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Environmental Aspects of the Extraction and Production of Nickel

Economic and Technical Review
Report EPS 3-AP-82-5

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82

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ENVIRONMENTAL ASPECTS OF THE EXTRACTION AND PRODUCTION OF NICKEL*

Background Paper Prepared
for
UNITED NATIONS ENVIRONMENTAL PROGRAM

by

Mineral and Metal Processes Division
Engineering Assessment Branch
Air Pollution Control Directorate

Report EPS 3-AP-82-5
July 1982

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* This report is based on a study done for Environment Canada in 1980 by INCO TECH, Toronto, Ontario under contract No. OSS 79-00130.

Cat. No. En 42-3/82-5E
ISBN 0-662-12099-X
BEAUREGARD PRESS LIMITED

ABSTRACT

This review includes an identification of the environmental problems in the extraction and production of nickel.

The review covers the two basic ore types: (a) the sulphide ore and (b) the oxide ore of nickel (laterites) and also touches on the outlook for recovery and processing of deep sea nodules.

World mineral resources and the supply and demand situation relative to nickel are discussed.

Various technologies used in the processing of Nickel ore to produce refined nickel are described as well as technologies to control air, water and soil pollution.

RÉSUMÉ

La présente étude traite des problèmes que causent à l'environnement l'extraction et la production du nickel.

Elle porte sur les deux types fondamentaux de minerais: (a) les sulfures et (b) les oxydes de nickel (latérites) et examine rapidement les perspectives de la récupération et du traitement des modules des grands fonds marins.

On y discute des ressources minérales mondiales et de l'offre et de la demande du nickel.

Divers procédés de raffinage du nickel ainsi que des techniques de dépollution de l'air, de l'eau et du sol sont décrits.

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1 INTRODUCTION

Mankind's use of nickel predates recorded history. Many very early bronze and copper artifacts contain substantial quantities of the metal. The "white copper" of Chinese commerce over one thousand years ago was a direct predecessor of modern nickel silver (1).

Nickel was isolated and named by A.F. Cronstedt in 1751 but for more than a century it remained little more than a scientific curiosity.

As recently as the early 1870's, world nickel production was only about 400 tonnes annually. This lethargic pace changed abruptly during the final quarter of the nineteenth century, largely as a result of three events:

1. The discovery, between 1863 and 1867, by Jules Garnier, of huge deposits of garnierite (a hydrous nickel-magnesium silicate) in New Caledonia. Mining commenced in 1875 and ore containing 8 to 10% nickel was shipped to Europe. Shortly thereafter a smelter was built in New Caledonia which shipped matte containing over 60% nickel and which was the forerunner of the present smelter at Noumea.
2. The discovery of extensive deposits of what was then believed to be sulphide copper ore at Sudbury, Canada. Mining operations began in 1886 (2). Unexpected problems were encountered upon smelting because of the presence of nickel. Research on a process to separate the two metals occupied metallurgists for several years and resulted by 1894 in the development of the Orford process (3). This process was in use, albeit with numerous modifications, until 1948, when it was replaced by the copper-nickel matte separation process now in operation at Sudbury and in the U.S.S.R.

   At the turn of the century, New Caledonia supplied about 65% of the world's nickel requirement with the remainder supplied from Sudbury. Within the first decade of this century the ratio was virtually reversed, and in 1916 the Sudbury mines produced 41 500 tonnes of nickel or 80% of the world's supply (4). Sudbury continues to be the world's largest single source of nickel.
3. The discovery by James Riley in 1889 of the properties which nickel imparted to steel was an event of profound significance to the nickel industry, as the armed forces of the world rushed to take advantage of the new material for

their armour (5). From this beginning, the demand for nickel as an alloying element in both ferrous and non-ferrous metallurgy increased rapidly and steadily as its special properties added a new dimension to metallurgical science. In the past one hundred years, annual nickel consumption has risen from 400 tonnes to more than 800 000 tonnes.

Tables 1 and 2 present statistics on the world's nickel reserves and metal production (1977) (6).

TABLE 1 WORLD NICKEL RESERVES, 1977 ('000 tonnes Ni content)

	Reserves	Other Resources[1]	Total Resources[1]
North America:			
United States	200	14 900	15 100
Canada	9 600	11 600	21 200
Total	9 800	26 500	36 300
Africa	2 300	6 700	9 000
Central America and Caribbean Islands:			
Cuba	3 400	14 200	17 600
Dominican Republic	1 100	100	1 200
Guatemala	300	900	1 200
Puerto Rico	-	900	900
Total	4 800	16 100	20 900
Europe: U.S.S.R.	8 100	13 200	21 300
Oceania:			
Australia	5 600	3 200	8 800
Indonesia	7 800	6 600	14 400
New Caledonia	15 000	31 300	46 300
Philippines	5 400	5 800	11 200
Total	33 800	46 900	80 700
South America:			
Brazil	200	3 300	3 500
Columbia	900	600	1 500
Venezuela	-	700	700
Total	1 100	4 600	5 700
World Total (rounded)[2]	60 000	114 000	174 000

[1] Derived in consultation with U.S. Geological Survey.
[2] Excludes nickel associated with seabed nodules.

TABLE 2 WORLD NICKEL MINE PRODUCTION, 1977

Country	Company	Nickel Products	Capacity (tonnes) Nickel Content)
Australia	Western Mining Corp. Ltd.	Nickel metal and matte	
	Queensland Nickel Pty.	Nickel oxide and mixed nickel-cobalt sulphides	80 000
Botswana	Botswana R.S.T.	Nickel-copper-cobalt matte	18 000
Canada	INCO Ltd.	Nickel oxide sinter, soluble nickel oxide,) nickel metal (cathode and pellets)	
	Falconbridge Nickel Mines Ltd.	Nickel-copper matte	270 000
	Sherritt Gordon Mines Ltd.	Nickel metal	
Cuba	Cubaniquel (state owned)	Nickel oxide and sulphide	36 000
Dominican Republic	Falconbridge Dominicana C. Por A.	Ferronickel	34 000
Greece	LARCO-Societe Miniere et Metallurgique de Larymna S.A.	Ferronickel	11 000
Indonesia	PT/INCO		30 000
New Caledonia	Societe Metallurgique de Nickel (SLN)	Ferronickel and matte	136 000
Philippines	Marinduque Mining Corp.	Nickel metal and nickel-cobalt sulphides	32 000
Rhodesia	Shangani Ltd. (JCI)		12 000
South Africa	Rustenburg Platinum Mines Ltd.	Nickel metal	
	Matte Smelters Pty. Ltd.	Nickel matte	23 000
U.S.S.R.	State owned	Nickel metal and matte	N/A

1.1 Mineral Resources

The world's land-based nickel ores are of two generic types, sulphide and lateritic. Arsenide deposits are known but are currently not classed as ores. Sea nodules which occur at great depths on the surface of the ocean floor, particularly in areas of the South Pacific, have captured interest in the past twenty-five years and are thought to constitute an almost limitless supply of nickel, cobalt and manganese. Research on mining and processing of the sea nodules is being carried out and commercial exploitation may begin in the 1990's.

1.2 Sulphide Ores

The sulphide ores of nickel occur at numerous, widespread locations. The more important now being exploited are those of Sudbury and Thompson, Canada; Kalgoorlie, Australia; and Norilsk, U.S.S.R. They are all of igneous origin and are mined at depths of up to 9000 feet. In all of these ores, the nickel is present in the mineral pentlandite $(NiFe)_9 S_8$, usually associated with large quantities of an iron sulphide mineral, pyrrhotite Fe_7S_8, which is often slightly magnetic. The ore commonly contains copper as chalcopyrite ($CuFeS_2$) as well as traces of the platinum group metals, gold and silver. Trace elements important from a refining and environmental standpoint are also present in small amounts. The sulphides are massive or are dispersed in a matrix of basic or ultra basic rock (peridotite).

There are wide variations in ore composition from one area to another even within a single deposit. At Sudbury the ratio of the three main sulphides - pyrrhotite, pentlandite, chalcopyrite - averages approximately 4.7 to 1.0 to 1.

The Thompson, Canada ore contains only small quantities of copper while the Norilsk, U.S.S.R. ore has a copper content two to three times that of nickel. The ores also vary widely in the ratio of pyrrhotite to nickel. This ratio is an important factor in determining both the process and the environmental effects. The ores of Sudbury and Norilsk both have high pyrrhotite: Ni ratios, commonly in excess of 10 to 1.

The sulphide ores were formed from molten magma with sulphur acting as the collector of the metallic elements. The sulphide minerals precipitated, from the melt, in discrete particles distinct from the rock matrix. Upon grinding, the sulphide minerals break away from the rock and from each other because of weaknesses at the grain boundaries. This makes it possible, by subsequent chemical or physical means, to discard the bulk of the barren rock as tailings and to separate one sulphide mineral from another. Figure 1 (7) is a photomicrograph of Sudbury ore illustrating the distribution and relative content of the rock and the three major sulphide minerals.

1.3 Lateritic Ores

The lateritic ores of nickel occur in many areas of the tropics and sub-tropics. Nickel is present as an oxide or complex silicate in quasi-molecular subdivisions. Consequently the ores are not amenable to concentration by flotation. In certain deposits some upgrading is possible through separation, by primary crushing and screening, of relatively unaltered cores of partially weathered boulders.

The nickel laterites were formed through weathering of nickel-bearing peridotite, an ultra-basic rock composed mainly of olivine and containing about 0.2% nickel.

The nickel in lateritic deposits is usually concentrated near the bottom of the zone of weathering, being overlain by a layer of iron-rich earth. Depending on weathering factors, some or even most of the nickel may be retained in the upper zone which is enriched in both iron and nickel as a result of selective dissolution of magnesia and silica. Such material containing up to 50% iron and 1.3 to 1.55% nickel is classed as limonitic ore. This contrasts with the silicate ore occurring at the lower levels where the nickel is concentrated in the form of silicate minerals through precipitation from the metal-bearing solutions percolating from above. In almost every lateritic nickel ore area the two types are present but in widely varying proportions. Ores that are predominantly limonitic are characterized by those of Moa Bay, Cuba, containing about 1.4% nickel, 0.15% Co, and 48% Fe. Ores that are predominantly silicate are characterized by those of New Caledonia which contain typically 2.8% Ni, 0.06% Co, 13% Fe, 37% SiO_2 and 25% MgO.

From the standpoint of extractive metallurgy, the high iron, relatively low nickel content and the high cobalt-to-nickel ratio of the limonitic ores favour hydrometallurgical treatment. The silicate ores are treated almost exclusively by pyrometallurgical processes.

1.4 Sea Nodules

Seabed nodules, as a source of nickel supply for the future, are still surrounded by many unknowns which make investment in seabed mining extremely risky - unless

Figure 1

PHOTOMICROGRAPHS OF POLISHED AND STAINED SUDBURY ORE

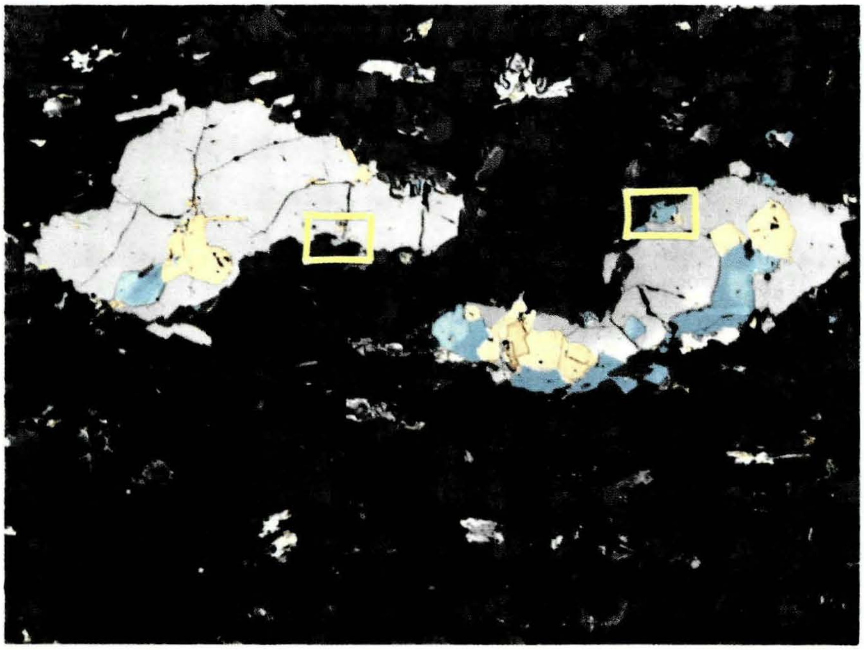


Fig. 1a/ Magnification 20x

Figures 1b and 1c are enlarged views of the outlined areas in the left and right grains respectively of Figure 1a.

	MINERAL
Olive	'Rock'
Pinky-beige	Pyrrhotite
Gold	Pentlandite
Blue	Chalcopyrite

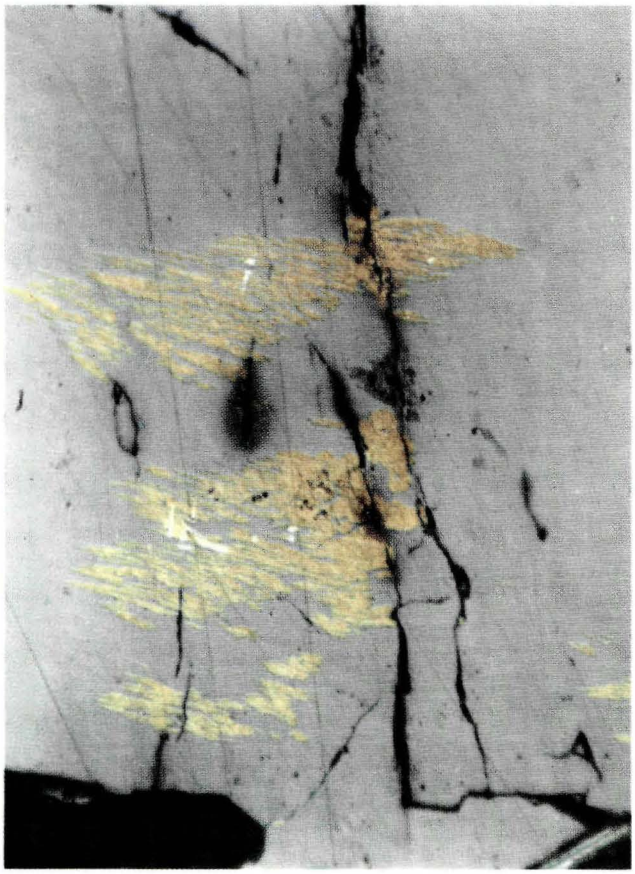


Fig. 1b/ Magnification 400x

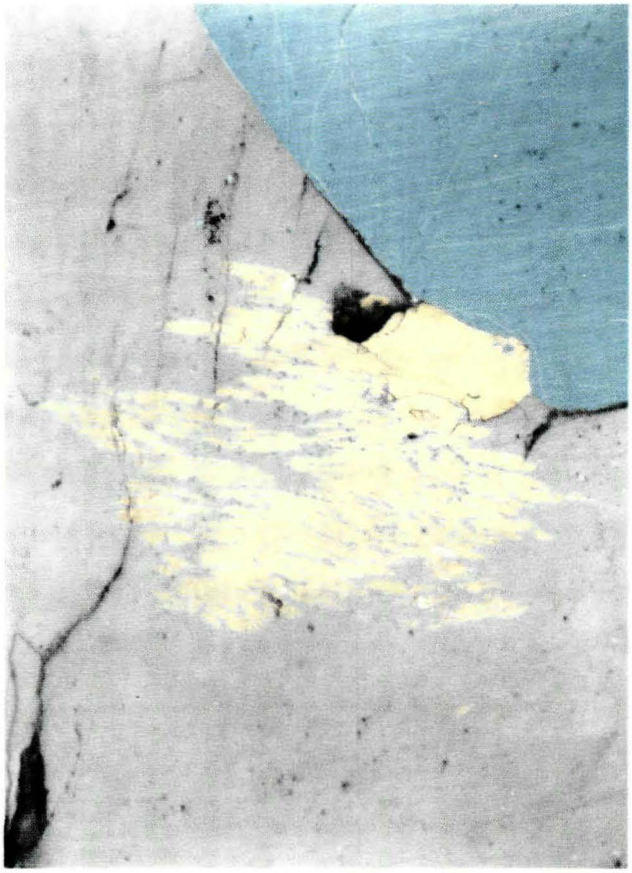


Fig. 1c/ Magnification 400x

extraordinarily optimistic forecasts regarding future demand developments are accepted. These unknowns range from the exploration and delineation of nodule beds adequate for mine planning, the problems of actual mining and hoisting to the surface, the smelting and refining processes to be used, the still unresolved legal issues connected with the U.N. Law of the Sea Conference, and the questions surrounding royalty or taxation levels to be imposed by the U.N. Sea Bed Authority on ocean mining operations.

Under present and foreseeable circumstances, it does not appear likely that any seabed mining will take place on a commercially significant scale before the 1990's.

2 NICKEL SUPPLY AND DEMAND

2.1 Supply

Table 3 shows the world mine production of nickel for the years 1975-79. The sources of supply of nickel have changed considerably over the years. Prior to 1900 most of the world's nickel came from New Caledonia (laterites). During the early 1900's, Sudbury (sulphides) became the major producer. In recent years, many new nickel producers have started operations, reducing Inco's share of the total and increasing the proportion produced from lateritic ores.

Canada remains the largest producer, all from sulphide ores, with a production of 230 000 to 240 000 tonnes per year (lower in 1978 and 1979 due to Inco strike) or about 30% of world production. About 60% of present nickel production is from sulphide ores, the balance from lateritic ores. The majority of new nickel mining projects are for lateritic ores, and consequently the proportion of nickel produced from sulphide ores will continue to decrease.

Sulphide ores are mined in Canada (Ontario and Manitoba), the Soviet Union, Australia, South Africa, Botswana, Zimbabwe, Finland and in several other countries in small amounts. Lateritic ores are mined in New Caledonia, Cuba, Indonesia, the Soviet Union, the Philippines, the Dominican Republic, Greece, Australia, the United States, Brazil and, in small quantities, in several other countries.

2.2 Demand

Global nickel consumption is dependent on the worldwide level of industrial production and the rate of capital expenditure for plant and equipment. About 70% of nickel consumption is derived from capital expenditure oriented economic activities.

In the period 1950-74, nickel consumption increased at a rate of 6.4% per year in the non-centrally planned world. Since 1975, there have been drastic increases in energy costs, high rates of inflation and rapid increases in social demands and costs, which have resulted in a major worldwide slowdown in capital formation and reinvestment in new plants and equipment. This has resulted in a significant decline in worldwide nickel consumption and will cause a slowdown in future consumption growth rates. The world nickel demand growth is now estimated at 4 to 5% per year for this decade. Nickel consumption is shown in Figure 2 for the 1950-90 period, by area for 1979 in Figure 3, and by use in Figure 4.

TABLE 3 WORLD MINE PRODUCTION OF NICKEL, 1975-79 (tonnes) (8)

	1975	1976	1977	1978	1979
Australia (b)	83 583	90 976	94 653	90 785	81 600
Botswana	18 314	13 866	13 331	17 691	17 600
Brazil (c)	3 516	5 812	6 100	3 924	4 400
Burma (d)	21	26	19	20	20
Canada (e)	266 957	265 464	259 442	143 360	145 040
Dominican Republic	29 652	26 896	26 676	15 763	27 650
Finland (b)	5 840	7 008	6 393	4 825	6 428
Greece (g)	31 014	30 380	21 000 (a)	24 140	22 700
Indonesia (c) (h)	21 193	21 928	15 400 (a)	35 179	39 352
Mexico (c)	55	62	65 (a)	24	22
Morocco (i)	550	220	220 (a)	192	187
New Caledonia (j)	146 938	131 065	120 218	65 490	80 942
Norway (b)	335	440 (a)	440 (a)	-	-
Philippines (a)	10 472	20 723	16 500	34 222	39 740
Republic of South Africa	22 877	24 660	25 400	24 250	24 250
United States (k)	16 987	16 469	14 347	13 504	14 100
Zimbabwe (a) (f)	11 000	17 600	17 600 (a)	17 600	17 600
Sub-total	669 304	673 595	637 804	490 969	521 631
Albania (a)	7 100	7 700	8 000	8 800	9 400
Cuba (a) (f)	40 300	40 600	40 800	40 800	40 800
Germany D.R. (a)	2 800	2 800	2 800	3 000	3 000
Poland (a) (c)	2 400	3 100	3 100	3 100	3 100
U.S.S.R. (c)	168 000	176 000	159 000	163 000	168 000
Sub-total	220 600	230 200	213 700	218 700	224 300
Total	889 904	903 795	851 504	709 669	745 931

Source: U.S. Bureau of Mines. In so far as possible, this table represents mine production of nickel; where data relate to some more highly processed form, the figures given have been used in lieu of unreported actual mine-out to provide some indication of the magnitude of mine output, and are so noted.

(a) Estimated. (b) Content of concentrate. (c) Content of ore. (d) Content of speiss. (e) Refined nickel and nickel content of oxides and salts produced, plus recoverable nickel in exported mattes and speiss. (f) Content of oxide and sulphide. (g) Recoverable content of ore. (h) Includes a small amount of cobalt not reported separately. (i) Content of nickel ore and cobalt ore. (j) Nickel-cobalt content of metallurgical plant products, plus recoverable nickel-cobalt in exported ores. Since 1977, data are given in nickel alone. (k) Content of ore shipped.

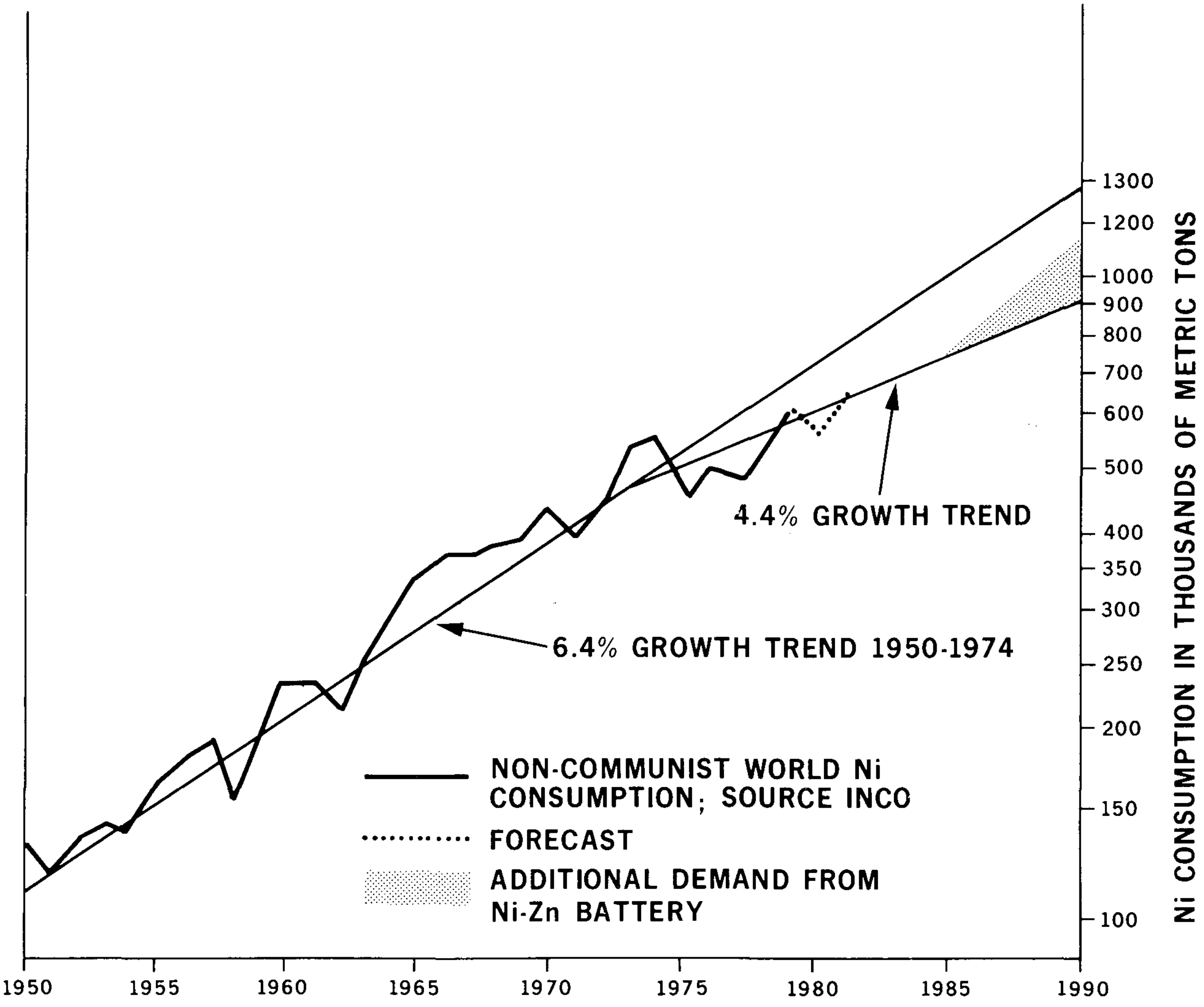


FIGURE 2 NICKEL CONSUMPTION (EXCLUDING CENTRALLY PLANNED ECONOMIES) (11)

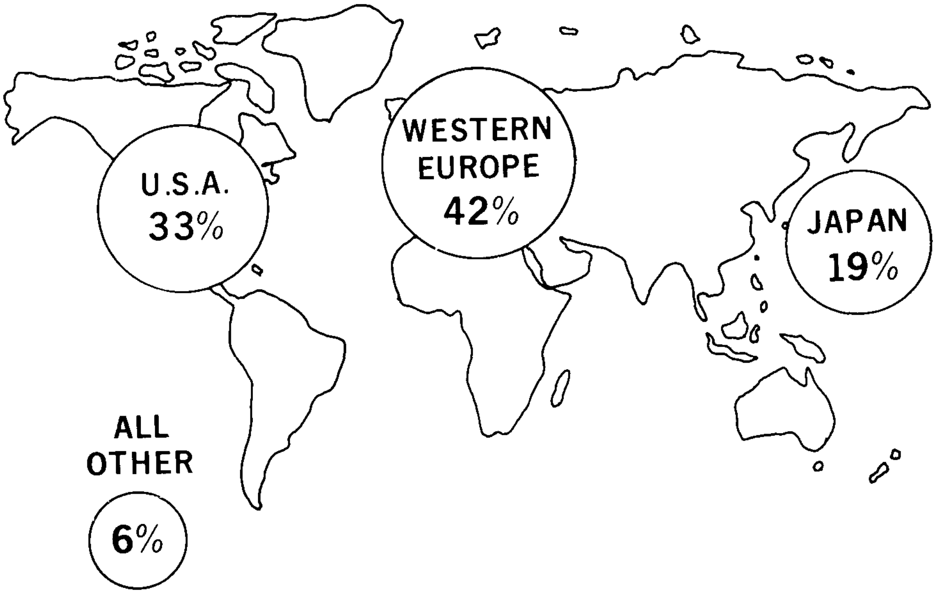


FIGURE 3 NICKEL CONSUMPTION BY AREA (EXCLUDING CENTRALLY PLANNED ECONOMIES) (6)

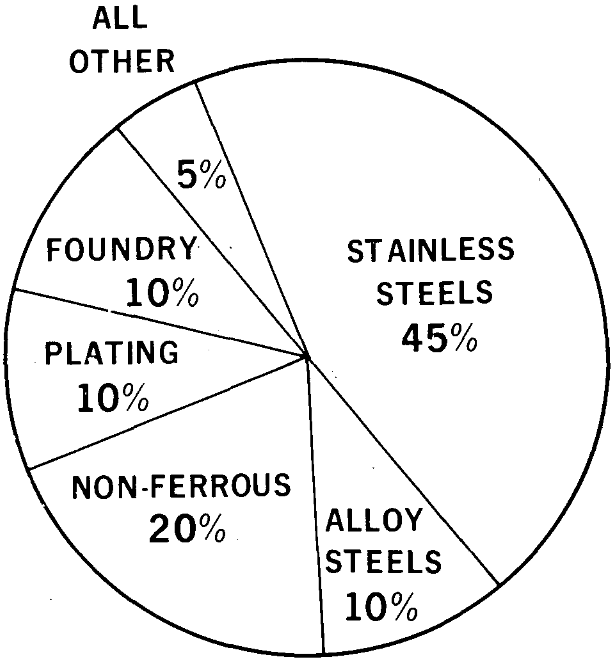


FIGURE 4 NICKEL CONSUMPTION BY FIRST USE (EXCLUDING CENTRALLY PLANNED ECONOMIES), 1979 (6)

Inco's share of the refined nickel market (excluding centrally planned economies) in 1977 was 31%. Falconbridge and SLN (Societe Anonyme Le Nickel - New Caledonia) each accounted for 9%, Japan 17%, supplies from the U.S.S.R. and Cuba 4%, and 14 other producers the remaining 30%, with the largest of these at 3%. The advent of many new producers into the nickel industry in recent years, combined with an oversupply situation in the 1975-77 period when supply exceeded demand by about 270 000 tonnes resulted in fiercely competitive nickel markets in the late 1970's (9).

It is estimated that by the end of 1980, nickel production capacity, excluding centrally planned economies, will have grown to about 725 000 tonnes. Production from Inco's two overseas projects, P.T. Inco Indonesia and Exmibal in Guatemala, and Larco's expansion in Greece are included in this estimate. An estimated additional 32 000 tonnes will be imported from the U.S.S.R. and Cuba.

Consumption and trade of nickel are shown in Tables 4 to 14.

2.3 Type of Product

Primary nickel is produced in some twenty different forms, divided into two major classifications based on product purity (10).

CLASS I NICKEL includes carbonyl nickel pellets and nickel cathodes, both with nickel content exceeding 99.9%. Briquettes, rondelles and NICKEL 98 are also usually included in this category although they are not suited for use directly in electroplating.

CLASS II NICKEL includes INCOMET containing about 95% nickel, Sinter 75 containing 75% nickel together with the various grades of ferronickel containing 20 to 55% nickel.

For many large-tonnage applications, notably stainless and alloy steels, the purity of Class I nickel is not required and the less-refined Class II material is adequate. In recent years Class II products have been making inroads into the nickel market, rising from 19% in 1955 to 38% in 1970 and to an anticipated 48% in 1980 (Inco).

Class I nickel can be readily produced from sulphide ores and from lateritic ores of the limonitic type. It can also be produced from lateritic ores of the silicate type, but a matte production step is necessary.

Class II nickel can be readily produced from all types of ore, although the silicate type of lateritic ore is more suited to the production of Class II types, especially ferronickel.

TABLE 4 WORLD CONSUMPTION OF NICKEL BY PRINCIPAL CONSUMING COUNTRIES, 1971-77

- 000 tonnes -

Country/Region	1971	1972	1973	1974	1975	1976	1977
Africa	4.0	4.0	5.0	6.0	5.5	5.0	5.2
Australia	4.0	4.0	5.0	5.0	3.5	4.0	4.3
Austria	3.0	3.7	5.3	5.2	4.3	4.2	5.4
BLEU	2.2	2.8	3.7	5.2	3.4	3.5	2.6
Brazil		2.9	5.3	6.6	4.0	4.7	6.0
Canada	7.9	11.5	13.0	14.3	10.8	11.5	8.0
Denmark	0.1	0.1	0.1	0.2	0.2	0.2	0.1
Finland	0.3	0.3	0.4	0.5	0.5	1.1	2.8
France	32.4	37.8	29.6	40.5	39.4	33.5	35.8
Fed. Rep. of Germany	34.3	43.0	54.8	61.2	42.8	56.4	54.2
Greece				0.2	0.1	0.1	0.1
Italy	18.0	18.4	20.4	21.1	17.0	22.0	22.6
Japan	89.5	83.3	111.2	119.1	90.0	121.2	97.3
Netherlands	0.7	0.7	1.3	1.7	1.2	1.5	1.1
Norway	0.9	0.7	0.9	0.9	0.9	0.7	0.4
Portugal		0.3			0.3	0.4	0.2
Spain	3.4	4.3	5.3	6.0	5.0	6.4	8.2
Sweden	19.0	21.4	26.8	31.9	22.0	24.0	17.5
Switzerland	1.8	1.4	1.0	1.1	1.6	1.8	1.3
U.K.	29.3	30.0	31.5	33.5	25.5	23.3	30.5
U.S.	116.8	144.5	179.4	189.1	132.9	147.8	140.9
Yugoslavia		1.0	1.4	1.4	1.1	1.4	1.6
Other	9.2	5.0	6.5	7.3	3.8	11.0	14.7
Total	376.8	421.1	507.9	558.0	415.8	485.7	460.8
Bulgaria	...	0.5	0.5	1.0	1.0	1.2	1.2
Czechoslovakia	...	6.0	7.0	7.0	8.0	10.0	11.0
Dem. Rep. of Germany	...	8.0	9.0	9.0	10.0	10.0	10.5
Hungary	...	1.5	1.5	1.8	2.0	2.2	2.2
Poland	...	5.0	5.0	7.0	7.5	7.8	8.0
Romania	...	2.9	4.0	4.0	4.5	6.0	6.0
U.S.S.R.	...	100.0	100.0	105.0	115.0	121.0	125.0
China	...	22.0	18.0	18.0	18.0	18.0	18.0
Other Centrally Planned Economies	...	0.4	0.5	0.5	0.5	0.5	0.5
Total	140.0	146.3	145.5	153.3	166.5	176.7	182.4
World Total	516.8	567.4	653.4	711.3	582.3	662.4	643.2

Refined nickel plus nickel content of ferronickel, nickel oxide and fonte, excluding scrap.
Includes estimates.
Source: World Bureau of Metal Statistics.

TABLE 5 WESTERN WORLD NICKEL CONSUMPTION BY PRODUCT CLASSIFICATION, SELECTED YEARS (12)

	Actual (000t)			Estimated (000t)				
Product	1961	1966	1971	1972	1973	1974	1975	1976
Electro-nickel	173.2	239.1	250.8	261.7	287.6	275.4	193.0	267.1
Ferronickel	39.3	70.9	100.1	125.9	165.3	201.7	164.1	170.0
Nickel oxide	13.4	42.5	45.8	44.0	60.7	70.6	53.7	48.6
Total	225.9	352.5	396.7	431.6	513.6	547.7	410.8	485.7

TABLE 6 WESTERN WORLD CONSUMPTION OF PRIMARY NICKEL BY INDUSTRY, 1971 and 1976 (12)

	1971 (INCO)		1976 (SLN)	
Industry	000t Ni	% of total	000t Ni	% of total
Stainless Steel	153.6	41	243.0	50
Nickel Plating	59.5	16	62.0	13
High-Nickel Alloys	45.0	12	62.0	13
Constructional Alloy Steels	37.3	10	51.0	10
Iron and Steel Castings	34.1	9	36.0	7
Copper and Brass Products	11.4	3	17.0	3
Total	340.9	91	471.0	96

The trace elements in the ore also have an influence on the type of nickel produced. Sulphide ores generally have recoverable amounts of copper, cobalt, gold and platinum group metals. Processing to Class I products is generally carried out to enhance the recovery of by-products, but some Class II products can also be produced without detrimental losses of by-product metals. The limonite type of lateritic ores generally have cobalt contents that are rich enough for cobalt recovery. The cobalt content of the silicate type of lateritic ores generally is too low to be commercially recoverable.

In Canada, Inco Limited accounts for about 80% of Canadian production - all from sulphide ore. Sherritt Gordon Mines Limited, with an annual production of about 12 000 tonnes, now derives its refinery input almost entirely from Inco, Thompson. Falconbridge Nickel Mines, Ltd. accounts for the remainder of Canadian production or about 50 000 tonnes.

TABLE 7 U.S. CONSUMPTION OF NICKEL BY END-USE INDUSTRY, 1966-76 ('000 tonnes) (12)

	1966	1967	1968	1969	1970	1971	1972	1973	1974	1975	1976
Chemicals	21.4	26.0	22.4	34.3	33.4	29.7	35.2	41.2	43.0	28.0	31.1
Petroleum	11.7	14.1	12.4	17.3	17.8	17.9	21.3	24.7	26.1	18.5	18.6
Fabricated Metal Products	28.7	40.3	25.0	18.7	21.3	20.3	23.6	27.5	26.1	17.3	18.7
Transportation:											
Aircraft	30.4	16.5	24.3	14.9	13.4	13.9	16.2	19.2	17.0	11.9	18.1
Motor Vehicles and Equipment	36.4	24.0	22.1	23.4	26.7	21.8	26.4	30.2	29.4	21.5	22.7
Ship and Boat Building and Repairs	11.7	7.3	9.2	8.6	6.6	6.1	6.8	8.4	13.4	10.2	8.7
Electrical	34.7	20.2	20.7	30.0	28.8	25.7	30.6	35.6	34.4	25.6	26.9
Household Appliances	28.9	18.4	18.7	13.7	14.5	13.6	16.5	19.2	21.7	12.7	15.1
Machinery	42.6	12.9	13.1	16.3	16.5	13.7	16.5	19.2	23.0	16.4	16.2
Construction	13.8	11.0	10.8	16.1	21.3	17.9	21.3	24.7	28.5	18.4	19.6
Other	31.6	44.4	20.1	30.1	23.1	18.0	20.9	23.6	21.0	15.6	17.5
Total	291.9	235.1	198.8	223.4	223.4	198.6	235.3	273.5	283.6	196.1	213.2

TABLE 8 TRENDS IN U.S. NICKEL INTERMEDIATE AND END-USES (12)

	1972-76	1974-75	1975-76
		% per annum	
Stainless and Heat Resisting Steel	-0.6	-50.5	+29.6
Alloy Steel	-1.5	-13.2	-6.3
Superalloys	-9.7	-48.7	+36.0
Nickel-Copper & Copper-Nickel Alloys	-5.6	0.0	-24.0
Permanent Magnet Alloys	+6.5	-38.1	+76.5
Other Nickel and Nickel Alloys	+1.3	-15.7	-14.8
Cast Irons	-11.8	-17.6	-8.6
Electroplating	-1.6	-28.3	+46.2
Chemicals & Chemical Uses	-0.8	+46.9	+30.9
Other Uses	-10.1	-20.1	-3.6

Since Inco Limited was and remains the dominant producer from sulphide ore, the evolution of mining and processing at Inco forms a suitable background for tracing the development of the industry.

TABLE 9 EXPORTS OF NICKEL ORES, CONCENTRATES, MATTE AND SPEISS BY PRINCIPAL EXPORTING COUNTRIES, 1977 (000 tonnes) (12)

	Exporting Countries						
Country of Destination	Canada	Indonesia*	Philippines	Netherlands	New Caledonia*	Norway	Total
Finland	-	-	-	-	-	14	14
France	-	-	-	-	18	-	18
Fed. Rep. of Germany	-	-	-	1	-	-	1
Japan	4	878	393	-	2 680	-	3 955
Norway	35	-	-	-	-	-	35
Sweden	-	-	-	-	-	-	-
U.K.	41	-	-	-	-	-	41
Others	-	-	-	-	-	-	-
Total	80	878	393	1	2 698	14	4 064

* Obtained from import statistics.

TABLE 10 IMPORTS OF NICKEL ORES, CONCENTRATES, MATTE AND SPEISS BY PRINCIPAL IMPORTING COUNTRIES, 1976 (000 tonnes) (12)

Country of Origin	Importing Countries											
	Austria	Canada*	Finland	France	Fed. Rep. of Germany	Italy	Japan	Norway	Sweden	Switzer-land	U.K.	Total
Australia	-	17.2	-	-	8.4	0.3	27.9	3.2	3.0	0.1	2.9	63.0
BLEU	-	-	-	-	-	0.1	-	-	-	-	-	0.1
Canada	-	-	-	-	4.2	3.6	13.2	63.9	0.4	0.4	51.1	136.8
Cuba	0.6	-	-	-	0.7	1.0	-	-	-	-	-	2.3
Indonesia	-	-	-	-	-	-	811.7	-	-	-	-	811.7
New Caledonia	-	-	-	15.1	-	-	2639.7	0.2	-	-	-	2655.0
Norway	-	-	12.9	-	-	-	-	-	-	-	0.3	13.2
Philippines	-	-	-	-	-	-	3.7	-	-	-	-	3.7
U.K.	-	9.4	-	-	0.1	0.1	-	0.1	-	0.4	-	10.1
U.S.	0.4	6.1	-	-	0.4	-	-	-	-	-	-	6.9
Others	0.2	0.4	-	-	-	0.4	-	4.2	1.2	0.8	0.8	8.0
Total	1.2	33.1	12.9	15.1	13.8	5.5	3496.2	71.6	4.6	1.7	55.1	3710.8

*Includes scrap.

TABLE 11 EXPORTS OF FERRO-NICKEL BY PRINCIPAL EXPORTING COUNTRIES, 1976 (000 tonnes) (12)

	Exporting Countries					
Country of Destination	Dominican Republic	France	Greece	Japan	New Caledonia	Total
Austria	-	2.2	-	-	-	2.2
BLEU	-	2.6	-	1.0	-	3.6
France	-	-	4.0	-	93.0	97.0
Fed. Rep. of Germany	5.1	16.8	20.0	-	-	41.9
Italy	5.0	13.8	-	-	-	18.8
Japan	1.8	-	-	-	3.0	4.8
Netherlands	-	0.1	-	9.0	-	9.1
Norway	-	0.3	-	-	-	0.3
Romania	-	0.6	-	-	-	0.6
Spain	-	1.0	-	-	-	1.0
Sweden	-	9.2	11.2	2.0	-	22.4
U.K.	-	4.3	-	-	-	4.3
U.S.	26.9	-	2.9	-	16.9	46.7
U.S.S.R.	-	4.2	-	-	-	4.2
Others	-	0.3	18.9	1.0	-	20.2
Total	38.8	55.4	57.0	13.0	112.9	277.1

TABLE 12 EXPORTS OF FERRO-NICKEL BY PRINCIPAL EXPORTING COUNTRIES, 1977 (000 tonnes) (12)

	Exporting Countries					
Country of Destination	Dominican Republic	France	Greece*	Japan	New Caledonia	Total
Austria	-	4.7	-	-	-	4.7
BLEU	-	-	-	1.0	-	1.0
France	-	-	6.0	-	82.8	88.8
Fed. Rep. of Germany	3.1	18.5	12.7	-	-	34.3
Italy	6.2	9.1	-	-	-	15.3
Japan	3.8	-	-	-	-	3.8
Netherlands	-	-	-	-	-	-
Norway	-	0.2	-	-	-	0.2
Romania	-	1.0	-	-	-	1.0
Spain	-	1.2	-	-	-	1.2
Sweden	-	5.6	6.6	1.0	-	13.2
U.K.	-	5.9	5.2	-	-	11.1
U.S.	29.0	-	1.5	-	27.8	58.3
U.S.S.R.	-	3.4	-	-	-	3.4
Others	-	1.0	-	-	4.4	5.4
Total	42.1	50.6	32.0	2.0	115.0	241.7

*Compiled from import statistics.

TABLE 13 EXPORTS OF UNWROUGHT NICKEL ALLOYS BY PRINCIPAL EXPORTING COUNTRIES, 1977 (000 tonnes) (12)

	Exporting Countries												
Country of Destination	BLEU	Canada	Finland	France	Fed. Rep. of Germany	Italy	Japan	Nether-lands	Sweden	Switzer-land	U.K.	U.S.	Total
Australia	-	-	-	-	-	-	0.04	-	-	-	-	-	0.04
Austria	-	-	-	-	-	-	-	0.23	-	-	0.60	-	0.83
BLEU	-	-	-	-	0.01	-	-	0.09	-	-	1.10	-	1.20
Brazil	-	-	-	-	-	-	0.34	-	-	-	-	0.20	0.54
France	0.03	-	-	-	1.14	-	-	0.07	-	-	3.20	0.14	4.58
Fed. Rep of Germany	-	-	-	0.14	-	-	-	0.39	-	-	7.40	8.42	16.35
India	-	-	-	-	-	-	0.20	0.08	-	-	1.60	-	1.88
Italy	-	-	-	-	-	-	0.25	-	-	-	2.60	-	2.85
Japan	-	-	-	-	-	-	-	0.36	-	-	-	0.25	0.61
Netherlands	-	-	-	-	0.02	-	0.96	-	2.16	-	1.10	3.27	7.51
Poland	0.14	-	-	-	-	-	-	-	-	-	-	-	0.14
Spain	-	-	-	-	0.01	-	0.40	0.04	-	-	1.20	-	1.65
Sweden	-	-	-	-	-	-	-	0.12	-	-	2.10	-	2.22
Switzerland	-	-	-	-	-	0.55	-	0.03	-	-	2.50	-	3.08
U.K.	-	-	-	0.01	-	-	0.07	0.20	-	-	-	0.19	0.47
U.S.	-	56.19	-	0.01	-	-	0.88	0.05	-	-	-	-	57.13
Yugoslavia	-	-	-	-	0.02	0.03	-	-	-	-	-	-	0.05
Others	0.06	27.43	0.03	0.20	0.33	0.01	1.04	0.07	0.04	0.01	1.10	1.54	31.86
Total	0.23	83.62	0.03	0.36	1.53	0.59	4.18	1.73	2.20	0.01	24.50	14.01	132.99

TABLE 14 EXPORTS OF NICKEL SEMI-MANUFACTURES BY PRINCIPAL EXPORTING COUNTRIES, 1977 (tonnes) (12)

	Exporting Countries												
Country of Destination	Austria	BLEU	Canada	France	Fed. Rep. of Germany	Italy	Japan	Nether-lands	Sweden	Switzer-land	U.K.	U.S.	Total
Australia	1	-	164	-	31	-	912	-	39	2	184	257	1 590
Austria	-	-	-	3	821	-	-	-	44	65	225	-	1 158
BLEU	2	-	19	591	1 363	-	10	7	4	7	527	166	2 696
Brazil	13	-	22	43	129	-	59	-	132	1	35	197	631
Canada	-	-	-	-	496	-	4	3	2	-	80	4 793	5 378
Denmark	1	-	-	-	355	-	-	-	11	228	59	-	654
France	7	14	1	-	3 128	7	-	61	153	396	1 407	954	6 128
Fed. Rep of Germany	81	-	32	875	-	241	103	211	57	330	1 145	512	3 587
India	2	-	36	26	62	49	280	-	2	5	234	15	711
Iran	34	-	-	-	616	-	3	3	-	1	10	-	667
Irish Republic	-	-	-	50	76	-	-	-	-	4	125	-	255
Italy	12	-	83	119	655	-	-	38	71	154	831	229	2 192
Japan	-	-	342	70	55	-	-	-	1	-	384	298	1 150
Mexico	4	-	17	12	5	-	-	6	94	-	12	575	725
Netherlands	2	-	30	111	1 144	102	12	-	48	43	692	611	2 795
Norway	-	-	-	-	403	-	-	-	27	4	118	4	556
Poland	-	-	-	5	6	-	-	3	130	8	92	-	244
Romania	191	-	-	61	246	93	1	-	6	7	27	-	632
South Africa	-	-	23	-	284	-	88	4	12	-	493	24	928
Spain	9	-	109	24	862	-	1	20	110	12	169	48	1 364
Sweden	53	-	117	-	378	-	-	2 341	-	60	720	106	3 775
Switzerland	18	-	1	23	753	-	-	5	44	-	417	113	1 374
U.K.	-	-	1 063	102	1 118	24	4	28	127	482	-	675	3 623
U.S.	83	-	8 994	663	2 270	2	116	40	146	17	1 193	-	13 524
Yugoslavia	15	-	-	-	496	-	-	4	25	12	21	-	573
Others nsl	185	2 524	169	417	2 947	304	1 204	174	145	246	2 097	860	11 272
Total	713	2 538	11 222	3 195	18 699	822	2 797	2 948	1 430	2 084	11 297	10 437	68 182

3 MINING SULPHIDE ORES

The mining of the sulphide ores is generally conducted at levels deep underground. In 1977, Inco operated ten underground mines and two open pits in Sudbury; one underground mine at Shebandowan, Ontario; and two underground mines and one open pit in Thompson, Manitoba for a total annual ore production of 19.6 million tonnes.

In 1977, Falconbridge mined 2.8 million tonnes at Sudbury from underground mines.

In all other areas of the world except the U.S.S.R., the mines are underground. In the U.S.S.R., ore tonnages are not known. The new mines serving Norilsk, the principal sulphide-nickel processing complex, are at Talnach, about twenty miles north of Norilsk, and are all underground mines. The development here has permitted retirement of the Norilsk underground mine and the two large open pits which served the metallurgical complex from its inception in the early 1930's. Norilsk is in Siberia at 70° N Latitude and mining presents unique problems of ground instability from penetration of permafrost to depths of 300 metres. Pechenga, formerly Petsamo, was developed by Inco prior to World War II when the property was in Finland. The ore body was exploited from underground and finally from surface until it was exhausted in about 1968. The smelter at Pachenga continues to operate on concentrates produced locally from several small open-pit operations and from direct-smelting-ore shipped from Norilsk.

The scope of this report precludes a description of the many underground mining techniques employed to recover ore under the widely varying conditions presented by the ore bodies. In general the evolution of mining has followed the trend towards bulk mining and mechanization with particular emphasis on the development of huge, rugged machinery. In recent years diesel trackless locomotion has tended to replace electric powered transportation underground.

3.1 Environmental Problems of Mining Sulphide Ores

Comprehensive monitoring of the atmosphere, the use of catalytic afterburners on the engine exhausts together with adequate ventilation have minimized problems of air quality in the mining areas.

Fracturing of the ore at the mining face uses a mixture of fuel oil and ammonium nitrate which is safer and more versatile than dynamite. From an environmental standpoint its use increases the quantity of nitrogenous compounds in mine water and in the mine atmosphere. The latter is minimized by ventilation but no satisfactory treatment is available for removal of nitrogen from the water, and the discharge of such mine water to receiving waterways is causing some concern. Fortunately water flow in the receiving streams is usually high enough to minimize the nutrient effect of the added nitrogen. Usually nitrogen content of mine water discharge does not exceed ~50 mg/L.

A second environmental concern relates to the quality of mine water discharge. Water in underground mines originates from various sources such as drainage of surface waters, water used in drilling and dust abatement and for back filling of mined-out stopes. In sulphide-ore mines, particularly where the pyrrhotite content is high, oxidation of iron sulphide to ferric sulphate readily occurs. Subsequent hydrolysis of iron produces sulphuric acid. This is a particularly severe problem in Sudbury where the rock has low acid buffering capacity. Here, mine water pH of 3.5 is common and it carries significant quantities of dissolved nickel and semicolloidal nickel sulphide.

The problem has been addressed at the Inco and Falconbridge mines in Sudbury by the treatment of waters discharged from the various mines and from the tailings areas at central wastewater facilities. Treatment is based on neutralization with lime to a pH of about 9.0 to precipitate heavy metals. Subsequent flocculation and clarification provides water of acceptable quality for discharge to public waterways.

To minimize the quantity of water to be treated, a maximum amount of tailings water is recycled to the mill without treatment. For example, at Inco, where process water requirements exceed 560 000 m^3 per day, 85% of the water derives from recycle.

At Thompson, Manitoba and Shebandowan, Ontario, the mine rock is sufficiently alkaline and active to neutralize all of the acid formed upon iron oxidation and hydrolysis. Here, mine water pH is naturally about 9.0 and is discharged with tailings to ponds from which it overflows to the receiving waters.

4 MINING LATERITIC ORES (13)

Lateritic ores are near-surface deposits of softer, sometimes clay-like decomposed material -- the product of intense, prolonged chemical weathering of nickel-bearing rock, with leaching and redepositing of the nickel and other elements and eventual breakdown of the rock. The ore usually lies under a soft overburden of variable depth and itself varies widely in depth within the limits of a single deposit. Techniques quite different from those used in hard rock mining have been worked out for mining these oxide-nickel ores.

Essentially the mining task is, in the engineer's parlance, "an earth-moving job". These deposits can usually be excavated by surface earth-moving equipment such as power shovels, draglines, and bulldozers. Blasting for the most part is not required. Drilling is needed only for such special reasons as mining control. However, the task is not as simple as in the more ordinary earth-moving jobs. The advantages of soft ore are offset to a large degree by the non-uniformity in grade of ore and inconsistency in depth of ore. Advance planning is necessary, and the mining requires constant sampling and skill in maintaining the proper grade of material mined and selected for the treatment plant.

After ore extraction, the mined-out areas may be backfilled with waste materials. Recontouring of the final topography after mining, followed by topsoil recovering and revegatation, complete the mining process (14).

4.1 Environmental Problems of Mining Lateritic Ores

Stripping and mining operations destroy the natural land contours and vegetation and expose lateritic surfaces to the weather (in some cases to very heavy tropical rainstorms).

Rehabilitation plans must be prepared before stripping begins. The mining sequence, access, erosion control earthworks and final contours must be planned to prevent run-off of turbid water during and after mining (15).

Surveys to monitor silt deposition and metal levels in waters receiving run-off must be done (16).

Proper techniques for handling topsoil in order to preserve its fertility and regeneration potential should be adhered to (15). Additional seeding is normally required in accordance with the land use planned for the mined area.

5 MINERAL PROCESSING

5.1 Inco - Sudbury

From the turn of the century to the early 1930's, Inco was the world's only large scale processor of sulphide-nickel ore. Consequently, process development at Sudbury is illustrative of the industry as a whole.

Processing of Sudbury ores began two decades before the advent of froth flotation. Because of the high ore grades available, it was not necessary to employ flotation until 1930. As for most non-ferrous base metals, flotation was a boon to nickel. It created ore from sulphide bodies which would otherwise have been too lean for exploitation. The process probably represents the century's most important advance in extractive metallurgy.

Until 1930 the treatment of Sudbury nickel ores involved only hand sorting to eliminate some obviously barren rock followed by open heap roasting and blast furnace smelting. The 50-year period after 1930 was prolific in process innovation and witnessed the phenomenal expansion in production capacity. The growth and development of the industry is illustrated in the series of decennial flow diagrams of Figures 5 through 10.

Figure 5 is a simplified flow diagram of operations as they were conducted in 1918 (10). It also substantially represents metallurgical practice from the turn of the century.

Figure 6 is a corresponding flow diagram covering a decade later, in 1928. The only significant change in operating practice involved the beginnings of a nickel electrorefining operation by the newly acquired Hybinette process (17).

The flow diagram for 1938, Figure 7, shows a considerable increase in process complexity and in the number of marketable products obtained from the ores. During this decade a selective flotation concentrator, a new reverberatory furnace smelter, an electrolytic copper refinery and a plant to produce sulphuric acid from smelter gas were built at Copper Cliff (17). Outdoor heap roasting was abandoned, with virtually all of the ore beneficiated by selective flotation to produce copper-rich and nickel-rich concentrates, which were handled in separate sections of the smelter. The nickel concentrate was roasted in multi-hearth units from which calcines passed by gravity flow to coal-fired reverberatory furnaces; the much higher grade copper concentrate went directly to reverberatory smelting. Selective flotation permitted a substantial part of the copper

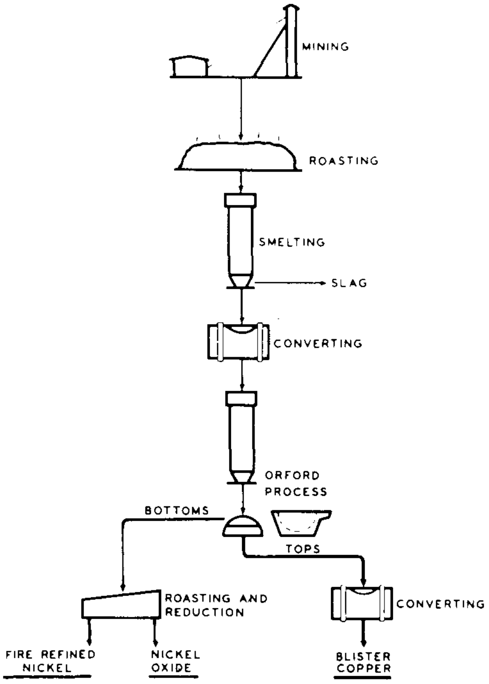


FIGURE 5 TREATMENT OF SUDBURY ORE IN 1918 AT INCO METALS CO.

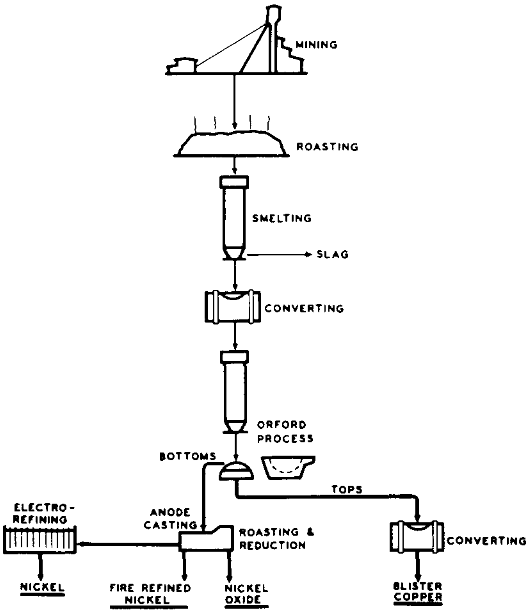


FIGURE 6 TREATMENT OF SUDBURY ORE IN 1928 AT INCO METALS CO.

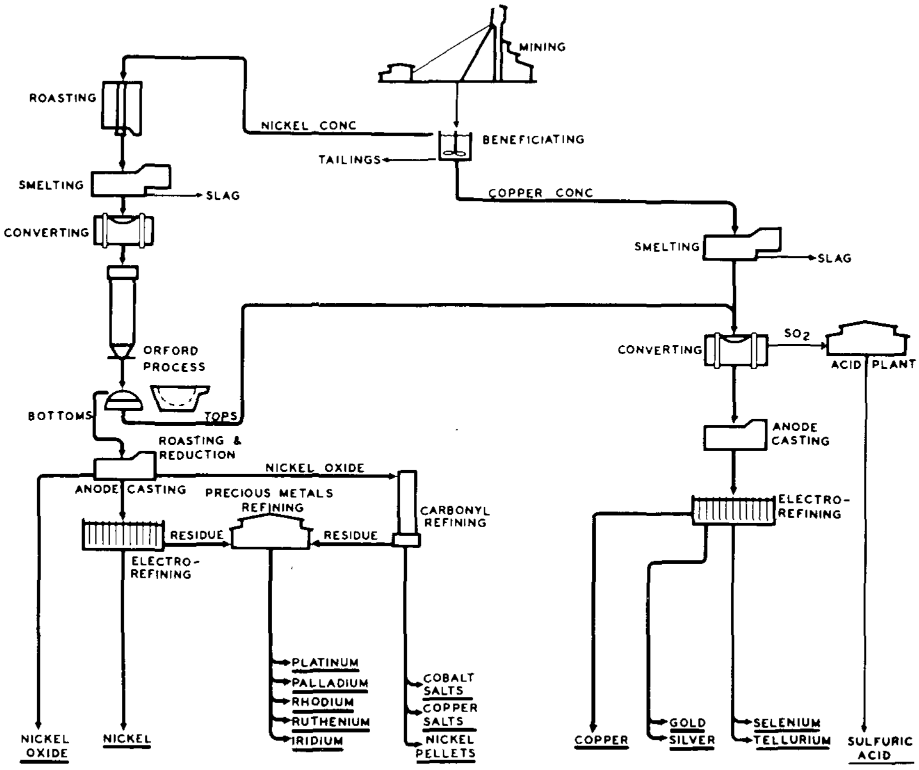


FIGURE 7 TREATMENT OF SUDBURY ORE IN 1938 AT INCO METALS CO.

content of the ore to by-pass the Orford process and to be processed directly to refined copper.

At Port Colborne, electrolytic refining had almost completely replaced fire refining operations. With the adoption of electrolytic refining for both nickel and copper, it became possible to recover gold, silver, platinum group metals, selenium and tellurium from refinery anode slimes. When the British-based Mond Nickel Company amalgamated with International Nickel in 1929, precious metal refining facilities in Acton, U.K. (17), became available for treatment of platinum metal concentrates, as well as a carbonyl-process nickel refinery at Clydach, Wales (17), for production of nickel pellets, cobalt and copper salts. The list of elements recovered commercially totalled 13.

Several important innovations are apparent in the flow diagram for 1948, Figure 8. The old Orford process for nickel-copper separation was superseded by a new technique which involved casting and controlled slow cooling of nickel-copper matte, followed by magnetic separation and flotation into nickel sulphide, copper sulphide and a metallic fraction containing the bulk of the platinum group metals (13). In the nickel refinery, the all-sulphate electrolyte was replaced by one containing sulphate and

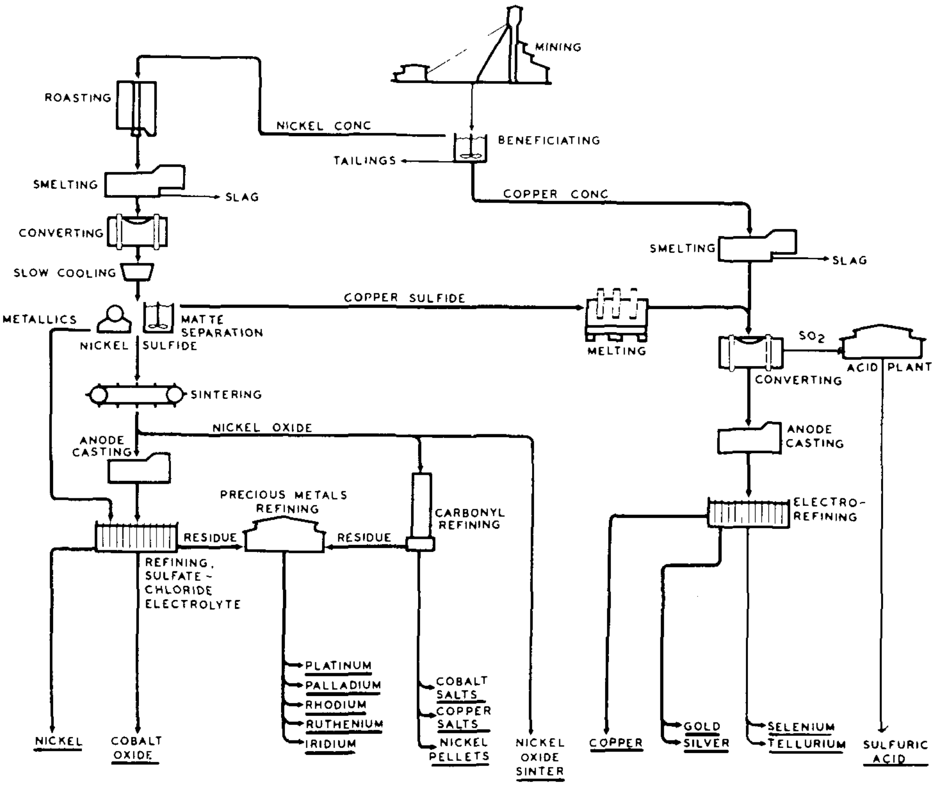


FIGURE 8 TREATMENT OF SUDBURY ORE IN 1948 AT INCO METALS CO.

chloride, with greatly improved electrochemical efficiency and increased throughput. This innovation also permitted the separation and recovery of cobalt, which previously accompanied the nickel to market.

Adoption of de-slimed mill tailings for use as mine backfill was also a development of this period.

The decade between 1948 and 1958 was a particularly active one, as shown in the simplified flow diagram for 1958 (Figure 9). Ore beneficiation operations were expanded to produce an iron concentrate in addition to nickel and copper concentrates; and the new circuit involved the roasting, selective reduction and ammoniacal leaching of this pyrrhotite concentrate to extract its nickel content, and to produce high-grade iron ore pellets (13). Removal and separate treatment of pyrrhotite, which previously was smelted with the nickel concentrate, in effect substantially increased the nickel productive capacity of the smelter. The introduction of iron ore pellets increased the number of elements recovered from the ores to 14.

The sintering machines employed to roast matte separation nickel sulphide to nickel oxide were replaced by compact, clean readily controlled fluid bed roasters which gave a superior product and eliminated workroom dust (17).

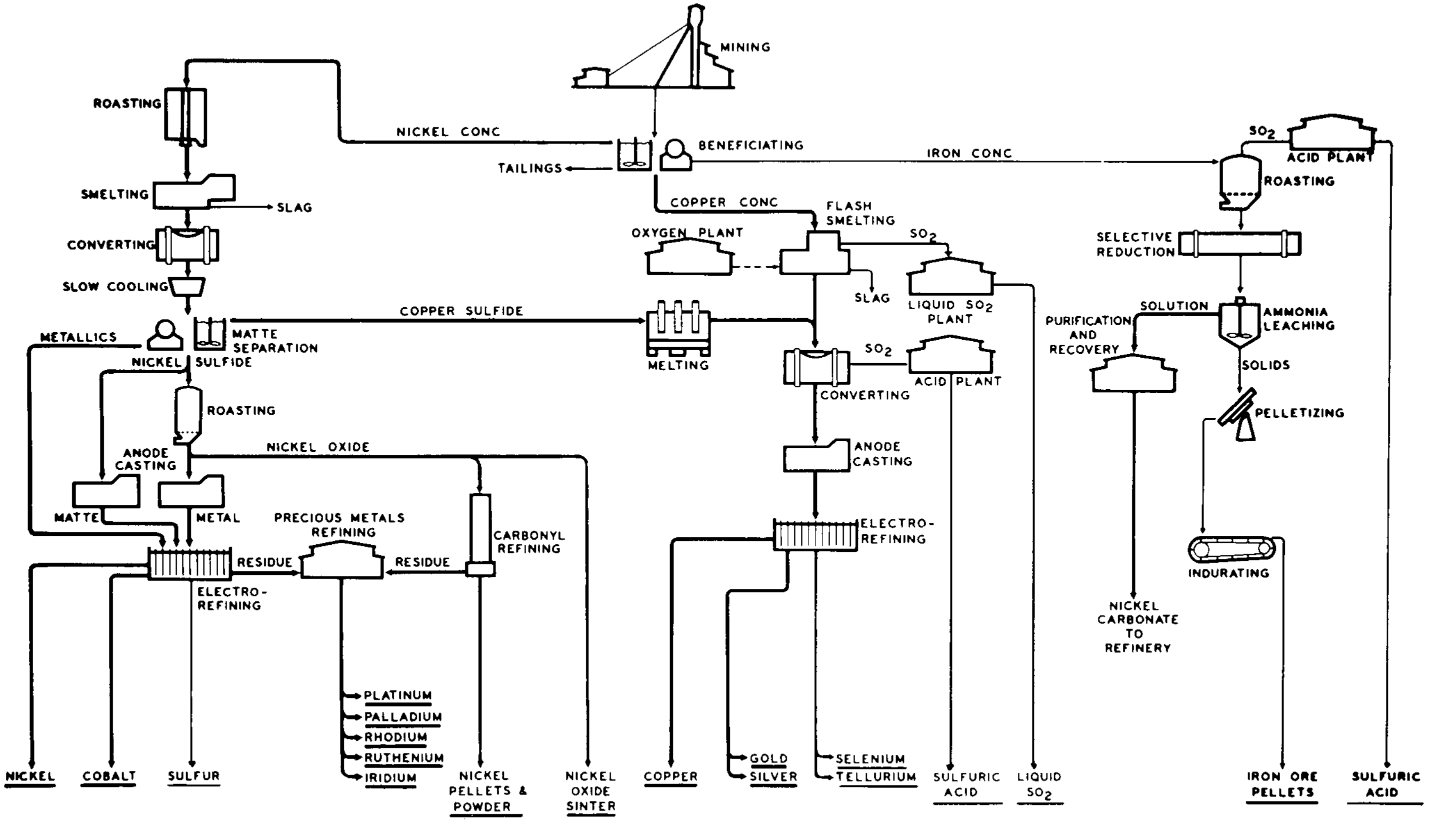


FIGURE 9 TREATMENT OF SUDBURY ORES IN 1958 AT INCO METALS CO.

A procedure was developed whereby nickel sulphide, without prior roasting and reduction to metal, could be cast directly into matte anodes and electrolyzed for the production of electrolytic nickel, with concomitant recovery of elemental sulphur and selenium from anode sludge (18). Cobalt recovery also was extended to electrolytically refined metallic cobalt.

The final flow diagram in this sequence (Figure 10) depicts Inco operations in the 1980 Ontario division. The integrated mining, milling, smelting, and refining operation at Thompson, Manitoba will be discussed separately. A greatly expanded oxygen plant (capacity 1350 t/d) is supplying oxygen to the reverberatory furnaces and converters as well as to the oxygen flash copper smelting furnace. In the production of nickel oxide sinter, two additional fluid bed reactors, for chlorination and hydrogen reduction respectively, produce a new, higher-grade material - nickel oxide sinter 90.

Electrolytic nickel output now includes a special sulphur-containing grade, designated S Rounds, which corrodes readily and uniformly as an anode in the electroplating industry. The addition of osmium, the sixth and final metal in the platinum group, has raised the number of the elements recovered to 15.

5.2 Inco - Thompson

The discovery of the rich nickel deposits of the Thompson area in northern Manitoba in 1955 presented Inco with the rare opportunity of establishing a modern, integrated operation for the production of 34 000 tonnes per year of refined nickel (19).

The mill designed to process 6 000 tonnes of ore per day came on stream in 1960, treating the sulphide ore from one underground mine having a nickel-to-copper ratio of about 15 to 1. The mill was expanded in 1967 to process 15 000 tonnes of ore per day in two separate circuits, one for underground ore and the other for open-pit ore.

The ore from the Thompson mine is hoisted directly into the coarse ore bins in the mill. The ore from the outlying underground mines and open pit is delivered to a receiving bin and crushed adjacent to the mill and the ore crushed to about 150 mm is conveyed to the coarse ore bins. All of the ore is crushed to one inch in cone crushers and ground through 65 mesh in a rod-ball mill circuit.

The mill produces separate copper and nickel concentrates from the underground ore circuit and a combined nickel plus copper concentrate from the open pit circuit. The nickel concentrate is combined to produce a concentrate containing 7% nickel, 0.3% copper and 25% S which is processed through the Thompson smelter.

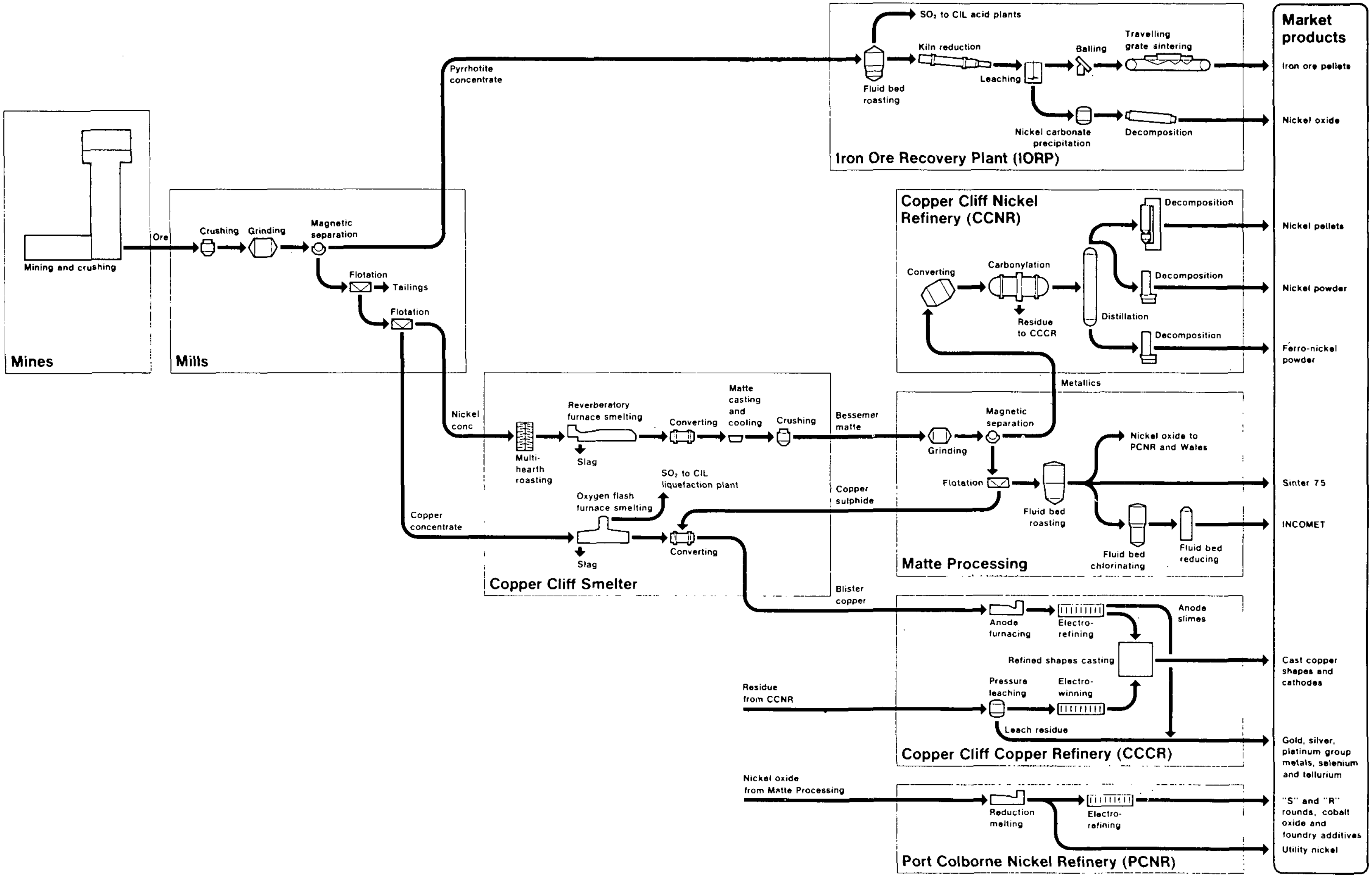


FIGURE 10 TREATMENT OF SUDBURY ORES IN 1980 AT INCO METALS CO.

Copper concentrate is shipped to Copper Cliff for processing and tailings are used as fill material in the underground workings.

The process developed for Thompson nickel concentrate (Figure 11) involves an autogenous partial roast of flotation concentrate in fluid bed reactors, smelting in electric furnaces and converting the furnace matte to nickel sulphide which is cast to matte anodes for electrolytic refining.

5.3 Mineral Processing - Falconbridge

Falconbridge Nickel Mines Limited commenced mineral processing operations in Sudbury in 1933 at an ore rate of about 3 000 t/d on ore similar to Inco's. Falconbridge operations were expanded in later years to incorporate other mines and three other mills in the area. Mill product was a copper-nickel concentrate which was sintered prior to smelting in blast furnaces. The furnace matte was converted to a copper-nickel matte for refining in Norway. There were major plant expansions in 1958 and 1965 to provide an ultimate capacity of about 45 000 tonnes of nickel per year.

In the mid-1950's Falconbridge added pyrrhotite separation to their milling practice and built a plant for the treatment of the nickeliferous iron sulphide concentrate. The process involved sulphate roasting in fluid bed reactors to form nickel sulphate and ferric oxide. The nickel was extracted from the calcine with water from which it was recovered as crude nickel sulphide for smelting. The operation was discontinued in 1972 because of inability to economically control sulphur dioxide (SO_2) and particulate emissions from the sulphate roasting operation.

In 1979 Falconbridge replaced its blast furnace smelter with a new fluid bed roaster - electric furnace smelter built along the lines of the Thompson smelter. Roaster gas is used for production of up to 600 t/d of sulphuric acid, a quantity sufficient to meet current provincial emission limits for SO_2.

5.4 Mineral Processing - Western Mining

In recent years western Australia has become an important producer of nickel from sulphide ore. This followed the discovery in the early 1960's of a rich ore body at Kambalda which contained pentlandite and pyrrhotite but only minor quantities of copper. Western Mining Corporation is developing this and other deposits in the area. The operations are fully integrated and involve milling and flotation to produce a nickel concentrate for smelting and refining. Initially, all of the concentrate was processed

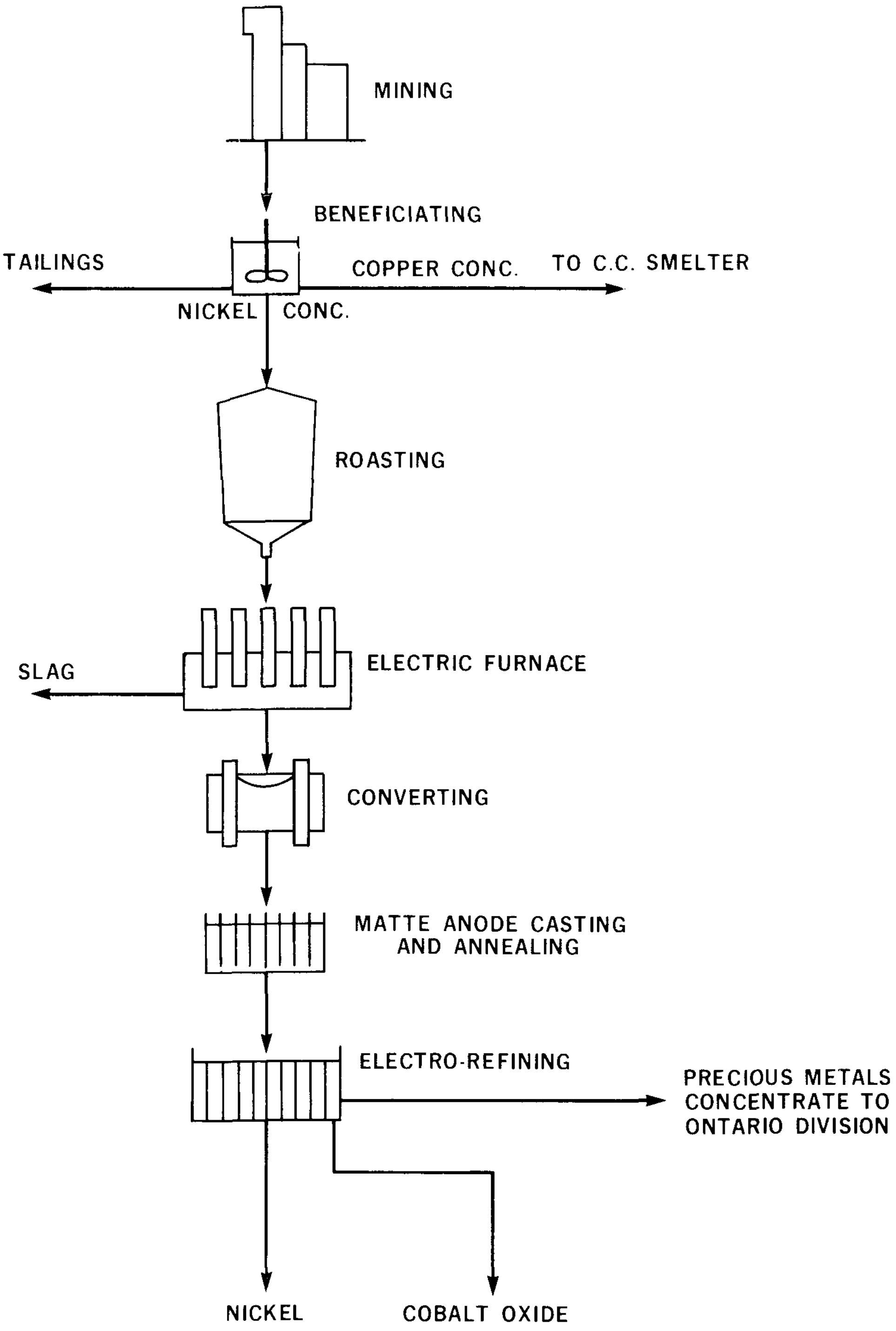


FIGURE 11 TREATMENT OF NICKEL ORE AT THOMPSON, MANITOBA BY INCO METALS CO.

through a Sherritt-Gordon-type hydrometallurgical facility. In recent years an Outokumpu smelting furnace and auxiliaries has been added to produce nickel matte which is then refined in the hydrometallurgical plant. Production capacity is about 19 000 tonnes of nickel per year.

5.5 Mineral Processing - U.S.S.R.

The U.S.S.R. currently produces about 68 000 tonnes of nickel per year from its sulphide ores. Norilsk, situated at 70°N latitude, near the mouth of the Yenisey River, and 1600 km north of the nearest railhead at Kranoyarsk, is the major centre of the Russian nickel industry. This copper-nickel ore deposit was discovered at Tolnash in the Norilsk region in 1965. Norilsk is served by the Arctic port of Dudinka and the Yenisey River and the short summer shipping season has been extended to year round by the use of ice breakers. Due to the remoteness, much of the material required for processing is produced locally. The operation supports a community of 170 000 people. Fuel requirements are met entirely by locally mined coal and recently discovered natural gas, about 80 km distant. Production capacity is currently estimated at about 45 000 tonnes of nickel per year and about 90 000 tonnes of copper per year. Two other nickel-producing areas are based on Pechenga and Monchegorsk in the Kola Peninsula. At Pechenga (formerly Petsamo, Finland) mining of several small deposits, milling and electric smelting are conducted to produce about 11 000 tonnes of nickel per year in the form of slowly cooled nickel-copper matte-ingots for refining at Monchegorsk. Sulphide nickel ores are also mined in the Monchegorsk area where facilities are available for milling, smelting and electrolytic refining of copper and nickel and for the processing of metal concentrates from other areas of Russia. The Monchegorsk ores are lean and the deposits are small. Consequently, some of the smelting capacity is utilized for processing high-grade ore shipped from Talnach. Refining capacity at Monchegorsk is estimated at about 23 000 tonnes per year. The Monchegorsk and Pechenga operations are under the control of the Severonickel combine. Although the plants are also in the Arctic, weather conditions are relatively benign because of the influence of the Gulf Stream. Good rail transportation is available throughout the area. The nickel facilities are also served by the port of Murmansk.

To the present, Russian practice has essentially paralleled that of Inco with respect to all phases of mining, milling, smelting and refining. In smelting, the U.S.S.R. uses electric furnaces.

Currently, smelting at Norilsk is undergoing drastic changes. A new copper and nickel smelter is being built based on the use of two large Outokumpu flash smelting furnaces, one for copper concentrate, the other one for nickel concentrate. The smelter is supplemented by a pressure hydrometallurgical facility, used for treatment of a nickeliferous pyrrhotite fraction, unusually high in platinum metals.

6 MILLING AND CONCENTRATION

Concentration of the sulphide minerals in sulphide nickel ore involves, basically, crushing and grinding to a sufficiently fine state to liberate the sulphide from rock. Generally, a grind through 60 mesh is sufficient. It may or may not be sufficient to free one sulphide mineral species from another. If it is not, the fraction of the concentrate in question must be further ground to achieve the desired separation.

All of the major processors of sulphide nickel ore use comparable milling and flotation procedures. Accordingly, current Inco practice will serve for the industry as a whole and reference will be made only to significant variations.

6.1 Milling and Concentration - Inco

The aims of milling and concentration at Sudbury and Thompson are: (1) separation of a pyrrhotite concentrate containing 85% Fe_7S_8 and about 1% nickel by magnetic separation, regrinding and flotation; (2) flotation to separate a bulk sulphide concentrate from barren rock tailings; (3) separation by flotation of about 80% of the copper into a copper concentrate containing 28% copper and 1% nickel and a nickel concentrate containing about 10% nickel and 2% copper.

Separation of nickeliferous pyrrhotite from sulphide nickel ore to remove it from the smelter burden represents a development of great significance to the nickel industry. Inco developed a process of magnetic separation from the ore pulp followed by regrinding and flotation of the magnetic concentrate to bring the nickel content down to less than 1%. Prior to the separation of the pyrrhotite, the so-called nickel concentrate flowing to the smelter contained only about 4% nickel together with 60% pyrrhotite. At that time, smelting of each tonne of concentrate involved the burning of 670 kg of iron sulphide to produce 480 kg of SO_2 and 1 tonne of slag to give only about 38 kg of nickel in bessemer matte. This was the situation at Sudbury until 1953. Currently, removal of about 75% of the pyrrhotite in ore provides a concentrate for the smelter containing 10% nickel, 2% copper and 40% pyrrhotite. Smelting of a tonne of the new concentrate involves the burning of 590 kg of FeS to produce 430 kg of SO_2 and 1 tonne of slag for the production of 100 kg of nickel in bessemer matte. From this, the effect of pyrrhotite diversion from the smelter is apparent. Quantity to be smelted and the flux required are proportionately decreased. The quantity of SO_2 emitted and the quantity of slag produced are inversely related to the pyrrhotite diverted.

All of the sulphide-nickel mining companies reject pyrrhotite to a greater or lesser degree depending on the ratio of pyrrhotite to nickel and on environmental requirements. At Inco, Sudbury, a concerted research effort is being directed to further rejection of a low-nickel pyrrhotite concentrate. Promising indications have been reported.

At Inco, Sudbury, the major portion of the pyrrhotite diverted from the smelter is processed at the Iron Ore Recovery Plant where the contained nickel and iron are recovered as nickel oxide and iron ore pellets respectively and where the sulphur is recovered as sulphuric acid. Capacity of the pyrrhotite facility is about 1 million tonnes per year of concentrate. Pyrrhotite in excess of this capacity is rejected from the ore and stockpiled for possible future use or is rejected with tailings.

Although the benefits of pyrrhotite rejection are well recognized, some criticism has been directed towards the potential loss of some 10 kg of nickel per tonne of such concentrate. However, even from the standpoint of resource conservation, the advantages of diverting pyrrhotite from the smelter far outweigh such loss, and are such as to permit the exploitation of high pyrrhotite deposits which could otherwise not be classed as ore.

Since 1959, Inco has commissioned five mineral processing mills as follows:

Year	Location	Capacity Ore t/d
1959	Levack (Sudbury)	6 000
1960	Thompson (Manitoba)	15 000
1967	Frood-Stobie (Sudbury)	22 000
1971	Clarabelle (Sudbury)	35 000
1972	Shebandowan (Ontario)	3 000

Each mill was designed to take advantage of the progress being made in instrumentation and control. The Clarabelle Mill (20) is under closed loop control of concentrate and tailings through the use of an IBM 1800 computer tied to continuous x-ray fluorescence analysis. This mill produces a bulk copper-nickel flotation concentrate and the crude pyrrhotite magnetic concentrate both of which are pumped 2.5 km to the revised Copper Cliff mill for separation into nickel, copper and iron concentrates.

6.2 Milling and Concentration - Falconbridge

Falconbridge Nickel Mines Limited commenced mineral processing operations in 1933 at the (ultimately) 3000-tonne daily capacity Falconbridge mill. Present operations include the Falconbridge mill and the 10 000 t/d capacity Strathcona mill, both in the Sudbury area.

The Falconbridge mills produce a bulk nickel-copper concentrate which is slurry fed to fluid bed roasters and smelted to nickel-copper matte which is shipped to Kristiansand, Norway for refining.

6.3 Environmental Problems of Milling and Concentration

The milling and concentration of sulphide nickel ores presents two environmental challenges: the management of fine solids disposal, and the quality of the water effluent. The scope of the problem is seen in the Sudbury area where Inco alone must dispose of up to 50 000 tonnes of fine tailings per day. The tonnage of tailings to be impounded is decreased by the quantity of the coarser fraction of the material used for mine backfill. In the Sudbury area up to 20% of tailings are so used.

The problem of dusting of fine tailings has been a serious one. However, it is now coming under control through wetting of the active areas and through vegetation of the worked-out areas. In Sudbury, 2500 hectares of such ground are now under cultivation, presenting a green park-like appearance in contrast to the barren, dusty scene of former years.

The second concern is the quality of the water existing in the tailings area. Large volumes of water are required in milling and flotation and amount to some 4500 m^3 per thousand tonnes of ore. Some of the water so used finds its way to tailings and contains small, residual quantities of highly toxic flotation agents. These are generally at concentrations sufficiently low to be tolerated by the receiving waters where they are rapidly degraded. In any event, the practice of recycling the bulk of tailings water in the interests of fresh water conservation and minimizing contamination of public waters is being almost universally adopted by the industry.

Another serious water quality problem created in milling and concentrating is the formation of thiosulphate and polythionates (21). When the iron sulphides (pyrrhotite, pyrite and marrasite) contained in the ore are ground and processed in the highly alkaline solutions usually employed in the treating of nickel ores, a small quantity of the iron sulphide can react to produce thiosulphate (S_2O_3) and polythionates (S_nO_6, n = 3 to 8). These complex sulphur compounds are stable under the conditions encountered in the milling process, but once the alkaline tailings slurry leaves the mill, nature begins to react on these salts. In the tailings pond, or in the discharge stream leaving the pond for a distance up to 15 km from discharge, there can be an abrupt deterioration. This change is the result of the oxidation of the thiosulphates and polythionates to sulphuric acid causing increased acidity in the receiving waters.

Successful treatment of the thiosulphates has been accomplished (21) by allowing sufficient retention time of the waste solutions in an impounded disposal area where proper conditions of temperature, sunlight, surface exposure and sulphur-oxidizing bacteria oxidize the thionates and polythionates to sulphuric acid. The resulting acid water is neutralized with lime or limestone before being discharged to the watershed.

Acid seepage from tailings can also be present. The waste solids discharged to the tailings impoundment area from the concentrating operation contain iron sulphides. The sulphides, especially pyrrhotite, by the action of bacteria are oxidized to ferric sulphate which subsequently forms sulphuric acid. Under these acid conditions, sulphides of copper, chromium, cobalt, manganese, nickel and zinc dissolve. Thus, the effluent is not only excessively acidic but contains metals in solutions that are toxic to aquatic life.

A satisfactory solution to seepage is provided by a good vegetation cover over the tailings as described in the next section, Tailings Disposal and Revegetation. The interception of rainfall by the vegetation and the transpiration of moisture by plants reduces the amount of water that will leach through the tailings to become a seepage problem.

7 TAILINGS DISPOSAL AND REVEGETATION

Mill tailings are the small unwanted particles of the ore left after the crushing and concentrating processes. They are transported as a slurry in pipes and discharged around the perimeter of large saucer-shaped basins. The elevation and exposure of these basins increase as they are filled. Their appearance becomes an aesthetic blot on the landscape and unless stabilized they create an environmental problem as a dust source under certain weather conditions.

The stabilization of tailings has been a concern of the mining industry for many years. At Inco, experiments to stabilize tailings began in the early 1940's. Although many materials were used in the experiments, their inability to control this dust source or their prohibitive cost eliminated them. In the mid-1950's, a successful procedure for revegetating tailings was developed (22).

The use of vegetation to stabilize tailings basins when they have reached their final elevation is the most satisfactory permanent solution. The economics will vary with each tailings pond as costs are dependent on the amount of the plant nutrients and the chemicals used to ameliorate the tailings to provide a suitable growth medium for the plant roots. Tailings produced from the Sudbury ores vary in their content of pyrrhotite from mine to mine. Inco's experience with the Copper Cliff tailings area demonstrated that a moderate amount of agricultural limestone (11.1 tonnes/hectare) is satisfactory. Falconbridge uses almost twice this amount as a general practice. Confronted with the necessity of revegetating a tailings area with high-pyrrhotite content after its acquisition by Falconbridge, DR. U.M. Oko initiated a research program to develop a successful procedure. The research undertaken by Oko and his staff culminated by the successful use of a gravel overburden by Michelutti (23) to establish vegetation on this site. Meanwhile Spires (24) carried out some interesting work on the type of overburden best suited to vegetate high-sulphide tailings and came to the conclusion that a gravel layer covered by a layer of loam would be the most effective. The function of the gravel layer was to prevent the upward movement of acid-laden water from the oxidized surface of the tailings (25).

After the pH adjustment of the root medium, sufficient plant nutrients must be provided to sustain plant growth. Since tailings generally have had little time to weather, the macro-nutrients nitrogen, phosphorus and potassium must be supplied in sufficient quantities to remain available to the plants during the growing period. The effect of iron and aluminum content of the tailings on the availability of phosphate for the plants is a problem that can be controlled by keeping the pH range between 5 and 6. Usually a sufficient supply of the trace elements required for plant growth is present as impurities in the fertilizer or they can be added. The establishment of legumes in the sward helps to maintain an adequate level of nitrogen.

The grass species that have been most successful in the Sudbury area are timothy (Phleum pratense L.), Canada blue (Poa compressa L.), Kentucky blue (Poa pratensis L.), red top (Agrostis alba L.), brome (Bromus inermis Leyss.) and tall fescue (Festuca elatier L.) The legumes that have been successful, providing the seed has been properly inoculated, are birdsfoot trefoil (Lotus corniculatus L.) and alfalfa (Medicago sativa L.). The most popular companion crop is fall rye (Secale cereale L.). Once a grass and legume cover was established, there was a voluntary migration of trees into these areas. The principal species are birch (Betula papryifera March.), trembling aspen (Populus tremuloides Mich.) and willow (Salix sp.). Experimental planting of the conifers jack pine (Pinus banksiana Lamb.) and white spruce (Picea glauca (Moench) Voss.) has been most successful on the Inco tailings areas at Copper Cliff (26).

Conventional farm machinery is used for seeding and maintenance of the gentle slopes or flat areas inside the tailings berms. At Inco, an application of half the limestone is made about six weeks prior to seeding time. At seeding time, which is in late July and August, the balance of the limestone requirement is applied along with a broadcast application of half the fertilizer requirements. These are then incorporated into the soil by discing, harrowing, or a combination of both depending on the physical condition of the tailings surface. A combination farm fertilizer and seed drill is used to apply the balance of the fertilizer, the companion crop seed and half the grass seed.

A light application of brome grass seed is then broadcast over the seeded area followed by a cultipacker-type seeder to compact the surface and to apply the balance of the grass seed. Adaptations of this technique for the steeper outside slopes include the use of Klodbusters for cultivation and the use of hydroseeders for applying seed and mulching material. Mulches should be used on all slopes with a southerly and southwesterly exposure.

Variations in tailings from site to site make it necessary to use reclamation programs on the basis of adapting and not adopting already proven practices.

In addition to improving the appearance of the tailings and eliminating a dust source, vegetation intercepts approximately 25% of the rainfall before it reaches the ground. Such moisture evaporates directly and this, coupled with the fact that plants transpire more moisture from a given area than will evaporate from the same area, reduces the amount of water that will leach through the tailings to become a seepage problem.

The voluntary invasion of tree species can lead to an eventual maintenance-free tailings ecosystem. The possibilities of forest production and/or the development of a wildlife management area on reclaimed tailings is currently under study.

Dust control programs on active tailings areas has only recently come under scrutiny. The use of synthetic-organic-membrane-forming-materials appears to be the most effective. They are applied as liquids and fill the pore spaces on the tailings surface to form a membrane in conjunction with the tailings particles. The membranes so formed are water permeable and non-toxic to plants. In windy, exposed areas, they are sometimes used in conjunction with the seeding operation to prevent wind erosion. The main concern stems from their relative fragility.

Dust on roadways in the areas is controlled through the use of calcium chloride or waste oil. Snow-fencing and cut brush gives satisfactory control over small localized areas.

A completely satisfactory solution to stabilizing active areas during the "freeze-dry" period has not yet been achieved. Such conditions occur in the late fall or late winter when there is no snow cover on the tailings. At such times the frozen tailings inhibit the upward movement of water by capillary action. A bright sun will thaw the very top of the tailings and a high wind will evaporate the moisture to produce a thin, dry layer of tailings particles which move freely in the breeze.

8 PYROMETALLURGY

8.1 Background

Apart from the early small-scale hand-operated procedures, the smelting of sulphide nickel ores began and was developed in Sudbury, Canada.

The fledgling nickel smelting industry relied heavily for its processes on its predecessor, the copper industry. The first blast furnace, blown in at Sudbury in 1888, had a daily capacity of 100 tonnes of roasted ore, considered large in those days.

To prepare the ore for blast furnace smelting, heap roasting was employed. Cordwood was piled in outdoor heaps in a roast yard, sulphide ore was spread over the wood, and the piles were ignited to smoulder and burn for three to six months. This partial roasting decreased the sulphur content and oxidized part of the iron, controlling factors for matte grade in subsequent smelting. When the roast heaps had cooled, the calcine was shoveled into wheelbarrows, loaded into railway cars, and delivered to the smelter. It was smelted with coke in blast furnaces to yield a furnace matte containing about 18% copper and 13% nickel. The matte was bessemerized to about 75% copper-nickel.

In 1901, eighty roast heaps and nine furnaces were operating at Sudbury, with a combined daily ore throughput of 900 tonnes. Only two of many operating companies, Inco and Mond, survived until World War I, at which time the British government sponsored a third short-lived entry. The two survivors amalgamated in 1929, not long before the advent of the Falconbridge operations.

For some forty years Inco followed the practice of heap roasting, blast furnace smelting and converting without substantial change, although larger blast furnaces and large basic converters were adopted. Mond's operations were similar, but its Coniston smelter (1913) employed Dwight-Lloyd machines to sinter mine fines, thereby permitting their addition to the blast furnace charge, and Mond's outdoor roasting was gradually eliminated. (The Coniston Smelter, inoperative since 1972, was recently demolished.)

Inco contemplated a growing stockpile of mine fines and flue dust, and in 1911 installed a reverberatory furnace in the Copper Cliff smelter. The new furnace proved valuable for recycling and cleaning converter slag, obviating previous expensive casting and remelting operations.

An industry turning point came in 1930 with the installation by Inco of a new mill (concentrator) to treat lower-grade ores. A new smelter employing multi-hearth roasters placed over reverberatory furnaces was erected to accommodate the smelting of flotation concentrates. Separate smelter circuits treated the nickel and copper concentrates. Coarse, high-grade ore continued to be smelted in blast furnaces along with smelter reverts and scrap, but as ore grade declined, this operation was phased out.

The 1930 smelter employed multi-hearth roasters for nickel concentrate, pulverized-coal-fired reverberatory furnaces, waste heat boilers, large Pierce-Smith converters, electrostatic dust collectors for roaster gases, a 150 metre stack, and an adjoining sulphuric acid plant which utilized the copper converter off-gases. Most of this smelter (27), with some modifications, continues in operation today.

8.2 Current Inco Practice - Ontario

The general arrangement of Inco's Ontario division plants (28) and their products is shown in Figure 10, while the respective nickel and copper circuits of the Copper Cliff smelter are shown in Figure 12. The matte separation process is discussed later.

The copper and nickel circuits of the smelter are fed, respectively, by two concentrate streams from the Copper Cliff mill, which have approximately the following copper-nickel content:

	%	%
Copper Concentrate	30	1.5
Nickel Contentrate	2	10

The copper concentrate is mixed with sand flux, dried, and oxygen flash smelted in an Inco-developed flash smelting furnace (29). The furnace slag is discarded, while the matte is bessemerized to blister copper which is shipped molten to the Copper-Cliff refinery. The flash furnace off-gas, containing some 80% sulphur dioxide, is treated in an adjoining plant to produce liquid sulphur dioxide for market. Most of the nickel contained in the copper concentrate reports in the copper converter slag, which is reverted to the nickel circuit reverberatory furnaces for cleaning.

The nickel concentrate stream is mixed with sand flux and partially roasted in multi-hearth furnaces. The hot calcine is charged directly by gravity flow to natural-gas-fired reverberatory furnaces where it is smelted to furnace matte and discard slag. The matte is bessemerized in Pierce-Smith converters to essentially iron-free, sulphur-deficient nickel matte for treatment through the matte separation process. Converter slag is recycled to the nickel reverberatory furnaces for cleaning.

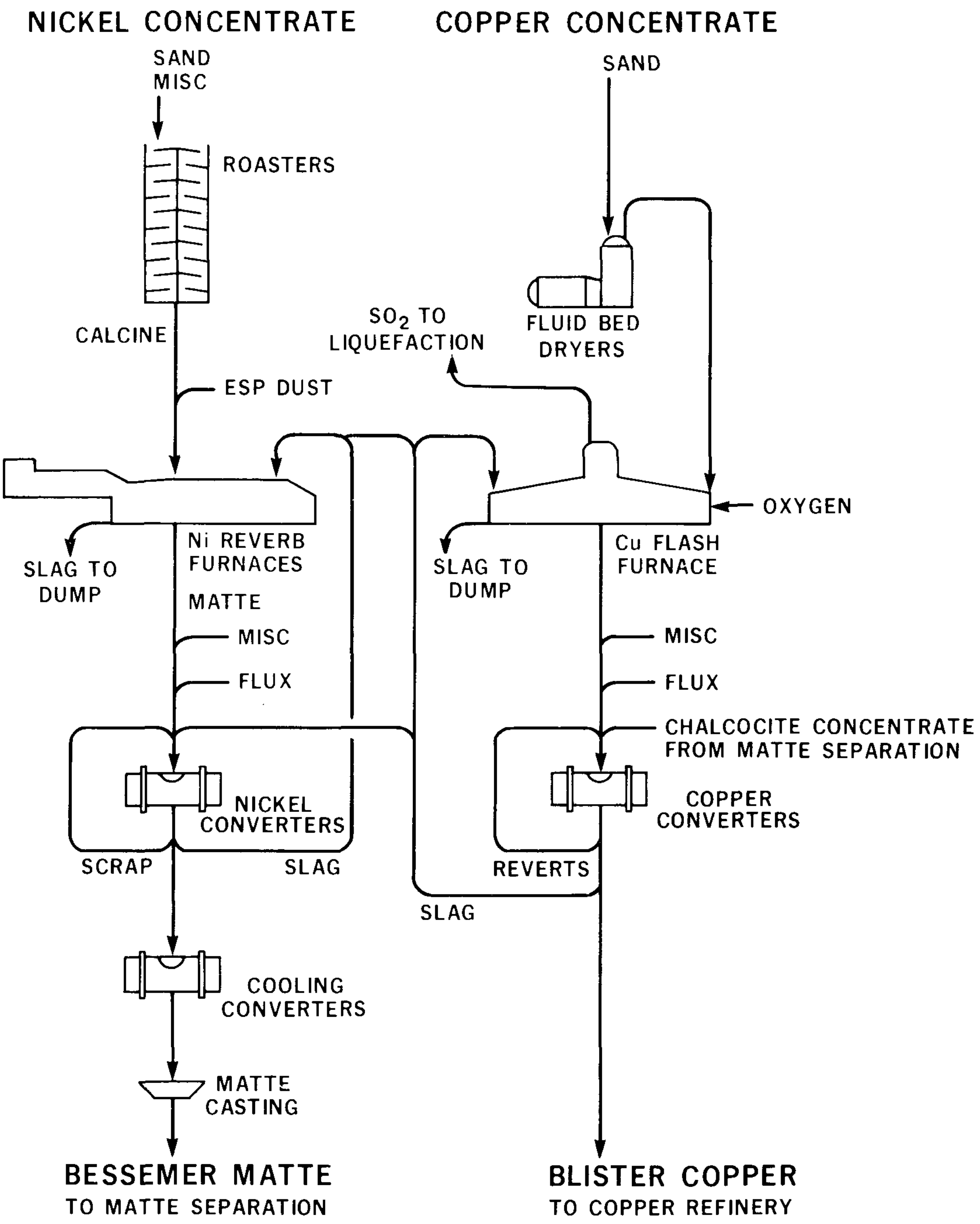


FIGURE 12 COPPER CLIFF SMELTER FLOW SHEET

Currently, the Copper Cliff smelter treats about 3600 t/d of nickel concentrates and 1200 t/d of copper concentrates. All the gases from the multi-hearth roasters, reverberatory furnaces, nickel converters and copper converters are vented to atmosphere via a 380-metre stack. These emissions contain about 2500 tonnes per day of SO_2.

The converter aisle in the Copper Cliff smelter houses 19 standard 100-tonne capacity Pierce-Smith converters and is 440 metres long. Normally eight to ten converters operate in the nickel circuit, and four in the copper circuit. Water-cooled converter hoods, which seal off the converters while blowing, are a recent innovation.

The venerable Orford process for copper-nickel separation was replaced in 1948 by the matte separation process (30). This process is based upon slow cooling of slightly sulphur deficient bessemer matte to encourage the growth of discrete and separate crystals of metallics (nickel-copper alloy), copper sulphide Cu_2S, and nickel sulphide, Ni_3S_2. In practice, molten bessemer matte is poured into large, 25-tonne capacity insulated moulds for slow cooling over several days. Under these conditions the solidifying matte exsolves into separate metallic and copper sulphide grains contained in a matrix of nickel sulphide. The metallics serve as a collector for about 90% of the precious metals in matte.

Following slow cooling, the matte is crushed and finely ground through 200-mesh. The metallics are separated magnetically and from the remaining non-magnetic fraction, copper sulphide and nickel sulphide are separated from each other by flotation.

The clean copper sulphide fraction joins the smelter copper circuit. The metallics fraction with some other nickel-rich materials are fed to the Copper Cliff carbonyl nickel refinery.

The nickel sulphide fraction is pelletized and roasted to nickel oxide in fluid bed reactors, at a temperature of 1200°C, which is above the melting point of the sulphide. Part of the resulting nickel oxide is marketed as Sinter 75 (Class II nickel). Some of the oxide goes to the Port Colborne nickel refinery to be electrolytically refined (Class I), and some is shipped to the nickel refinery at Clydach, Wales to be refined into nickel powders and pellets (Class I).

8.3 Current Inco Practice - Thompson

The integrated plant at Thompson, Manitoba, which began production in 1961, has the mine (one of several), mill, smelter and refinery in one location (19). The mill daily input capacity is about 12 000 tonnes of ore, and the major final product is nickel cathodes (Class I nickel).

Because of the geographic location, ore characteristics, and its more recent design, the Thompson equipment and flow sheet (Figure 11) differ considerably from their counterparts at Copper Cliff, although the objectives are very similar. In comparison with Sudbury ores, Thompson ore contains a much lower proportion of copper, which is successfully removed by flotation into a minor copper concentrate in the mill, and by sulphide precipitation during electrolyte purification in the refinery obviating the need for matte separation.

The considerable distance from sources of fossil fuel and the local availability of hydro-electric power sites made electric furnace smelting attractive. Fluid-bed roasters were selected in preference to the multi-hearth roasters used at Copper Cliff. Thompson furnace slag is granulated and pumped as a slurry to disposal.

As originally designed and constructed, the Thompson smelter had a capacity of some 2000 tonnes of concentrate per day, and comprised three fluid bed roasters each of which individually fed an 18 MVA electric smelting furnace, together with four Pierce-Smith converters, two matte holding furnaces, and a matte anode casting facility. In 1970 the capacity was increased to 3500 tonnes of concentrate per day by the addition of two larger fluid bed roaster /30 MVA furnace couples and three more Pierce-Smith converters. Depending upon the smelting rates in effect, one of the original 18 MVA electric furnaces may be used to smelt flotation copper concentrates and copper precipitates to matte for shipment to the Ontario division.

In the usual practice, nickel concentrate filter cake containing about 7.0% nickel and 0.3% copper, together with added silica flux, is treated in the fluid bed roasters to oxidize about half the concentrate's sulphur and its associated iron. The resulting hot calcine is charged to the submerged arc electric furnaces below each roaster, where power is supplied through six in-line self-baking electrodes per furnace. Furnace slag is granulated for disposal. The furnace matte containing about 20% nickel is bessemerized to about 75% nickel, then cast into matte anodes for electrorefining (19).

All smelter gases containing about 1250 tonnes per day of SO_2 are vented to the atmosphere via a stack.

8.4 Current Falconbridge Practice

The original 1930 smelter at Falconbridge, Ontario, consisting of a small blast furnace and two converters, was designed for direct smelting of lump ore following limited hand sorting. The resulting low-grade furnace matte generated sufficient heat during converting to allow a substantial tonnage of magnetic ore to be smelted in the converters (31). The nickel-copper bessemer matte was cast in moulds, broken and shipped to a refinery at Kristiansand, Norway, for separation and electrolytic refining of the nickel and copper and for recovery of precious metals.

A concentrator and sintering plant were added in 1933, and sintered flotation concentrates, magnetic ore fines and flue dust were combined with lump ore in the blast furnace charge. By 1956, lower ore grades increased the proportion of fine concentrates to the smelter and pelletizing of the sinter plant feed was initiated. A briquetting plant was also installed to handle concentrates from the Hardy and Fecunis Lake mills. Blast furnace charge then included sinter, briquettes, lump ore and revert slag, with sinter comprising about half the total charge.

Additions of new, larger furnaces and converters were made to the smelter in 1958 and 1965, at which time most of the original equipment was placed on standby. Further expansions to handle increased output of concentrates were made during the early 1970's.

In 1978, under a program entitled Smelter Environmental Improvement Project (SEIP), sintering and blast furnace smelting were completely replaced by an Inco-Thompson type of electric furnace smelter, with an annual capacity of 45 000 tonnes of nickel (32).

The new smelter has two slurry-fed fluid bed roasters, each of which discharges to a 36-MVA electric smelting furnace with six Soderberg electrodes. The Pierce-Smith converter aisle from the previous smelter was retained to bessemerize the furnace matte. Extensive ventilation arrangements have been provided in the roaster-furnace area.

Roaster off-gas is cleaned and delivered to an adjoining sulphuric acid plant; tail gas from the plant is reheated and mixed with clean electric furnace and converter gases before discharge through the smelter stack. Acid goes to market in a 56-car unit train.

Blended concentrate slurries from the mills at 75% solids are fed to the fluid bed roasters along with screened silica sand flux. About half the contained sulphur is removed during roasting, and the high space velocity carries the calcine out of the roasters for recovery by cyclones and hot electrostatic precipitators. Each roaster can operate completely independently.

8.5 Current Bamangwato Concessions Ltd. Practice - The Outokumpu Process (33)

In 1974 BCL Ltd. began the production of nickel-copper matte from its smelter in Pikwe, Botswana.

The ore, which is mined at Pikwe and Selebi, contains chalcopyrite and pentlandite as the main copper and nickel minerals, and pyrrhotite as the dominant sulphide.

A bulk nickel-copper concentrate is produced which is spray dried and smelted in an Outokumpu-type flash smelter (13, p. 246). The nickel-copper matte is periodically tapped and transferred to two Pierce-Smith converters. Slag from both the flash furnace and the converters is cleaned in two cylindrical electric slag cleaning furnaces which use ash (30% carbon) as a reductant. In 1979, oxygen was introduced into the flash furnace.

A schematic of BCL's smelter is shown in Figure 13.

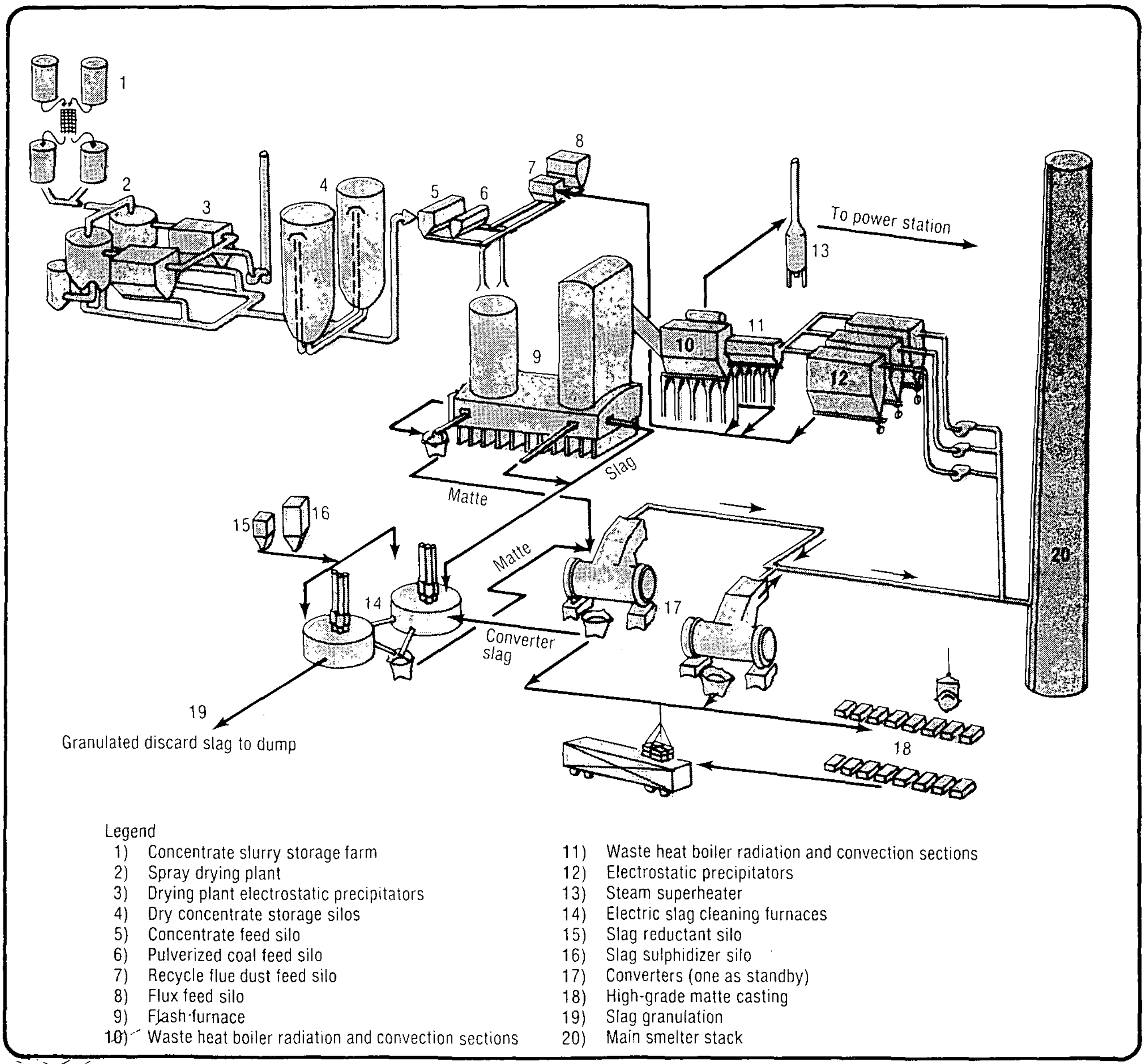
The drying plant encompasses two Niro Atomizer spray driers, uprated from 35 to 55 t/h capacity each. The units provide a means for producing a bone dry concentrate from the 74% solids-26% water slurry. In each plant, the slurry is reduced to an atomized spray, which is then exposed to a stream of hot gases. The dried concentrate is separated from the hot gas stream, and the bone dry product is pneumatically conveyed to the flash furnace. The process is "single-step" from slurry to dry concentrate, with heat and mass transfer occurring instantly in the atomized spray cloud, enabling particle temperature to be maintained at a low and uniform level while ensuring that the product is consistently bone dry.

The dried concentrate is fed along with flux, flue dust and coal to the reaction shaft of the furnace via four concentrate burners. In each burner, concentrate mix and reaction air are combined in a predetermined ratio.

Matte and slag separate into two molten layers in the settler region of the flash smelting furnace. The matte is tapped below the reaction shaft and transferred by ladle to the converter.

Slag containing some 2% Cu plus Ni is tapped from the uptake end of the furnace and flows by gravity through launders directly to the two 9-MVA Barnes Birlec electric furnaces which operate in series for slag cleaning. Discard values (0.18% Ni, 0.35% Cu, and 0.15% Co) are significantly lower than in the previous years.

BCL now uses ash in place of industrial coke as a reducing agent within the electric furnaces. The increase in industrial coke prices led to coke replacement by ash



Legend

1) Concentrate slurry storage farm
2) Spray drying plant
3) Drying plant electrostatic precipitators
4) Dry concentrate storage silos
5) Concentrate feed silo
6) Pulverized coal feed silo
7) Recycle flue dust feed silo
8) Flux feed silo
9) Flash furnace
10) Waste heat boiler radiation and convection sections
11) Waste heat boiler radiation and convection sections
12) Electrostatic precipitators
13) Steam superheater
14) Electric slag cleaning furnaces
15) Slag reductant silo
16) Slag sulphidizer silo
17) Converters (one as standby)
18) High-grade matte casting
19) Slag granulation
20) Main smelter stack

FIGURE 13 SCHEMATIC OF BCL'S SMELTER (33) from the Moruple coal. The ash contains up to 30% residual carbon. A re-sulphurizing agent (currently pyrite) is also added to the electric furnaces to assist slag cleaning.

Waste gases from the flash furnace (at 1350°C), are passed through a waste-heat boiler to produce saturated steam at 285°C. The waste gases contain a dust burden of some 120 g/Nm^3. The dust is collected in the waste heat boiler and electrostatic precipitators and is returned to the process furnace feed.

The converter section consists of two 9-x-4-metres Pierce-Smith units, one hot and the other on standby. Matte from the electric furnaces joins flash furnace matte as feed to the operating converter. The matte is blown and slag is returned to the No. 1 electric furnace while final, high-grade matte is cast directly in 2.3-tonne-capacity Meehanite moulds. Final matte contains 77-78% Ni plus Cu and about 0.8% Co. Cold cast matte is then broken by sledges into lump matte, which is shipped to Port Nickel, Louisianna, United States for separation and refining.

8.6 Environmental Problems of Pyrometallurgy (34)

The problem in smelting is that virtually all the sulphur input to the smelter leaves in the gas phase, with relatively minor amounts contained in the solid phase products or discarded materials. In addition, because of the high-sulphur content in the feed material, a copper or nickel smelter is a point of high SO_2 emissions and therefore at a given location the smelter will likely be the major contributor to air pollution.

In order to protect the environment, it is therefore necessary to capture and fix the SO_2 as some marketable product, or one which can be safely stored, to an extent determined by environmental legislation and/or economic factors.

Sulphur compounds released to the atmosphere ultimately give rise to sulphurous or sulphuric acid or ammonium sulphate, depending on the reactions. Acids formed in the atmosphere fall with rain and cause environmental damage.

Sulphur dioxide may also oxidize in the atmosphere to form sulphates, a particulate form of sulphur compounds. These compounds may have adverse effects on human health and property, depending upon particle size, the particular form of sulphate present, dispersion as affected by weather conditions, and the presence of other pollutants which may magnify the effects.

9 SULPHUR DIOXIDE EMISSION CONTROL

Sulphur dioxide is present in the gases emitted from smelting operations in concentrations ranging from less than 0.1% to over 80%. The SO_2 concentrations are usually not uniform, which imposes a limitation on the practicality of sulphur fixation.

Sulphur dioxide emissions can be converted to marketable forms such as sulphuric acid, elemental sulphur, liquid sulphur dioxide and ammonium sulphate. SO_2 may also be converted to "throwaway" solids, such as gypsum, by lime or limestone scrubbing, although this is not presently practised by any Canadian smelter.

Several different types of control strategies exist; the following section reviews their principal features. Information is drawn from an EPA report (35) and a study commissioned by Environment Canada and conducted by J.H. Reimers and Associates (36).

9.1 Contact Sulphuric Acid Plants

Sulphuric acid is the most common form in which SO_2 is recovered from smelter gases. Its major disadvantage is that it is expensive to transport and requires expensive handling, transport and storage facilities.

In the conventional contact process, a gas containing SO_2 and a suitable amount of oxygen, is heated and passed through beds of vanadium pentoxide catalyst. The SO_2 reacts with oxygen forming sulphur trioxide, SO_3, and is subsequently absorbed forming sulphuric acid, H_2SO_4. Heat is generated during the reaction and the hot gases exiting the catalyst beds are passed through heat exchangers to heat the incoming gases. Complete conversion requires a multiplicity of catalyst beds and heat exchangers in series. Tail gases from the absorber must be demisted. Some plants treat the tail gas in additional catalyst beds. Capital and operating costs are higher.

Moderately high temperatures have to be maintained in the catalyst beds in order to obtain adequate reaction rates and conversions to SO_3. The thermodynamics of the reaction impose limits on the amount of diluents that can be present in the feed gases. A steady flow rate of gases containing between 5 and 8% SO_2 and 30 to 50% more oxygen than SO_2, is suitable for the operation of conventional acid plants. More-dilute gases require the use of larger, externally fired heat exchangers. This increases the operating costs. Irregular gas flow rates require the plant to be designed for the maximum flow rates and the lowest SO_2 concentrations, resulting in higher unit capital costs. The capital costs and operating costs, for acid plants, decrease as the gas volumes decrease and the SO_2 concentrations increase. Measures are often taken to achieve this; these include oxygen enrichment, tight hooding and the replacement of certain smelting systems with others capable of delivering stronger gases, e.g., fluid bed roasters for multi-hearth roasters.

The gases going to the acid plant have to be clean in order to prevent plugging and poisoning of the catalysts and heat exchangers, and contamination of the acid. Dry cleaning systems (cyclones, waste heat boilers and electrostatic precipitators) followed by wet scrubbers and demisters are used. Chlorides and fluorides present in the gas as dust are removed as they will attack the steel. The demisters remove SO_3 fumes from the untreated gases and prevent corrosion of the heat exchangers and structural materials as well as avoiding environmental damage from the acid mist.

Partially combusted organic materials originating from flotation reagents and wood chips in the smelter feed can produce a black discolouration of the acid. Discoloured acid can be bleached by hydrogen peroxide but this increases the production costs and dilutes the acid.

Acid is normally marketed as 93% or 98% H_2SO_4.

9.2 Liquid Sulphur Dioxide

Sulphur dioxide can be recovered as a liquid by compressing and refrigerating smelter off-gases. This requires clean, dry gases, rich in SO_2. CIL presently recovers SO_2 from Inco's oxygen flash furnace at Sudbury. Cominco operates a plant handling a strong SO_2 stream produced by acidification of ammonia scrubbing liquor at a lead smelter in Trail, B.C.

The plants can only operate efficiently on strong gas streams. Oxygen flash furnaces provide such streams directly. Other streams can be upgraded prior to treatment by either pressure absorption in water, or absorption in special reagents, e.g., ammonia scrubbing solutions and dimethylaniline (DMA) followed by stripping to produce a rich SO_2 stream.

Use of this process is limited by the demand. North American demand is now satisfied by the two Canadian plants and several plants in the United States.

9.3 Chemical Absorption Processes

Several processes have been developed which absorb SO_2 in solution, under atmospheric conditions. These include regenerative and non-regenerative processes. The processes can concentrate dilute SO_2 streams. The best known processes are the Cominco and DMA process.

The Cominco process absorbs SO_2 from weak lead sinter machine gases in an ammonia bisulphite-sulphate solution. The resulting solution is then acidified with H_2SO_4 forming ammonium sulphate and driving off a strong SO_2 stream. Ammonium sulphate is recovered and marketed as a fertilizer, and the SO_2 is liquified or fed to acid plants.

The DMA process was developed by American Smelting and Refining Company (ASARCO). The process absorbs SO_2 in dimethylaniline and then desorbs it as a strong SO_2 stream, with steam. The process is used by Cities Service in Tennessee, Asturiana de Zinc in Spain and Falconbridge at their Norwegian refinery. SO_2 is recovered as a liquid.

Several processes have been developed to fix SO_2 in the form of a disposable solid. Examples of these are lime and limestone scrubbing, which treat dilute gases. The processes have several disadvantages which make them unattractive for smelter control:

- unstable calcium sulphite/calcium sulphate sludge is formed;
- large landfill or tailings disposable areas are required;
- maintenance costs are high due to calcium scaling, equipment abrasion etc.;
- waste pollution results;
- high energy costs for pumping.

No nickel smelters have installed these processes.

9.4 Elemental Sulphur Plants

ASARCO, Texasgulf, Outokumpu Oy, and Allied Chemical have developed processes to reduce sulphur dioxide to elemental sulphur. Allied Chemical operated a plant for a short while, treating fluid bed roaster gases at Falconbridge, Ontario. Outokumpu Oy also operated a plant for a while at Outokumpu, Finland producing sulphur from flash furnace gases.

The major advantage of the processes is that they recover sulphur in an elemental, inert form which is easy to handle, transport and store. The cost of reductants is a major cost item.

All of the processes involve dual-stage reactions:

(i) SO_2 is first reacted with reductants, forming elemental sulphur and various compounds such as H_2S, COS, etc.

(ii) The gases are then reacted in catalyst beds with the remaining SO_2, forming more elemental sulphur and inert CO_2 and H_2O. The reductant/SO_2 ratio is controlled to obtain conversion.

Several conditions are necessary for the process to be applicable: (i) SO_2 concentrations have to be no lower than 10-15%; (ii) the oxygen content has to be low, 0-3% to prevent excessive reductant consumption; (iii) gas flow rates and composition have to be constant. This limits the applicability to flash furnace and fluid bed roaster off-gases.

The plants are generally only 90% efficient and the SO_2 content of the tail gas is proportional to that of the feed gas. Other hazardous and objectionable pollutants are formed, e.g., H_2S and COS, and these have to be burned prior to release. The gases are emitted as SO_2.

9.5 Emission Control - Summary

Sulphur dioxide emission control is much more complex than that of particulate emission control involving the use of treatment plants that are expensive to build and operate. Smelters are usually located at relatively long distances from the sulphur product markets and transportation costs are high. The market is limited and is well supplied by various sources, and consequently demand is relatively constant. The income from sales often does not cover the cost of production, transportation and marketing. Losses have to be covered from profits elsewhere.

Several types of control systems are available but each is limited regarding the types of off-gas they can handle. Many of the high-volume, low-SO_2-content gas streams cannot be handled economically, e.g., multi-hearth roasters, blast furnaces, reverberatory furnaces. Development work is continuing on wet scrubbing processes to handle these gases using regenerative and non-regenerative systems. These processes are only in their infancy and more time, effort and money is required before they will be suitable for adoption.

The absence of economic processes to fix weak SO_2 streams, along with other factors such as the desire to reduce fuel and energy costs, and improve working conditions, have led many smelters to change their smelting process. These changes result in the generation of lower-volume, more concentrated gas streams; examples are the adoption of fluid bed roasters, electric furnaces, flash smelters and continuous smelting.

Table 15 summarizes the characteristics of gas streams suitable for treatment. Dust-free gases are required for all except the Outokompu Oy elemental sulphur process. This latter reacts the reductants and gas prior to cleaning and passing them through catalyst beds for final conversion.

TABLE 15 CHARACTERISTICS OF GAS STREAMS REQUIRED FOR OPTIMUM TREATMENT IN SULPHUR RECOVERY PLANTS

		% SO_2	% O_2	Flow Rate & Composition
Contact Acid	- Single	5-8	8-12	Can be variable[a]
	- Double	5-8	8-12	Can be variable[a]
Liquid SO_2	- Physical	70-80	Low as possible	Constant
	- Chemical	10-11[b]	-	Constant
Elemental Sulphur		10	1-3	Constant

a More economical when constant
b Plant requires pressure or chemical absorption followed by treatment of strong regenerated stream.

The cost of sulphur fixation differs for the different plants. Table 16 compares the approximate costs (36). The following points should be noted:

- the costs are order of magnitude estimates and will vary with location, whether the plant is part of a new smelter, geographical conditions, etc.;
- the costs will vary depending on off-gas characteristics, SO_2 concentrations, etc.;
- costs of hot gas cleaning systems are not included. This could increase the costs by 30% or more;
- the estimates include capital (amortized over 15 years at 10%) and operating costs for the sulphur fixation plant only and do not include the cost of adapting the metallurgical plant. Smelter modifications, such as new processes or extensive duct and hood changes, the removal of mercury and selenium from the feed gas or product acid, etc., could well cost more than the sulphur fixation plant itself;
- the estimate assumes that SO_2 in smelter gases is delivered free of charge to the sulphur fixation plant.

TABLE 16 TOTAL PRODUCTION COSTS FOR SULPHUR BY-PRODUCTS FROM NON-FERROUS SMELTER GASES (36)

Sulphuric Acid Plants	$/tonne Sulphur Fixed (1979 Canada)			
Nominal production t/day H_2SO_4	540		1 100	
	Double Catalysis	Single Catalysis	Double Catalysis	Single Catalysis
From				
continuous gas flow	63	59	-	-
variable gas flow	99	92	-	-
continuous and variable gas flow	77	71	58	54
Sulphuric acid neutralization	92	-	86	-
Liquid SO_2 Plant				
Nominal production t/day SO_2	160			
	Chemical Absorption		Physical Recovery	
continuous gas flow	100		85	
Elemental Sulphur Plant				
Nominal production t/day sulphur			270	
continuous gas flow			129	

Considerable cost is involved in by-product manufacture. Large quantities of material are generated and these have to be marketed, stored or disposed of safely. The sulphur product market is presently fairly well satisfied and the sale price in a particular area is fixed by the costs of the cheapest supplier.

Sulphur by-products are expensive to ship. Traditionally, 300 km was the maximum distance H_2SO_4 could be shipped economically by rail. Unit trains carrying 5000 or more tonnes between major suppliers and large distribution centres, have increased this further. For example, the freight rate for 90-tonne lots of acid from the Sudbury area to Niagara Falls is approximately $20 per tonne while that for unit trains of over 3300 tonnes is $10 per tonne (June 1979) (32). Costs are higher for more distant markets.

Expensive transportation costs and limited markets can cause acid to be produced at a loss and the sale price may not even cover transportation costs. Smelters may be forced to neutralize acid and impound it as an inert solid. This increases the overall fixation cost as shown in Table 16.

Variations in the S/metal ratio cause sulphur dioxide control costs to vary. The sulphur/nickel ratio in nickel concentrates varies from 2.2 to 4.8, to 1 (32). Table 17 summarizes estimated costs for various sulphur dioxide control methods, in terms of ¢/pound of nickel or copper produced. The costs shown are approximate only and will vary for different plants, as noted previously. Costs have been calculated assuming the products have no value. Products from some plants may have a value, depending on whether selling costs exceed shipping and marketing costs.

TABLE 17 COST OF SULPHUR FIXATION PER POUND OF METAL PRODUCED, COPPER AND NICKEL *(36)

	Single Catalysis Acid Plant						Double Catalysis	
	Continuous Gases only		Variable Gases only		Continuous & Variable Gases		Continuous & Variable Gases	
Sulphur fixation, %	50		45		95		97	
Production cost $/tonne S	59		92		54		58	
Neutralization cost $/tonne S	92		92		86		86	
Sulphur/Metal Ratio in Concentrate	Low	High	Low	High	Low	High	Low	High
Sulphur Fixation Costs								
Copper, ¢/lb Cu	1.07	2.14	1.50	3.01	1.86	3.73	2.05	4.10
Nickel, ¢/lb Ni	2.95	6.43	4.14	9.02	5.12	11.18	5.64	12.31
Neutralization - total cost of making acid and neutralizing it								
Copper, ¢/lb Cu	2.75	5.50	3.02	6.03	4.67	9.64	5.07	10.14
Nickel, ¢/lb Ni	7.57	16.51	8.29	18.09	13.25	28.91	13.94	30.42

(Canada - 1979)

* Gas cleaning costs not included

Liquid sulphur dioxide manufacture would not be feasible due to the limited market, and new plants are unlikely to be built as the existing plants could handle larger throughputs. The plants should be operated in conjunction with acid plants to obtain a low-cost drying agent.

Cominco presently operates an ammonia scrubbing plant on low-grade gases from a lead sinter machine. This type of control is not economical for use on large volume, low-SO_2 gases due to the cost of ammonia. The market for ammonium sulphate fertilizer is presently well satisfied. North American demand is low due to the low nitrogen content. Cominco supplies some of this market from its Trail operations and Sheritt Gordon from the hydrometallurgical nickel plant at Fort Saskatchewan.

10 HYDROMETALLURGY

The first commercial application of hydrometallurgy for complete processing of nickel sulphide concentrates was by Sherritt Gordon Mines Ltd. which started operation in 1954 at Fort Saskatchewan, Alberta.

The Sherritt process was adopted by Western Mining Company of Australia at its nickel refinery at Kwinana which began operation in 1970.

Inco has used hydrometallurgy as part of its treatment process to recover nickel, copper and cobalt from nickeliferous pyrrhotite since 1956 at its Iron Ore Recovery Plant.

10.1 Sherritt Gordon Practice (37)

The Sherritt Process starts with pressure leaching of pentlandite concentrate at 90°C with air in ammoniacal ammonium sulphate solution. A flow sheet of the Sherritt Ammonia Leach Process is shown in Figure 14. Nickel and copper form ammine sulphate complexes. The copper is first precipitated as copper sulphide from this metal ammine solution by boiling off free ammonia.

After copper removal the solution is further purified by oxidizing all contained thionate to sulphate and similarly hydrolyzing sulphamate to sulphate. This step is called "oxydrolysis" to account for the simultaneous oxidation and hydrolysis that takes place and is accomplished by sparging air through the solution at high temperatures.

The solution is now a nickel (and cobalt if it was present in the feed) diammine sulphate solution and is reduced with hydrogen gas under pressure to recover the nickel in metallic powder form. After filtering and drying, the nickel powder is briquetted and sintered in hydrogen to produce refined nickel briquettes for market.

The solution after nickel reduction still contains any cobalt that may have been present in the feed material as well as a small amount of residual nickel which has been retained in the solution to prevent cobalt from coprecipitating with the nickel. These residual metals are precipitated as sulphides by the addition of an appropriate amount of ammonium sulphide. These mixed cobalt-nickel sulphides are filtered off and constitute the feed material for Sherritt's cobalt refinery.

After stripping the residual metals in this manner, a pure, highly concentrated ammonium sulphate solution is obtained from which fertilizer grade ammonium sulphate is recovered by evaporation and crystallization.

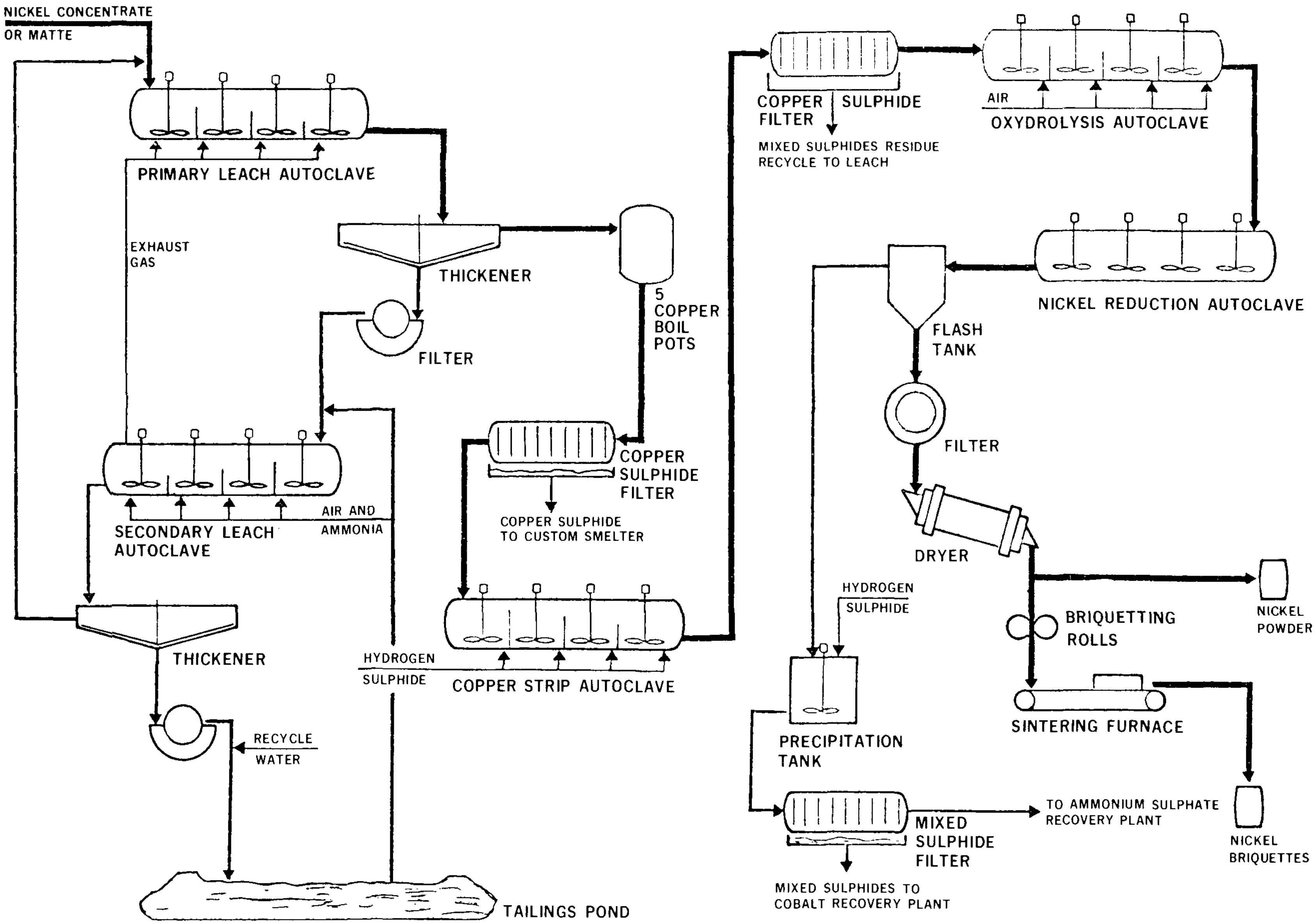


FIGURE 14 SHERRITT AMMONIA LEACH PROCESS (37)

10.2 Inco Practice

Inco expanded its ore beneficiation operations in 1956 to produce an iron concentrate (pyrrhotite) in addition to nickel and copper content. This increased the nickel productive capacity of the smelter without increasing smelter sulphur dioxide emissions.

The pyrrhotite concentrate containing about 0.75% nickel, 0.05% copper, 58% iron, 35% sulphur and 2% silica, (13) is pumped from the concentrator to the Iron Ore Recovery Plant. There it is treated in fluid bed roasters to eliminate sulphur and reduced in rotary kilns to yield magnetite and to reduce the nickel, copper and cobalt so that they will dissolve in the subsequent leach.

The reduced calcine is leached at atmospheric pressure in an aerated ammoniacal ammonium carbonate solution to extract its nickel, cobalt, and copper content. The leached magnetite is further processed to produce high-grade indurated iron ore pellets. A flow sheet of leaching and washing at the Inco Iron Ore Recovery Plant is shown in Figure 15.

The pregnant solution is treated with sodium hydrosulphide to precipitate copper, cobalt and other impurities which are returned to the smelter. Soda ash is added to the purified solution which is steam stripped to remove its ammonia content and the nickel is precipitated as a basic carbonate of high purity. A flow sheet of the nickel recovery from pregnant solution is shown in Figure 16.

The precipitate is calcined to oxide in an externally heated rotary kiln at 500°C to produce a black, dense acid-soluble nickel oxide for market.

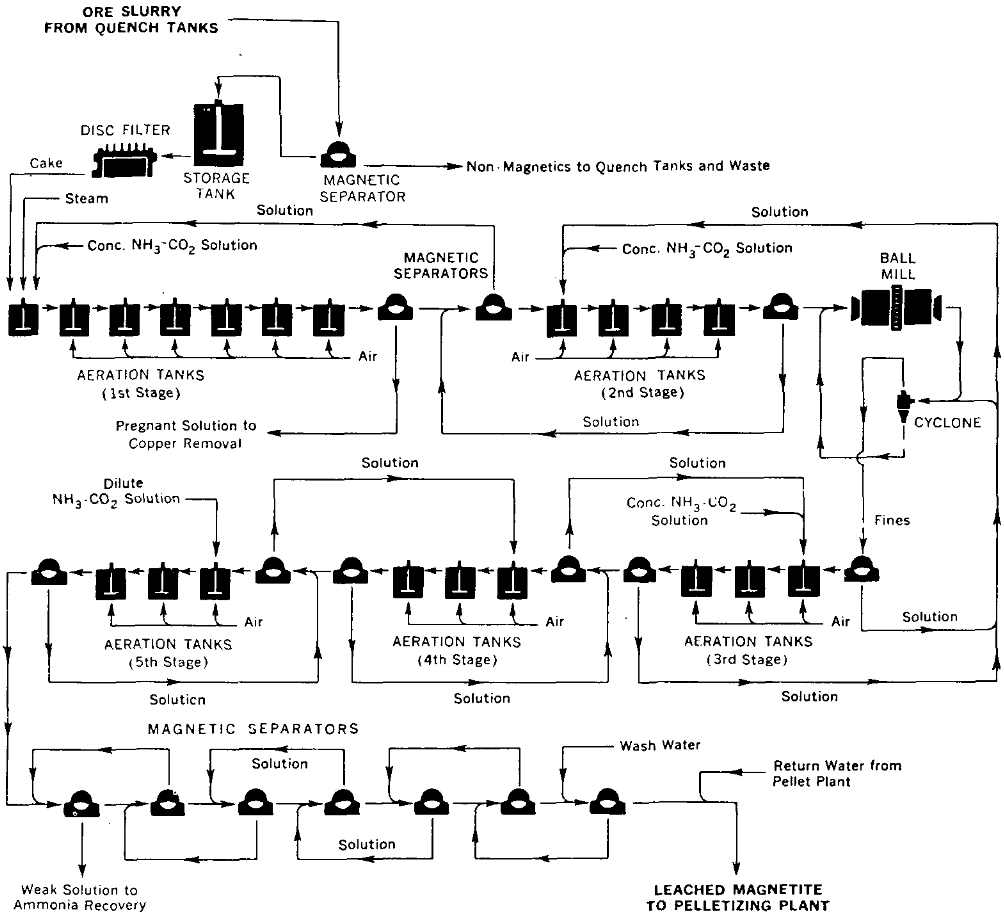


FIGURE 15 FLOW SHEET OF LEACHING AND WASHING AT INCO IRON ORE PLANT (13)

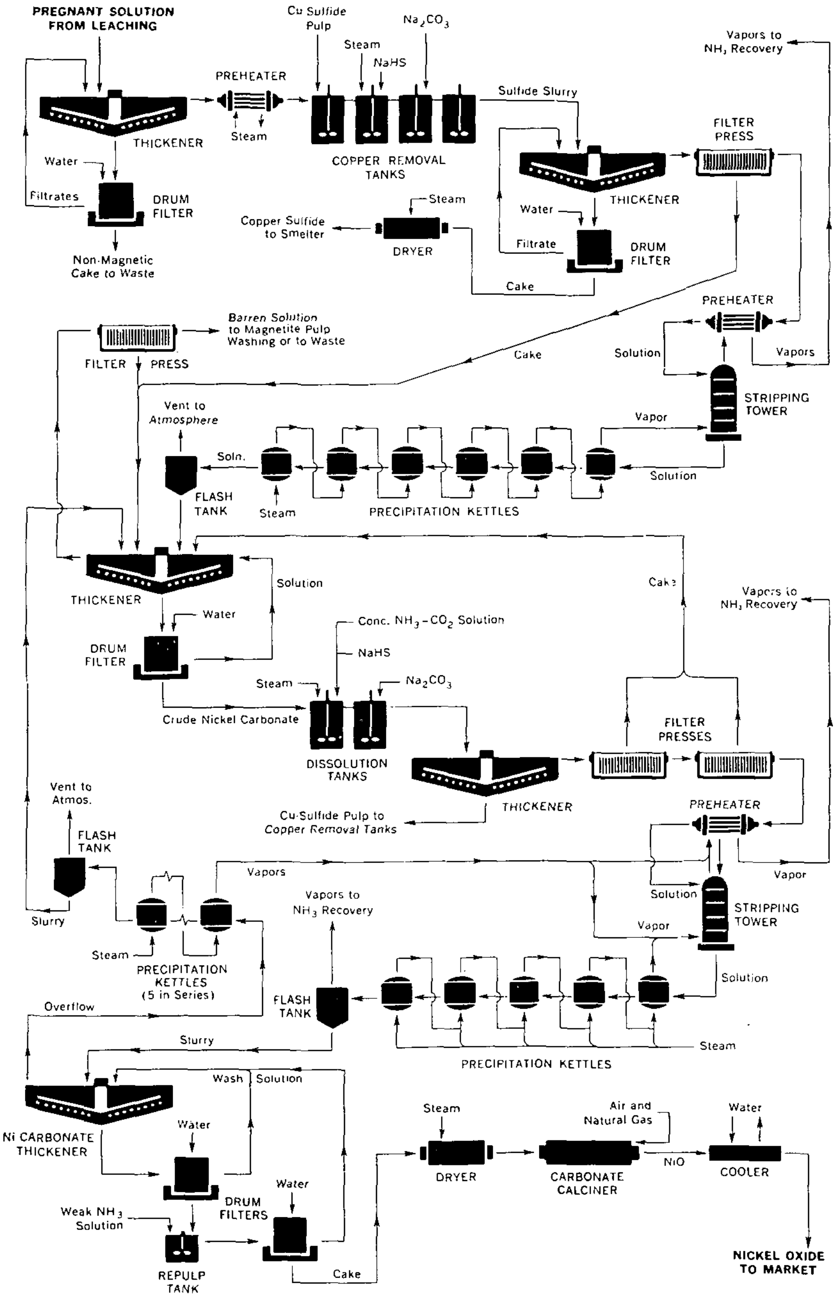


FIGURE 16 FLOW SHEET OF NICKEL RECOVERY FROM PREGNANT SOLUTION AT COPPER CLIFF (13)

11 REFINING

Refining practice varies considerably among nickel producers depending on the cost of energy available, the crude metal production processes and historical factors.

Since nickel is used for alloying, the purity of the refined nickel also varies. For instance, iron is not detrimental in nickel used in making steels. Impurities such as oxygen, silicon and carbon may be acceptable if they are removed in alloying procedures.

Electrorefining, electrowinning, vapometallurgical refining and purification by hydrogen pressure reduction are all used to produce refined nickel.

11.1 Inco Refining Practice

The Company's Port Colborne, Ontario refinery receives fluid bed roasted nickel oxide analyzing about 74% nickel, 3% copper and cobalt, iron, sulphur and other impurities (13). The oxide is reduced to metal with petroleum coke in a reverberatory-type furnace from which anodes are cast for electrorefining. Through careful control of electrolyte purity, nickel cathodes of over 99.95% purity are produced. The anode slime is reverted to the Sudbury operations for recovery of its valuable constituents.

Inco employs the sulphide anode electrowinning process in its refinery at Thompson, Manitoba. The nickel converter matte at 20% sulphur is cast directly into anodes. During electrowinning, the anodes are placed in woven plastic bags to collect the voluminous sludge.

The refined cathodes analyze over 99.95% nickel. The anode sludge contains 95% elemental sulphur, plus small amounts of copper, nickel, selenium and precious metals.

Sulphide anode electrowinning is also done at the Company's Port Colborne refinery using nickel sulphide concentrate from the matte separation process at Copper Cliff.

Inco has employed vapometallurgical refining since 1902 at its refinery at Clydach, Wales. The process is based on the fact that carbon monoxide at atmospheric pressure and at temperatures between 40°C and 100°C will react with nickel to form a gas identified as nickel carbonyl (38). The reaction is readily reversible by heating the nickel carbonyl to temperatures in the 150°C - 300°C range to yield pure nickel and carbon monoxide.

Under the mild conditions employed for reaction at atmospheric pressure, the impurities in crude nickel metal do not enter the gas phase. Iron forms a volatile carbonyl-iron pentacarbonyl $Fe(CO)_5$ - but the rate of reaction is slow. Cobalt can form tetracarbonyl, $Co_2(CO)_8$, and tricarbonyl, $Co_4(CO)_{12}$, but both these solids have low volatility. Copper, like the great majority of elements, is not known to carbonylate directly. Thus the extraction of nickel as carbonyl from a crude feed is a highly selective process (13, p. 374).

The feed for the Atmospheric Pressure Carbonyl Process at Clydach is granular nickel oxide from Copper Cliff produced by roasting nickel sulphide from the matte separation process. The oxide is reduced to metal by hydrogen in rotary kilns. The nickel sponge is then activated by sulphiding and volatilized as carbonyl in kilns by carbon monoxide. The resulting nickel carbonyl product is decomposed thermally to either pure metal pellets or powder.

The Inco Pressure Carbonyl process has been in use at the Company's Copper Cliff refinery since 1973. This refinery uses the same basic chemistry as the Atmospheric Pressure Carbonyl process used at Clydach, but forms the nickel carbonyl at high pressure to accelerate the reaction and utilize less-pure refinery feeds (39).

Feed materials - nickel sulphide crudes, precious metal-bearing metallics and refinery residues are charged to two 50-tonne capacity top-blown rotary converters where they are melted with natural gas-oxygen lance burners. After raising the bath temperature close to 1400°C, high-pressure oxygen is lanced into the melt and the bath agitated by rotating the converter. The oxygen desulphurizes the bath.

The finished hot metal is transferred to a teeming ladle, from which it is granulated by pouring through high-velocity water jets. Following dewatering, the granules are dried in a gas-fired kiln dryer and conveyed to the IPC department.

The dry metallic granules are batch-reacted with carbon monoxide at temperatures up to 180°C and pressures up to 70 atmospheres in three 150-tonne capacity rotating reactors. Nickel, with some iron, is extracted as a carbonyl while copper, cobalt, precious metals and impurities are retained in the residue, which is ground in a rod mill and pumped as a water-borne slurry to the electrowinning department at the copper refinery.

The extracted carbonyl vapour is liquified and then purified in two parallel distillation columns with most of it sent to pellet decomposers, where it is thermally decomposed. Nickel deposits itself on a circulating stream of preheated nickel pellets to produce approximately ten tonnes of nickel pellets per day per decomposer. The remaining pure nickel carbonyl vapour is decomposed in electrically heated decomposers to make pure nickel powder.

In addition to pure nickel carbonyl, the distillation process produces a liquid residue consisting of a mixture of nickel and iron carbonyls. This is vapourized and decomposed to ferronickel powder.

11.2 Falconbridge Refining Practice

Nickel-copper converter matte at about 48% nickel and 27% copper is shipped from Falconbridge, Ontario to Kristiansand, Norway for separation and refining. Two different processes are used.

In the older process (the Hybinette Process), the matte is roasted to oxide in a multi-hearth roaster. The calcine is leached by spent electrolyte which contains sulphuric acid from the copper electrowinning process to dissolve its copper content. The leached calcine is reduction melted with coke in an electric furnace from which nickel anodes are cast for electrolysis.

Electrolyte purity is controlled so that cathodes of 99.95% purity are produced.

In 1968, the Falconbridge Matte Leach Process was put into operation at the Kristiansand refinery to treat part of the matte from Canada. The process is based on the following principles (40):

(1) When finely divided copper-nickel converter matte is treated with strong hydrochloric acid, the nickel is selectively dissolved, leaving copper and platinum metals as an insoluble sulphide residue.

(2) Because the solubility of nickel chloride in pregnant solutions decreases with increasing HCl concentrations, the values can be precipitated from the solution by increasing, rather than by neutralizing, the acidity of the solution. This property permits the maintenance of high acidity throughout a co-current leaching system, thereby obviating the liquid-solid separations necessary when a multi-stage counter-current leach aimed at maximum utilization of acid is resorted to.

(3) Since, unlike nickel, the majority of the impurities picked up in the pregnant solution form anionic complexes under acid conditions, purification of the pregnant solution can be readily effected by solvent extraction or anion exchange, also without prior neutralization of acid in the solution.

(4) Since it is possible in this system to maintain high acidity throughout the leach, high leaching efficiencies are obtained at temperatures no higher than about 70°C. This feature permits the use of ordinary rubber-lined acid-proof equipment for the handling of gases, liquids and solids.

The nickel chloride undergoes high temperature hydrolysis in a fluid bed reactor to form nickel oxide and hydrogen chloride gas. Reduction of the oxide by hydrogen in a rotary reduction furnace yields nickel granules for market.

11.3 Refining Practice Amax Nickel Inc., Port Nickel, Louisiana (41)

In 1974, Amax Nickel Inc. started operation of its mainly hydrometallurgical refining process with an annual capacity of 36 000 tonnes of pure nickel, 23 000 tonnes of copper, 450 000 kg of cobalt and 100 000 tonnes of ammonium sulphate.

Refinery feed consists of Botswana (Bamangwato Concession Ltd.) matte grading 40% Ni, 40% Cu, 0.5% Co, and the remainder mostly sulphur and New Caledonia matte grading 75% Ni, 1.5% Co, some iron, and the remainder sulphur. An additional 20 000 t/y of matte is supplied from other sources such as powdered matte from South Africa.

The matte is crushed, ground and fed in slurry form to the process. Figure 17 shows the refinery flow sheet.

The first step in hydrometallurgy makes a primary separation of nickel-cobalt from copper and any iron present. The slurried matte is piped from the "day" storage tanks to the atmospheric leaching section - a train of five serially connected tanks. The tanks are agitated, steam-heated, and air-sparged. The process requires a controlled reaction between the matte and spent electrolyte, recycled from the copper tankhouse. Oxidizing conditions are achieved by the addition of large volumes of air, sparged under pressure. Retention time in the five tanks is six to eight hours, during which 50-60% of the contained nickel and cobalt is dissolved, while all the copper cements out. The combined solid and liquid is then piped to a thickener for liquid-solid separation. Overflow, or decant, from the thickener contains only nickel and cobalt sulphates in solution, and this solution is piped to the nickel-cobalt separation stage. Underflow from the thickener, containing cement copper, the remaining undissolved nickel and cobalt, and certain impurities, is then sent to the pressure leaching section.

11.3.1 Pressure Leaching in Autoclaves. The pressure leaching circuit has three autoclaves in parallel, each with an effective volume of 25 m^3. The autoclaves have 63.5-m thick carbon steel shells, lined with lead and acid brick. Agitators and other metal parts are made from titanium. Pressure leaching takes place at 205°C and 400 atmospheres. Under these conditions, nickel, the copper and remaining cobalt are

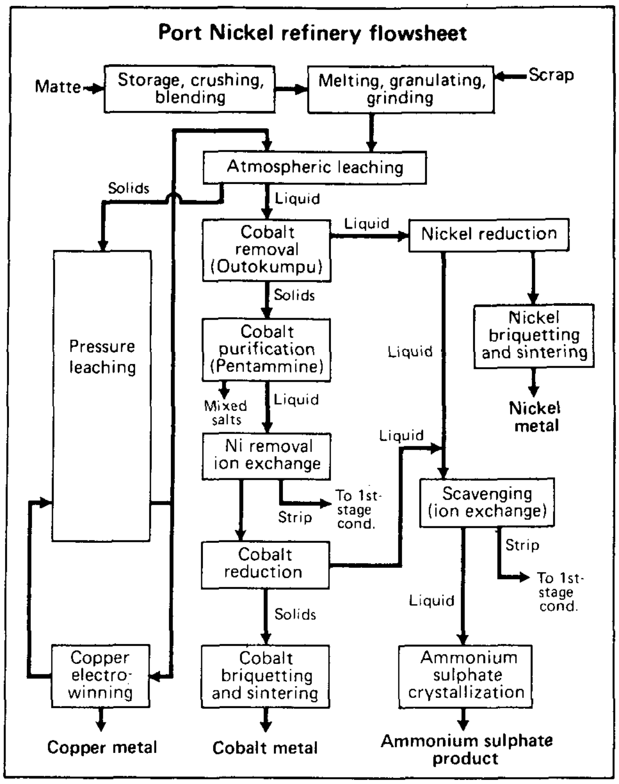


FIGURE 17 PORT NICKEL REFINERY FLOW SHEET (41)

leached into solution using spent electrolyte, while inert impurities and any remaining iron remain in a solid state. The autoclaves are then discharged and the slurry piped to a thickener. Overflow decant from the thickener is the electrolyte feed to the copper tankhouse. Underflow from the thickener, containing inerts, wastes, and perhaps 5% Cu, Ni, and Co values, is then piped to a second-stage pressure leaching circuit, where processing is similar to the first stage. Liquid overflow from the second stage also is piped to the tankhouse, while the second-stage thickener underflow is sent to the tailings pond as waste. (Overflow from the tailings pond is recycled water, treated and re-used in the plant.)

11.3.2 Electrowinning the Copper. The pregnant liquor, or electrolyte solution, from the pressure leaching circuit, containing in solution all the copper and the remaining portions of nickel and cobalt, is pumped to the series of electrowinning tanks in the tankhouse. A portion of the process stream electrolyte is diverted to produce copper starter sheets on titanium cathodes. The starter sheets (24-hour deposits weighing 5 kg) are stripped and transferred to the commercial electrowinning section, which yields a finished copper cathode of about 50 to 55 kg in five to seven days. The spent electrolyte from the tanks, containing nickel, cobalt, unplated copper, and sulphuric acid, is recycled to the leaching steps.

11.3.3 A Mixed Cake of Cobalt. The stream of solution containing the bulk of nickel and cobalt values is piped to the cobalt removal section, where cobalt is precipitated as trivalent hydroxide. The process uses an oxidation/reduction reaction in two serially connected precipitation tanks. Two tanks are at atmospheric pressure and steam-heated between 70°C and 80°C. The nickel and cobalt sulphate solution is agitated with trivalent nickel hydroxide (produced electrolytically within the same section using sodium hydroxide) for a retention time of five to six hours in each of the tanks. Within the tanks, cobalt hydroxide is selectively precipitated. The slurry produced is piped to pressure filters that remove the mixed hydroxides in solid form. (The mixed cake contains all the cobalt and some nickel.) Meanwhile, the filtrate, a pure $NiSO_4$ solution, is piped to the nickel circuit for hydrogen reduction.

11.3.4 Batch Processing of Nickel. The nickel sulphate solution is piped to one of six parallel autoclaves for batch processing. Each autoclave is lined with stainless steel, contains three agitators, and is 2.7 m in diameter and 7.5 m long, with a capacity of 45 m^3. The process requires a series of densifications, in which the solution is reacted with hydrogen gas to reduce contained nickel to metallic powder. In the first densification, a catalyst is used to make nickel "seed". The sulphuric acid formed in the reduction reaction is neutralized by adding anhydrous ammonia (NH_3), forming an ammonium sulphate solution. After numerous batch densifications, the wet nickel powder is discharged from the autoclave and collected by decantation. The powder, ranging in particle size from 100 to 325 mesh, is either washed, filtered, dried, and packaged in drums or - more frequently - compacted into briquettes, heated in a sintering furnace, cooled, and packaged into drums. In either form, the pure nickel is ready for shipment to market.

11.3.5 Cobalt Metal and Ammonium Sulphate. The mixed cake precipitated in the cobalt section, which contains all the cobalt and some nickel impurities, is subjected to further processing to form a cobalt pentamine and to selectively crystallize a nickel ammonium sulphate salt. To accomplish this, the cake is treated with ammonia in an autoclave at 90°C, dissolving cobalt as an ammine complex. The solution is then acidified and cooled, crystallizing the nickel as a double salt. The crystals, separated by centrifuging, are dissolved and recycled to the cobalt removal section. The cobalt, as a dissolved ammine complex, is piped to an ion exchange circuit where any final traces of nickel are removed. The purified cobalt solution is reduced to cobalt metal by hydrogen gas at elevated temperature and pressure. The resultant cobalt metal powder is then stored in surge bins, from which it is withdrawn and packaged as a powder or formed into briquettes before shipment.

Meanwhile, the ammonium sulphate solutions from both reduction operations, containing traces of nickel and cobalt, are passed through an ion exchange circuit to remove the dissolved metal traces. The purified ammonium sulphate is sent to multi-stage evaporators (using steam) to produce crystalline ammonium sulphate. The product is stored for shipment, to be used as a fertilizer or chemical.

12 PROGRAMS FOR CONTROL OF THE WORKROOM ENVIRONMENT

Evaluation and control of the working environment is essential to protect the health and safety of personnel in the nickel industry. To this end a comprehensive program is required to monitor and evaluate exposures in surface plants as well as underground. The end result of a yearly program will be an overall audit of workroom environmental conditions.

A comprehensive program should be established to determine the full shift (eight hours) or short-term (near instantaneous) exposures to contaminants. Care must be taken to ensure that all potential contaminants are included in the program. Contaminants present in the raw materials being produced form only a portion of potential hazards such as gases, solvents, particulates, fumes, heat stress and noise.

The monitoring program should cover fixed (static) sampling locations as well as personal sampling of employees. Major contaminants can be identified on a routine basis and minor contaminants on a non-routine basis. Where provincial or federal guidelines as to the make-up of a program are not available, an in-house design is in order. A typical program could be modelled as follows:

Full-Shift Workroom Monitoring at designated locations should be completed monthly in all surface plants and perhaps at some fixed locations underground. These sample locations should be typical of areas frequented by employees. In conjunction with these tests, short-term tests can be made to identify upset conditions and to provide additional study data. This form of monitoring can be used to identify gases, noise, particulates, mists, etc. These data should be recorded and retained, preferably on a computer for future evaluations.

Occupational Exposure Monitoring (sampling on the employee for eight hours) can be completed for noise, particulates, fumes, heat stress, gases, etc. Where no frequencies have been directed by the government agency involved, in-house criteria can be used. For particulates and gases, one employee in five in a given job classification is sampled once per year. Noise exposure is monitored on an exposure basis: one in ten persons under 90 dBa and one in five persons over 90 dBa in a given job code are monitored once per year. Short-term monitoring data can also be collected in order to identify variations in exposures. Where data have indicated close similarities in job classification, various classifications can be judiciously combined to form an exposure group. Items such as the wearing of respiratory and hearing protection should be noted for each employee sampled. All data gathered should be recorded and stored preferably in a computer for historical purposes and future epidemiological studies. It may be a provincial or federal standard that the exposure histories be maintained for extended periods beyond the employees' retirement. Standards for monitoring the workroom environment are available from individual provincial ministries of Labour or Health and the federal government. Exposure criteria generally come from the individual province or federal government, the American Conference of Governmental Industrial Hygienists (ACGIH) (42) and to some extent the National Institute for Occupational Safety and Health (NIOSH) (43), a U.S. agency.

Equipment employed in full-shift workroom and occupational exposure monitoring typically but not necessarily consists of personal gravimetric samplers (Bendix, Dupont, MSA), noise dosimeters (B & K, Metrosonics, Monarch) and gas dosimeters (Dupont, Abcor or 3-M).

Short-Term Workroom Monitoring is generally not considered as a compliance method yet serves as an excellent policing method of sampling. Near-instantaneous sampling indicating immediate readings can be a benefit when conditions warrant action. Equipment offered by such firms as Drager, GCA, MSA, Bendix and numerous others are employed to identify gas, dust, noise, etc.

Metallurgical Account samples of workroom air should be completed once per year (or as required by law) to identify contaminants and yearly changes in exposure. This sampling method usually employs a Hi-volume sampler 0-100 m^3/h to gather a bulk sample which is submitted for a comprehensive analysis identifying major and minor elements that may be of concern.

Vehicle Emission Monitoring requirements as spelled out in provincial mining and labour legislation for internal combustion engines used in underground applications must be met. Fork-lift trucks or other motorized vehicles operating within surface buildings require exhaust monitoring for carbon monoxide to ensure acceptable workroom conditions.

Non-Routine Monitoring for contaminants not specified in the Occupational Exposure Monitoring Program is a critical area of concern. Contaminants in the nickel industry such as mercury, lead, solvents and asbestos, to name a few, can generally be categorized as non-routine type exposures. Provision must be made to identify, monitor and improve conditions in areas where non-routine contaminants occur.

Analytical procedures employed are generally best accepted practice or from in-house experience utilizing (e.g.,) atomic absorption, X-ray diffraction, gas chromotography and wet chemical methods. In addition, the ACGIH and the National Institute for Occupational Safety and Health (44) have recommended procedures for numerous contaminants.

Typical Controls for workroom environment exposures are local and general ventilation (i.e., engineering controls), use of personal protective equipment such as respirators, ear plugs or muffs and administrative controls (limiting employee access or limiting time of exposure). General exhaust hooding and contaminant cleaning by scrubber, fabric filters and electrostatic precipitators are primary methods of control. Prudent design of contaminant control systems may allow for the return of vitiated air which has been mechanically cleaned and meets existing criteria for recirculation.

1. HI-VOL SAMPLING

Samplers are equipped with either flow controllers or flow recorders. They are calibrated with manometers for accurate readings.

2. PERSONAL SAMPLING - DUST AND SO_2

All particulate and fume samples are taken with pumps attached to the individual and the sample head located as close to the breathing zone as possible. Calibration is carried out with soap bubble meters or standardized dry-gas meters. Passive dosimeters (Abcor, 3M, DuPont) and standard gas impingers are available for the normal gases expected in a mine or smelter environment.

3. PERSONAL SAMPLING - NOISE

Dosimeters are attached with microphones close to the ear. Some testing is carried out with microphones inside muffs. Dosimeters are set for A.C.C.I.H. criteria - 100% for 85 dBa LEQ over eight hours and all noise above 80 dBa recorded. Calibration is carried out before and after use with appropriate calibration equipment.

MISCELLANEOUS

Direct-reading colourimetric indicator tubes (Draeger, Castec) and gas dosimeters and monitors are used and calibrated as specified. Organic vapours can be sampled employing charcoal tubes connected to a standard personal pump.

FREQUENCY OF SAMPLING

This is based on a one in five sample density for personal samples with a minimum of six samples per year. Groups vary in size from one person to 100 or more. The larger groups are in the underground mining operation. Frequency of sampling per group varies from six to twenty or more per year.

13 PROCESS DESCRIPTION - LATERITES

The processes used to extract nickel from lateritic ore bodies can be broadly divided into pyrometallurgical and hydrometallurgical technology. Typical examples of each type will be described.

13.1 Pyrometallurgical Processes

13.1.1 Reduction Smelting Process for Nickel Matte. An example of this process is the Inco-Hanna joint venture in Guatemala. Another location using virtually the same technology is Inco in Indonesia. A typical flow sheet is shown in Figure 18. For more details the reader is referred to Reference 45.

In the Guatemala operation, ore is mined by the open pit method at a rate of about 670 000 t/d, grading about 1.95% Ni. Mining begins with removal of heavy jungle growth, accomplished by slashing, burning, and bulldozing the area. Some lumber is recovered. The overburden is then stripped with scrapers and power shovels and trucked to a disposal area. As worked-out mining blocks become available, they are used as overburden disposal sites. When contact with the ore zone is approached deep sample holes are sunk and pockets of overburden remaining in the irregular ore-overburden contact zone are identified. It is very important to minimize dilution of the lateritic ore as no technology exists to concentrate the ore.

Ore is removed by front-end loaders from panels about 11 m wide, and loaded onto trucks. No blasting is required. Recovery of the bottom zone of the ore body is effected by backhoes.

Run-of-the-mine ore is fed to a double roll crusher. The minus-150-mm product is then stockpiled as wet ore, using a stacker-tripper machine to permit some blending of the ore. Wet ore is then reclaimed by a front-end loader and charged to a rotary dryer. Hot gases for drying are provided by an external combustion chamber fired with residual oil. The moisture content is reduced from 30% to about 20%, after which the ore is sized over a discharge trommel. The plus-100-mm size fraction is recycled to the wet ore crusher. The intermediate-size fraction (plus 40 mm) is crushed in a Hazemag crusher and then joins the minus-40-mm fraction. This partially dried material is then sent to a large stockpile. Since the process metallurgy is sensitive to changes in nickel, iron, silica, and magnesia content, it is important that the feed be as uniform as possible. This is achieved by laying the dried ore down with a stacker and then reclaiming the ore

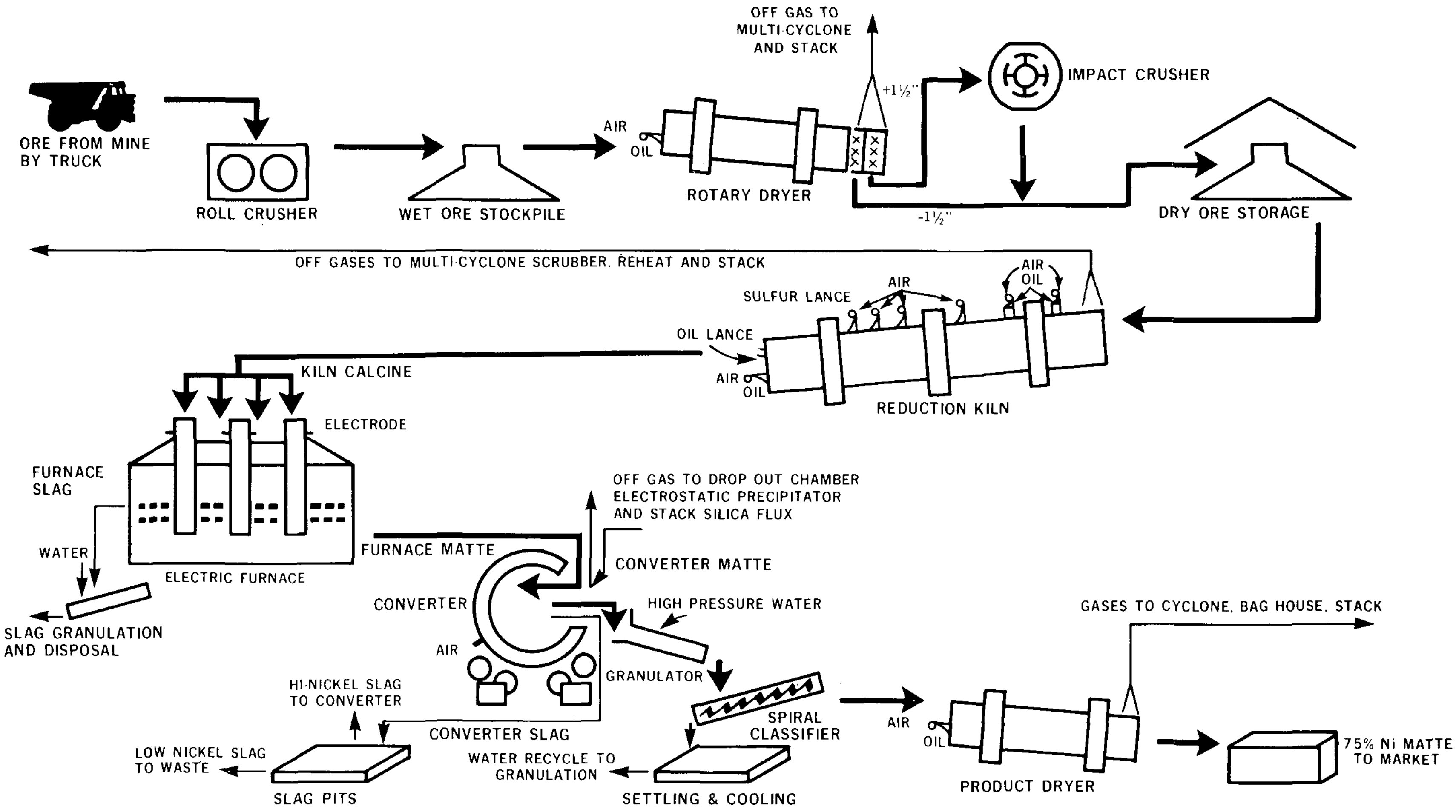


FIGURE 18 EXMIBAL PROCESS FLOW DIAGRAM (GUATEMALA)

by front-end loaders operating perpendicular to the direction of travel of the stacker. The stockpile is typically 36 000 wet tonnes (about 2 weeks' production).

Dryer off-gases are passed through multiclones and the cleaned gases are discharged to the atmosphere. The recovered dust is pugged and joins the main stream of dry ore to storage.

Selective reduction of the nickel and iron is accomplished using a rotary kiln, fitted with burners on the discharge end and along the length of the kiln. Flow of gas and ore is countercurrent. The kiln can be divided into three zones: drying, preheating, and reduction. Two burners on the kiln in the drying zone provide the energy to remove the remaining 20% water. In the preheat zone, the temperature of the ore is increased from 470K to 1120K and the 10-12% water of crystallization is removed. Energy for this is provided by the flow of hot gases from the reduction zone, plus air injected into the preheat zone to expend combustibles in the gas. The third zone is where the reduction occurs and the nickel is sulphided by injecting sulphur in the discharge end of the kiln. An oscillating stream of oil is injected onto the surface of the hot ore, and the products of the oil cracking provide the reductants for the ore. Calcine is discharged at 1170K into a sealed surge bin from which it is transferred to the electric furnace.

Dust from the kiln is collected in multiclones and a wet venturi scrubber, after which the dust is pugged and recycled to the kiln feed.

A circular electric furnace is used to complete the reduction of the ore, using the carbon residue from the oil injection as the reductant. The furnace is choke fed from the transfer bins. The matte is tapped into ladles for transfer to the converting section. The slag is granulated in a pit and trucked to a slag dump. Typical slag assay is: 0.2% Ni, 18% Fe, 0.3% S, 42% SiO_2, 28% MgO.

The final product is made by removing iron in a converter. Silica flux is charged to slag the iron. The slags are poured into casting pits, and are either discarded or recycled to the converter with the flux, depending on the nickel content. Matte from the converter is granulated, dried and packed for shipment to refineries. Typical matte assay is 77% Ni, 0.6% Fe, 22% S.

Gas from the converter is cleaned in a drop-out chamber and an electrostatic precipitator. Dust is also collected in the product dryer and load-out area. All process gas cleaning equipment is checked regularly to ensure that dust losses are under control.

13.1.2 Ferronickel Process. This process is used by several companies in a number of locations as follows: SLN (New Caledonia), Larco (Greece), Hanna (U.S.), Falconbridge

(Dominican Republic). The latter process will be described as a typical example of a modern ferronickel process as represented by the flow sheet in Figure 19.

Mining of the lateritic ore is basically the same as previously described for Guatemala. Run-of-the-mine ore (typically 1.5% Ni) is dumped through a grizzly to remove plus-760-mm boulders. Boulders remaining in the ore passing through the grizzly are broken down by a wobbler and a pneumatic breaker. A rubbler is used to remove the adherent wet ore and separate the softer high-grade material found in the outer layers of the boulders. The process consists of tumbling the ore and rubbing such boulders together in a cylindrical attrition mill. Low-grade rock reject (50 to 600 mm) is discharged from the end of the rubbler and is trucked to a dump. Ore for the process is discharged through rows of grates around the circumference of the rubbler.

Wet ore is stockpiled in sheds and is stored and reclaimed in a pattern to achieve blending. Front-end loaders are used to reclaim ore for the rotary dryer feed system. A trommel is fitted to the discharge end of the dryer. The dryer operates concurrently, with energy provided by an external combustion chamber fired with naptha. Moisture content of the ore is reduced from 26% to about 19%. Dust from the dryer gases is collected in cyclones and added back to the dryer product.

The minus-10-mm fraction from the trommel is fed directly to dry ore storage. The oversize is hammer milled and screened. Low-grade hard rock is rejected from the vibrating screens. Special dust collection systems are provided in the hammer mill area and the dust is returned to the crushing plant product.

The partially dried ore is briquetted to facilitate handling and provide a product form suitable for reduction in the shaft furnaces. It is essential that the descending bed of ore remains permeable to rising flow of reducing gases. The ore is pressed into briquettes by a roll-type machine, and if the ore preparation area is operating satisfactorily no water or binders are needed. The press also acts as a feeder to the shaft furnaces.

Gasification units are used to crack naptha with air to yield the reductant stream for the 12 shaft furnaces. One gasifier is used for each of the 12 shaft furnaces. The gas is cooled to 1150°C before it enters the tuyeres of the shaft furnace. Low-sulphur naptha is used to minimize contamination of the ferronickel product.

The shaft furnaces are refractory lined. The briquettes are distributed across the top of the furnace by a travelling shuttle conveyor. Reducing gases enter through a primary set of tuyeres in the walls of the shaft. Part of the gas rising to the top of the

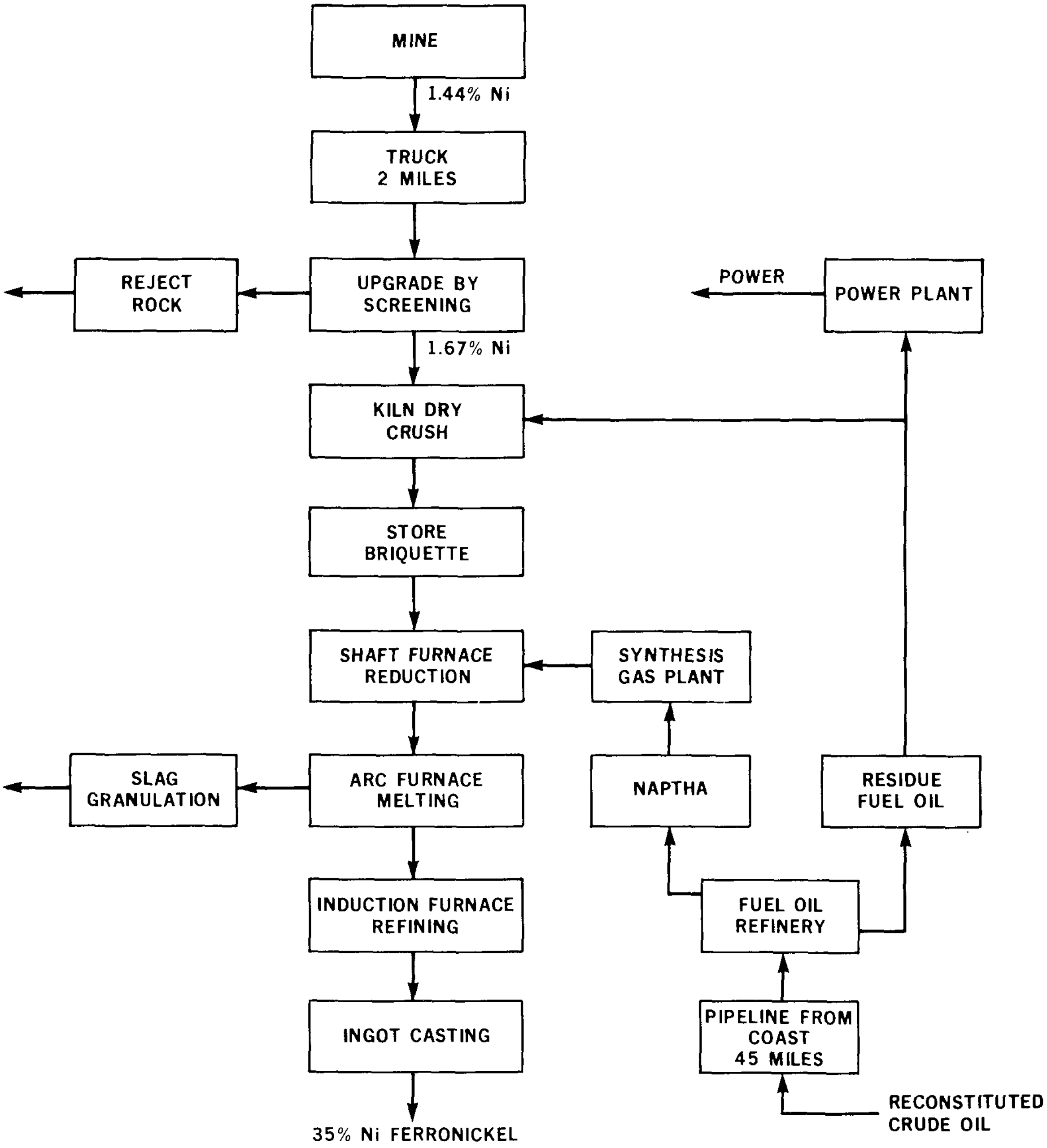


FIGURE 19 FLOW SHEET - REDUCTION, SMELTING - FERRONICKEL FALCONBRIDGE DOMINICA - DOMINICAN REPUBLIC

furnace is collected in a central gas off-take, dedusted in a cyclone, and recycled to a second set of tuyeres together with some air. The secondary air reacts with the recycle gas to provide the heat necessary to dry, dehydrate, and preheat the charge in the upper zone before it enters the reducing zone. The reduced calcine is discharged from the bottom of the furnace by pushers. Exhaust gas from the furnace is processed through cyclones to capture the dust. Wet venturi scrubbers are used to complete the gas clean-up operation. The wet filter cake is returned to the mine ore stream. Calcine at 850°C is transported to the three electric furnaces in insulated hopper cars. The charge is smelted to crude ferronickel, typically containing 32 to 40% Ni. Slag is transported in pots to the slag dump, located about 2 km from the smelter. The typical slag assay is: 0.15% Ni, 0.01% S, 3% Al_2O_3, 22% FeO, 28% MgO, 46% SiO_2. The ferronickel is tapped into refractory-lined ladles which subsequently serve as refining vessels. A roof fits over the ladle and permits heating the charge by three electrodes. The contents are stirred by an induction drive. Refining consists of removal of phosphorus using burned lime and iron ore, followed by removal of sulphur using ferrosilicon, aluminum, silico-calcium and burned lime. Once the refining is completed, the hot metal ladle is transferred to the casting area where the ferronickel is cast into pigs. The typical product assay is 36% Ni, 0.8% Co, 0.3% Si, 0.01% P, 0.08% S, 0.06% C, and the remainder Fe.

Ventilation of the melting and refining areas is important, particularly as there have been problems in controlling dust evolution around the furnace electrodes. Special slot chambers are installed around the electrodes and are vented to a baghouse to recover the dust.

13.2 Hydrometallurgical Processes

13.2.1 Ammonia Leach Process. This process (Figure 20) was first used in Cuba in 1944 by the Nicaro Nickel Company, a subsidiary of Freeport Sulphur Company. The same basic technology has more recently been used by Marinduque in the Philippines, with a change in the final nickel recovery section in order to recover the nickel as a metallic powder using pressure hydrogen reduction. The Greenvale project in Australia also adds a hydrogen reduction step at the end of the process to recover a 90% Ni product.

In the Nicaro example, described more fully in Reference 45, mining of the ore is basically the same as any other laterite. The wet ore, typically 28 to 36% water and 1.4% Ni, is crushed and then fed to rotary driers. The drier product is about 5% water. The drier off-gas is cleaned in a cyclone and an electrostatic precipitator. The dust catch

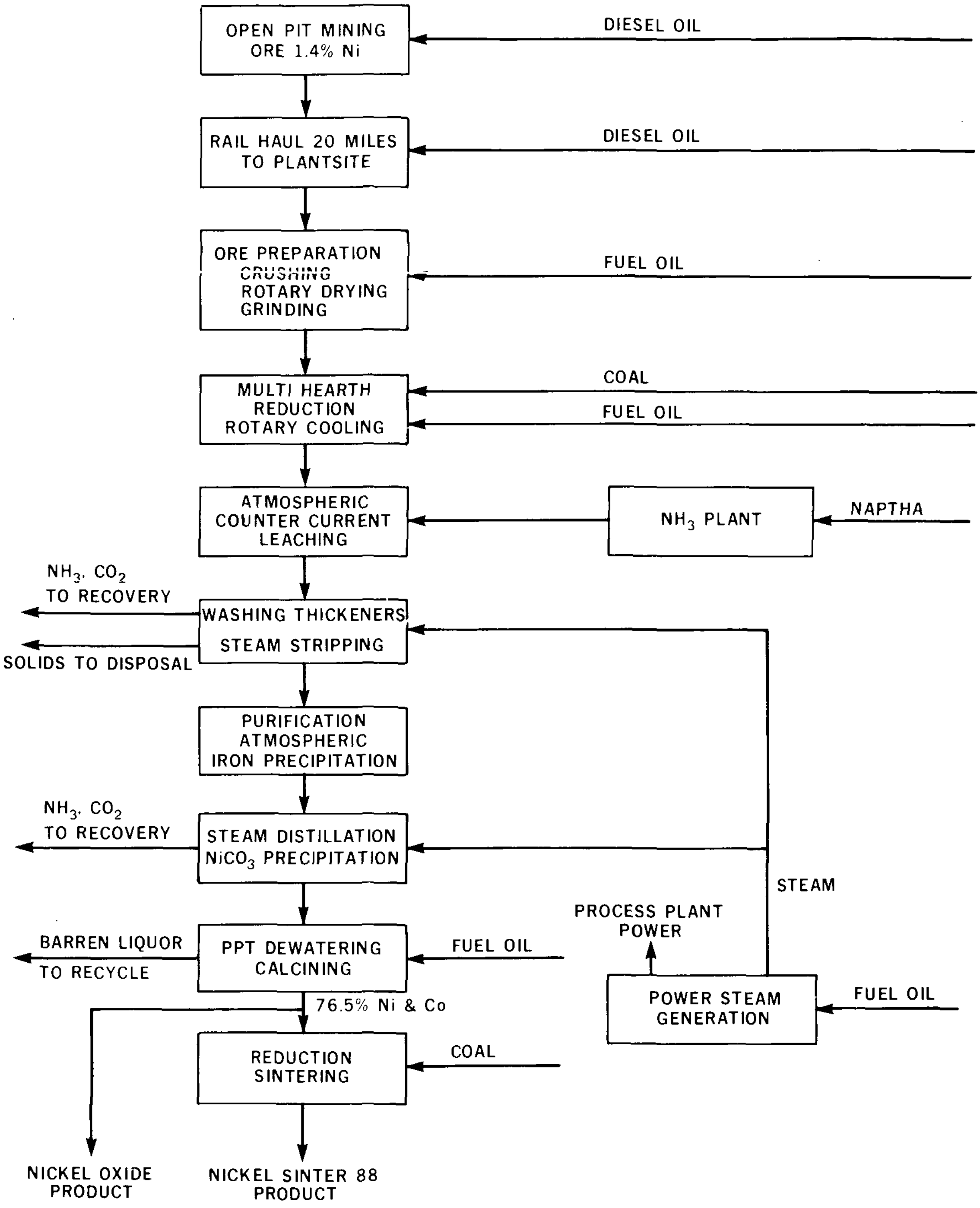


FIGURE 20 FLOW SHEET - AMMONIA LEACHING - NICARO, CUBA

is added back to the ground dry ore. The dry ore has to be finely ground to prepare it for reduction and leaching. This is accomplished by a hammer mill, followed by air classification, and a ball mill to process the coarse fraction so that the final product is 90% minus 200 mesh.

The nickel is selectively reduced in 21 multi-hearth reduction furnaces each with 17 hearths. The reducing gas flows countercurrent to the ore, and is produced from anthracite coal in external gas producers. Additional heat is provided from external combustion chambers fired with heavy oil. Processing conditions are controlled at less than 750°C to reduce the nickel oxide, while minimizing the amount of iron that is reduced. Rabble arms in the multi-hearth reduction furnaces are used to work the ore downward against the flow of the reducing gases, with a designed retention time of about 90 minutes.

The reduced ore is discharged under a non-oxidizing atmosphere to rotary kiln coolers, which rotate in a water bath. The ore leaves the coolers at 150°C and is discharged into a quench tank containing the ammoniacal leach solution, containing about 6.5% NH_3, 3.5% CO_2, and 1% Ni. The flow of solution is adjusted to give a 20% solids in the pulp. The leaching of the nickel is completed in a series of concrete tanks equipped with impellers and spargers through which air is passed. The nickel combines with the ammonia in solution to form the stable hexammine complex. Ferrous iron is further oxidized to the ferric state and comes out of solution as a gelatinous hydroxide precipitate.

The slurry is processed through a series of thickeners where it is washed with a 2% ammonia solution and a 14% ammonia, 8.5% CO_2 solution from the ammonia recovery section. It is necessary to keep the ammonia concentration in the pregnant solution at about 5% in the fourth thickener to prevent nickel precipitation.

Ammonia is removed from the leach tails in a series of wash thickeners. The final tails are heated and the ammonia is stripped in a bubble cap tower using steam. A key design feature of this type of leaching system is to maximize ammonia recovery and minimize dilution of the pregnant solution. Environmentally, it is also important to recover ammonia from the tailings and thus good control of the bubble cap stripper is necessary.

The typical pregnant liquor assays: Ni 12 g/L, Co 0.2 g/L, NH_3 65 g/L, CO_2 35 g/L, plus traces of iron and ore fines. Iron is precipitated by aerating the liquor, followed by filtering the liquor. Bubble cap stills heated with steam are used to drive the ammonia off the clarified liquor. The liquor flows to a flash tank at the bottom of the still and the nickel precipitates out at an ammonia concentration of 2% as an insoluble basic nickel carbonate. Ammonia vapour from the still goes to the recovery system where it is scrubbed with water. Fresh ammonia plus carbon dioxide from the power plant are added to make new leach solution for use in the thickeners.

The nickel carbonate slurry is thickened and filtered. The filter cake is then calcined in a rotary kiln to yield a nickel oxide calcine containing 76.5% Ni, 0.6% Co, 0.04% S and 0.25% Fe. The calcine is pelletized with coal and sintered to make the final product easier to handle in steelmaking, and upgrade the nickel content to about 88%.

In hydrometallurgical processing, nickel tailings disposal is a major environmental problem. In the ammonia leach process, traces of ammonia in the discarded tailings and in other miscellaneous effluents can also pollute surface water. Overall recovery of nickel by the ammonia leach process is very low compared to pyrometallurgical processes, typically 76% versus 85% or higher.

13.2.2 Pressure Leaching with Sulphuric Acid. This process (Figure 21) was also developed by the Freeport Sulphur Company, for their project in Moa Bay, Cuba. This is the only example of this technology, but it has been considered in feasibility studies by Inco for processing the limonitic fraction of its ore in Indonesia. Compared to ammonia leaching, the recovery of both nickel and cobalt is high, about 90%. Another interesting feature of this technology is the relatively low use of energy compared to other processes.

At Moa Bay, the ore is passed through a grizzly to reject boulders, and it is then slurried. The slurry is paddled to remove lumps and is then screened to remove the plus-20-mesh material. The screen undersize at 30% solids then flows by gravity approximately 5 km from the mine to the process plant.

The slurry is thickened at the plant to about 45% solids and is then preheated to 75°C with steam. The slurry passes through booster pumps and high-pressure feed pumps, and then flows to a heater tower where direct steam injection at 40 atmospheres raises the slurry to reaction conditions. It next flows by gravity to the leaching autoclaves. These are brick and lead lined to withstand the vigorous leaching conditions of 230°C and 40 atmospheres. Concentrated sulphuric acid is added to the first of the four autoclaves in series.

Nickel and cobalt are selectively leached from the iron-rich ore using sulphuric acid at elevated temperature (230°C) and pressure. About 95% of the nickel and cobalt are dissolved and only small amounts of iron are dissolved.

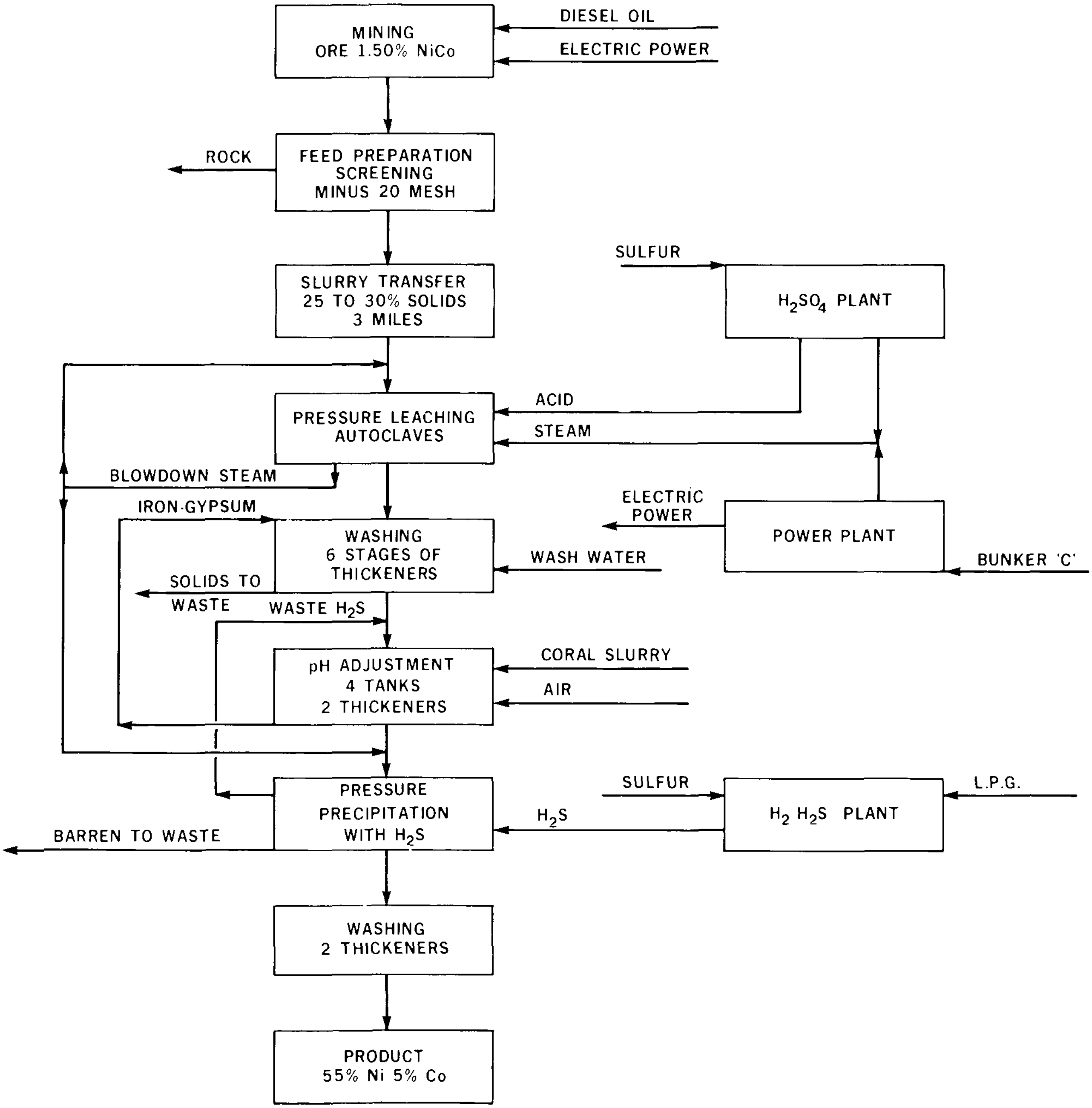


FIGURE 21 FLOW SHEET - PRESSURE ACID LEACHING - MOA BAY, CUBA

The acid-to-ore ratio is the chief variable in controlling the leaching reaction. The retention time is approximately one to two hours. The leached slurry overflowing the final stage passes through a heat exchanger to cool it to 135°C. Low-pressure process steam is produced in the heat exchanger.

The pregnant solution is separated from the tailings in a series of six thickeners, and is washed with fresh water. A high-grade concentrate of nickel and cobalt is obtained from the dilute leach liquor by neutralizing the free acid with coral slurry, and precipitating the nickel and cobalt with hydrogen sulphide gas. Control of the pH during precipitation is crucial to obtain good recovery of the nickel and cobalt, without excessive contamination by iron and aluminum.

The coral slurry used for neutralization is obtained from mud dredged from nearby reefs. The mud is screened and water washed in thickeners to remove chlorides and produce a 40% solids slurry analyzing higher than 90% calcium carbonate. The gypsum sludge from the neutralization is added to the leached slurry in the washing thickeners and leaves the system with the leach tails.

The neutralized liquor is preheated to 75°C using steam from the leaching flash tanks. It is further heated to 120°C and flows to an autoclave. Gaseous high-purity hydrogen sulphide is injected to maintain 10 atmospheres and precipitate the nickel and cobalt. Recycled sulphide precipitate is added as seed material for the precipitation reaction.

The resultant slurry is blown into a flash tank to remove the hydrogen sulphide, which is recycled. The sulphide slurry is then thickened to yield a final concentrate product, typically assaying 55% Ni, 6% Co, 1% Cu and 36% S.

In 1959, Freeport Sulfur built a refinery for this sulphide concentrate in Port Nickel, Louisiana. However, following the takeover of their plant at Moa Bay by the Cuban Government, the sulphide concentrate was shipped to Russia for refining.

14 EMERGING TECHNOLOGY - HYDROMETALLURGY

Hydrometallurgy has been practised for a number of years for treating certain sulphide (Sherritt Gordon, Western Mining) and oxide (Nicaro, Moa Bay, Marinduque, and Greenvale) nickel ores. In these cases, hydrometallurgy had a demonstrable advantage over pyrometallurgy. A number of advances in sulphide hydrometallurgy have occurred in the last few years and an overview and the potential of these developments will be discussed here.

The principal advantage of hydrometallurgy over the present smelting processes used for nickel ores is the improvement in the working environment, the ease with which the process can be automated and operated on a continuous basis, and the elimination of emissions of dilute and often discontinuous streams of sulphur dioxide.

The hydrometallurgical processes for treating nickel sulphide ores or concentrates convert the sulphide sulphate to elemental sulphur which is innocuous or to sulphate which can be sold as ammonium sulphate or impounded as gypsum. Clearly, the biggest potential is in producing elemental sulphur. Elemental sulphur can be produced under oxidizing conditions and only in acidic media. The acidic medium employed usually has sulphate, chloride, nitrate or some combination of these ions.

Leaching in sulphate media, under oxygen pressure (~7 atmospheres) and at 45°-60°C, oxidizes the sulphide sulphur from the pyrrhotite and pentlandite to elemental sulphur. Depending on conditions, up to 80% of the sulphur is converted to the elemental form, and about 20% is oxidized to sulphate and sulphuric acid, and better than 90% of the nickel, cobalt and copper in the ore dissolves. Considerable work was carried out on defining the leach conditions, ultra-fine grinding of the leach feed, use of catalysts, additives to disperse the elemental sulphur, etc.

Considerable work was also done in the chloride media. Hydrochloric acid, chlorine, oxidizing agents such as ferric chloride, cupric chloride and oxygen have been used (Cyprus, Duval, Falconbridge). All of these extractive processes have an application in the nickel hydrometallurgy of the future.

In Canada, because of the nature of the nickel ores - pentlandite extensively intergrown with pyrrhotite - mineral beneficiation yields concentrates containing 10% Ni (consists of two components, high-grade fraction of about 16% Ni, and a low-grade fraction of about 4% Ni) and pyrrhotite concentrates containing about 1% Ni. Treatment of the pyrrhotite concentrates by hydrometallurgy will undoubtedly be one of the first of the new generation of plants.

A typical plant of the future will process low-grade nickel concentrates (1% to 5% Ni). The process could consist of fine grinding of the concentrates followed by oxygen pressure leaching of the water pulp (45°C, 7 atmospheres) to solubilize the nickel, cobalt and copper, oxidize about 20% of the sulphur to sulphate and convert about 80% of the sulphur to the elemental form. The elemental sulphur will be separated from the leach residue by flotation, and can be impounded or the concentrate (~95% S°) melted, hot filtered and sold. The dissolved metals are separated and washed from the leach residue solids by a system of countercurrent wash thickeners. The underflow solids are neutralized with lime and impounded in the tailings pond. The pregnant solution containing the nickel, cobalt and copper will be treated either by solvent extraction to separate the elements and recover the metals by electrowinning, or by bulk precipitation of the hydroxides followed by refining of the cake.

15 EMERGING TECHNOLOGY - PYROMETALLURGY

Recognizing atmospheric pollution control as a critically important concern for the nickel industry, pyrometallurgists are exploring the possibilities for improving control of sulphur dioxide and particulate emissions. The world's largest nickel smelter, Inco at Copper Cliff, contains multi-hearth roasters and reverberatory furnaces which emit continuous streams of low-concentration SO_2 gases, and converters which produce somewhat higher concentration SO_2 gases at variable rates. Neither the roaster, the furnace nor the converter gases are readily amenable to fixation of their SO_2 content as sulphuric acid because of the low concentrations and/or the variability of their compositions. Having noted the serious problems encountered by the U.S. copper smelting industry in attempting to fit sulphuric acid plants to old smelters not designed for the application, nickel industry metallurgists probably will want to explore the use of alternative smelting processes which will be as amenable to pollution control as to the economic processing of nickel concentrates.

Falconbridge's new smelter and acid plant, commissioned in 1978, replaced sintering and blast furnace operations with fluid bed roasters and electric furnaces. The continuous high-strength SO_2 gas stream from the fluid bed roasters is treated in an acid plant, permitting the fixation of about 50% of the SO_2 generated in the smelting process.

Outokumpu flash smelting technology initiated at Harjavalta, Finland, has been applied at Western Mining, Kalgoorlie (Australia), and at Bamangwata Concessions' Pikwe Smelter, Botswana. The process yields about 75-80% of the SO_2 as a continuous high-strength gas stream amenable to fixation as acid, but only at Harjavalta has an acid plant been installed. The Pikwe flash smelter was originally equipped with furnace gas reduction facilities, using coal, to convert the SO_2 to elemental sulphur, but this part of the process proved difficult to operate and was abandoned. In the copper industry, the new Phelps Dodge Outokumpu flash smelter at Hidalgo was equipped with facilities for sulphur dioxide reduction by naphtha, but the sulphur plant was mothballed before commissioning when escalation of fuel prices made SO_2 reduction uneconomic. Hidalgo now has acid plants, treating both flash furnace gases and converter gases.

The Inco flash smelting process uses 95% oxygen for combusting the concentrates, whereas the Outokumpu process uses preheated and/or oxygen-enriched air as oxidant. In the copper application, the Inco flash smelter yields a gas stream containing about 80% SO_2, which is directly liquified, and sold commercially. The Inco flash furnace could be applied to nickel concentrate processing and would similarly yield over 50% of the total nickel smelter generated SO_2 as a high-strength (80% SO_2) stream. However, the lack of commercial markets for the very large quantity of liquid SO_2 derived from the nickel smelter, additional to the existing copper flash smelter, probably would dictate disposition of most of the nickel flash smelter SO_2 gas to an acid plant rather than to liquefaction.

To facilitate collection of larger proportions of nickel smelter generated SO_2 into continuous streams more readily amenable to fixation requires replacement of the batch converting operation by some form of continuous converting process. In copper practice, the Noranda continuous reactor goes a long way toward fulfilling this requirement, and a similar process probably could be applied to nickel. The Noranda unit comprises a cylindrical converting furnace into which copper concentrates are injected, the concentrates being melted by the heat of oxidation of the matte by air blown through submerged tuyeres. Furnace products are "white metal" containing about 75% Cu and copper-bearing slag. The latter is cleaned by milling and concentration, the slag copper concentrate being recycled to the furnace. The white metal is converted to blister copper in conventional converters. Typically 75% or more of the SO_2 generated in producing blister copper is emitted from the continuous reactor in a steady stream at an SO_2 concentration suitable for sulphuric acid manufacture. Application of this process to nickel concentrates smelting would require development of a different slag cleaning system, since nickel slags are not amenable to cleaning by milling and concentration. Pyrometallurgical slag cleaning methods, in a separate vessel, would be applicable to the nickel slag.

The concept of simultaneously flash smelting nickel concentrate, converting matte, and cleaning slag in a segregated section of a single smelting vessel is being explored in relatively small-scale pilot-plant operations; the investigation is still in an early stage.

Other concepts worthy of consideration include modification of the conventional roaster - reverberatory or electric furnace - batch converter operation, to eliminate emission of SO_2 from the batch converters. This might be achieved by removing a very large proportion of the nickel concentrates' sulphur content in fluid bed roasters, leaving in the roaster calcine only the amount of sulphur needed to form nickel bessemer matte upon converting the furnace matte.

The foregoing review indicates some of the options to be investigated by anyone contemplating exploitation of a nickel sulphide ore deposit. Exploitation of oxide nickel deposits poses a different set of problems, but sulphur dioxide pollution is usually of only minor concern.

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