overburden to ore is one to one.
3. Drilling or blasting is minimal or not required.
4. Ore stripping will be done by the day shift; overburden by the afternoon shift.
5. Stripping of overburden and ore will be a face shovel operation.

From this report, capital cost to mine 2000 tons each of ore and overburden is estimated at $940,000; and the mine operating expenses covering direct costs, maintenance, and general overhead are estimated at 28 cents per ton of treated ore.

In addition to costs listed in the above-mentioned report, a further $660,000 for the replacement of mining equipment in the eighth year of operation has been added to the capital costs, and mine operating expenses have been projected to increase at the rate of 4% per annum. An itemized account of mining costs is presented in Appendix B.

Because of lateral and vertical chemical variations and variations in the depth and thickness of the ore-bearing horizon, the mining operation may require both drilling ahead of mining and ore blending, in which case cost estimates presented above may be understated. No allowance for this contingency has been included in the study since it is not possible to determine the extent to which this work is required until the reserve estimates have been confirmed with further drilling.

Harbour Facilities. Imports of raw materials and the export of finished product, either in bulk or bag, are assumed to be via the Inspection Head wharf at Beauty Point. It is further assumed that a stockpile area adjacent to the wharf could be made available to store both finished product and raw materials. Because Inspection Head is used for the export of meat and fruits, uncertainty exists as to the actual quantities and nature of product in bulk form that would be permitted to cross this wharf. The Port of Launceston Authority, however, is most anxious to increase the outward tonnage at Beauty Point and has shown a willingness to assist any project offering export potential. The Port Authority has pointed out that one problem associated with the movement of large volumes of material is the availability of space for a stockpile area. At present, the area available is insufficient, but the Port Authority has indicated that they would consider reclaiming the land close to the wharf.

Sulphur Prices. Elemental sulphur can be obtained only through the Sulphur Purchasing Association, administered by The British Phosphate Commission. Admittance to the association is by application and by approval of the present members. Presently, the fob price, gulf ports USA, for the bright sulphur is $US42.00 per ton. Freight rates to Australia, including landing charges, are estimated at $US12.00 per ton. Port and transportation expenses for the product from the off-loading point at Beauty Point to the mine site are estimated at 50 cents to $1 per ton, making a total "on site" cost of approximately $US55.00 per ton. The fob price for sulphur is not expected to hold because of increased production, and by the 1970's the price is expected to decline at least to $US37.00 per ton. On this basis we have, therefore, assumed that the on site price of sulphur in the 1970's would be $US50.00 per ton or $A45.00 per ton.

Labour Estimates. Four operating crews would be required for each process and the labour requirement for each crew is estimated on the basis of the number of men required for each operating unit. Direct labour costs are estimated at $60.00 per week per man, and direct supervisory labour is estimated at $90.00 per week per man with one supervisor for four process labourers. Added to the direct labour cost is a 10% shift allowance and a 5.5% loading for payroll tax and workmen's compensation. Total annual labour costs are calculated on the basis of 52 weeks a year and are projected to rise at a rate of 4% per annum, which is equivalent to wage increases for male employees in Tasmania for the past 6 years.

Fuel Oil Prices. Each process is fuel intensive and requires significant volumes of fuel oil, the cost of which represents a major component in the operating expenses. Discussions with representatives of the oil industry indicate that a great deal of uncertainty surrounds the future price of fuel oil as a result of Australia's changing pattern of crude oil supplies. In fact, all major oil companies are presently unwilling to negotiate long-term contracts for fuel oil until such time as the Federal Government legislates to permit future imports of either crude oil or fuel oil from the Middle East. Assuming that import restrictions are not imposed, the delivered price of fuel oil 19 miles from a major seaport terminal (Bell Bay) would be $15.00 per ton. If the Government does not permit importation and requires Australian refineries to produce fuel oil from indigenous Australian crude, the delivered price of fuel oil would be of the order of $26.00 per ton, an increase of 73%.

Because of the disparity in these two cost estimates, and the expected reaction from major industrial users of fuel oil, the most likely alternative is that the Government will permit imports of fuel oil or crude oil. The cost per ton of fuel oil therefore has been assumed in this study to be $15.00 per ton, and is not expected to increase over the life of the project. Close attention, however, should be paid to future legislation affecting imports of fuel oil or crude oil.

Availability of Power and Cost Estimates. The Hydro Electricity Commission does not anticipate difficulties in providing electricity to a major industrial user in the Beaconsfield area. Delivery of high-voltage electricity could result in some additional expense in the way of transformers, but this would be only minor. Cost estimates for electricity used in this study are based on the quoted tariff rate listed below and are projected to remain constant:

Tariff M - Industrial Demand Rate:
High voltage supply of 500 kW and over. Price, $67-40 per annum per kilowatt of maximum demand.

Availability of Water and Cost Estimates. Anderson River, which traverses the mining site, reportedly flows year round and is considered as one source of water supply. If this is insufficient, water supplies could be obtained from the West Tamar Scheme, which distributes filtered water into the Beaconsfield area, and has a capacity of 2.2 million gallons per day. The cost of water is not significant to the study, and the price assumed is 30 cents per 1000 gallons, which is held constant for the life of the project.

Maintenance Cost Estimates. Process maintenance costs covering the cost of materials and labour required for incidental repairs, refractory work, replacement, and preventive maintenance has been included at the rate of 5% of capital equipment. This percentage rate is held constant for the life of the project on the assumption that low maintenance requirements in the first few years will be offset by modification work during these years.

General Overhead. This item includes plant overhead and process control. Plant overhead consists of expenditure of a non-productive nature, such as administration, accounting, insurance, workshops, safety, stores, canteen, and security. The charge for this item can vary from 50% to 275% of direct labour. For this plant, which is semi-automated and operates 24 hours per day, a rate towards the lower end of the scale is more appropriate, and a rate of 100% on direct labour has been allowed. Process control consists of routine laboratory testing of raw materials and products. For a plant of this nature the cost would be of the order of 10% of direct labour. The total estimate for general overhead is therefore taken as 110% of the direct labour charge and is increased annually at the rate of 4%.

General Expenses. This item includes an allowance for the corporate administration, i.e. selling expenses, executive travel, and an allowance for research and development connected with the process. These expenses, with the exception of research, should not be particularly high for this plant and an amount of $115,000 has been included for each year of operation.

Transportation Cost Estimates. Transportation costs for raw materials are included in the raw material price. Transportation expenses for the finished product include two costs: road transport from mine site to stockpile area at Beauty Point, and shipping rates to overseas markets. Quotations from road transport companies indicate that bulk material could be transported in drop-back trucks at a contract rate of 8 cents per ton mile, which assumes free loading at the site. Bagged material would be transported for $1-75 per ton. Overseas shipping rates include the following costs:

a. Outward wharfage calculated at 61 cents per ton.

b. Insurance based on 0.19% of product value.

c. Loss factor for bulk movements based on 0.5% of product value.

d. Freight at the rate of $9-00 per ton for bulk shipments to Japan.

e. Freight at the rate of $22-00 per ton for bagged shipments to Japan.

Product Price Estimates. There is a considerable amount of speculation on future nickel prices, both in Australia and overseas. The price for nickel has risen markedly in the last few years. In 1965 the fixed, or producer's, price of metallic nickel was £Stg642 per long ton. By the end of 1966 the price had increased to £Stg652 per long ton, and in 1967 the price rose to £Stg702 and £Stg773.5 per long ton before being adjusted to £Stg902 in November 1967, following the devaluation of sterling. However, this price of £Stg902 is a fixed price; whereas the unofficial price, commonly referred to as "the free market price", can vary up to as much as twice the value of the producer's price.

At the present, the demand for nickel exceeds the supply, but future demand/supply relationship for nickel is most uncertain. Production of nickel will show significant increases in the 1970's, notably with the growth of nickel mining in Australia and New Caledonia and the accelerated mine development and expansion programme by International Nickel in Canada. The future demand for nickel is unpredictable, however, and published estimates of annual increases range from 4% to 9.5% per annum. Best indications point to a gradual increase in supply relative to demand, which will have the effect of reducing the free market price. In fact, in recent months the free market price has dropped £Stg300 from £Stg1700 to £Stg1400. The general opinion is that this price will diminish still further to the point where the free market price in the 1970's will equate with the fixed price. Opinion is somewhat divided on the future trend of the producer's price, but the weight of opinion tends to predict a stabilization of price at its present level although some forecast a possible increase.

Little statistical evidence is available to support these conclusions, but the expected increase in supply emphasises the danger of assuming prices in excess of the fixed price or in assuming further unofficial price rises. For this reason quoted prices for nickel products, obtained from mineral brokers, Derby and Co. and Tennant Trading, are closely related to the fixed price for nickel and are assumed to remain constant for the life of the project. The prices of individual nickel products are discussed in later sections of this report.

Royalties and Rental Fees. No allowance has been included in the study for process licensing fees or for state royalties on the production or sale of the product. A lease rental, at the rate of $1 per acre, however, has been included, assuming an area of 8000 acres.

Land Values. The study assumes that a mining and processing operation would render unfit for production 400 acres of land, valued at $10 per acre. The cost of land at Beauty Point for a stockpile area is also included and varies according to the area required.

Working Capital. There are three major components of working capital. These are:

1. the average inventory for raw materials, valued at delivered prices;
2. the average inventory for fuel oil, valued at delivered prices; and
3. the average inventory of finished product, valued at cost of production.

The study assumes the required average inventories would be:

a. raw materials - 2 months,
b. fuel oil - 1 month,
c. finished product - 2 - 2½ months (depending on frequency of shipments).

Taxation. Under the Income Tax Assessment Act (ITAA) mining companies are permitted certain taxation concessions which have a significant bearing on the viability of a mining operation. By election, mining companies are permitted to treat certain capital expenditures as allowable deductions in the year of acquisition or to depreciate the asset over the life of the mine. The economics of long-term projects which involve major capital outlays are sensitive to cash flow in the early years and therefore the choice of alternatives favours deductions that increase the operation's cash flow in early years. Listed below are the relavant taxation concessions and the scale of tax applicable for the mining and treating of nickel ore, which are used in this study:

Investment Allowance:

Section 62AA of the ITAA permits a special 20% deduction on new capital expenditure for a manufacturing plant in the concentration of a metal. This deduction, however, is not allowed if the company elects to treat capital expenditure as an allowable deduction in the year of acquisition. Not included in the allowance for concentration plants are expenditures for crushing, grinding, breaking, screening and sizing. This study therefore assumes that capital expenditure associated with the concentration plant - but excluding the above-mentioned items - qualifies for the allowance, which is deducted from assessable income in the first year of operation.

Allowable Capital Expenditure:

Under Section 122E capital expenditure for "prescribed mining operations" is tax deductible. Prescribed mining operations means mining operations on a mining property in Australia for the purpose of extracting minerals other than petroleum. The study therefore assumes that the expenditure for preparing the site, mining plant equipment, buildings, housing, and processing equipment which does not qualify for the investment allowance - namely crushing, grinding, breaking, screening and sizing - is fully deductible in the first year of income.

Depreciation:

Income Tax Order 1217, Part 1, permits mining machinery and plant to be depreciated at the rate of 11¼% declining balance. Capital items that are depreciated in the study are processing equipment that qualify for the investment allowance, and materials handling equipment associated with loading and unloading at the port site.

Partial Exemption of Income:

Section 23A of the ITAA permits mining companies producing prescribed metals or minerals to treat 20% of income, less allowable deductions, as exempt from taxation. The study therefore assumes that 20% of the net income, before tax for each year of operation, is fully deductible.

Section 80 - Losses:

Under Section 80 of the ITAA, losses incurred by a company in any of the 7 years preceding a year of income are allowable as a deduction in that year of income. However, if exempt income has been derived, the deduction shall first be applied against the exempt income and the remainder deducted from the assessable income. The study therefore assumes that cumulative losses are first reduced by the amount of exempt income and then deducted from assessable income to arrive at the project's taxable income.

Taxation Rate and Year of Payment:

Taxation payments are based on the prevailing corporate tax rate of 45% of taxable income. Payment of taxes is assumed to be in the year following receipt of income.

Housing. The study makes no major allowance for housing of mining and process workers other than $60,000 for key mining personnel, and $100,000 for plant managers.

Residual Value. The analysis assumes that the residual value of the equipment at the end of the project's economic life is equivalent to the undepreciated value or book value. Work capital and land values are not appreciated or depreciated over the life of the project and their monetary return at the end of the project is valued at their initial cost.

6.3 Pug Roast Acid Leach Process

6.3.1 Process Description

This process was developed by Sherritt Gordon Mines Limited and is described by Zubryckyj et al (1965) and Young et al (1965). It was developed to treat a range of ores occurring in laterite deposits which were not amenable to treatment by other processes such as smelting or the Nicaro process. A pilot plant of capacity 700 to 1000 lb of ore per day was operated for over 6 months but no commercial plants using this process have yet been built.

The ore is crushed to minus 20 mesh and pugged with concentrated sulphuric acid using 30% acid by weight of dry ore, pelletized and dried in a rotary kiln. The pellets are then fed into a multiple hearth roaster and the temperature raised at about 400°C. "Sulphation" of about 60% of the Ni and Co occurs. At 700°C or thereabouts the iron sulphate present decomposes releasing sulphur trioxide which reacts with the remaining nickel and cobalt; however, some decomposition of the Ni and Co sulphates may also occur. This decomposition of the nickel and cobalt sulphates is retarded by the presence of between 1.5 and 5% magnesium; the presence of magnesium at these levels is therefore advantageous.

The roasted pellets are discharged into quench tanks and the slurry formed is passed through agitated leaching tanks to a thickener, the underflow being washed in a conventional CCD system before being discarded. The overflow is pumped into a series of autoclaves where it is contacted with H_2S gas at 120°C and 70 psig. Finally, the sulphide slurry is exhausted into flash tanks, excess H_2S is removed and the sulphide precipitated and recovered by thickening and filtration.

The process uses a large quantity of sulphuric acid and H_2S. An acid plant and a hydrogen plant would almost certainly have to be provided and the acid and H_2S manufactured on site.

The flowsheet for the process is presented in Figure 1.

6.3.2 Product Recovery

This process is capable of achieving an 85% recovery. From a feed of 2200 tons of dry ore per day 19.5 tons of nickel per day could be recovered, and would be contained in 37 tons per day of a sulphide concentrate, giving an annual production of 12,210 tons. The concentrate is expected to be in pellet form and would probably have the following assay:

	%
Nickel	50 - 55
Cobalt	5
Iron	1 - 2
Sulphur	35 - 40

6.3.3 Process Applicability

The Beaconsfield deposit is a typical nickel-bearing laterite, formed by the weathering of ultrabasic rocks. This weathering process has given the deposit a characteristic profile.

The surface horizons consist essentially of the secondary iron oxides, containing a relatively small proportion of the nickel. The lower horizons, differing in detail from one area of the deposit to another, contain the bulk of the nickel, usually associated with serpentine or with clay minerals.

The pug-roast-leach process is adaptable in principle to treat either of these two broad ore types. However, as noted earlier, the presence of 1.5 to 5% of magnesium in the ore is essential for satisfactory selectivity in the sulphation reactions. The surface horizons are deficient in magnesium and for best results would require enrichment in this element. The altered horizons are relatively high in magnesium, some samples containing upwards of 15% magnesium, and these, if treated alone, would require a prohibitively large amount of sulphuric acid for satisfactory sulphation.

From the limited amount of analytical data available it appears that the average composition of the ore-bodies is approximately as follows:

	%
Ni	1.06
Co	0.06
Cr	0.38
Mg	4.9
Fe	20.6
SiO_2	39.9
CaO	4.0
Al_2O_3	6.3

These figures indicate that the overall grade of the deposit, particularly with respect to magnesium, is within the limits recommended for treatment by the process. As the mineralogy is consistent with that of the ores on which the Sherritt Gordon pilot-scale work was carried out, there is every reason to believe that from the technical viewpoint the process should be readily applicable to the Beaconsfield ore.

The least desirable feature of this ore is its relatively high content of silica. This could be expected to cause considerable difficulty in a simple acid leach process. However, the present process, employing a roasting stage and a water leach, should not be subject to the same limitation. While silicates would be decomposed by the concentrated sulphuric acid in the pugging stage, the subsequent roasting would dehydrate all the silica quite effectively, and no solid-liquid separation problems or product contamination, would be expected.

6.3.4 <u>Capital Cost Estimates</u>

Total capital expenditure for the production of 12,210 tons per year of nickel sulphide is estimated at $25,374,000. Listed below are the major cost components of the expenditure:

	$,000
Mining capital expenditure	940
Process equipment	18,000
Auxiliary equipment and loading facilities	2,900
Working capital	3,274
Housing	160
Land	100
	25,374

Mining Capital Expenditure. Capital cost estimates for mining are consistent for each processing alternative and are listed in Appendix B.

Process Equipment. Processing cost estimates have been derived from the paper by Young et al, (1965), who estimates that the capital requirement for a plant processing 20 million pounds of nickel per year in 1964 would be $US40,000,000. This figure, however, includes cost of mining equipment, port facilities, township and auxiliary plant. These items are estimated at $US12,600,000, leaving a cost of $US27,400,000 for the concentrating process. This figure can be scaled down to $US22,000,000 ($A18,000,000) for the estimated production of 19.5 tons per day, using the six-tenth factor and afterwards adjusting for cost index increases.

Auxiliary Equipment and Loading Facilities. Cost estimates for auxiliary equipment and loading facilities are discussed below:

Sulphuric Acid Plant:

The capital cost of an acid plant to produce 221,000 tons of acid is estimated at $1,900,000. This figure is derived from data given by Bauman (1964) and is confirmed by scaling published costs of plants recently erected in Australia and New Zealand.

Hydrogen Plant and Hydrogen Sulphide Generator:

Capital costs for these two items of equipment are estimated at $600,000 based on figures given by Twist and Sagar (1965) and Charlesworth and Schmidt (1965).

Port Loading Facilities:

Large bulk shipments of raw materials would necessitate the installation of fast off-loading equipment at the wharf site and an amount of $400,000 has been allowed.

Working Capital. Listed below is the cost build-up for working capital:

	$,000
Raw materials:	
Average inventory based on 2 months' supply valued at delivered prices	647
Fuel Oil:	
Average inventory based on 1 month's supply valued at delivered prices,	157
Finished Product:	
Average inventory based on 2.5 months' stock valued at cost of production,	2470
	3274

Housing. The cost of housing is estimated at $160,000 and is constant for each process.

Land Values. This process requires large shipments of sulphur, and a sizable storage area at the off-loading point would be necessary. Cost of land, therefore, is $96,000 for the reclamation and development of a stockpile area adjacent to the wharf site, plus $4,000 for rendering land unfit for production at the mine site.

6.3.5 Operating Expenses

Total operating expenses, including mining, processing and transportation, but excluding depreciation and interest payments, are estimated at $A8,308,000 for the first year of operation, and are itemized in Appendix C. The single most important cost component for the processing method is sulphuric acid. The process requires 221,000 tons per annum of sulphuric acid, which is estimated to have an on-site production cost of $15.57 per ton or a total annual value of $3,443,000.

6.3.6 Market, Product Price and Estimated Revenue

Currently, the best market for nickel sulphide is Japan. Quotations from Derby and Co. indicate that nickel sulphide assaying 50% nickel ore is priced at 61.59 A¢ per pound of nickel contained, cif Japan, This price is based on International Nickel's official nickel price of 94 US¢ per pound and is to be varied 50% up or down on any variation to the official price.

Assuming stabilisation of the fixed price at its present level, the cif price of contained nickel is assumed to be 61.59 A¢ per pound, resulting in an annual revenue of $A8,878,000.

6.3.7 Profitability Analysis

Key economic indicators for the project utilizing the Pug Roast Acid Leach process are listed below:

	$m
Capital cost	25.4
Annual revenue	8.9
Annual operating expenses - excluding depreciation and interest payments	8.3
Net profit - before depreciation, interest, taxation, and deduction allowances	0.6
First year's depreciation	2.3
Project's DCF rate of return	- %
Project's payout period	- year

The annual production of 12,210 tons of nickel sulphide would incur operating expenses, excluding depreciation and interest, of $8.3 million and yield only $8.9 million in revenue. Annual net profit would be of the order of $0.6 million.

The project's high operating expenses relative to revenue receipts renders this process financially uneconomic. Essentially, the nickel content of the ore at Beaconsfield is too low a grade to be economically treated by this process.

6.4 The Nicaro Process

6.4.1 Process Description

The Nicaro process was developed and the first plant commissioned during World War II to exploit the large deposits of nickel occurring in Cuba. The plant was closed down in 1947 but was re-opened in 1952. Two years later production was increased from 80,000 lb of nickel per day to 140,000 lb of nickel per day. In 1960 the plant was taken over by the Castro regime and no further details of its operation are available. The process was described in detail by Lutjen (1954) and some modifications were described by Alonso and Daubenspeck (1960). The plant is also described by Boldt (1967). The flowsheet for the process is presented in Figure 2.

The Cuban deposits consist of a serpentinitic base which contains less than 0.2% nickel. Immediately above this is a band of altered serpentine which has a nickel content of 1.8 to 2%. This is a transition zone and is surmounted by a layer of limonite containing an average of 1.25% nickel and 1.5% chromium.

The nickel grade in the limonite increases with depth, and the first stage in mining is to scrape off the top layers down to a cut-off grade of 1.1% nickel.

The orebody has a great variation in composition, and has to be carefully mined so as to provide a constant feed to the process of 1.37% nickel and 38% iron. The orebody has been extensively drilled at 50-ft intervals to define blocks of ore of reasonably constant composition. Ore is mined simultaneously from several blocks according to a carefully calculated ratio, worked out in advance, so that the assay of the mixture fed to the process is kept fairly constant. Mining is by dragline to trucks and the ore is transported 11 miles by rail to the process plant stockpile.

The ore in the stockpile is further blended to the required feed grade of 38% iron and 1.37% nickel and is passed through a primary crusher into silos. It is then conveyed from the silos to a rotary kiln drier before being ground to minus 200 mesh in a ball mill. The ground ore is roasted in multiple hearth furnaces in a reducing atmosphere. The temperature is controlled at 1280°F on the main firing hearths, control being critical. Firing is by fuel oil and the reducing atmosphere is maintained at a ratio of H_2/H_2O of 1:1 by the addition of the gas from partially combusted fuel oil. The ratio of H_2/H_2O is important as there is only a very narrow range in which nickel will be reduced to the metal whilst ferric iron is reduced to the magnetic oxide.

After roasting, the ore must be cooled to 250°F as quickly as possible in a reducing or inert atmosphere to prevent the formation of insoluble nickel compounds by recombination of nickel oxide, magnesia and silica. These compounds do not form in oxidation at lower temperatures. The apparatus used at Nicaro consisted of a double-shelled rotary kiln, filled with water and partially immersed in a water-bath. This method has not been entirely satisfactory and more modern alternative equipment would be recommended for Beaconsfield.

The "cooled" ore is quenched by passing it into a solution of ammonia and ammonium carbonate. The density of the slurry thus formed is controlled at 25% solids and is passed into an aerator where the nickel is oxidized and dissolved by the ammonia-ammonium carbonate solution. The aerated slurry is then passed into leaching tanks and separated from the pulp. The pulp passes to two more stages of aeration and leaching before it is passed to a conventional CCD washing circuit, where the last traces of soluble nickel are removed. The pulp then is heated and the dissolved ammonia is recovered by steam stripping before it is passed to a tailings dam.

The strong liquor is passed through another aerator to precipitate any remaining iron, which is removed by filtering. The liquor then is passed through a still, in which the ammonia is stripped out with steam, and recovered in a series of condensers and absorbers.

As the ammonia is removed in the stripping tower, nickel and cobalt carbonates are formed which are precipitated. The slurry is pumped via a flash tank to a thickener. The overflow is passed through a filter and the clarified liquor is rejected. The underflow is filtered and the cake is dried and calcined in a rotary kiln to yield a nickel oxide powder. This powder can then be sintered to remove more of the oxygen and give a high purity product.

6.4.2 Product Recovery

Lutjen (ibid) reported that a recovery of 75% of the nickel was achieved at Nicaro, and Alonso and Daubenspeck (ibid) described modifications to the process improving this to about 80%. Manson (1960) reported that a laboratory experiment aimed at the extraction of nickel from Beaconsfield ore by the Nicaro process achieved a recovery of 67.7%. He also indicated that this could probably be improved upon by more carefully controlled reduction of the ore. In view of these reports it seems reasonable to expect that a recovery of 75% or better could be achieved at Beaconsfield.

The composition of the dry ore treated at Nicaro is shown below, together with that of the sintered nickel oxide product:

Dried Feed, %		Sintered Product, %	
Nickel	1.4	Nickel	88
Cobalt	0.1	Cobalt	0.7
Iron	38	Oxygen	7.5
Magnesia	8	Iron	0.3
Silica	14	Silica	1.7

The ore at Beaconsfield, has an average composition which is shown in tabular form in Section 6.3.3 above. The differences are principally in the amounts of iron and silica present, and these would not be expected to influence the product composition greatly. The cobalt values, in particular, should be closely comparable.

From a feed of 2200 tons of dry ore per day at Beaconsfield it is estimated that a recovery of 17.2 tons of nickel per day (75%) could be achieved. The nickel would be contained in 19.5 tons per day of a nickel oxide sinter with an annual production of 6435 tons. The nickel oxide would be crushed into a suitable size fraction (e.g. minus 1-inch, plus ¼-inch mesh), the oversize and undersize being recirculated through the sintering machine. The sintering process tends to partially reduce the nickel oxide to the metal, resulting in a high grade product which would have an assay similar to the product produced in Cuba.

6.4.3 Process Applicability

Boldt (ibid) reports that there are only two processes for extracting nickel from oxide ores by hydrometallurgy which have been used commercially. These are the Nicaro process and a sulphuric acid leach process at elevated temperature and pressure. The choice between these two processes is based on a knowledge of the gangue materials in the ore. The Nicaro process is applicable to a material which contains a high proportion of magnesia and silica, which are acid soluble, and would thus prohibit the use of the acid leach process due to a high consumption of acid. The acid leach process has been applied to a deposit at Moa Bay, Cuba, which contained a very low proportion of magnesia and silica. The higher operating cost for this process is offset by its higher recovery rate of 95%.

The ore at Nicaro contains 8% magnesia and 14% silica, whereas that at Moa Bay contains only 1.7% magnesia and 3.7% silica. As the Beaconsfield deposit is estimated to contain 8% magnesia (5% magnesium) and 40% silica, the Nicaro process is selected as being the more suitable of the two. The mineralogy of the three deposits differs somewhat in detail but is basically similar and gives no evidence suggesting difficulty in applying the Nicaro process to the Beaconsfield ore. The limited amount of experimental work carried out on this ore also confirms the applicability of the process.

6.4.4 Capital Cost Estimates

Total capital expenditure for the production of 6435 tons per year of nickel oxide sinter is estimated at $19,327,000. Listed below are the major cost components of the expenditure:

	$,000
Mining capital expenditure	940
Process equipment and loading facilities	16,400
Working capital	1,793
Housing	160
Land	34
	19,327

Process Equipment and Loading Facilities Estimate. The Nicaro process operating in Cuba was designed to treat 7000 tons of dry ore per day. In 1954 the total capital investment, according to Lutjen, was $US90,000,000. This investment, however, included appropriations for mine development, mining equipment, township, railroad, rolling stock, working capital and a port. Because of the uncertainty of these costs and the difficulty in updating overseas investment figures, the cost of the Nicaro processing equipment for treating 2200 tons a day has been developed by deriving individual cost estimates for each piece of equipment as defined in the Lutjen flowsheet. On this basis the total capital requirement for the process and loading facilities is estimated to be $16,400,000, which is reported in detail in Appendix D.

Working Capital Estimates. Working capital associated with this project is estimated on the following basis:

	$,000
Raw Materials:	
Average inventory based on 2 months' supply valued at delivered prices,	72
Fuel Oil:	
Average inventory based on 1 month's supply valued at delivered prices,	69
Finished Product:	
Average inventory based on 2 months' stock valued at cost of production,	937
Initial inventory of 5500 tons of ammonia valued at delivered prices,	715
	1793

Land Value Estimates. Only 6000 tons per annum of material in bag form will be exported through Beauty Point, and supplies of raw material will be restricted to small shipments of anhydrous ammonia, which would be off-loaded by pipeline at Beauty Point to storage units at the stockpile area. The cost of reclaiming land and developing a small site for a stockpile area is estimated at $30,000, added to which is $4,000 for the destruction of property at the mine site, making a total land value of $34,000.

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6.4.5 Operating Expenses

Total operating expenses for the first year of operation, including mining, processing, and transportation expenditures, but excluding depreciation and interest payments, are estimated at $4,402,000 and are project to rise steadily over the life of the project. The project's operating expenses are itemized in Appendix E. It can be noted from the itemized costs that operating expenses are evenly distributed over several items. This is highly advantageous in that it minimizes the possibility of any one cost adversely affecting the project's profitability as a result of an unfavourable variance.

6.4.6 Market, Product Price and Estimated Revenue

Markets for nickel oxide sinter exist in Japan and Great Britain. Separate quotations were received from Derby and Co. and Tennant Trading for material assaying 80% nickel. Derby has provided an indicative cif price Japan of $US1.05 per pound of contained nickel, less 15% for duty. This reduction brings the price back to 89 US¢ or 80.4 A¢, which is just below the producers' price of 94 US¢ per pound. Derby has pointed out, however, that the price of this material can be adversely affected if the cobalt content exceeds 1%. Tennant's quotation is based on a market demand in Great Britain. According to Tennant, International Nickel are the only known suppliers of nickel oxide sinter in Europe and do not publish their prices. The British Steelworks, however, has advised Tennant that the price of nickel oxide sinter cif Great Britain is at a "slight" discount on the official producers' price for refined nickel which is £Stg902.

Assuming that the arbitrary figure of 6% represents the slight discount on producers' price, the cif price would be approximately £Stg850. On a per-pound basis, and expressed in Australian currency, this is equivalent to 81¢ and equates with the cif price Japan quoted by Derby and Co.

On this basis, the price of nickel oxide sinter is assumed to be 81¢ per pound of nickel contained and the anticipated market is Japan. The total annual revenue from a production of 6435 tons containing 5676 tons of nickel is therefore $10,300,000 and is held constant for the life of the project.

6.4.7 Profitability Analysis

Key economic indicators are listed below:

	$m
Capital cost	19.3
Annual revenue	10.3
Annual operating expenses - excluding depreciation and interest payments	4.4
Net profit - before depreciation, interest, taxation and deduction allowances	5.9
First year's depreciation	1.8
Project's DCF return	20%
Pay-back period	4 years

The Nicaro process is capable of producing 6435 tons of nickel-oxide sinter with an annual value of $10.3 million. Total annual operating expenses are estimated at $4.4 million in the first year of operation and are projected to rise gradually. The project's net profit before depreciation, interest payments, taxation, and deductions for investment allowances, allowable capital expenditure, and exempt income is $5.9 million per year. Depreciation in the first year of operation is $1.8 million and declines over the life of the project at the annual rate of 11½% on a decline balance.

For a total capital outlay of $19.3 million, this project would yield a 20% DCF rate of return on investment and achieve a 4-year pay-back period, which indicates that the Nicaro process has strong economic potential for the development of the deposit. The DCF rate of return calculation is presented in Tables 14 and 15.

6.5 Pyrosulphidizing Process

6.5.1 Process Description

This is a hypothetical process which is believed to be technically feasible, but it must be emphasized at the outset that each stage of the process must be proved experimentally before any of the figures in the estimates can be regarded as anything better than indications of orders of magnitude.

The process envisaged converts the nickel present in the ore into a nickel sulphide, which may be recoverable from the gangue by familiar techniques, e.g. froth flotation. However, since the nickel is present in the laterite in a dispersed form, disseminated throughout the ore and not present in any identifiable discrete particles, it is clear that direct conversion in a simple reaction step is unlikely to be practicable. The first step in the process therefore would be a selective reduction to metallic nickel, as practised at Nicaro.

Following this reduction under non-slagging conditions, the nickel, together with the cobalt and possibly some iron, will be present in a finely divided and presumably highly reactive form. It should then react readily, especially at elevated temperature, with elemental sulphur to give nickel sulphide. Cobalt sulphide would also form, and probably some iron would be converted to pyrrhotite. This much of the process, in the absence of any experimental work, would appear to have at least a moderate chance of being operable.

Vanyukov, Vanyukov and Yudina (1955) reported the sulphidation of oxidized nickel ores by gaseous or solid sulphidizers. Ferric oxides were reduced to ferrous oxides by heating with solid or gaseous reducing agents for 30 to 45 minutes at 800 to 1000°C. The reduced ore was then sulphided for 45 to 60 minutes at 1000 to 1100°C. The particle size of the nickel-rich sulphides was increased by precipitation for 2 to 3 hours at 1200°C. After this treatment froth flotation was an effective means of separation of sulphides from gangue material.

Vanyukov (1956) reported very high conversion (95%) of oxidized nickel to nickel sulphide in 5 minutes at 800 to 900°C. A particle size of 0.3 to 0.5 mm was achieved after 2 hours of heating at 1200°C.

Khan and Zarakhani (1959) reported laboratory and pilot-scale work on reduction-sulphidation. Coke was used as the reducing agent and pyrite as the sulphidizer. Nickel was recovered by flotation, followed by magnetic separation. Ore containing 0.8 to 1.5% nickel yielded a concentrate containing 7 to 8% nickel with 88 to 95% extraction. A richer concentrate containing 11 to 14% nickel was achieved, but with only 75 to 82% extraction.

It is probable that the use of sulphur instead of pyrites as the sulphidizer would increase the nickel content of the product.

The flowsheet for the process is presented in Figure 4.

The optimum conditions cannot be predicted with certainty, but it seems logical to assume reducing conditions similar to those used in the Nicaro process, followed by a short period of contact at high temperature between the reduced ore and gas contained sulphur vapour, preferably in the absence of large amounts of inert gases. The particle size of the sulphides produced cannot be estimated with any certainty, but it is assumed here that the particles would be sufficiently coarse to be amenable to separation by froth flotation. The concentrate produced by the flotation cells would be filtered and then kiln dried ready for sale.

6.5.2 Product Recovery

This process could recover 80% of nickel treated, which would yield 18 tons of nickel per day from a feed of 2200 tons of dry ore. The nickel would be contained in 92 tons a day of a sulphide concentrate, which would have the following approximate assay:

	%
Nickel	20
Iron	10
Sulphur	30
Silica	40

Annual production of the sulphide concentrate would be 30,360 tons in powder form and would be shipped in bulk.

6.5.3 Process Applicability to Beaconsfield Ore

As explained above, this process is hypothetical, but is believed to have a good chance of being technically feasible. If the economics of the process are of interest, based on the rough figures in the following estimates, experimental work would have to be performed to prove that the process is operable, and to obtain operating data. This data could then be used as a basis for more accurate estimates of capital and operating costs.

6.5.4 Capital Cost Estimates

Total capital for the production of 30,360 tons of nickel sulphide per year is estimated at $10,968,000. Listed below are the major cost components of the expenditure:

	$,000
Mining capital expenditure	940
Process equipment and loading facilities	9,000
Working capital	818
Housing	160
Land	50
	10,968

Process Equipment and Loading Facilities Estimates. Published information on the capital cost of pyrosulphidizing is non-existent. However, this process closely resembles the Nicaro process up to and including the reduction furnace. The study therefore assumes that the purchase cost of comparable equipment is the same as estimated for the Nicaro process. The remaining plant, consisting of coolers, quench tanks, flotation cells, filter, drying kiln and loading facilities is estimated to cost an additional $500,000. This gives a total purchase cost of $3,261,000. As the pipe work required for this process is a good deal less than required for the Nicaro process, only a 27% loading factor has been allowed for piping. Loading for other installations remain as is for the Nicaro process. The total installation loading is therefore 177% or $5,770,000, making a total capital cost of the installed plant and loading facilities of $9,031,000.

This estimate compares quite closely with the Torco process, which is an existing process analogous both in size and principle with pyrosulphidizing. The Torco process is currently operating in South Africa, with a capacity of 2000 tons per day. Reportedly, the installed cost of this plant is £Stg3.5 million which is equivalent to just over $A9 million (before devaluation of sterling).

Working Capital Estimates. Working capital estimates have been derived on the following basis:

	$,000
Raw Materials:	
Average inventory based on 2 months' supply valued at delivered prices,	97
Fuel Oil:	
Average inventory based on 1 month's supply valued at delivered prices,	55
Finished Product:	
Average inventory based on 2 months' stock valued at cost of production,	666
	818

Land Value Estimates. Land value of $50,000 is based on $4,000 for destruction of property at mine site and $46,000 for the establishment of a stockpile area. It is assumed that 30,000 tons per year nickel sulphide would be transported in bulk form, with an average lift of 5000 tons, or six shipments per year. An area close to the wharf would therefore be required to stock up to 5000 tons of ore plus a small quantity of raw materials. Ship loading and unloading would be by grab-haul.

6.5.5 Operating Expenses

Total operating expenses for the first year, including mining processing and transportation but excluding depreciation and interest payments, are estimated at $3,551,000 and are expected to rise steadily over the life of the project. An itemized account of the operating expenses is listed in Appendix F. Operating expenses for the Pyrosulphidizing process are similar to the Nicaro process in that costs are evenly distributed among several items, and therefore an unfavourable cost variance in any one item will not significantly affect the project's profitability.

6.5.6 Market, Product Price and Estimated Revenue

As previously mentioned in this report, the best market for nickel sulphide is Japan. Tennant Trading has quoted an indicative cif price of 55.25 US¢ per pound of nickel contained for nickel sulphide assaying 13% nickel. This is thought to be the prevailing price for nickel sulphides from Western Australia. Tennant points out that the 55.25 US¢ quotation is a base price and is subject to bonuses and penalties to the order of 0.2¢ per pound per percent plus a 2¢ or 3¢ bonus for ease of separation. The price is also subject to a premium that is variable annually in line with the prevailing Japanese import duties on nickel metal. For 1969 the premium would be 4¢, but this premium is expected to diminish over the next 5-year period.

According to Tennant, the value of a 20% nickel sulphide would be in excess of the base price. The extent to which the price would exceed the base depends upon ease of separation and the material's physical properties. In this analysis the price of 20% nickel sulphide is assumed to be 57.46 US¢ or 51.77 A¢, which is a 2¢ premium on the base price. Quite possibly, the actual price could exceed this amount, but in the absence of sufficient evidence a conservative price is assumed. Total annual revenue based on the price of 51.77 A¢ per pound of nickel contained would, therefore, be $6,880,000 and this estimate is held constant for the life of the project.

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6.5.7 Profitability Analysis

Key economic indicators are listed below:

	$m
Capital cost	11.0
Annual revenue	6.9
Annual operating expenses - excluding depreciation and interest payments	3.6
Profit before depreciation, interest, taxation and deduction allowances	3.3
First year's depreciation	1.0
Project's DCF rate of return	22%
Project's payout period	5 years

Annual revenue from the production of 30,360 tons per year of nickel sulphide is estimated at $6.9 million. Total operating expenses are $3.6 million per year and are projected to rise gradually over the life of the project. Annual net profit before depreciation, interest taxation and deductions for investment allowances, allowable capital expenditures and exempt income is $3.3 million. Depreciation in the first year of operation is $1.0 million and declines over the life of the project at the annual rate of 11¼% on a declining balance.

Based on an after tax cash flow, the project's DCF rate of return is 22% on an investment of $11 million and the payback period is 5 years. These economic factors indicate that the project is financially attractive based on the aforementioned assumptions. The DCF rate of return calculation is presented in Tables 16 and 17.

6.6 The Ugine Process

6.6.1 Process Description

The Ugine process is employed by the Hanna Nickel Smelting Company, Oregon, USA and is described in detail by Coleman and Vedensky (1960 and 1961) and Boldt (ibid). The process flowsheet is presented in Figure 3. The process has been in use since 1954 and produces a ferro-nickel alloy from an ore containing 1.3% nickel.

The ore is mined by open-cut and transported to the processing plant where it is crushed and screened. The ore is up-graded to 1.4% nickel by rejecting a coarse hard fraction of very low grade at this stage. The ore is then blended and dried in rotary kilns. The soft high-grade ore breaks up by attrition in the kilns and a further set of screens is set to reject the hard lumps as oversize. This oversize represents 20% of the total ore, but only 10% or less of the nickel. Its removal up-grades the remaining ore to 1.65% nickel.

The ore is then preheated in calcining kilns or multiple hearth furnaces and is passed into feed hoppers above the electric melting furnaces. The furnaces are fed continuously except during pouring and are operated at 700 volts. The molten ore is tapped into a ladle, which is one of two reaction ladles. It is mixed with ferro-silicon as it is poured from one ladle to the other and is also contacted with some ferro-nickel left from the previous mix. The nickel is reduced to ferro-nickel during the mixing, which occurs during seven pours from one ladle to the other. The mix is then allowed to stand, to permit the metal to coalesce, before removal of most of the slag by decantation.

The ferro-nickel in the furnace remains with the original seed metal for up to eight cycles, after which it is removed and transferred to a refining furnace. This removes phosphorus by addition of slagging materials, and the ferro-nickel is deoxidized by addition of a small quantity of ferro-silicon. The ferro-nickel is then cast into ingots, which are cleaned in a tumbler mill.

6.6.2 Product Recovery

The Ugine process is capable of recovering 80% of nickel treated. Production would therefore be 18.0 tons nickel per day from a feed of 2200 tons dry ore, and the nickel would be contained in 40 tons per day of a ferro-nickel, assaying approximately 46% nickel, 52% iron.

6.6.3 Process Applicability

This process appears to be technically unsuitable to treat the Beaconsfield nickel deposit. The composition of the ore at Beaconsfield is such that the melting point is likely to be relatively high. Some parts of the orebody would be quite refractory, but an average composition of material containing in excess of 1% nickel is approximately 18% iron, 40% silica, 4% alumina, and probably 6% magnesia. In either an oxidizing atmosphere or a reducing atmosphere, it seems likely that the high proportion of silica would give rise to a viscous melt, and then only at temperatures of 1500°C or above. This in itself would make operation of the process difficult. As a second limitation, temperatures of 1700°C are employed in the process at Hanna Nickel, but even at this temperature the Beaconsfield ore, and any slag formed from it, would in all probability be quite viscous.

Besides the technical limitations of the Ugine process, there are other factors which weigh heavily against its economic viability. Power costs are of the order of $4 million per year. Moreover, the process requires ferro-silicon as a reductant, which is not manufactured in Australia and must be imported from overseas at an annual cost of $2 million.

These technical and economic limitations demonstrate that direct smelting is not feasible and it is therefore eliminated from consideration at this point.

6.7 Choice of Alternatives

Although the analyses indicate that two processes - the Nicaro and the Pyrosulphidizing - could be financially attractive, the Nicaro process is the more promising of the two despite its higher capitalization requirement and apparent lower return on investment. This conclusion is based on the following points:

1. The Nicaro process is an established process for treating lateritic nickel and its technical capability is more predictable. The Pyrosulphidizing process, on the other hand, is virtually untried, and the assumptions underlying its analysis must be treated with a greater degree of uncertainty.

2. The Nicaro process would require less developmental work and could be brought into operation within a 3-year period. Pyrosulphidizing would require considerable tests and development work, and production could be delayed 4 or 5 years.

3. Product output from the Nicaro process of approximately 6000 tons per year and the importation of raw material required to treat the ore could be handled adequately across the existing wharf facilities at Beauty Point. However, the movement of 30,000 tons per annum of sulphide concentrate from the Pyrosulphidizing process presents handling problems at the wharf site, and could even require construction of a special jetty.

4. The Nicaro process is economically more attractive than the Pyrosulphidizing process, even though pyrosulphidizing shows a 22% return as opposed to 20% return. This incongruity results from a disparity in levels of investment. In the case of Pyrosulphidizing, the return of 22% is related to $11 million investment; whereas the 20% return on Nicaro is related to an investment of $19.3 million. This incongruity can be resolved by calculating how much each project would add to the company's net worth. For this test the two cash

flows are discounted using the sponsor's cost of capital rate, i.e. 10% to determine their net present value (NPV). The project having the highest NPV should be selected because it would result in a greater increase to the company's worth. On this basis the Nicaro process has a NPV of $11,400,000, compared with $6,900,000 for the Pyrosulphidizing alternative, which demonstrates the financial superiority of the Nicaro process.

5. The attractiveness of the Nicaro process as a method of treating nickel ore is in part demonstrated by International Nickel and Sherritt Gordon's recent experimentation on this process. Although it is not officially known, apparently these two organizations either separately or jointly are considering the Nicaro process or an adaptation of it as a possible method of treatment.

6.8 Sensitivity Analysis

The economic and technical analysis of the Nicaro process is based on assumptions of varying certainty. It is therefore important to identify the major variables in the evaluation and the degree to which each affects the project's rate of return.

Critical to the analysis of this project are:

1. the amount of recoverable ore,
2. the process recovery rate,
3. the expected product price,
4. the projected operating expenses; and
5. the capital estimates.

Variations in these estimates from those used in the study and their effect, in isolation, upon the project's 20% rate of return are listed in the following tabulation and discussed below:

Base Case Assumption		Variable	Adjusted Rate of Return %
Orebody:	11 m tons	7 m tons	9
Orebody:	11 m tons	9.5 m tons	15
Process recovery rate:	75%	65%	15
Nickel price:	81¢/lb	70¢	15
Operating expenses:	$ 6,190,000	20% increase	16
Process capital:	$16,400,000	15% increase	17

Recoverable ore. Geological evidence indicates that reserves of 11 million tons of nickel ore containing 1.03% nickel could be present, and the evaluation has been based on the assumption that this amount of ore could be recovered. The probability does exist, however, that the deposit could be less, and even as low as 7 million tons. Assuming that only 7 million tons can be recovered and treated, the DCF rate of return based on an appropriately sized operation to treat 1400 tons per day drops from 20% to 9%. This level of return is obviously unattractive since cost of capital is estimated at 10%. However, if recoverable reserves were of the order of 9.5 million tons, the process could be sized to treat 1900 tons per day, which would raise the project's DCF rate of return to above the acceptable level of 15%.

Recovery Rate. The study assumes that 75% of the nickel treated could be recovered by the Nicaro process. This is a realistic estimate based on the present operation in Cuba, but it could vary up or down depending upon the chemistry of the Beaconsfield ore. Assuming that the recovery rate were only 65%, which is considered unrealistically low for the Beaconsfield ore, the annual revenue would drop $1,379,000, and the project's rate of return would fall from 20 to 15%.

Price Estimates. The price of nickel oxide sinter has been estimated and used in this study at 81¢ per pound of contained nickel or £Stg850 per ton. As explained, this closely relates to the producers' price for refined nickel, which is expected to remain firm in the foreseeable future. Statistical evidence to support this conclusion is limited, and the possible decline of prices should not be discounted. Assuming that the estimated price is not obtainable, and that the value of the product drops to 70¢ per pound (£Stg732) annual revenue would decrease to $1,380,000, resulting in a 5% decline in the rate of return.

Operating Expenses. As previously mentioned, the operating expenses for the Nicaro process are evenly dispersed over several cost items, which minimizes the possibility of only one item adversely affecting the project's profitability. However, the possibility must be considered that operating expenses in total may exceed their estimates. Assuming that operating expenses rise 20% above estimate, net profit before tax for the project would fall approximately $1,200,000 a year. This would have the effect of dropping the project's rate of return to just under 16%.

Capital Estimates. Capital cost estimates for this project could possibly vary 15% either way. Assuming that the capital cost for processing were 15% higher, total capital requirement for the project would increase $3,000,000 to $22,327,000 and the DCF rate of return would fall 3% to 17%.

6.8.1 Summary

The results of the sensitivity analyses indicate quite clearly the following points:

1. The single most important variable is deposit size.
2. A 7-million ton deposit is not sufficient to support an economic operation.
3. Reserves in excess of 7 million tons but less than 9.5 million could only be considered marginally attractive and could not justify the risk of large sums of capital.
4. The existence of 9.5 million tons places the project in a financially attractive region, but estimates would need to be more soundly based and substantiated before final commitment.
5. A very profitable mining and processing operation could be developed on the basis of 11 million tons, if the major underlying assumptions were to prove correct.
6. If 11 million tons do exist, no one adverse variance would render the project unacceptable and only the combined effect of the two most unfavourable variances could reduce the project to an unprofitable state.

In summary, and based on information presently available, there is sufficient reason to believe that a profitable venture could be established at the Beaconsfield mine site. This conclusion would, however, require substantiation with more definitive research and analysis, which is discussed in the following section.

7. REQUIRED DEVELOPMENTAL WORK

Whilst analyses of the project, based on the adoption of the Nicaro process, have shown that there is scope for a fair return on investment, sensitivity tests on critical assumptions indicate that the profitability of the venture could change appreciably with any significant change in estimate areas and, therefore, it is essential to determine more accurately:

1. the market value of nickel oxide sinter,
2. the operating expenses and capital costs associated with the total project, and
3. the amount and grade of recoverable ore.

To obtain this information and to reappraise the viability of the project, the following programme of research, development, and exploration is recommended:

1. Contact the following persons and organizations to obtain the information listed below:

   (a) Research and Development Division,
   Sherritt Gordon Mines Limited,
   Fort Saskatchewan, Alberta, Canada.

   (b) The International Nickel Co. Inc.,
   67 Wall Street,
   New York, NY 10005.

   (c) Office of the Administrator,
   General Services Administration,
   General Services Building,
   18th and F Street, NW,
   Washington, DC.

   (d) G.P. Lutjen, [1]
   Associate Editor (1954),
   Engineering and Mining Journal,
   330 West 42nd Street,
   New York, NY.

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1. Author of "Nicaro Proves Lateritic Nickel Can Be Produced Commercially", June, 1954).

i. Data on the latest technical developments on Nicaro-type processes.

ii. Operating cost data and estimates of capital expenditure for a plant treating 2200 tons of ore per day.

iii. Process flow sheet and plant specification.

iv. Process licence agreements and royalty fee.

v. The existence of any Nicaro-type pilot plant which could test samples of ore to determine:

process applicability,
optimum operating conditions,
nickel recovery rate,
product yield,
product assay.

2. Delineate, accurately, the recoverable ore-reserves and the grade of ore in the Beaconsfield area. A more comprehensive pattern of diamond drill holes should be completed in the Barnes Hill, Mount Vulcan and Scotts Hill areas. Because of the lateral variability in thickness and grade of the ore-bearing horizon, the "Flat" area should not be discounted as a source for potential nickel reserves on the basis of the unfavourable results obtained in this area to date.

3. Submit ore samples for pilot-scale test work, and reappraise revenue estimates based on results obtained.

4. Reassess process operating costs and capital cost estimates derived from flow sheet and plant specifications.

5. Re-evaluate mining costs if selective mining and ore blending is required before treatment.

6. Update all costs associated with the venture and re-evaluate the project's economic potential.

A Nicaro-type pilot plant does not perhaps exist, or is unavailable for test work, in which case a detailed programme of experimentation should be developed to provide the necessary operating data. This would involve laboratory work and pilot studies on a scale not less than 1000 lb of ore per day.

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Mobil Oil Australia Ltd, Melbourne	-	Fuel oil costs.
Imperial Chemical Industries of Australia and New Zealand Ltd, Melbourne	-	Ammonia costs.
Tennant Trading (Aust) Pty Ltd	-	Finished product prices.
The British Phosphate Commissioners, Melbourne	-	Elemental sulphur prices.
Westralian Farmers' Transport Pty Ltd, Melbourne	-	Freight rates.
The Australian National Line, Melbourne	-	Freight rates.

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APPENDIX A

SUMMARY DRILL LOGS

In the accompanying tabulation each diamond drill hole is listed together with the horizons of chemical weathering intersected by that drill hole. The range in nickel values for each of the weathered horizons is presented.

Drill Hole	Horizon	Depth Range ft	Ni Assay %
1	Laterite:		
	Ferruginous red zone	0 - 2	0.11
	Limonitic yellow zone	2 - 7	0.11
	Mottled zone	7 - 16	0.6 - 1.73
	Serpentinite:		
	Bleached zone	16 - 51	0.52 - 1.86
	Fresh zone	51 - 59	0.48 - 0.67
2	Laterite:		
	Limonitic yellow zone	0 - 5	0.19
	Mottled zone Serpentinite: Transition zone	5 - 10	0.41
	Bleached zone	10 - 42	0.25 - 0.38
3	Laterite:		
	Pisolitic zone	0 - 5	0.09
	Ferruginous red zone	5 - 15	0.1 - 0.3
	Limonitic yellow zone	15 - 18	0.48
	Mottled zone	18 - 43	0.41 - 1.18
	Serpentinite:		
	Transition zone	43 - 48	1.13
	Bleached zone	48 - 53	0.70
	Fresh zone	53 - 66	0.21 - 0.29
4	Laterite:		
	Pisolitic zone	0 - 5	0.11
	Ferruginous red zone	5 - 10	0.17
	Limonitic yellow zone	10 - 15	0.19
	Mottled zone	15 - 20	0.26
	Serpentinite:		
	Transition zone	20 - 28	0.73

Drill Hole	Horizon	Depth Range ft	Ni Assay %
4A	Laterite:		
	Pisolitic and ferruginous zones	0 - 5	0.06
	Limonitic yellow zone	5 - 15	0.12 - 0.17
	Mottled zone	15 - 20	0.35
	Serpentinite:		
	Transition zone	20 - 25	1.12
	Bleached zone	25 - 30	0.63
	Fresh zone	30 - 41	0.22 - 0.41
5	Laterite:		
	Pisolitic zone	0 - 5	0.20
	Mottled zone	5 - 12	0.73
	Serpentinite:		
	Transition zone	12 - 17	0.93
	Bleached zone	17 - 22	0.38
	Fresh zone	22 - 36	0.23 - 0.27
6	Overburden	0 - 3	0.13
	Laterite:		
	Ferruginous and limonitic zones	3 - 6	0.36
	Mottled zone	6 - 15	0.70 - 0.75
	Serpentinite:		
	Transition zone	15 - 20	0.80
	Bleached zone	25 - 51	0.25 - 0.39
7	Laterite:		
	Ferruginous red zone	0 - 6	0.11
	Limonitic yellow zone	6 - 8	0.48
	Mottled zone	8 - 13	0.71
	Serpentinite:		
	Bleached zone	13 - 20	0.90
7A	Laterite:		
	Ferruginous and limonitic zone	0 - 6	0.11
	Mottled zone	6 - 9	0.71
7B	Laterite:		
	Ferruginous red zone	0 - 7	0.10
	Mottled zone	7 - 12	0.39

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Drill Hole	Horizon	Depth Range ft	Ni Assay %
7B (Contd)	<u>Serpentinite</u>:		
	Fresh zone	12 - 16	0.46
	Transition zone	16 - 24	0.87 - 0.88
	Bleached zone	24 - 28	0.57
	Fresh zone	28 - 42	0.22 - 0.34
8	<u>Laterite</u>:		
	Ferruginous red zone	0 - 9	0.08 - 0.11
	Limonitic yellow zone	9 - 16	0.12
	Mottled zone	16 - 35	0.13 - 0.54
	<u>Serpentinite</u>:		
	Transition zone	35 - 45	1.26 - 2.17
	Bleached zone	45 - 69	0.26 - 1.80
	Fresh zone	69 - 91	0.24 - 0.37
9	<u>Laterite</u>:		
	Pisolitic zone	0 - 4	0.30
	Mottled and transition zone	4 - 7	0.66
	<u>Serpentinite</u>:		
	Bleached zone	7 - 9	0.40
	Fresh zone	9 - 21	0.20
10	<u>Laterite</u>:		
	Pisolitic and limonitic zone	0 - 5	0.56
	Mottled zone	5 - 7	0.78
	<u>Serpentinite</u>:		
	Transition zone	7 - 10	1.02
	Bleached zone	10 - 22	0.36 - 0.58
	Fresh zone	22 - 62	0.10 - 0.22
11	Rodingite ?	0 - 6	0.01
	<u>Serpentinite</u>:		
	Fresh zone	6 - 18	0.04 - 0.20
12	<u>Laterite</u>:		
	Ferruginous red zone	0 - 4	< 0.01
	Pisolitic zone	4 - 12	< 0.01

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Drill Hole	Horizon	Depth Range ft	Ni Assay %
12 (Contd)	Ferruginous red zone	12 - 26	< 0.01
	Limonitic yellow zone	26 - 30	0.12
	Mottled zone	30 - 51	0.59 - 0.20
	Serpentinite:		
	Transition zone	57 - 64	1.46 - 1.52
	Bleached zone	64 - 69	1.22
	Fresh zone	69 - 87	0.48 - 0.64
13	Laterite:		
	Ferruginous red zone	0 - 6	0.02
	Limonitic yellow zone	6 - 11	0.06
	Mottled zone	11 - 22	0.10 - 0.32
	Serpentinite:		
	Transition zone	22 - 32	0.62 - 0.96
	Bleached zone	37 - 45	0.71 - 1.21
	Fresh zone	45 - 58	0.46 - 0.62
14	Laterite:		
	Ferruginous red zone	0 - 6	0.34
	Limonitic yellow zone	6 - 16	0.1 - 0.12
	Mottled zone	16 - 44	0.11 - 0.42
	Serpentinite:		
	Transition zone	44 - 47	0.74
	Bleached zone	47 - 57	0.85 - 1.06
	Fresh zone	57 - 69	0.20 - 0.34
15	Laterite:		
	Ferruginous red zone	0 - 8	0.10
	Limonitic yellow zone	8 - 16	1.08
	Mottled zone	16 - 50	0.17 - 0.27
	Serpentinite:		
	Transition zone	50 - 55	0.11
	Bleached zone	55 - 73	0.11 - 1.21
	Fresh zone	73 - 86	0.19 - 0.26
16	Laterite:		
	Ferruginous red and pisolitic zones	0 - 14	0.07 - 0.09
	Limonitic yellow zone	14 - 18	0.06
	Mottled zone	18 - 28	0.12 - 0.13

Drill Hole	Horizon	Depth Range ft	Ni Assay %
16 (Contd)	<u>Serpentinite</u>:		
	Transition zone	28 - 32	1.56
	Bleached zone	32 - 57	0.70 - 1.36
	Fresh zone	57 - 76	0.4 - 0.7
17	<u>Laterite</u>:		
	Pisolitic zone	0 - 5	0.31
	Ferruginous red zone	5 - 9	0.44
	<u>Serpentinite</u>:		
	Transition zone	9 - 19	0.33 - 0.34
	Bleached zone	19 - 24	0.42
	Fresh zone	24 - 32	0.12 - 0.13

< - Less than.

APPENDIX B

MINING COST ESTIMATES

Listed below are estimates for mining capital equipment and estimates for mine operating expenses:

1. Capital Costs Estimates

Mining capital costs for mining at the rate of 2,000 tons per day each of ore and overburden are estimated as follows:

Mining Equipment -	$,000	$,000	
Ore 110 RB Face Shovel (erected)	250		
Ore dozer (200 - 300 bhp)	50		
Ore grader (Cat 12E)	30		
Four Haulpak Haul units	320		
Miscellaneous light vehicles	10	660	
Maintenance shop, store, office		100	
Six houses for Key personnel		60	
Preproduction - cleaning, stripping, roads		100	
		920	
Contingency allowance		80	
Total Capital			$1,000,000

2. Mine Operating Expenses Estimates

Mine operating expenses at the rate of 2,200 tons per day each of ore and overburden are estimated as follows:

Operation	Cost/Ton Ore	Cost/Ton Overburden	Total Cost/Ton Ore Treated
Loading	6¢	6¢	12¢
Transport	8	4	12
Forward preparation	2	-	2
Supervision/pumping	2	-	2
	18¢	10¢	28¢

Annual cost of mine operation:

733,000 tons ore by 28¢ = $205,000

APPENDIX C

OPERATING COST DATA - PUG ROAST ACID LEACH PROCESS

Listed below are the operating expenses and allowable tax deductions for mining, transportation, and treatment by the pug roast acid leach process for a 2,200 ton per day operation:

		$,000
1.	**Sulphuric Acid** Annual consumption of 221,000 tons is 30% by weight of ore treated. Cost of production, on-site, is estimated at $15.57 per ton for an annual cost of -	3,443
2.	**Hydrogen** Annual consumption of hydrogen for the production of H_2S is estimated at 600 tons. Cost is estimated at 80¢ per 1000 standard ft^3 for an annual cost of -	192
3.	**Sulphur** Additional sulphur for the manufacture of H_2S gas is estimated at 9,245 tons per annum. Annual cost at $45 per ton is	425

4. **Labour**

	$,000	
104 operators at $60 per week	324	
20 supervisors at $90 per week	93	
	417	
Shift allowance 10%	42	
Benefit loading and payroll Tax 5.5%	23	482

5.	**Fuel Oil** Annual consumption, assuming a moisture content of 20% for the ore, is estimated at 105,000 tons. Delivered price per ton is $15. Annual cost is -	1,580
6.	**Power** The power requirement for driving the various rotating equipment and agitators is estimated at 2,000 hp. Assuming an 80% power factor and a rate of $67.40 per annum per kW, the annual cost is -	80

		$,000
7.	**Water**	
	Fresh water enters the process at the CCD washers and is rejected at the sulphides thickener and filter. The liquor flow is equivalent to 330 million gallons per annum, which is rejected with a magnesium content of 12 g per litre and a pH of 1.0. There appears to be no reason why some of the reject liquor should not be recycled. The water requirement is therefore assumed to be 100,000,000 gallons per annum at a cost of 30¢ per 1000 gallons, i.e.	30
8.	**Maintenance**	
	Maintenance expenses at 5% of capital cost is assessed on equipment valued at $20,900,000	1,045
9.	**General Overhead**	
	General overhead is taken at 110% of labour costs	530
10.	**General Expenses**	
	General expenses have been arbitrarily assessed at -	115
11.	**Transportation Costs**	
	Annual tonnage of 12,210 tons of nickel sulphide is assumed to be shipped in bulk at $14.85 per ton. The cost buildup is listed below:	

	$/ton	
Cost of road transport to wharf site	0.64	
Outward wharfage	0.61	
Freight rate to Japan	9.00	
Marine Insurance	1.30	
0.5% Loss factor	3.30	
	14.85	181

		$,000
12.	**Mining Costs**	205
	Total Operating Expenses	8,308

13.	<u>Depreciation</u>	<u>$,000</u>
	Depreciation at the rate of 11.25% and based on a declining balance is calculated on process and auxiliary equipment valued at $20,000,000 plus $400,000 for materials handling equipment. Depreciation for the first year of operation is -	2,295

APPENDIX D

CAPITAL COST ESTIMATE - NICARO PROCESS

Capital estimates for the treatment of 2200 tons per day by the Nicaro process are listed below:

	Cost $,000
Reclamation of ore from stockpile and transport onto a conveyor leading to primary crushers:	
Assume 20% moisture in the ore, 380 ton per hour and 2 front end loaders, capacity 2 cubic yards per bite,	46
Conveyor to primary crushers:	
Assume 36 in. wide by 400 ft long, plus one tripper,	34
Precrushing screen:	
Assume 30 ft^2,	2
Primary crusher:	
Assume load is 200 ton per hour	65
Post crushing screen:	
Assume 30 ft^2,	2
Conveyor to silos:	
Assume 36 in. wide by 300 ft long with 9 trippers, into 9 silos,	38
Storage silos:	
Assuming 60 days capacity, requirement is 435,000 ft^3. Assume 9 units each 40 ft dia. by 120 ft high,	450
Conveyor to drying kilns. Load is 110 ton per hour:	
Assume 18 in. wide by 100 ft long, with 9 chutes and 3 trippers,	13
Drying kilns. Input 2640 ton per day containing 20% moisture:	
Assume 3 units, each 10 ft diameter by 120 ft long,	375
Conveyor to grinding mills:	
Assume 18 in. wide by 100 ft long plus 3 chutes and 4 trippers,	14
Ball mills:	
Assume 4 units, capacity 550 ton per day each,	294

	Cost $,000
Post grinding screen:	
Assume 30 ft^2,	3
Conveyor to roaster hoppers:	
Assume 18 in. wide by 300 ft long, plus 4 chutes and 8 trippers	23
Furnaces plus gas generators:	
Assume 8 Herroshoff type furnaces, each having 16 hearths and 22 ft diameter,	1500
Conveyor to ore coolers - screw type:	
8 required, 4 at 50 ft long and 4 at 10 ft long	53
Ore coolers:	
As Nicaro equipment unsatisfactory, suggest use of fluid beds with water quenching. Require 4 off, duty 20 ton per hour each. Assume residence time is 10 minutes, bed density is 3016 per ft^3, require 250 ft^3. Unit 6 ft diameter by 20 ft high adequate,	108
Screw conveyors to quench tank:	
10 ft long, 4 off,	2
Quench tanks:	
Assume 4 off, 25% solids slurry, 15 minutes residence time. Volume required - 604 ft^3 each. Tanks 10 ft diameter by 12 ft deep,	13
Impellers and motors for above:	74
Aerators:	
Assume 2 streams, each 40 ton per hour solids at 25% solids. Volume required for 10 minutes retention is 805 ft^3 each,	6
First leach tanks:	
Assume 100 ft thickeners	110
Surge tank:	
Allow 2000 gallons	2

	$,000
Second aerators	6
Second leach tanks	110
Surge tank	2
Third aerators	6
Third leach tanks	110
Surge tank	2
Wash circuit:	
Assume two sets of four thickeners. Cost including pumps and mixing tanks	500
Transfer tank	2
Aerator for pregnant liquor:	
Assume same as in leach circuit	3
Surge tank	2
Filter for removal of slimes:	
Duty 500 g per minute	15
Surge tank:	
Allow 1 hr capacity, 30,000 gallons	15
Ammonia stripper:	
Assume 4 units, each 6 ft diameter by 40 ft high, bubble cap colums	62
Flash tank:	
Assume 4 at 2000 gallons	8
Thickener for concentration of nickel carbonate	55
Filter:	
Size 4 ft diameter by 6 ft wide	20
Conveyor to salination kiln	4
Calcination kiln:	
Feed 5 ton per hour. Size 8 ft diameter by 140 ft long kiln, plus motor and brick lining	150

	$,000
Surge tank for thickener overflow:	
Assume 4 hours capacity to permit cleaning of plate and frame type filter. 120,000 gallons.	18
Filter press for final recovery of nickel carbonate	12
Condensers for steam and ammonia vapours:	
4 units at 9000 ft^2.	74
Ammonia absorber 170 gal/min:	
Assume similar to stripper	15
Gas washer:	
Assume one third size of absorber	8
Surge tank:	
Allow 2000 gallons	2
Ammonia absorber 500 gal/min.	62
Carbon dioxide absorber, 195 gal/min.	31
Liquor cooler	20
Ammonia tank:	
Assume 2000 gallons	2
Ammonia blending tank:	
Assume 1000 gallons	1
Surge tank for tailings liquor:	
Assume 10,000 gallons plus impeller	32
Tailings preheater	20
Surge tank:	
Allow 2000 gallons plus impeller	5
Ammonia strippers:	
Assume 4 units	62
Spray absorber for vent air and ammonia for aerators	2
Scrubber for vent air and ammonia for absorber:	
Assume 2 units, 8 ft diameter by 50 ft high	65
Compressors, 5000 standard ft^3 per minute	60

		$,000
Liquor cooler:		
Allow 7000 ft^2 .		16
Steam generating plant:		
Require 3.15 by 10^6 lb per day		300
Electrostatic precipitators:		
60,000 ft^3 per minute		90
Sinter plant:		
20 ton per day throughput plus recirculatory load		200
Miscellaneous pumps, etc		25
Total Equipment Cost		5,422
Additions -	%	
Installation	40	
Instrumentation	15	
Piping	50	
Electrical	20	
Buildings	10	
Utilities	20	
Design and engineering fees	20	
Contingency	25	
	200	10,844
Port loading facilities		100
Total Investment		16,366

030

APPENDIX E

OPERATING COST DATA - NICARO PROCESS

Listed below are the operating expenses and allowable tax deductions for mining, transportation, and treatment by the Nicaro process for a 2200 ton per day operation:

1. Ammonia — $,000

A large percentage of the ammonia used in the process is recovered by steam stripping. The portion that is lost is estimated at 5.5 lb per ton of ore fed to the reduction furnace, giving an annual consumption of 1800 tons. The "on-site" price for ammonia is estimated at $130 per ton. Cost to process is - 234

2. Carbon Dioxide

Consumption of carbon dioxide is equivalent to the quantity evolved in the calcination of the nickel and cobalt carbonates, i.e. 6000 ton per annum. On-site price for carbon dioxide is estimated at 1.5¢ per pound making a cost to the process of - 198

3. Labour

140 operators at $60 per week	$437	
25 supervisors at $90 per week	117	
	554	
Shift allowance	55	
Benefit loading and payroll Tax 5.5%	30	639

4. Fuel Oil

Consumption of fuel is estimated at 75 ton per 1000 tons of ore processed. At $15 per ton the cost of 55,000 tons of oil is - 825

$,000

5. Power and Process Stream

Electric power consumption is estimated at 2500 hp with an 80% power factor. At a tariff rate of $67.40 per annual kW, annual power cost is $100,000. Steam consumption is estimated at 2.85 by 10^6 lb per day. Annual cost for steam is estimated at $350,000 at the rate of 37¢ per 1000 lb. Total cost for power and steam is – 450

6. Water

The water requirement for the process plant would be of 880 g per minute but a large portion of this could be recycled. At the cost of 30¢ per 1000 gallon it is estimated that the annual consumption of water would be 59 million gallons for a total cost of – 18

7. Maintenance

Maintenance expense at 5% of capital cost is calculated on equipment valued at $16.4 million – 820

8. General Overhead

General overhead is taken at 110% of labour cost 702

9. General Expenses

General expenses have been arbitrarily assessed at – 115

10. Transportation Cost

The annual production of 6435 tons of nickel oxide sinter is assumed to be shipped in bags at a cost of $30.48 per ton.

The components of this cost are listed below:

Cost of bags per ton	$5.00	
Road transport to wharf site	1.75	
Outward wharfage	0.61	
Freight rate to Japan	22.00	
Marine insurance	1.12	
	$30.48	196

11. Mining Cost 205

Total Operating Expenses 4,402

		$,000
12. **Depreciation**		

Capital items that are depreciated over the life of the project at 11.25% declining balance are:

	$,000
Process equipment	15,800
Materials handling equipment	100
	15,900

Not included above is $500,000 for crushers, grinders, etc. which does not qualify for the investment allowance and can be treated as allowable capital expenditures rather than depreciated. The cost of depreciation in the first year is - 1,788

13. **Investment Allowance**

Capital equipment which qualifies for the investment allowance is the total investment for the process plant, $16,300,000 less $500,000. Twenty percent on the balance of $15,800,000 is deductible in the first year of operation - 3,160

14. **Allowable Capital Expenditures**

Deductible in the first year of operation is allowable capital expenditures under Section 122E of the Income Tax Assessment Act, are the following items:

	$,000	
Mining investment	940	
Housing	160	
Specific process equipment	500	1,600

15. **Exempt Income**

Twenty percent of the project's net income is treated as exempt income as allowed under Section 23A of the Income Tax Assessment Act (ITAA). In the second year of operation the project's net income is $4,226,000 of which 20% can be treated as a deduction before tax - 845

APPENDIX F

OPERATING COST DATA - PYROSULPHIDIZING PROCESS

Listed below are the operating expenses and allowable tax deductions for mining, transportation and treatment by the Pyrosulphidizing process for a 2200 ton per day operation:

	$,000
1. Sulphur The consumption of sulphur is estimated to be equivalent to the amount contained in the concentrate plus an additional 30%, i.e. the daily tonnage of concentrate is 92 tons containing 27.6 tons sulphur and the additional requirement is 8.3 tons making a total of 35.9 ton per day. Valued at $45 per ton, annual cost of sulphur is -	533
2. Flotation Reagents The cost of reagents is estimated on the basis of 0.2 lb of frother per ton of ore and 0.02 lb of collector per ton of ore at an on-site price of 30¢ per lb. Annual cost to the process is -	48

3. Labour

	$,000	
110 operators at $60 per week	343	
25 supervisors at $90 per week	117	
	460	
10% shift allowance	46	
Benefit loading and payroll tax 5.5%	23	529

4. Fuel Oil The fuel oil requirement is the same as for the Nicaro process up to and including the furnaces plus additional quantities of fuel oil for drying the sulphide concentrate. Annual consumption of fuel oil is estimated at 44,000 tons at a price of $15 per ton, giving an annual cost of -	660
5. Power The power requirement for driving the various motors is estimated at 2000 hp. Assuming an 80% power factor, annual cost is -	80

		$,000
6.	**Water** The water requirement will be mainly for the flotation circuit and consumption is estimated at 60 million gallons. At a price of 30¢ per 1000 gallons annual cost is -	18
7.	**Maintenance** Maintenance expense at 5% of capital cost is calculated on equipment valued at $9,000,000.	450
8.	**General Overhead** General overhead is taken at 110% of labour costs -	581
9.	**General Expenses** General expenses have been arbitrarily assessed at -	115
10.	**Transportation Cost** The annual production of 30,360 tons of nickel sulphide is assumed to be shipped in bulk form at a cost of $10.95, comprising the following items:	

	$/ton	$,000
Cost of road transport to wharf site	0.64	
Outward wharfage	0.61	
Freight rate to Japan	9.00	
Marine insurance	0.20	
0.5% loss factor	0.50	
	10.95	332

		$,000
11.	**Mining Cost**	205
	Total Operating Expenses	3,551

12. **Depreciation**

Capital items that are depreciated over the life of the project at 11.25% declining balance are:

	$,000
Process equipment	8,500
Materials handling equipment	200
	8,700

On this basis, first year depreciation is - 979

		$,000
13. **Investment Allowance** Twenty percent of the process equipment valued at $8,500,000 is assumed to be deductible as an investment allowance in the first year of operation -		1,700
14. **Allowable Capital Expenditure** Deductible in the first year of operation as an allowable capital expenditure under Section 122E of the ITAA are the following items:		
	$,000	
Mining investment	940	
Housing	160	
Process equipment	300	1,400
15. **Exempt Income** Twenty percent of the project's net income is treated as exempt income as allowed under Section 23A of the ITAA. In the second year of operation the project's net income is $2,408,000 of which 20% can be treated as a deduction before tax -		482
16. **Section 80 - Loss** Losses in the first year after all allowable deductions is $749,000. Of this amount $267,000 can be treated as Section 80 - Loss, and is deductible before tax in the following year -		267

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APPENDIX G

SAMPLES FROM ORE-BEARING HORIZONS

Samples from the ore-bearing horizon that were examined as a part of the mineralogical examination are tabulated below:

Drill Hole	Depth ft	Ni Assay %	Hand Specimen
1	19	1.86	Crumbly, buff coloured, fibrous to platy mineral phases with fine-grained black material formed along parting fractures.
5	16	0.93	Chocolate-brown to yellow fibrous and platy mineral phases that display a poorly defined lineation.
6	17	0.80	Dark-brown, massive, earthy material streaked with secondary iron-oxides.
7B	20	0.88	Black to dark-brown, crumbly material flecked with orange and white phases. * kaol.
8	48	1.80	Buff to pale-yellow, massive clay with fibrous green phases and black flecks. (chlorite?)
10	8	0.78	Massive, brown, earthy material flecked with orange, green and yellow phases. ? kaol.
12	58	1.46	Black, massive, earthy material intersected with veins of an apple-green mineral. A few flakes of a white mineral phase. * kaol.
13	41	1.20	Massive buff-coloured clay flecked and streaked with a darker fibrous mineral phase.
14	47	0.74	Massive, emerald green mineral flecked with white and brown phases. * kaol.
15	69	1.21	Pale yellow to pale green, massive fibrous rock with black iron-oxides staining fractures and associated with brown clay.
16	28	1.56	Black earthy material speckled with green mineral phases.

TABLES 1 – 17

(NB: Tables 3 & 4 missing)

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TABLE 1: CHEMICAL ANALYSES OF THE BEACONSFIELD ULTRABASIC COMPLEX

Hole Number	Footage From/to	SiO_2 %	CaO %	LOI %	Al_2O_3 %	Mg %	Fe %	Co %	Cr %	Ni %	
1	0 - 2	-	-	-	-	-	27.2	0.01	0.25	0.11	
	2 - 7	-	-	-	-	-	20.6 14.1	0.01 0.02	0.19 0.14	0.11 0.11	
	7 - 11	13.4	0.12	9.22	6.8	-	44.5	0.30	0.13	0.60	MOTTLED
	11 - 16	23.18 32.96	0.37 0.62	12.10 14.99	5.5 4.2	-	37.6 30.7	0.20 0.11	0.15 0.17	1.16 1.73	
	16 - 21	41.89	0.07	13.25	0.7	-	14.3	0.05	0.07	1.86	
	21 - 26	42.74	7.28	15.65	1.0	-	10.4	0.03	0.05	1.50	
	26 - 31	45.58	9.99	15.61	0.3	-	6.1	0.02	0.06	1.52	
	31 - 36	44.35	20.77	14.65	0.3	-	6.6	0.02	0.05	1.31	BLEACHED
	36 - 41	39.42	17.26	14.41	0.9	-	6.0	0.02	0.03	1.10	SERP.
	41 - 46	43.89	17.66	13.44	0.9	-	8.4	0.02	0.04	1.35	
	46 - 51	42.98 -	12.19 -	14.50 -	0.68 -	-	7.91 3.6	0.02 0.01	0.05 0.03	1.31 0.52	
	51 - 56	-	-	-	-	-	3.0	0.01	0.02	0.48	
	56 - 59	-	-	-	-	-	2.85 2.7	0.01 0.01	0.02 0.02 0.08	0.57 0.67	FRESH
3	0 - 5	-	-	-	-	-	-	0.01	-	0.09	
	5 - 10	-	-	-	-	0.03	31.0	0.01	2.20	0.10	
	10 - 15	-	-	-	-	0.01	33.0	0.01	0.51	0.13	
	15 - 18	11.24	0.11	10.63	10.7	0.02 0.03	35.3 42.0	0.01 0.03	1.09 0.57	0.20 0.48	
	18 - 23	19.25	0.07	12.76	12.0	0.03	39.5	0.03	0.20	0.50	
	23 - 28	14.17 9.09	0.05 0.03	10.43 8.11	9.3 6.6	0.61	49.0	0.08	0.14	0.47	
	28 - 33	-	-	-	-	0.30	46.0	0.25	0.18	0.41	MOTTLED
	33 - 38	-	-	-	-	0.42	39.5	0.17	0.36	0.43	
	38 - 43	-	-	-	-	0.63 1.8	40.8 30.0	0.13 0.12	0.21 0.18	0.60 1.18	
	43 - 48	-	-	-	-	7.5	16.0	0.03	0.13	1.13	TRANSN. SERP.
	48 - 53	-	-	-	-	12.3	5.5	0.02	0.04	0.70	BLEACHED
	53 - 58	-	-	-	-	-	3.8	0.01	0.01	0.29	
	58 - 63	-	-	-	-	-	4.6	0.01	0.01	0.21	FRESH.
	63 - 66	-	-	-	-	-	4.23 4.3	0.01 0.01	0.01 0.02	0.25 0.24	
4A	0 - 5	-	-	-	-	-	24.2	0.01	1.39	0.06	
	5 - 10	-	-	-	-	-	36.6	0.01	2.04	0.12	
	10 - 15	-	-	-	-	-	24.6 42.9	0.01 0.01	1.57 1.29	0.12 0.17	
	15 - 20	17.56	0.11	10.32	9.3	-	38.4	0.10	0.75	0.35	MOTTLED
	20 - 25	42.97	0.17	16.00	2.1	-	12.4	0.07	0.11	1.12	TRANSN
	25 - 30	43.89	11.61	13.77	0.7	-	6.6	0.01	0.09	0.63	BLEACHED
	30 - 35	45.01	0.30	13.34	0.3	-	4.3	0.01	0.04	0.44	
	35 - 41	-	-	-	-	-	4.5 4.4	0.01 0.01	0.03 0.03	0.33 0.23	FRESH.

Continued

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TABLE 1: CONTINUED

Hole Number	Footage From/to	SiO_2 %	CaO %	LOI %	Al_2O_3 %	Mg %	Fe %	Co %	Cr %	Ni %	
5	0 - 5	-	-	-	-	0.30	33.6	0.02	-	0.20	
	5 - 12	31.72	0.08	15.64	7.7	1.2	25.1	0.03	-	0.73	MOTTLED
	12 - 17	40.66	0.05	14.16	1.2	4.2	22.6	0.05	-	0.93	TRANS^N
	17 - 22	46.66	0.11	13.02	0.6	16.4	6.8	0.01	-	0.38	BLEACHED
	22 - 27	-	-	-	-	19.8	3.5	0.01	-	0.27	
	27 - 32	-	-	-	-	18.8	4.0	0.01	-	0.27	FRESH.
	32 - 36	-	-	-	-	18.23 16.1	4.03 4.6	0.01 0.01	-	0.26 0.23	
6	0 - 3	-	-	-	-	0.3	14.5	0.02	-	0.13	
	3 - 6	31.19	0.17	12.29	9.9	0.35 0.4	16.4 18.3	0.05 0.09	-	0.24 0.36	
	6 - 11	27.41	0.12	16.94	8.4	0.9	23.3	0.07	-	0.70	
	11 - 15	28.72 30.03	0.10 0.09	11.06 11.18	7.2 6.0	0.95 1.0	24.7 26.1	0.08 0.09	-	0.72 0.75	MOTTLED.
	15 - 20	43.51	0.10	11.45	2.5	2.4	21.6	0.05	-	0.80	TRANS^N
	20 - 25	45.74	0.08	13.30	0.3	16.8	5.3	0.01	-	0.39	
	25 - 30	49.28	4.92	13.92	0.2	17.1	4.4	0.01	-	0.31	
	30 - 35	47.51 -	2.50 -	13.61 -	0.25 -	18.6	5.3	0.01	-	0.29	
	35 - 40	-	-	-	-	18.3	6.6	0.01	-	0.28	BLEACHED.
	40 - 45	-	-	-	-	18.3	4.4	0.01	-	0.26	
	45 - 51	-	-	-	-	17.9 18.3	5.22 5.3	0.01 0.01	-	0.30 0.25	
7B	0 - 7	-	-	-	-	0.04	24.0	0.01	-	0.10	
	7 - 12	-	-	-	-	0.23	37.0	0.01	-	0.01	MOTTLED.
?	12 - 16	45.74	0.09	13.62	0.1	4.1	22.4	0.01	-	0.01	? FRESH
	16 - 20	36.96	0.26	15.16	8.9	5.8	21.9	0.05	-	0.05	TRANS^N
	20 - 24	37.5 38.04	0.16 0.06	16.17 17.19	7.45 6.0	6.05 6.3	21.5 21.2	0.045 0.04	-	0.04 0.04	
	24 - 28	39.89	0.24	15.05	8.3	7.7	13.7	0.02	-	0.02	BLEACHED
	28 - 33	42.97	19.63	13.26	6.2	16.3	7.2	0.01	-	0.01	
	33 - 38	-	-	-	-	20.1	5.0	0.01	-	0.01	FRESH.
	38 - 42	-	-	-	-	18.8 20.0	5.8 5.2	0.01 0.01	-	0.01 0.01	
8	0 - 4	-	-	-	-	0.43	26.9	0.01	-	0.08	
	4 - 9	-	-	-	-	0.24	42.6	0.01	-	0.11	
	9 - 13	-	-	-	-	0.22	19.4	0.01	-	0.12	
	13 - 16	-	-	-	-	0.23 0.03	22.7 21.8	0.01 0.01	-	0.11 0.12	
	16 - 21	-	-	-	-	0.02	31.1	0.01	-	0.17	
	21 - 26	-	-	-	-	0.09	32.5	0.01	-	0.18	
	26 - 31	-	-	-	-	0.04	26.0	0.01	-	0.13	MOTTLED
	31 - 35	24.18	0.11	12.44	20.7	0.50 0.06	30.7 31.1	0.01 0.01	-	0.25 0.54	
	35 - 40	38.81	0.03	18.25	13.6	1.91	18.1	0.03	-	1.26	TRANS^N
	40 - 45	39.77 40.73	0.07 0.11	17.03 15.82	11.0 8.4	3.51 5.12	19.0 19.9	0.08 0.14	-	1.71 2.17	
	45 - 50	45.05	0.07	16.03	3.9	10.0	10.3	0.10	-	1.80	
	50 - 57	38.50	0.89	11.16	0.3	8.4	10.9	0.08	-	1.18	BLEACHED

Continued

TABLE 1 p25

TABLE 1: CONTINUED

030

Hole Number	Footage From/to	SiO_2 %	CaO %	LOI %	Al_2O_3 %	Mg %	Fe %	Co %	Cr %	Ni %	
8	57 - 63	47.59	9.72	13.89	15.1 ?	17.7	4.6	0.07	-	1.04	BLEACHED
(continued)	63 - 69	41.99 36.81	2.97 1.19	13.04 11.07	62.0	12.2 12.7	8.1 6.6	0.07 0.03	-	1.07 0.26	
	69 - 74	-	-	-	20.32 -	15.2	5.0	0.01	-	0.37	
	74 - 79	-	-	-	-	15.5	5.6	0.01	-	0.24	FRESH.
	79 - 84	-	-	-	-	16.0	4.6	0.01	-	0.25	
	84 - 91	-	-	-	-	15.67 16.0	4.8 4.0	0.01 0.01	-	0.27 0.24	
9	0 - 4	15.63	0.20	10.04	12.8	1.34	42.6	0.07	-	0.30	[illegible]
	4 - 7	35.42	0.04	16.51	6.2	6.0	26.6	0.11	-	0.66	~~[illegible]~~ + TRANS^N
	7 - 9	48.66	0.10	12.58	0.4	24.0	4.8	0.02	-	0.40	BLEACHED
	9 - 14	-	-	-	-	23.4	7.8	0.02	-	0.20	FRESH.
	14 - 21	-	-	-	-	23.55 24.7	6.1 4.8	0.02 0.02	-	0.20 0.20	
10	0 - 5	38.96	0.16	20.19	19.6	2.3	16.6	0.03	-	0.56	
	5 - 7	34.96	0.27	19.27	11.0	2.1	25.0	0.10	-	0.78	MOTTLED.
	7 - 10	45.53	0.19	21.89	8.7	3.6	15.8	0.04	-	1.02	TRANS^N
	10 - 15	46.59	1.38	17.32	4.2	18.8	7.0	0.02	-	0.58	
	15 - 19	45.89	1.69	15.79	3.0	20.5	6.2	0.02	-	0.38	BLEACHED
	19 - 22	46.18 46.66	3.97 8.84	16.36 15.96	3.1 2.1	19.8 20.1	7.33 6.8	0.02 0.02	-	0.44 0.36	
	22 - 27	-	-	-	-	20.7	4.8	0.02	-	0.20	
	27 - 32	-	-	-	-	22.2	5.2	0.02	-	0.24	
	32 - 37	-	-	-	-	24.0	5.6	0.02	-	0.20	
	37 - 42	-	-	-	-	21.7	5.0	0.02	-	0.20	
	42 - 45	-	-	-	-	22.2	4.6	0.01	-	0.24	FRESH.
	45 - 50	-	-	-	-	22.41 23.6	4.87 4.0	0.02 0.01	-	0.20 0.12	
	50 - 58	-	-	-	-	6.1	4.0	0.01	-	0.04	
	58 - 62	-	-	-	-	22.9	4.6	0.01	-	0.10	
	62 - 66	-	-	-	- ? contam →	8.8	4.4	0.01	-	0.01	
12	0 - 4	-	-	-	-	0.20	25.2	0.01	-	0.01	
	4 - 12	-	-	-	-	0.20	32.6	0.01	-	0.01	
	12 - 17	-	-	-	-	0.1	41.4	0.01	-	0.01	
	17 - 22	-	-	-	-	0.10	38.0	0.01	-	0.01	
	22 - 26	-	-	-	-	0.10	36.6	0.01	-	0.01	
	26 - 30	20.48	0.04	13.29	23.8	0.13 0.10	33.5 27.4	0.05 0.14	-	0.03 0.12	
	30 - 40	-	-	-	-	0.20	22.4	0.17	-	0.20	
	40 - 45	17.71	0.06	10.39	16.2	0.20	35.2	0.22	-	0.59	MOTTLED
	45 - 51	20.29 22.87	0.09 0.12	10.54 10.69	15.8 15.4	0.30 0.50	30.1 32.6	0.16 0.09	-	0.42 0.48	
	51 - 57	-	-	-	-	-	-	-	-	-	
	57 - 60	33.0	0.14	14.02	4.2	3.0	28.6	0.10	-	1.46	TRANS^N.
	60 - 64	36.36 39.73	0.88 1.63	16.43 18.85	7.5 10.8	5.4 7.9	21.5 14.4	0.07 0.04	-	1.49 1.52	
	64 - 69	38.81	1.0	14.90	10.2	7.3	15.4	0.04	-	1.22	BLEACHED
	69 - 74	39.58	23.21	12.64	0.4	23.1	8.0	0.02	-	0.48	
	74 - 79	-	-	-	-	23.5	7.4	0.01	-	0.53	FRESH.
	79 - 87	-	-	-	-	23.03 22.5	6.96 5.3	0.015 0.01	-	0.55 0.64	

Continued

TABLE 1: CONTINUED

Hole Number	Footage From/to	SiO_2 %	CaO %	LOI %	Al_2O_3 %	Mg %	Fe %	Co %	Cr %	Ni %	
13	0 - 6	-	-	-	-	0.45	30.2	0.01	-	0.02	
	6 - 11	-	-	-	-	0.28 0.12	24.3 18.4	0.01 0.01	-	0.04 0.06	
	11 - 16	-	-	-	-	0.14	30.4	0.01	-	0.10	
	16 - 22	21.87	0.06	13.00	18.9	0.33 0.52	32.1 33.8	0.04 0.07	-	0.21 0.32	MOTTLED
	22 - 27	35.27	0.12	16.48	17.0	5.06	18.0	0.07	-	0.62	TRANS^N.
	27 - 32	35.92 36.58	0.08 0.05	17.12	12.5 8.0	5.56 6.07	21.2 24.4	0.06 0.05	-	0.79 0.96	
	32 - 37	-	-	-	-	-	-	-	-	-	
	37 - 41	-	-	-	-	13.3	17.3	0.04	-	1.20	
	41 - 45	36.19	5.30	11.53	6.8	15.95 18.6	16.8 16.3	0.03 0.02	-	0.95 0.71	BLEACHED
	45 - 50	41.58	3.00	14.21	3.0	22.6	6.5	0.01	-	0.62	
	50 - 58	-	-	-	-	22.6 22.7	6.4 6.3	0.01 0.01	-	0.54 0.46	FRESH.
14 SCOTTS HILL	0 - 6	-	-	-	-	1.22	31.2	0.02	-	0.34	
	6 - 11	-	-	-	-	0.39	32.0	0.02	-	0.10	
	11 - 16	-	-	-	-	0.42 0.26	31.9 32.6	0.02 0.02	-	0.19 0.12	
	16 - 21	-	-	-	-	0.15	42.3	0.02	-	0.11	
	21 - 26	32.03	0.07	13.46	26.4	0.17	19.4	0.10	-	0.15	
	26 - 31	27.41	0.05	13.32	18.9	0.40	29.9	0.30	-	0.26	MOTTLED
	31 - 36	7.55	0.10	7.48	2.4	0.35	54.8	0.16	-	0.42	
	36 - 40	10.63	0.11	9.85	4.5	0.25	49.9	0.20	-	0.33	
post	40 - 44	16.79	0.03	13.18	7.4	0.20	37.6	0.20	-	0.27	
TC-1 → *	44 - 47	35.57	0.14	19.47	7.7	2.2 3.65 MgO	22.9 32.7 Fe_2O_3	0.14	-	0.74	TRANS^N
	47 - 52	45.12	0.23	22.79	2.8	2.1	15.1	0.08	-	1.06	
	52 - 57	34.50	0.15	16.89	1.4	1.92	29.1	0.04	-	0.85	BLEACHED
	57 - 62	41.20	0.86	13.51	0.3	21.4	6.7	0.02	-	0.34	
	62 - 66	-	-	-	-	22.0	11.0	0.02	-	0.20	FRESH.
	66 - 69	-	-	-	-	23.7	4.4	0.02	-	0.24	
15 SCOTTS HILL	0 - 8	-	-	-	-	1.06	28.9	0.02	-	0.10	
	8 - 16	-	-	-	-	0.91 0.77	24.9 21.0	0.02 0.02	-	0.59 1.08	
	16 - 25	15.55	0.05	15.26	15.1	0.37	39.0	0.04	-	0.17	
	25 - 35	-	-	-	-	-	-	-	-	-	MOTTLED
	35 - 50	32.19	0.11	14.82	16.6	1.38	23.8	0.06	-	0.27	
	50 - 55	53.13	0.07	8.13	3.7	1.08	9.8	0.02	-	0.11	TRANS^N
	55 - 60	51.13	0.27	8.05	5.2	2.86	6.1	0.01	-	0.11	
	60 - 65	54.36	0.30	9.08	5.2	3.82	4.4	0.01	-	0.17	BLEACHED
	65 - 73	42.04	0.15	18.49	1.9	2.28	20.4	0.02	-	1.21	
	73 - 76	53.13	0.14	7.99	1.5	6.8	6.1	0.01	-	0.26	
	76 - 81	-	-	-	-	3.8	3.4	0.01	-	0.24	FRESH.
	81 - 86	-	-	-	-	21.8	5.7	0.02	-	0.19	

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Continued

TABLE 1 4/5

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TABLE 1: CONTINUED

Hole Number	Footage From/to	SiO_2 %	CaO %	LOI %	Al_2O_3 %	Mg %	Fe %	Co %	Cr %	Ni %	
16 MT VULCAN	0 - 5	-	-	-	-	0.1	46.4	0.01	-	0.09	
	5 - 10	-	-	-	-	0.1	46.8	0.01	-	0.09	
	10 - 14	-	-	-	-	0.1	32.6	0.01	-	0.07	
	14 - 18	-	-	-	-	0.1	14.0	0.01	-	0.06	
	18 - 23	8.01	0.05	10.36	9.9	0.1	46.6	0.02	-	0.13	
	23 - 28	14.48	0.05	8.36	11.3	0.2	44.8	0.04	-	0.12	MOTTLED
	28 - 32	35.57	0.16	14.30	6.7	4.7	26.8	0.06	-	1.56	TRANS"
	32 - 37	34.19	6.29	11.80	2.8	10.2	22.8	0.04	-	1.36	
	37 - 42	40.66	0.05	13.05	1.9	11.5	19.8	0.04	-	1.21	
	42 - 47	37.42	0.85	15.44	8.6	6.4	18.4	0.05	-	1.16	BLEACHED
	47 - 51	28.34	3.99	11.03	13.2	7.5	17.2	0.02	-	0.70	
	51 - 57	28.34	3.99	11.03	13.2	15.2	11.6	0.02	-	0.71	
	57 - 62	38.50	1.34	14.54	2.5	17.4	6.2	0.01	-	0.40	
	62 - 67	40.81	0.20	15.87	7.1	15.2	8.0	0.02	-	0.70	FRESH.
	67 - 72	-	-	-	-	20.0	4.6	0.01	-	0.44	
	72 - 76	-	-	-	-	17.6	6.9	0.02	-	0.42	

Note: LOI = Loss on Ignition.
- = Not determined.

TABLE 2: SULPHURIC ACID LEACHING TESTS

Test	Nickel Extracted		Sulphur Dioxide Added lb/Short Ton Ore
	lb/short ton	%	
1	7.2	31.6	100
2	10.4	45.6	200
3	10.8	47.4	500
4	11.0	48.2	1000
5	11.6	50.9	2000
6	11.2	49.1	500

TABLE 3: SOLUTION OF ORE CONSTITUENTS

Test	lb of Constituents Dissolved per Short Ton of Ore						
	Co	SiO_2	Fe	Al	Mn	Mg	Ni
1	2.2	0.1	0.1	2.1	30.9	0.2	7.2
2	2.8	1.4	9.7	7.0	42.5	0.4	10.4
3	3.2	2.7	38.4	10.1	42.2	0.5	10.8
4	3.5	3.4	48.9	7.8	44.6	0.5	11.0
5	3.8	4.1	57.4	7.5	46.6	0.5	11.6

TABLE 4: SIZE ANALYSES

Size	Ore, %		Leach Residue, %	
BSS	Weight	Ni	Weight	Ni
+ 30	7.7	1.39	3.9	0.12
+ 60	9.4	1.02	6.9	0.30
+100	9.9	1.15	7.5	0.48
+150	11.8	1.20	0.9	0.63
+200	5.9	1.15	10.8	
-200	55.3	1.00	70.0	0.75

TABLE 5: SAMPLES COLLECTED FOR MINERALOGICAL EXAMINATION[a]

Drill Core and Location	Footage	Logs	Assay Results				
			Ni	Co	Mg	Cr	Fe
1		Laterite:					
Barnes Hill	14	Mottled zone	1.73	0.11	-	0.17	30.7
		Serpentinite:					
	17	Bleached zone	1.86	0.05	-	0.07	14.3
	19	Bleached zone	1.86	0.05	-	0.07	14.3
	20	Bleached zone	1.86	0.05	-	0.07	14.3
	24	Bleached zone	1.50	0.03	-	0.05	10.4
3		Laterite:					
Barnes Hill	3⅚	Pisolitic zone	0.09	0.01	-	-	-
	9	Ferruginous red zone	0.10	0.01	0.03	2.20	31.0
	17	Limonitic yellow zone	0.48	0.03	0.03	0.57	42.0
	20	Mottled zone	0.50	0.03	0.03	0.20	39.5
	38½	Mottled zone	1.18	0.12	1.8	0.18	30.0
		Serpentinite:					
	46	Transition zone	1.13	0.03	7.5	0.13	16.0
	49	Bleached zone	0.70	0.02	12.3	0.04	5.5
	57	Fresh zone	0.29	<0.01	-	<0.01	3.8
	66	Fresh zone	0.24	<0.01	-	0.02	4.3
4		Laterite:					
Barnes Hill	14	Limonitic yellow zone	0.19	<0.01	-	1.14	46.4
		Serpentinite:					
	23	Transition zone	0.73	0.25	-	0.48	27.5
4A		Laterite:					
Barnes Hill	6	Limonitic yellow zone	0.12	<0.01	-	2.04	36.6
		Serpentinite:					
	21½	Transition zone	1.12	0.07	-	0.11	12.4
	25	Bleached zone	0.63	0.01	-	0.09	6.6
	29½	Fresh zone	0.44	<0.01	-	0.04	4.3
5		Laterite:					
Barnes Hill	12	Base of mottled zone	0.73	0.03	1.2	-	25.1
		Serpentinite:					
	16	Transition zone	0.93	0.05	4.2	-	22.6
	22	Base of bleached zone	0.38	0.01	16.4	-	6.8

Continued

TABLE 5: CONTINUED

Drill Core and Location	Footage	Logs	Assay Results Ni	Co	Mg	Cr	Fe
6		Laterite:					
Barnes Hill	6	Ferruginous and limonitic zones	0.36	0.09	0.4	-	18.3
		Serpentinite:					
	17	Transition zone	0.80	0.05	2.4	-	21.6
	19	Transition zone	0.80	0.05	2.4	-	21.6
	21½	Bleached zone	0.39	0.01	16.8	-	5.3
	35	Bleached zone	0.29	0.01	18.6	-	5.3
	40	Bleached zone	0.28	0.01	18.3	-	6.6
	50	Bleached zone	0.25	0.01	18.3	-	5.3
7A		Laterite:					
Barnes Hill	1	Ferruginous and limonitic zones	0.11	<0.01	0.07	-	24.8
	6	Mottled zone	0.71	0.11	0.91	-	31.2
7B		Laterite:					
Barnes Hill	9	Mottled zone	0.39	0.01	0.23	-	37.0
		Serpentinite:					
	17	Transition zone	0.87	0.05	5.8	-	21.9
	20	Transition zone	0.88	0.04	6.3	-	21.2
	27	Bleached zone	0.57	0.02	7.7	-	13.7
	32	Fresh zone	0.34	0.01	16.3	-	7.2
8		Laterite:					
Barnes Hill	5	Ferruginous red zone	0.11	0.01	0.24	-	42.6
	22	Mottled zone	0.18	<0.01	0.09	-	32.5
		Serpentinite:					
	35	Transition zone	1.26	0.14	1.91	-	18.1
	42	Transition zone	2.17	0.10	5.12	-	19.9
	48	Bleached zone	1.80	0.08	10.0	-	10.3
	60	Bleached zone	1.04	0.03	17.7	-	4.6
	76	Fresh zone	0.24	0.01	15.5	-	5.6
	90	Fresh zone	0.24	0.01	16.0	-	4.0
10		Laterite:					
Barnes Hill	6	Mottled zone	0.78	0.10	2.1	-	25.0
		Serpentinite:					
	8	Transition zone	0.78	0.10	2.1	-	25.0
	9	Transition zone	1.02	0.04	3.6	-	15.8
	12	Bleached zone	0.58	0.02	18.8	-	7.0

Continued

TABLE 5: CONTINUED

Drill Core and Location	Footage	Logs	Assay Results Ni	Co	Mg	Cr	Fe
11		Serpentinite:					
Barnes Hill	15	Fresh zone	0.20	0.01	23.2	-	3.4
12		Laterite:					
Barnes Hill	21	Ferruginous red zone	<0.01	0.01	0.10	-	38.0
	28	Limonitic yellow zone	0.12	0.14	0.10	-	27.4
	41	Mottled zone	0.59	0.22	0.20	-	35.2
		Serpentinite:					
	58	Transition zone	1.46	0.10	3.0	-	28.6
	61½	Transition zone	1.52	0.04	7.9	-	14.4
	68½	Bleached zone	1.22	0.04	7.3	-	15.4
	80	Fresh zone	0.64	0.01	22.5	-	5.3
13		Laterite:					
Barnes Hill	20	Mottled zone	0.32	0.07	0.52	-	33.8
		Serpentinite:					
	25	Transition zone	0.62	0.07	5.06	-	18.0
	31	Transition zone	0.96	0.05	6.07	-	24.4
	41	Bleached zone	1.20	0.04	13.3	-	17.3
	45	Bleached zone	0.71	0.02	18.6	-	16.3
	46	Fresh zone	0.62	0.01	22.6	-	6.5
14		Laterite:					
Scotts Hill	4	Ferruginous red zone	0.34	0.02	1.22	-	31.2
	19	Mottled zone	0.11	0.02	0.15	-	42.3
	26	Mottled zone	0.26	0.30	0.40	-	29.9
	34	Mottled zone	0.42	0.16	0.35	-	54.8
	44	Mottled zone	0.27	0.20	0.20	-	37.6
		Serpentinite:					
	47	Transition zone	0.74	0.14	2.2	-	22.9
	48	Bleached zone	1.06	0.08	2.1	-	15.1
	49	Bleached zone	1.06	0.08	2.1	-	15.1
	56	Bleached zone	0.85	0.04	1.92	-	29.1
	60	Fresh zone	0.34	0.02	21.4	-	6.7
15		Laterite:					
Scotts Hill	16	Limonitic yellow zone	1.08	0.02	0.77	-	21.0
	35	Mottled zone	0.27	0.06	1.38	-	23.8

Continued

TABLE 5: CONTINUED

Drill Core and Location	Footage	Logs	Assay Results				
			Ni	Co	Mg	Cr	Fe
15		Serpentinite:					
Scotts Hill	50	Transition zone	0.11	0.02	1.08	-	9.8
	58	Bleached zone	0.11	0.01	3.82	-	4.4
	65	Bleached zone	1.21	0.02	2.28	-	20.4
	69	Bleached zone	1.21	0.02	2.28	-	20.4
	76	Fresh zone	0.24	0.01	3.8	-	3.4
16		Laterite:					
Mt Vulcan	20	Mottled zone	0.13	0.02	0.1	-	46.6
	26	Mottled zone	0.12	0.04	0.2	-	44.8
		Serpentinite:					
	28	Transition zone	1.56	0.06	4.7	-	26.8
	30	Transition zone	1.56	0.06	4.7	-	26.8
	35	Bleached zone	1.36	0.04	10.2	-	22.8
	44	Bleached zone	1.16	0.05	6.4	-	18.4
	50	Bleached zone	0.70	0.02	7.5	-	17.2
	54	Bleached zone	0.71	0.02	15.2	-	11.6
	60	Fresh zone	0.40	0.01	17.4	-	6.2
	69	Fresh zone	0.44	0.01	20.0	-	4.6
17		Laterite:					
Flat Area	6	Ferruginous red zone	0.44	0.06	-	-	-
		Serpentinite:					
	18	Transition zone	0.33	0.02	-	-	-
	22	Bleached zone	0.42	0.03	-	-	-
	23	Bleached zone	0.42	0.03	-	-	-
	30	Fresh zone	0.13	0.02	-	-	-

(a) Data are from Senior Geologist (1967).

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TABLE 6: MINERALOGY OF NICKELIFEROUS HORIZONS (XRD)

Drill Hole	Depth ft	Head Assay Ni %	Mineralogy									
			Serpentine	Smectite	Chlorite	Hematite	Chromite	Quartz	Dolomite	Talc	Amphibole	Goethite
1	19 B	1.86	A	D	-	T	T	-	-	-	-	-
3	38 Mottled	1.18	-	D	-	-	T	T	-	-	-	-
5	16 T	0.93	-	D	-	-	T	-	-	-	-	-
6	17 T	0.80	-	D	-	-	A	T	-	-	-	T
7B	20 T	0.88	-	D	A	-	T	-	-	-	-	T
8	48 B	1.80	CD	CD	-	-	T	-	-	-	-	-
10	8 T	0.78	-	D	T	-	T	T	T	-	-	-
12	58 T	1.46	-	D	-	T	A	-	-	-	-	T
12	61½ T	1.52	-	D	A	-	T	-	-	-	-	-
13	41 B	1.20	SD	D	-	-	T	-	-	-	-	-
14	47 T/B	0.74	-	D	T	-	T	-	-	-	-	-
15	69 B	1.21	-	T	-	-	-	-	-	D	T	T
16	28 M/T	1.56	T	D	-	-	T	-	-	-	-	-

Note: A = Accessory. 10-20 T = Trace. <10
CD = Co-dominant. 30-50 D = Dominant. >50
SD = Sub-dominant. 20-30

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TABLE 7: MINERALOGY OF DDH 8 BARNES HILL

Drill Hole	Depth ft	Head Assay Ni %	Mineralogy						
			Serpentine	Smectite	Chromite	Kaolin	Hematite	Geothite	Chlorite
8	5 (ferrug red)	0.11	-	-	-	T ?	CD	CD	-
	22 (mottled)	0.18	A	D	T	-	-	A	-
	35 (mottled)	0.54	-	D	T	-	-	A	A
	42 transit.	2.17	-	D	T	-	-	T	-
	48 bleached	1.80	CD	CD	T	-	-	-	-
	60	1.04	D	-	T	-	-	T	-

Note:
A = Accessory. 10-20
CD = Co-dominant. 30-50
D = Dominant. >50
T = Trace. <10

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TABLE 8: MINERALOGY OF DDH 16 MT VULCAN

Drill Hole	Depth	Head Assay	Mineralogy			
	ft	Ni %	Serpentine	Smectite	Chromite	Goethite
16	26 mottled	0.12	A	T	-	D
	28	0.12	T	D	A	-
	30 transn.	1.56	-	D	A	-
	35 bleached	1.36	A	D	T	-
	44	1.16	A	D	T	-
	50	0.70	A	D	-	-
	54	0.71	D	A	A	-
	60 fresh	0.40	D	T	-	-

Note: A = Accessory.
T = Trace.
D = Dominant.

(this could indicate detrital chromite)

100

XRD

TABLE 9: MINERALOGY OF DDH 14 AND DDH 15 SCOTT'S HILL

Drill Hole	Depth ft	Head Assay Ni %	Mineralogy									
			Serpentine	Smectite	Chlorite	Amphibole	Talc	Calcite	Magnesite	Goethite	Chromite	Enstatite
14	ferrugined 4	0.34	–	–	–	–	–	–	–	D	T	–
14	mottled 34	0.42	–	–	–	–	–	–	–	D	A	–
14	mottled 41	0.27	–	D	T	–	A	–	–	A	T	–
14	trans~ 47	1.06 ?0.74	–	D	– ?	–	–	–	T	–	T	–
15	top mottled 16	1.08	A	–	–	–	–	–	–	D	T	–
15	trans~ 50	0.27	–	T	–	–	A	–	–	D	T	–
15	bleached 58	0.11	–	T	–	A	D	SD	T	–	–	–
15	65	0.17	–	CD	CD	A	A	SD	T	–	–	A
15	69	1.21	–	T	–	T	D	–	–	–	–	A

Note:
A = Accessory. 10-20
D = Dominant. >50
T = Trace. <10
CD = Co-dominant. 30-50
SD = Sub-dominant. 20-30

Table 11 shows chlorite ≥ 1% Ni at 47', MDH14. ?

? dolomite

TABLE 10: MINERALOGY OF DDH 17 FLAT AREA

Drill Hole	Depth ft	Head Assay Ni %	Mineralogy Serpentine	Smectite	Goethite
17	6	0.44	-	CD	CD
17	22	0.42	D	T	T
17	30	0.13	D	-	-

Note:
D = Dominant.
CD = Co-dominant.
T = Trace.

NB ? Absence of smectite (— smectite 6.)

TABLE 11: ELECTRON PROBE MICROANALYSES

Drill Hole	Depth ft	Head Assay Ni %	Nickel %									
			Serpentine	Smectite	Goethite	Chromite	Chlorite	Hematite	Talc	Amphibole	Manganese ?	Enstatite
8 m/T	35	0.54	-	0.2	1	0 m/T	0	-	-	-	-	-
8 T	42	2.17	-	1	4	0 T	-	-	-	-	-	-
8 B	60	1.04	0.8	-	1	0 B	-	-	-	-	-	-
12 T	58	1.46	-	0.4	5	0 T	-	T	-	-	-	-
14 T/B	47	0.74 (1.06)	-	T	-	0 T/B	1	-	-	-	-	-
15 B	69	1.21	-	T	1	- B	-	-	0.8	0	-	0.5
16 T	30	1.56	-	1	2	0 T	-	-	-	-	-	-
16 B	35	1.36	1.8	1.5	-	0 B	-	-	-	-	-	-
17 B	22	0.42	1	0.2	0	- B	-	-	-	-	5	-

Note:
- = Mineral not present in sample.
0 = Mineral present but devoid of nickel.
T = Trace (less than 0.2%).

(all zones)

TABLE 12: AVERAGE OF CHEMICAL ANALYSES FROM ULTRABASIC COMPLEX

Hole Number	SiO_2 %	CaO %	Loss on Ignition %	Al_2O_3 %	Mg %	Fe %	Co %	Cr %	Ni %
1	32.30	9.22	13.90	1.12	-	13.66	0.04	1.06 ?	0.98
3	13.19	0.70	10.50	7.43	2.3	26.43	0.05	0.35	0.45
4A	37.36	3.04	13.36	1.00	-	21.23	0.02	0.71 0.71	0.39
5	39.01	0.08	14.27	2.98	10.97	14.31	0.02	-	0.43
6	37.86	0.91	13.18	4.55	10.22	12.77	0.03	-	0.41
7B	40.32	4.05	14.86	5.90	9.06	17.51	0.01	-	0.45
8	38.74	1.73	14.09	10.33	6.64	17.83	0.03	-	0.57
9	33.23	0.11	13.04	4.48	15.88	17.32	0.04	-	0.35
10	43.08	2.08	18.40	8.10	15.97	7.97	0.02	-	0.33
12	30.31	3.74	13.54	11.57	5.93	24.70	0.05	-	0.48
13	34.29	0.94	14.46	10.74	8.95	20.16	0.08	-	0.50
14	27.86	0.19	14.43	7.97	5.11	27.92	0.09	-	0.36
15	43.07	0.15	11.68	7.02	4.18	15.32	0.02	-	0.35
16	30.88	1.44	12.75	7.11	7.90	23.34	0.02	-	0.53

TABLE 13: CHEMISTRY OF ORE HORIZON BEACONSFIELD ULTRABASIC COMPLEX

Hole Number	Depth to Ore, ft	Thickness of Ore, ft	Ni Average %	Ni Range %	Co Average %	Co Range %	Cr Average %	Cr Range %	Mg Average %	Mg Range %	Fe Average %	Fe Range %	SiO_2 Average %	SiO_2 Range %	CaO Average %	CaO Range %	Al_2O_3 Average %	Al_2O_3 Range %	LOI[a] Average %	LOI[a] Range %
1	7	39	1.37	0.6 ↓ 1.86	0.07	0.02 ↓ 0.30	0.75	0.03 ↓ 0.17	–	–	15.87	6.0 ↓ 44.5	32.30	13.4 ↓ 44.35	9.22	0.07 ↓ 20.77	1.12	0.3 ↓ 6.8	13.90	9.22 ↓ 15.65
3	28	25	0.95	0.41 ↓ 1.18	0.12	0.02 ↓ 0.25	0.18	0.04 ↓ 0.36	4.46	0.3 ↓ 12.3	27.40	5.5 ↓ 46.0	13.19	9.09 ↓ 19.25	0.70	0.03 ↓ 0.11	7.43	6.6 ↓ 12.2	10.50	8.11 ↓ 12.76
4A	20	5	1.12	–	0.07	–	0.11	–	–	–	12.4	–	42.97	–	0.17	–	2.1	–	16.00	–
5	5	12	0.83	0.73 ↓ 0.93	0.04	0.03 ↓ 0.05	–	–	2.7	1.2 ↓ 4.2	23.8	22.6 ↓ 25.1	36.19	31.72 ↓ 40.66	36.19	31.72 ↓ 40.66	4.45	1.2 ↓ 7.7	14.9	14.16 ↓ 15.64
6	6	14	0.75	0.70 ↓ 0.80	0.07	0.05 ↓ 0.09	–	–	1.4	0.9 ↓ 2.4	23.6	21.6 ↓ 26.1	33.65	27.41 ↓ 43.51	0.10	0.09 ↓ 0.12	5.63	2.5 ↓ 8.4	13.19	11.18 ↓ 16.94
7B	16	8	0.87	0.87 ↓ 0.88	0.04	0.04 ↓ 0.05	–	–	6.0	5.8 ↓ 6.3	21.5	21.2 ↓ 21.9	37.5	36.96 ↓ 38.04	0.16	0.06 ↓ 0.26	7.45	6.0 ↓ 8.9	16.17	15.16 ↓ 17.19
8	35	29	1.49	1.04 ↓ 2.17	0.08	0.03 ↓ 0.14	–	–	8.62	17.7 ↓ 1.1	12.7	4.6 ↓ 19.9	38.74	24.18 ↓ 47.59	1.73	0.03 ↓ 9.72	10.33	0.3 ↓ 20.7	14.09	11.07 ↓ 16.03
9	4	3	0.66	–	0.11	–	–	–	6.0	–	26.6	–	35.42	–	0.04	–	6.2	–	16.51	–
10	5	5	0.9	0.78 ↓ 1.02	0.07	0.04 ↓ 0.10	–	–	2.8	2.1 ↓ 3.6	20.4	15.8 ↓ 25.0	40.24	34.96 ↓ 45.53	0.23	0.19 ↓ 0.27	9.8	8.7 ↓ 11.0	20.53	19.27 ↓ 21.89
12	40	39	1.05	0.48 ↓ 1.52	0.08	0.04 ↓ 0.22	–	–	3.78	0.2 ↓ 7.9	25.2	14.4 ↓ 35.2	30.31	17.71 ↓ 39.73	3.74	0.06 ↓ 23.21	11.57	0.04 ↓ 16.2	13.54	10.39 ↓ 18.85

Continued.

TABLE 13: CONTINUED

Hole Number	Depth of Ore, ft	Thickness of Ore, ft	Ni Average %	Ni Range %	Co Average %	Co Range %	Cr Average %	Cr Range %	Mg Average %	Mg Range %	Fe Average %	Fe Range %	SiO_2 Average %	SiO_2 Range %	CaO Average %	CaO Range %	Al_2O_3 Average %	Al_2O_3 Range %	LOI[a] Average %	LOI[a] Range %
13	22	28	0.87	0.62 ↓ 1.20	0.04	0.02 ↓ 0.07			10.76	5.06 ↓ 18.6	19.0	16.3 ↓ 24.4	37.4	35.27 ↓ 41.58	2.14	0.05 ↓ 5.30	8.7	3.0 ↓ 17.0	14.83	11.53 ↓ 17.12
14	44	13	0.88	0.74 ↓ 1.06	0.08	0.04 ↓ 0.14	-	-	3.07	1.92 ↓ 2.2	22.36	15.1 ↓ 29.1	38.39	34.50 ↓ 45.12	0.16	0.14 ↓ 0.23	3.7	1.4 ↓ 7.7	17.38	16.89 ↓ 22.79
15A	8	8	1.08	-	0.02	-	-	-	0.77	-	21.0	-	-	-	-	-	-	-	-	-
15B	65	8	1.21	-	0.02	-	-	-	2.28	-	20.4	-	42.04	-	0.15	-	1.9	-	18.49	-
16	28	29	0.97	0.4 ↓ 1.56	0.03	0.01 ↓ 0.06	-	-	11.01	4.7 ↓ 17.4	16.3	6.2 ↓ 26.8	30.88	8.01 ↓ 40.81	1.44	0.05 ↓ 6.29	7.11	1.9 ↓ 13.2	12.75	8.36 ↓ 15.87

(a) LOI = Loss on ignition.

\- = Not determined.

TABLE 14: CALCULATION OF ANNUAL CASH FLOWS
For Mining and Treating 2200 Tons per Day By the Nicaro Process

Items	Years, $A,000																	
	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
Revenue																		
Volume (tons)	–	–	6435	6435	6435	6435	6435	6435	6435	6435	6435	6435	6435	6435	6435	6435	6435	–
Gross revenue	–	–	10287	10287	10287	10287	10287	10287	10287	10287	10287	10287	10287	10287	10287	10287	10287	–
Less royalties and rentals	–	–	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	–
Total revenue	–	–	10279	10279	10279	10279	10279	10279	10179	10279	10279	10279	10279	10279	10279	10279	10279	–
Operating Expenses																		
Processing:																		
Raw materials	–	–	432	432	432	432	432	432	432	432	432	432	432	432	432	432	432	–
Labour	–	–	639	665	692	720	749	779	810	842	875	909	945	982	1022	1062	1104	–
Fuel	–	–	825	825	825	825	825	825	825	825	825	825	825	825	825	825	825	–
Power	–	–	450	450	450	450	450	450	450	450	450	450	450	450	450	450	450	–
Water	–	–	18	18	18	18	18	18	18	18	18	18	18	18	18	18	18	–
Maintenance	–	–	820	820	820	820	820	820	820	820	820	820	820	820	820	820	820	–
General overhead	–	–	702	731	761	792	824	856	891	926	962	999	1039	1080	1124	1168	1210	–
General expenses	–	–	115	115	115	115	115	115	115	115	115	115	115	115	115	115	115	–
Depreciation	–	–	1788	1588	1409	1239	1100	976	866	769	682	605	538	477	423	376	333	–
Transportation:																		
Finished product	–	–	196	196	196	196	196	196	196	196	196	196	196	196	196	196	196	–
Mining costs	–	–	205	213	221	230	239	249	260	272	283	294	306	318	331	344	358	–
Total operating expenses	–	–	6190	6053	5939	5837	5768	5716	5683	5665	5658	5663	5684	5713	5756	5806	5816	–
Investment Allowance	–	–	3160	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Allowable Capital Expenditure	–	–	1600	–	–	–	–	–	660	–	–	–	–	–	–	–	–	–
Net Income (or loss)	–	–	(671)	4226	4340	4442	4511	4563	3936	4614	4621	4616	4595	4566	4523	4473	4463	–
Less exempt income	–	–	–	845	868	884	902	913	787	923	924	923	919	913	905	895	893	–
Less Section 80 losses	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Taxable Income	–	–	–	3381	3472	3558	3609	3650	3149	3691	3697	3693	3676	3653	3618	3578	3570	–
Tax at 45%	–	–	–	–	1521	1562	1601	1624	1643	1417	1661	1664	1662	1654	1644	1628	1610	1607
Profit After Tax	–	–	(671)	3381	1951	1996	2088	2026	1506	2274	2036	2029	2014	1999	1974	1950	1960	(1607)
Capital Investment																		
Land	(34)	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	34
Process equipment & loading facilities	(8000)	(8400)	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	2731
Mining investment	(100)	(840)	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Working capital	–	(1793)	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	1793
Housing	–	(160)	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Annual Cash Flow	(8134)	(11193)	5877	5814	4228	4119	4010	3915	3159	3966	3642	3557	3471	3389	3302	3221	3186	2951

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TABLE 15: NICARO PROCESS - DISCOUNTED CASH FLOW RATE OF RETURN CALCULATION

Year	Cash Flow $A,000	20% Discount Factor	Discounted Cash Flow $A,000	19% Discount Factor	Discounted Cash Flow $A,000
0	(8,134)	1.0000	(8,134)	1.0000	(8,134)
1	(11,193)	0.8333	(9,327)	0.8403	(9,405)
2	5,877	0.6944	4,081	0.7062	4,150
3	5,814	0.5787	3,365	0.5934	3,450
4	4,228	0.4823	2,039	0.4987	2,109
5	4,119	0.4019	1,655	0.4190	1,726
6	4,010	0.3349	1,343	0.3521	1,412
7	3,915	0.2791	1,093	0.2959	1,158
8	3,159	0.2326	735	0.2487	786
9	3,966	0.1938	769	0.2090	829
10	3,642	0.1615	588	0.1756	640
11	3,557	0.1346	479	0.1476	525
12	3,471	0.1122	389	0.1240	430
13	3,389	0.0935	317	0.1042	353
14	3,302	0.0779	257	0.0876	289
15	3,221	0.0649	209	0.0736	237
16	3,186	0.0541	172	0.0618	197
17	2,951	0.0451	133	0.0520	153
			163		905

Note: Interpolation of Exact Rate of Return.

$$\frac{1\%}{905 - 163} = \frac{x}{905}$$

$$x = 1.22\%$$

$$\therefore \text{ DCF rate of return} = 19\% + 1.22\%$$

$$= 20.22\%$$

TABLE 16: CALCULATION OF ANNUAL CASH FLOWS
For Mining and Treating 2200 Tons Per Day by the Pyrosulphidizing Process

Item	Years. $A,000																
	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
Revenue																	
Volume	–	30360	30360	30360	30360	30360	30360	30360	30360	30360	30360	30360	30360	30360	30360	30360	–
Gross revenue	–	6889	6889	6889	6889	6889	6889	6889	6889	6889	6889	6889	6889	6889	6889	6889	–
Less royalties and rentals	–	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	–
Total revenue	–	6881	6881	6881	6881	6881	6881	6881	6881	6881	6881	6881	6881	6881	6881	6881	–
Operating Expenses																	
Processing:																	
Raw materials	–	581	581	581	581	581	581	581	581	581	581	581	581	581	581	581	–
Labour	–	529	550	572	595	619	644	670	697	725	754	784	811	843	876	919	–
Fuel	–	660	660	660	660	660	660	660	660	660	660	660	660	660	660	660	–
Power	–	80	80	80	80	80	80	80	80	80	80	80	80	80	80	80	–
Water	–	18	18	18	18	18	18	18	18	18	18	18	18	18	18	18	–
Maintenance	–	450	450	450	450	450	450	450	450	450	450	450	450	450	450	450	–
General overhead	–	581	605	629	654	680	708	737	766	797	829	862	892	925	963	1011	–
General expenses	–	115	115	115	115	115	115	115	115	115	115	115	115	115	115	115	–
Depreciation	–	979	869	771	684	607	539	478	424	376	334	297	263	234	206	184	–
Transportation:																	
Finished product	–	332	332	332	332	332	332	332	332	332	332	332	332	332	332	332	–
Mining costs	–	205	213	221	230	239	249	260	272	283	294	306	318	331	344	358	–
Total operating expenses	–	4530	4473	4429	4399	4381	4376	4381	4395	4417	4447	4485	4520	4569	4625	4708	–
Investment Allowance	–	1700	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Allowable Capital Expenditure	–	1400	–	–	–	–	–	–	660	–	–	–	–	–	–	–	–
Net Income (or loss)	–	(749)	2408	2452	2482	2500	2505	2500	1826	2464	2434	2396	2361	2312	2256	2173	–
Less exempt income	–	–	482	490	496	500	501	500	365	493	487	479	472	462	451	435	–
Less Section 80 losses	–	–	267	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Taxable Income	–	–	1659	1962	1986	2000	2004	2000	1461	1971	1947	1917	1889	1850	1805	1738	–
Tax at 45%	–	–	–	747	883	894	900	902	900	657	887	876	863	850	833	812	782
Profit After Tax	–	(749)	1659	1215	1103	1106	1104	1098	561	1314	1060	1041	1026	1000	972	926	(782)
Capital Investment																	
Land	(50)	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	50
Process equipment	(8800)	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	1456
Off-loading and storage equipment	(200)	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Mining investment	(1000)	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Working capital	(818)	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	818
Housing	(100)	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Annual Cash Flow	(10968)	3330	3277	2476	2283	2213	2144	2076	1350	2183	1881	1817	1761	1696	1629	1545	1542

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TABLE 17: PYROSULPHIDIZING PROCESS - DISCOUNTED CASH FLOW RATE OF RETURN CALCULATION

Year	Cash Flow $A,000	20% Discount Factor	Discounted Cash Flow $A,000	25% Discount Factor	Discounted Cash Flow $A,000
0	(10,968)	1.0000	(10,968)	1.0000	(10,968)
1	3,330	0.8333	2,775	0.8000	2,664
2	3,277	0.6944	2,276	0.6400	2,097
3	2,476	0.5787	1,433	0.5120	1,268
4	2,283	0.4823	1,101	0.4096	935
5	2,213	0.4019	889	0.3277	725
6	2,144	0.3349	718	0.2621	562
7	2,076	0.2791	579	0.2097	435
8	1,350	0.2326	314	0.1678	227
9	2,183	0.1938	423	0.1342	293
10	1,881	0.1615	304	0.1074	202
11	1,817	0.1346	245	0.0859	156
12	1,761	0.1122	198	0.0687	121
13	1,696	0.0935	159	0.0550	93
14	1,629	0.0779	127	0.0440	71
15	1,545	0.0649	100	0.0352	54
16	1,542	0.0541	83	0.0281	43
			756		(1,022)

Note: Interpolation of Exact Rate of Return,

$$\frac{5\%}{1022 + 756} = \frac{x\%}{756}$$

$$x = 2.12\%$$

$$\therefore \text{ DCF rate of return } = 20\% + 2.12\%$$

$$= 22.12\%.$$

FIGURES 1 - 4

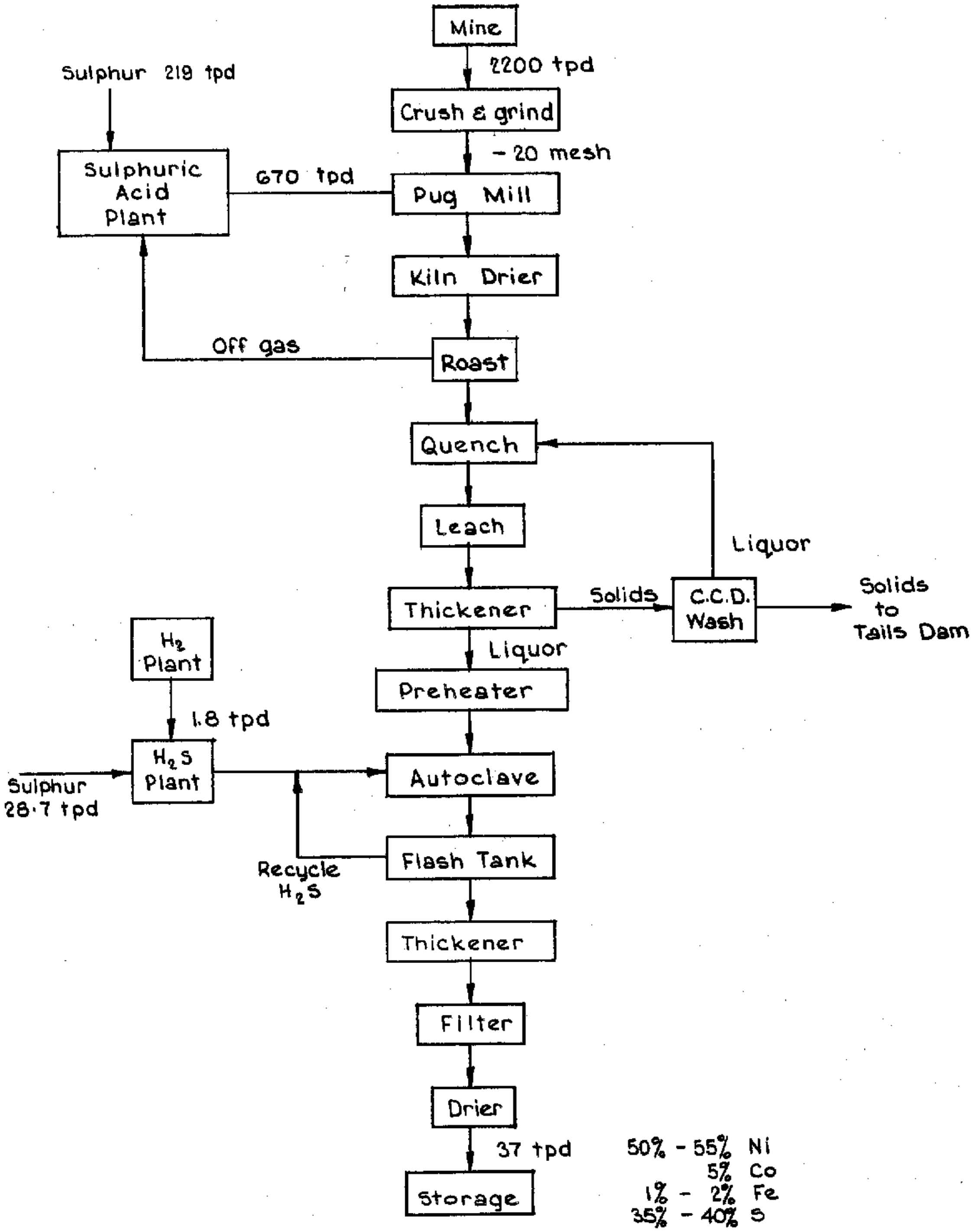

FIG.1: PUG ROAST ACID LEACH PROCESS FLOWSHEET
733,000 tons of ore per annum

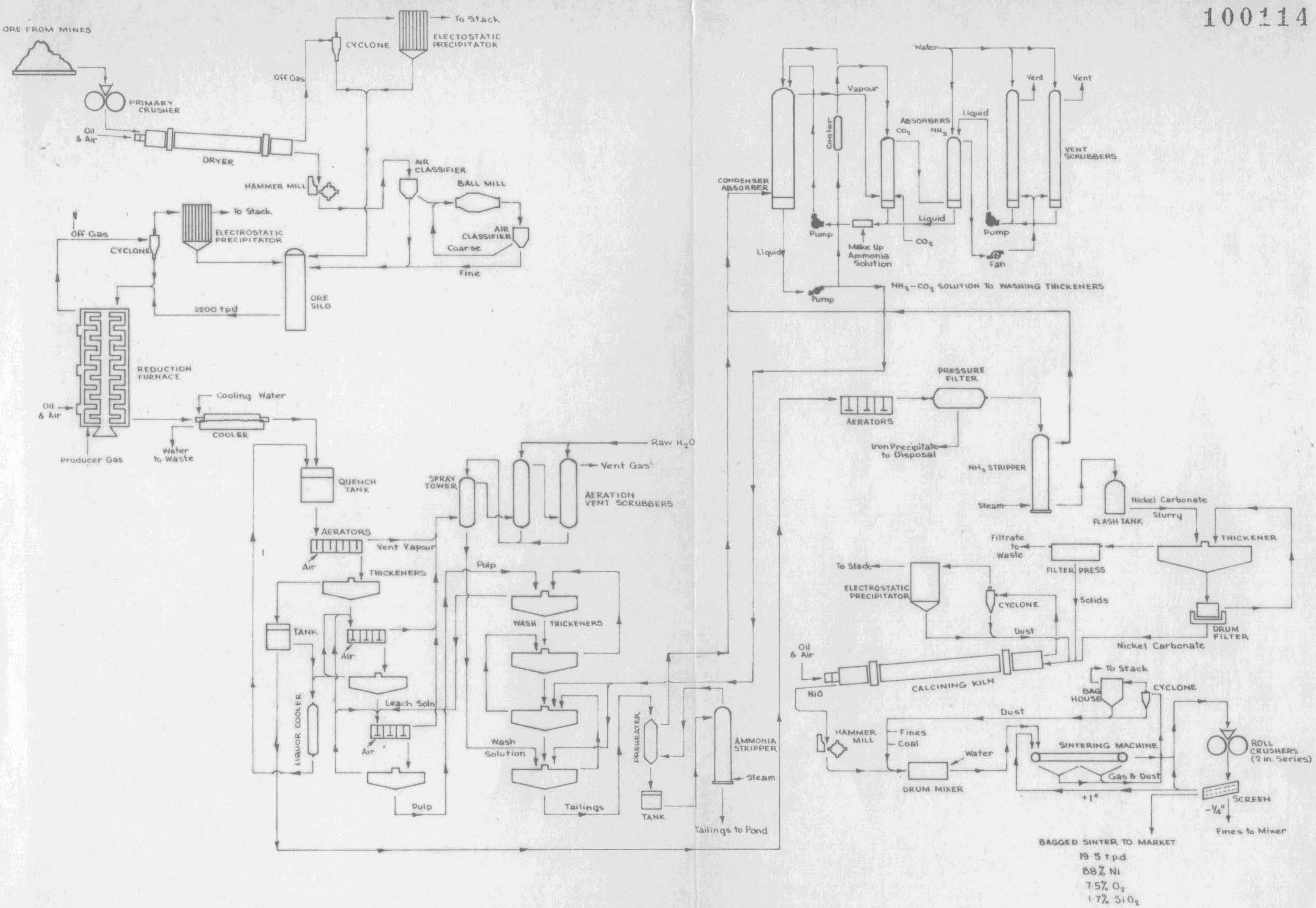

FIGURE 2: NICARO PROCESS FLOWSHEET
(733,000 tons of ore per annum)

RG 2369

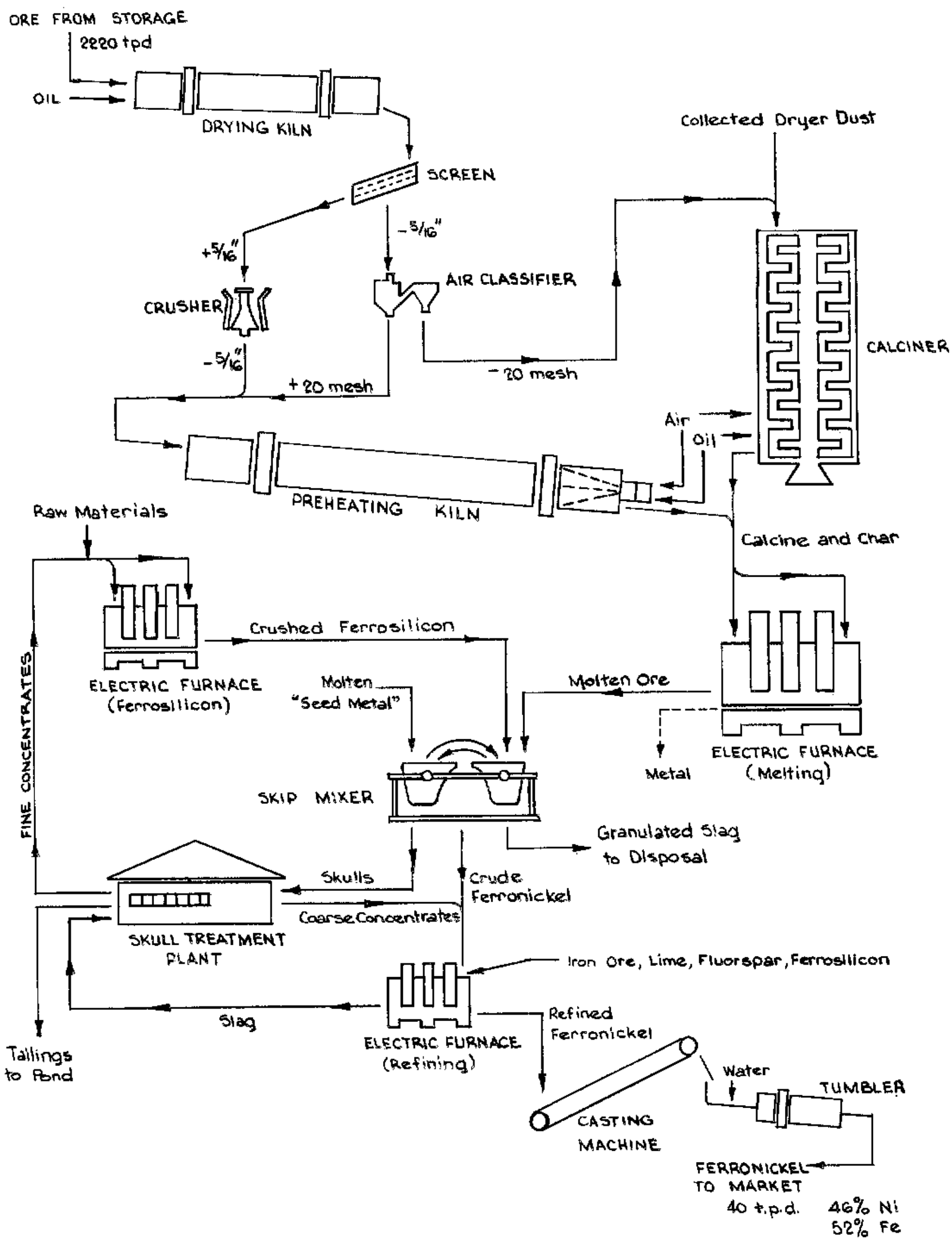

FIG.3: UGINE PROCESS FLOWSHEET
733,000 tons of ore per annum

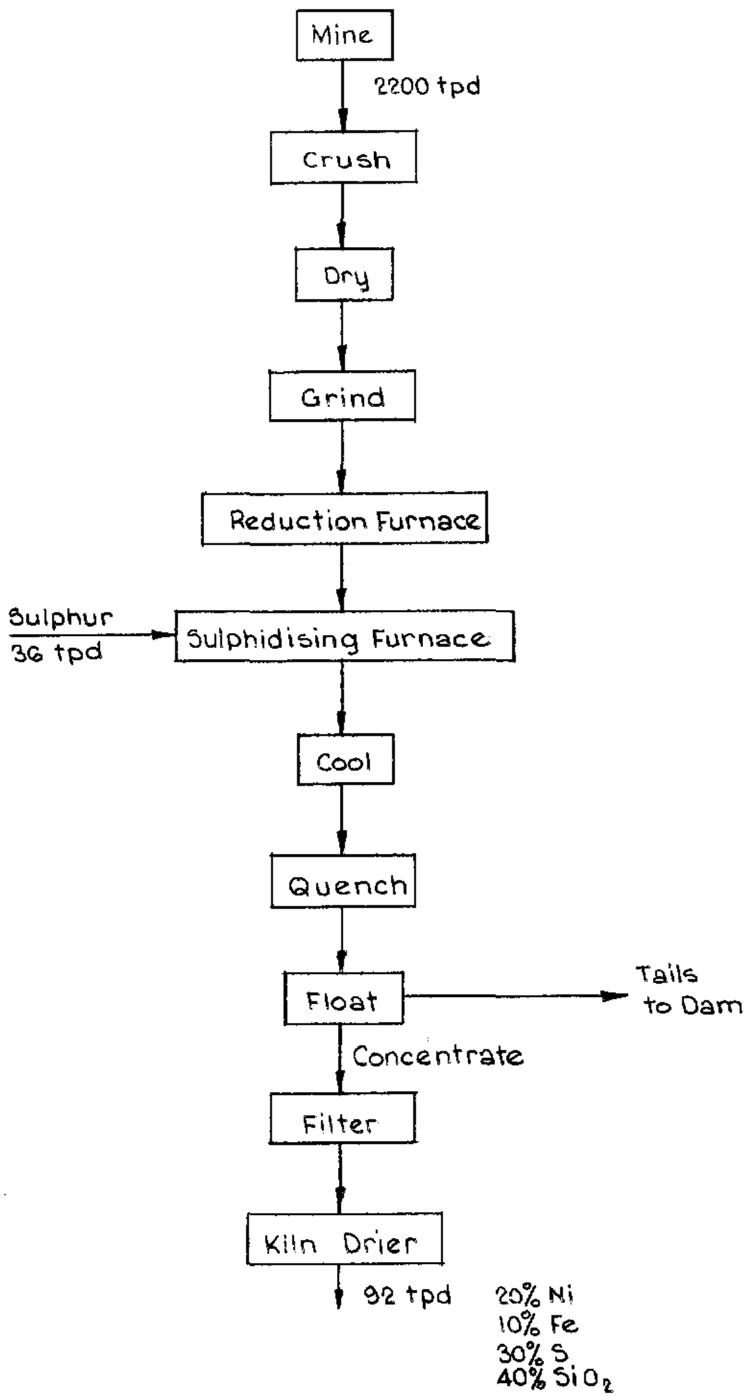

FIG.4: PYROSULPHIDIZING PROCESS FLOWSHEET
733,000 tons of ore per annum