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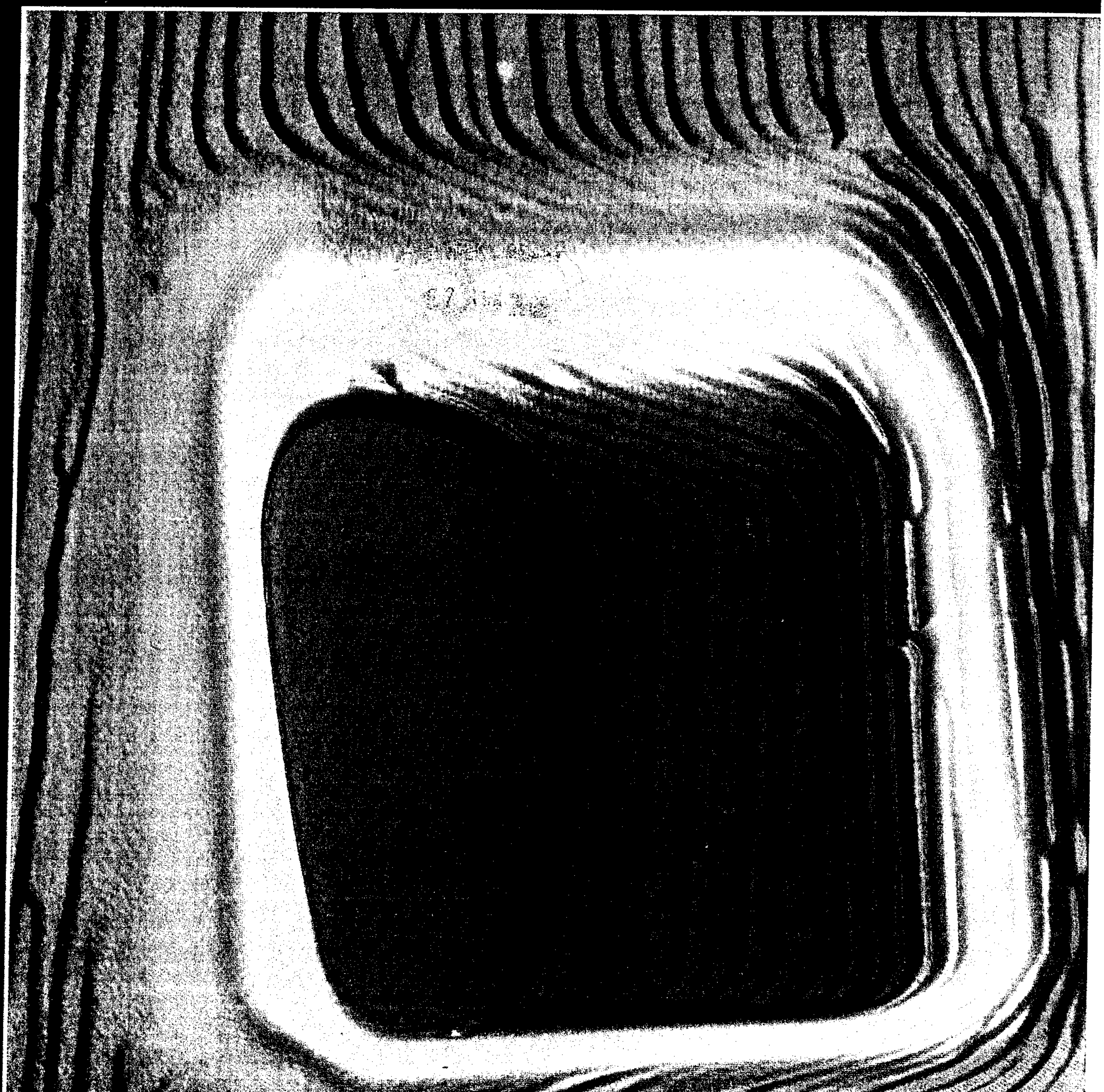

THE COMMERCIALIZATION OF NANOMATERIALS

JOM WORLD SMELTER SURVEY

- Nickel: Laterite Smelting

JOM World Nonferrous Smelter Survey, Part III: Nickel: Laterite

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In July 2004 JOM *published the first installment in an ambitious TMS program: the World Nonferrous Smelters Survey. The program is intended to develop a database of all known nonferrous smelters. This paper, the third installment in the project, presents a survey for nickel smelters processing lateritic or other types of oxidic nickel ores. Data for nickel sulfide smelting is scheduled to be published by* JOM *in the second half of 2006.*

INTRODUCTION

This nickel smelter survey is one of a series that the Extractive Processing Division of TMS is publishing in *JOM* as an important component of the services that the society delivers to its members. This series was initiated in 2004 with the publication of a copper smelter survey.[1,2] Within TMS, we believe that such surveys constitute a valuable information source for industry, research organizations, engineering companies, and academia, and an excellent means of facilitating benchmarking and the identification of potential areas of technical cooperation.

Several nickel smelting surveys and industry reviews were published between 1987 and 2003;[3–7] the present survey covering 13 smelters presents the latest review of world laterite nickel smelters. The 13 world laterite smelters reported here total some 365,000 t/y of nickel output, representing about 30% of total world primary nickel output. A very high proportion of current operations participated, directly or indirectly, in this new survey. The tables included in this paper present data for nickel smelters processing lateritic or other types of oxidic nickel ores. Part II of the survey, reviewing the data for nickel sulfide smelting, will be published by *JOM* in the second half of 2006.

In this survey, smelter representatives were invited to review/complete technical questionnaires that were pre-filled by the authors using public information. Doniambo and Larymna did not participate. In these two cases, available public data are presented in the accompanying tables. Three operating Ural smelters, Rezh, Ufaleynickel, and Yuzhuralnickel, that are still using blast furnaces to produce matte from agglomerated oxidic nickel ores, were not included in the survey.

Within TMS, we believe that such surveys constitute a valuable information source for industry, research organizations, engineering companies, and academia, and an excellent means of facilitating benchmarking and the identification of potential areas of technical cooperation.

Table I lists the plants in the survey and their respective annual nickel productions. The nickel laterites smelting survey results are presented in Tables II to IV that were composed by grouping the smelters as Latin America (Table II); Japan and New Caledonia (Table III); and Indonesia, Eastern Europe, and Greece (Table IV).

Tonnages in these tables are given in metric tonnes. The acronym NMBF stands for "new metal bearing feed."

NICKELIFEROUS LATERITES: BACKGROUND

Nickeliferous laterites are ores that were generated by the prolonged weathering of "ultramafic" rocks containing ferromagnesian silicate minerals. In this weathering process, nickel leaches from the upper layers and subsequently precipitates in the lower layers, substituting NiO for MgO and FeO in the lattice of respectively silicate and iron oxide minerals. The chemistry and mineralogy of these ores vary within a very wide range, particularly with respect to Fe/Ni and SiO_2/MgO weight ratios, and chemical and physical H_2O contents. Nickel is recovered from high iron-containing laterites (limonite, nontronite/smectite) by hydrometallurgical processing, while pyrometallurgy is generally used to extract nickel from low iron-containing saprolites and garnierites. Dalvi et al. report that laterites contain about 70% of the estimated world land-based nickel reserves;[8] about 40% of these ores would be suitable for smelting.

The first processing treatment for recovering nickel from laterites was developed in 1879 in New Caledonia, based on the iron blast furnace technology of the day. Production of nickel from laterites has grown slowly since that time. However, during the 20th century, sulfide ores were the predominant source of primary nickel, essentially due to the available reserves and the cost of production. With improvements in technology, the proportion of primary nickel produced from laterites increased steadily

Table I. World Laterite Smelters in 2005 TMS Survey, Annual Nickel Output

Plant	Country	Annual Tonnage (t/y)*
Falcondo	Dominican Republic	28,500
Cerro Matoso	Colombia	49,100
Loma de Niquel	Venezuela	17,400
Codemin	Brazil	6,500
Hyuga	Japan	22,000
Hachinohe	Japan	41,000
Nippon Yakin	Japan	15,000
Doniambo	New Caledonia	60,000
PT Inco	Indonesia	72,000
Aneka Tambang	Indonesia	11,000
PFK	Ukraine	16,000
FENI	Macedonia	7,000
Larymna	Greece	19,200
Total		**364,700****

*Annual tonnages are typical for one year in the period 2002–2004.
** Equals 30% of world Ni (1,200,000 t/y).

in the second half of the last century. In 2003, these ores accounted for 42% of the world 1,200 kt primary nickel production; ~70% of the laterite nickel was produced by pyrometallurgical processing.[8] Table I presents this nickel output by plants in the survey.

Dalvi et al. predicted that by 2012 half of the world primary nickel will be produced from laterites, and that the proportion of nickel extracted by hydrometallurgical processing of these ores will also increase. Clearly then, the next decade promises to be an interesting one for nickel laterites.

PYROMETALLURGICAL PROCESSING OF LATERITES

The standard laterite pyrometallurgical flowsheet consists of the following stages, each with a specific objective:

- Drying: elimination of most or a substantial portion of the free moisture content of the ore
- Calcining-Reduction: elimination of the remainder of the free ore moisture and of its crystalline water, preheating of the ore, and reduction of a substantial portion of the nickel and a controlled portion of the iron
- Electric furnace smelting: completion of reduction of the nickel and separation of the product ferronickel from the gangue that reports to a ferromagnesian silicate slag
- Refining: elimination of undesirable minor elements from the ferronickel to meet market specifications.

The production of sulfur-deficient matte in smelting by adding sulfur to the feed calcine, followed by converting of this material to a low-iron nickel matte product (Inco's Sorowako operation, see Table IV) is an important variation of the standard flowsheet. A second variation, which developed from a technique for iron production, consists of conducting the calcining-reduction stage at a sufficiently high temperature to cause the partial melting of the calcine, thus permitting the growth of ferronickel granules.

The pasty calcine is subsequently water-quenched and comminuted, and the metal granules (referred to as "luppen" from the original German developments in this approach to iron production) are magnetically separated from the gangue (Nippon Yakin's Oheyama operation, see Table III). The reader is referred to an earlier survey[6] for a thorough discussion of the chemistry of the pyrometallurgical processing of laterites. Good descriptions and flowsheets of a number of laterite smelters are found in References 7, 9, and 10.

Feed

Due to their chemical and mineralogical composition, laterites are not amenable to concentration by physical means. However, screening is normally used to separate low-nickel-containing boulders from run-of-mine ore. Typically, the feed to a laterite smelter contains 1.5–2.5% nickel, 25–35% free moisture, and 10–12% crystalline water. In addition, the normal products of smelting, ferronickel and ferromagnesian silicate slag, have high liquidus temperatures. Not surprisingly, laterite smelting is a highly energy-intensive operation, with an average smelting electrical consumption of 502 kWh/t of calcine, based on the data reported in Tables II to IV.

This survey shows that 77% of the laterite smelters are part of integrated mine-smelter operations; the others are custom operations. In the dedicated plants, blending of ores from various mining sites is practiced to generate a relatively constant composition feed to the smelter. Custom operations, such as the Japanese smelters, process various blends of ores imported from New Caledonia, Indonesia, and the Philippines. The Eastern European smelters, including Larymna (Greece), process oxidic nickel ores that have lower moisture contents and substantially higher Fe/Ni and SiO_2/MgO weight ratios than typical laterites.

Drying

Most plants in the survey use direct-fired rotary dryers for elimination of a portion of the free moisture of the ore. Ideally, the dryer should yield an easy-to-handle, non-dusting product. This limits physical water evaporation in dryers, with the product still containing from about 15% to slightly above 20% moisture. Some smelters do dry to lower moisture contents (e.g., Cerro Matoso [Table II]). Drying is a low-temperature operation, with the moisture-laden off-gas exiting the dryer at about 100°C.

Calcining-Reduction

The partially dry ore is calcined and reduced in slightly sloped rotary kilns (RKs). The exception is the Falcondo Smelter, where these process steps take place in rectangular section shaft furnaces[11] that are fed with partly dried ore as briquettes.

In RKs, fuel is burned substoichiometrically at the solids discharge end in order to generate the reducing atmosphere required to control iron reduction to the desired level. The addition of a solid carbonaceous reductant such as bituminous coal or anthracite to the ore is common practice. Combustion gases travel countercurrent to the slowly moving ore that is successively dried, preheated, calcined, and finally partially reduced. Temperatures above 700°C are required to fully eliminate crystalline water. The accompanying dissociation of the lattice structure of the hydrated silicates generates highly reactive amorphous oxides that in turn lead to fast reduction rates. Much higher temperatures would result in undesirable silicate recrystallization and calcine stickiness. While traveling to the feed end of the kiln, the gas combustibles are gradually burned with air fed through kiln-mounted pipes, thus optimizing fuel utilization. Calcine is normally discharged at 700–900°C, while the low combustibles off-gas leaves at the feed end at 250–400°C. Dusting rates are typically 10–20%. The dust is normally agglomerated prior to being recycled to the RK.

Current RK technology efforts focus on adopting/improving computerized process monitoring/control and dependable kiln on-board instrumentation, and achieving higher energy efficiency and ore throughputs. Replacing refractory bricks by monolithic castable lining has resulted in longer kiln campaigns. At some plants, efforts are underway to control dust generation and improve the treatment of this dust. Recently, a computational fluid dynamics model was used for the basic design of Sorowako's RK #5.[12]

Electric Furnace Smelting

In laterite smelting, the high liquidus temperatures of ferronickel and ferromagnesian slag require slag bath temperatures of about 1,600°C. Increased furnace power density, leading to a higher specific processing rate and lower specific energy consumption, has been achieved by adopting a high-voltage (shielded-arc) operation.[13,14] In this mode of operation, a substantial proportion of the power input is transferred directly to the calcine surrounding the arc, and it is almost exclusively used for smelting, while the power released in the bath suffice to maintain slag and metal at temperatures adequate for tapping.

High-voltage operation was first developed by Falcondo and Hatch Associates in the 1970s, and adopted at Cerro Matoso and Sorowako in the mid-1980s. Today, furnaces in these smelters operate at power inputs of 60 MW to 75 MW, with power densities of 230 kW/m^2 to 360 kW/m^2 of the furnace hearth. New furnace control and power electronics technology have been developed to respond to the decreased furnace stability and power swings associated with high-voltage operation.[15,16]

Figure 1 presents the average electrical consumption in the electric furnaces of the laterite plants reviewed in this survey. The data show that the electrical consumption varies from 379 kWh/t to 600 kWh/t of calcine, with the average being 502 kWh/t of calcine.

Furnace sidewall integrity is an important issue in intensive smelting operations, in particular in those cases where the smelting requirements call for highly superheated slag. A variety of copper water-cooling devices, each of which respond to specific refractory protection needs, are being used to protect furnace integrity.[13,14] Currently, nickel laterite smelting furnace campaigns of 10–20 years between major rebuilds are quite common.

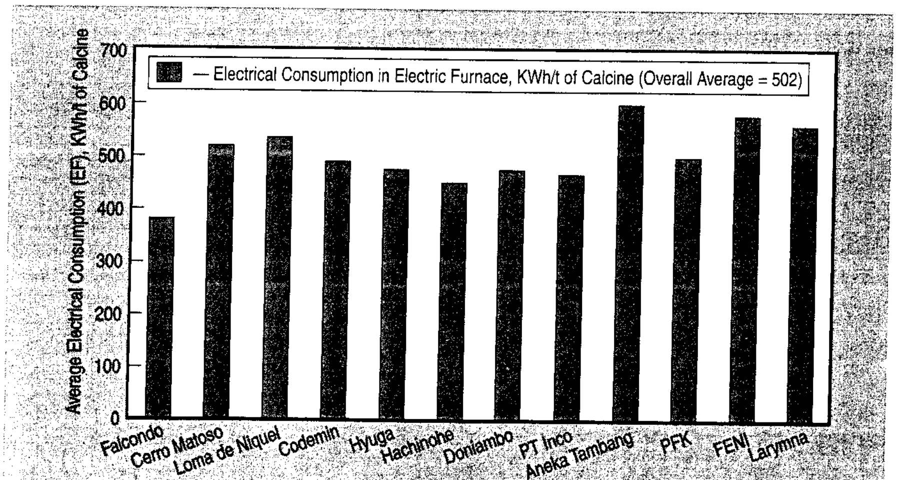

Figure 1. World laterite smelters–electric furnace electrical consumption.

With the exception of P.T. Inco's Sorowako and Nippon Yakin's Oheyama, either high- or low-carbon ferronickel is the usual product of laterite smelters. Ores with a relatively low Fe/Ni weight ratio, not higher than 6, are amenable to producing lower liquidus temperature high-carbon metal at an acceptable nickel grade. This is the case of the Japanese smelters and of Pomalaa. Low-carbon ferronickel is produced from ores with higher Fe/Ni weight ratio. Converting is practiced at the Pobuzhie, FENI MAK, and Larymna smelters to increase the nickel content of the low-grade electric furnace product. The feed to the last two smelters has an unusually high Fe/Ni weight ratio.

Converting and Refining

Tables II to IV give detailed information on the process stages and type of equipment and reagents used for refining crude ferronickel and converting matte to meet market product requirements in different smelters. The tables also contain data on product form and composition.

Nickel Recovery from New-Metal-Bearing Feed

Except for lower-capacity operations, nickel recovery from new-metal-bearing feed in laterite smelting is higher than 90%. In most cases, the survey data show that the smelting nickel partition ratio (i.e., the ratio of the %Ni content of the ferronickel to the %Ni content of the slag) is close to or above 200, based on reported data for the plants in the survey (see Tables II–IV). However, due to the low nickel content of the ore, in laterite smelting the weight ratio of slag to metal product is also high. The highest recoveries are observed in smelters with relatively high-grade and low Fe/Ni weight ratio feed.

ACKNOWLEDGEMENTS

The authors express their appreciation to the smelters that directly or indirectly participated in this survey; without their cooperation this survey could not have been conducted. Thanks are also due to Inco Limited, Falconbridge Limited, the Russian State Research Institute of Non-Ferrous Metals (Gintsvetmet), and WorleyParsons HGE for supporting this work.

Table II. Smelters in Latin America

	Producer			
	Falcondo Falconbridge Dominicana Dominican Republic	Cerro Matoso BHP Billiton Montelibano, Colombia	Loma de Niquel Anglo American PLC Venezuela, SA	Codemin Codemin S.A. (Anglo American), Brazil
Annual Ni Production from NMBF (t/y)	28,500	49,100 (2003)	17,400 (2004)	6,500 (2004)
Form	FeNi	FeNi	FeNi	FeNi
Recovery (%)	91.2	93.8	—	87.5
Feed Composition				
Ni (%)	1.38	2.2	1.48	1.44
Co (%)	0.04	0.085	0.08	0.04
Fe (%)	14.5	15.2–15.4	17	16.9
Fe/Ni	10.5	7	11.5	11.7
S/M	1.6	2.76–2.8	1.3	1.6
Technology				
Drying	2 rotary dryers	2 rotary dryers	1 rotary dryer	1 rotary dryer
Reduction	12 shaft furnaces	2 rotary kilns	2 rotary kilns	2 rotary kilns
Smelting	2 rectangular furnaces	2 round furnaces	2 round furnaces	2 round furnaces
Converting	Not applicable	Not applicable	Not applicable	Not applicable
Refining	2 ASEA-SKF ladles	COBS and ASEA ladles	ASEA ladles	6.5 MVA refining Electric furnace
Blending				
Blending System	—	Blending piles	Blending piles (stacker/reclaimer)	Tripper system/ loader reclaimer
Materials Blended	<64 mm ore + fines recovered from >64 mm fraction	<63 mm ore from screening and crushing	<60 mm ore	Chevron piles <80 mm ore
Drying				
Type of Dryer and Number of Units	2 rotary dryers	2 rotary dryers	1 rotary dryer	1 rotary dryer
Outside Dimensions (Diam., Length–m)	4.27 × 24.4	5.1 × 45	4.8 × 34	3.4 × 22
Nominal Capacity (Dry t Ore/h)	285	260	234	104
Ore Moisture In (%)	23–28	22–30 (seasonal)	25–30	25–27
Ore Moisture Out (%)	18	10–12 (seasonal)	Minimum of 15	23–24
Evaporation Rate (kg H_2O/m^3 Dryer)	93	64	60	27
Fossil Fuel–Type	Naptha	Natural gas	Natural gas	Fuel oil
Average Fuel Consumption (L or kg or Nm^3/Dry t Ore)	—	12–18 Nm^3/t (seasonal)	10–11 Nm^3/t	9.0 kg/t
Dust Handling Rate (%)	—	4	—	0.5
Disposition	Recycled back to dryer	Kiln feed mixer-struder	Pelletized and fed to RK	Transported to the dust bin
Calcination/Reduction				
Equipment: Number of Units and Type	12 shaft furnaces	2 rotary kilns	2 rotary kilns	2 rotary kilns
Size (diam. × length or length × width × height) (m)	5.5 long × 1.37 wide × 8.4 high	RK1 6.1 × 185 RK2 6.0 × 135	5.4 × 120	3.6 × 70
Feed Rate (Dry t Ore/h)	30 (briquettes)	165 each RK	65	75
Calcine Discharge Temperature (°C)	800–1,000	800–850	850	900
Fossil Fuel Type	See reductant	Natural gas	Natural gas	Heavy oil
Average Fossil Fuel Consumption (L or kg or Nm^3/t of Dry Ore)	—	50–55 Nm^3/t	80–85 Nm^3/t	52 kg/t
Reductant Type	Partially combusted naphtha (reformed gas)	Anthracite	Coal	Woodchip
Average Reductant Consumption (kg/t of Dry Ore)	—	50–60	55	180
Dusting Rate (%)	3	RK1 12, RK2 22	15	20
Dust Disposition	Blended with fresh ore	Rotary kiln mixer–Struder	Pelletized and rec. to RK	Recycled to rotary kiln
Smelting				
Electric Furnace	2 rectangular furnaces (Six-electrode in-line)	2 round furnaces	2 round furnaces	2 round furnaces
Outside Dimensions (m)	24.3 × 8.8 × 7.3	22.15 × 7.6	16.8 × 6.8	15 × 6
Furnace Wall Cooling System	Copper cooling fingers	Finger and plate copper coolers	Spray cooling water–shell	Spray cooling water–shell
Maximum Power (MW)	80	75	45	22
Average Power (MW)	56	65–70	40	15.5
Power Density (kW/m^2 Hearth–Average)	329	211	—	117

Table II. Smelters in Latin America (cont.)

	Producer			
	Falcondo Falconbridge Dominicana Dominican Republic	**Cerro Matoso** BHP Billiton Montelibano, Colombia	**Loma de Niquel** Anglo American PLC Venezuela, SA	**Codemin** Codemin S.A. (Anglo American), Brazil
Smelting (cont.)				
Average Voltage (V)	1,500	1,080	500	500
Secondary Current (kA)	—	24	25	15
Nominal Capacity (t of Calcine/h/fce)	140	178	75	29
Average Electrical Energy Consumption (kWh/t of Calcine)	379	520	535	490
Electrode Consumption (kg/t of Calcine)	1.08	1.3	3.9	3.3
Matte/Metal Temperature (°C)	1,455	1,450–1,470	1,550	1,480–1,500
Matte/Metal Composition				
Ni (%)	37–40	35	20–25	28
Co (%)	0.92	0.77	0.45	0.56
Fe (%)	60.3	Balance	78	Balance
Slag Temperature (°C)	1,550	1,560	1,650	1,600
Slag Composition				
SiO_2 (%)	43.4	56	45.3	44.5
MgO (%)	29.3	20	36.6	28.7
Fe (%)	13.9	14.7	11.8	14.9
SiO_2/MgO	1.7	2.8	1.24	1.55
Partition Coefficient (Ni)	257	175	205	215
Partition Coefficient (Co)	92	26	>22.5	—
Converting	See refining	See refining	See refining	See refining
Number & Type	—	—	—	—
Outside Dimensions (m)	—	—	—	—
No of Tuyeres	—	—	—	—
Tuyeres Diam. (mm)	—	—	—	—
Average Blowing Rate (Nm³/h)	—	—	—	—
Blast O_2 (Vol.%)	—	—	—	—
Feed	—	—	—	—
Product Matte Composition				
Ni (%)	—	—	—	—
Co (%)	—	—	—	—
Fe (%)	—	—	—	—
S (%)	—	—	—	—
Slag Composition				
SiO_2 (%)	—	—	—	—
Fe (%)	—	—	—	—
Slag Disposition	—	—	—	—
Refining				
First Step	Dephosphorization	De-P and De-C	De-P and De-C	Dephosphorization
Equipment	2 4 MW ASEA-SKF ladles	COBS and ASEA ladles	ASEA ladles	FeNi tapping ladle
Reagent (s)	Basic oxidizing slag	CaO/SiO_2 slag and O_2	CaO and O_2	CaO and O_2
Process Temperature (°C)	1,500–1,550	1,440	1,650–1,700	1,500
Second Step	Deoxidation	Deoxidation	Desulphurization	Desulphurization
Equipment	Same as above	Same as above	Same as above	6.5 MVA refining electric furnace
Reagent(s)	Ferrosilicon	FeSi and Al	CaO, CaSi, FeSi, Al_2O_3	CaO and O_2
Process Temperature (°C)	1,500–1,550	1,550	>1,600	1,630
Third Step	Desulphurization if required	Desulphurization	—	—
Equipment	Same as above	Same as above	—	—
Reagent(s)	Basic reducing slag	Basic reducing slag	—	—
Process Temperature (°C)	1,500–1,550	1,620	—	—
Form of Product FeNi	100% 0.1 kg "ferrocones"	100% 3–50 mm shots	100% 3–30 mm shot	100% shot
Comp of Product FeNi				
Ni (%)	38.9	34–36	20– 25	30
Co (%)	0.93	<1.0	0.49	0.56
C (%)	0.06	<0.04	≤0.04	0.005
S (%)	0.04	<0.06	≤0.06	0.067
Si (%)	0.35	<0.7	≤0.2	—
P (%)	0.01	<0.04	≤0.03	0.016
Cr (%)	0.02	<0.03	—	—

Table III. Smelters in Japan and New Caledonia

	Producer			
	Hyuga **Hyuga Smelting Co Ltd** **Miyazaki, Japan**	**Hachinohe** **Pacific Metals Co. Ltd.** **Hachinohe, Japan**	**Oheyama** **Nippon Yakin Kogyo, Ltd.** **Oheyama, Japan**	**Doniambo** **SLN (Eramet Nickel)** **Noumea, New Caledonia**
Annual Ni Production from NMBF (t/y)	22,000	41,000 (Avg. 2002–2004)	15,000 (2003)	60,000 (80% as FeNi, 20% as matte)
Form	FeNi (1% ingots, 99% shots)	FeNi (30% ingots, 70% shots)	FeNi Luppen (0.5–20 mm)	FeNi (ingots and granules)
Recovery (%)	97–98	97	93	—
Feed Composition				
Ni (%)	2.1–2.5	2.3	2.3	2.7
Co (%)	<0.1	0.08	—	0.06
Fe (%)	11–23	14	13.6	13
Fe/Ni	4.8–10	6.1	5.9	4.8
S/M	1.49–1.67	1.6	1.9	1.75
Technology				
Drying	1 rotary dryer	1 rotary dryer 2 impact dryers	5 preheating grates (each attached to one RK)	2 rotary dryers
Reduction	2 rotary kilns	3 rotary kilns	5 rotary kilns	5 rotary kilns
Smelting	2 round furnaces	3 round furnaces	Not applicable	3 rectangular furnaces
Converting	Not applicable	Not applicable	Not applicable	PS converters (20% of crude FeNi + sulfur)
Refining	Induction furnace and LD converter	Ladles with stirrers	Not applicable	Shaking ladle
Blending				
Blending System	Ore stockyard	—	1/3 ore ground wet, 2/3 ore ground dry; two fractions blended in rod mill	Blending piles
Materials Blended	±100 mm ore from New Caledonia and Indonesia (target Fe/Ni 0.14–0.18, SiO_2/MgO 1.49–1.67)	Ores from New Caledonia, Indonesia, and Philippines	Blend of ground (<3mm) New Caledonia and Indonesia ores, limestone and anthracite is briquetted	Various saprolite ores
Drying				
Type of Dryer and Number of Units	1 rotary dryer	1 rotary dryer 2 impact dryers	5 preheating grates (each attached to one RK)	2 rotary dryers
Outside Dimensions (Diam., Length–m)	5 × 40	Rotary: 4.75 × 35 Impact: 9 m × 4 m × 3 m	17 m × 4 m	4 × 32
Nominal Capacity (Dry t Ore/h)	160	Rotary: 105; impact: 210	27 (each lower cap. line)	220
Ore Moisture In (%)	23–30	30	17 (feed briquettes)	26
Ore Moisture Out (%)	22–23	24	0	18
Evaporation Rate (kg H_2O/m^3 Dryer)	30	19 (rotary dryer)	NA	72
Fossil Fuel Type	Pulverized coal + bunker C oil for ignition + electric furnace off-gas	Waste electric furnace gas	Rotary kiln hot gas	Heavy fuel oil
Average Fuel Consumption (L or kg or Nm³/ Dry t Ore)	12–13 L (includes oil equivalent of coal)	None	—	—
Dust Handling Rate (%)	2–5	1	14 (RK plus preheater)	—
Disposition	Recycled to dryer	Blended with ore	Recycled to ore blending	To dryer discharge
Calcination/Reduction				
Equipment: Number of Units and Type	2 rotary kilns	3 rotary kilns	5 rotary kilns	5 rotary kilns
Size (diam. × length or length × width × height–m)	4.8 × 105	5.25 × 100, 5.5 × 115, 4.6 × 131	4 RKs 3.6 × 72 1 RK 4.2 × 72	4 × 95
Feed Rate (Dry t Ore/h)	60-65 (each kiln)	90, 110, 90, respectively	27 (Each low cap. line)	—
Calcine Discharge Temperature (°C)	800–900	1,050	1,200–1,250. Pasty discharge is water-cooled, ground. Jigging/magnetic separation yields 23% Ni, 0.5–20mm FeNi luppen	900
Fossil Fuel Type	55–65% pulverized coal, balance Bunker C oil	Pulverized coal	Coal	Pulverized coal
Average Fossil Fuel Consumption (L or kg or Nm³/t of Dry Ore)	60–62 L/t (includes oil equivalent of coal)	30–50 kg/t	80 kg/t	—
Reductant Type	Coal	Coal	Anthracite (briquettes)	Anthracite
Average Reductant Consumption (kg/t of Dry Ore)	70–80	110	130	50
Dusting Rate (%)	15–20	25	14 (RK plus preheater)	10
Dust Disposition	Pelletized and recycled to rotary kiln	Pelletized and recycled to RKs	Recycled to ore blending	Recycled to RKs

Table III. Smelters in Japan and New Caledonia (cont.)

	Producer			
	Hyuga **Hyuga Smelting Co Ltd** **Miyazaki, Japan**	**Hachinohe** **Pacific Metals Co. Ltd.** **Hachinohe, Japan**	**Oheyama** **Nippon Yakin Kogyo, Ltd.** **Oheyama, Japan**	**Doniambo** **SLN (Eramet Nickel)** **Noumea, New Caledonia**
Smelting				
Electric Furnace Outside Dimensions (m)	2 round furnaces #3 18.5 × 5.5; #5 17.5 ×5.4	3 round furnaces 18 × 5.6, 19 × 6.15, 20 × 6.6	Not applicable	3 rectangular furnaces 33 × 13 × 5.5
Furnace Wall Cooling System	Spray cooling water-shell	Spray cooling water-shell	—	—
Maximum Power (MW)	#3 60; #5 40	43, 54, 54, respectively	—	50
Average Power (MW)	Max. power 60MW day time, 80 MW night time	45	—	36
Power Density (kW/m² Hearth–Average)	#3 170, #5 140	—	—	94
Average Voltage (V)	400–900	664, 760, 760, respectively	—	300
Secondary Current (kA)	28–32	35, 42, 42, respectively	—	20
Nominal Capacity (t of Calcine/h/fce)	60–65	80, 100, 100, respectively	—	76 (at 36 MW)
Average Electrical Energy Consumption (kWh/t of Calcine)	470–480	450	—	475
Electrode Consumption (kg/t of Calcine)	~1	1.5	—	—
Matte/Metal Temperature (°C)	1,400–1450	1,450	—	1,500
Matte/Metal Composition				
Ni (%)	17–25	18.5	—	22–28
Co (%)	<0.8	0.5	—	—
Fe (%)	70–75	—	—	—
Slag Temperature (°C)	1,550–1,600	1,550	—	1,600
Slag Composition				
SiO_2 (%)	50–55	54	—	55.8
MgO (%)	32–36	35	—	31.9
Fe (%)	7–10	5	—	5.7
SiO_2/MgO	1.49–1.67	1.5	—	1.75
Partition Coefficient (Ni)	210	264	—	185
Partition Coefficient (Co)	—	25	—	—
Converting	See Refining	See Refining	Not Applicable	
Number & Type	—	—	—	PS converters
Outside Dimensions (m)	—	—	—	—
No. of Tuyeres	—	—	—	—
Tuyeres Diam. (mm)	—	—	—	—
Average Blowing Rate (Nm³/h)	—	—	—	—
Blast O_2 (Vol.%)	—	—	—	—
Feed	—	—	—	20 wt.% crude FeNi + S
Product Matte Composition				
Ni(%)	—	—	—	75–78
Co (%)	—	—	—	—
Fe (%)	—	—	—	—
S (%)	—	—	—	—
Slag Composition				
SiO_2 (%)	—	—	—	—
Fe (%)	—	—	—	—
Slag Disposition	—	—	—	—
Refining				
First Step Equipment	Desulphurization Low-frequency induction furnace	Desulphurization Ladles with stirrers	—	Desulphurization Shaking ladle
Reagent(s)	CaC_2	CaC_2	—	CaC_2
Process Temperature (°C)	1,400–1,450	1,500	—	—
Second Step Equipment	De-C & De-Si LD converter	—	—	Decarburizing Shaking ladle
Reagent (s)	Oxygen	—	—	Oxygen
Process Temperature (°C)	1,600–1,650	—	—	—
Third Step Equipment	—	—	—	—
Reagent (s)	—	—	—	—
Process Temperature (°C)	—	—	—	—
Form of Product FeNi	1% ingots (I), 99% shots (S)	30% ingots, 70% granules	Luppen (crude FeNi)	15–40 kg ingots & granules
Comp. of Product FeNi				
Ni (%)	Hi-C I & S>16 LoC I & S 17–28	17–23	—	FN1 24–30, FN4 22–28
Co (%)	Hi-C & Lo-C I & S <Ni × 0.05	Ni/Co wt. ratio <20	—	—
C (%)	Hi-C I & S<3 Lo-C I & S<0.02	2	—	FN1 0.03, FN4 1.2–1.9

Table III. Smelters in Japan and New Caledonia (cont.)

	Producer			
	Hyuga Hyuga Smelting Co Ltd Miyazaki, Japan	**Hachinohe Pacific Metals Co. Ltd. Hachinohe, Japan**	**Oheyama Nippon Yakin Kogyo, Ltd. Oheyama, Japan**	**Doniambo SLN (Eramet Nickel) Noumea, New Caledonia**
Refining (Cont.)				
S (%)	<0.03	<0.030	—	FN1 0.03, FN4 0.23
Si (%)	Hi-C I & S <5 Lo-C I & S<0.3	2	—	FN1 0.03, FN4 1.0–3.0
P (%)	Hi-C I & S<0.05 Lo-C I & S<0.02	<0.050	—	—
Cr (%)	Hi-C I & S<2.5	1.3	—	—

Table IV. Smelters in Indonesia and Eastern Europe

	Producer				
	Sorowako P.T. Inco Sulawesi, Indonesia	**Pomalaa P.T. Aneka Tambang Sulawesi, Indonesia**	**Pobuzhsky Ferronickel Combine PFK Pobuzhie, Ukraine**	**FENI FENI MAK Macedonia**	**Larymna Larco GMM SA Larymna, Greece**
Annual Ni Production from NMBF (t/y)	72,000 (2004)	11,000	16,000	7,000	19,200 (2002)
Form	Bessemer matte granules	FeNi ingots and shots	FeNi–90% ingots and 10% pigs	FeNi ingots	Converter alloy shot
Recovery (%)	90	96	87	88.5	88–89
Feed Composition					
Ni (%)	1.8–1.9	2.2	2.4	1.25	1.1
Co (%)	0.06	0.05	0.04	0.06	0.06
Fe (%)	20	13.4	18–20	21	32.0
Fe/Ni	10.5	6.1	7.9	16.8	29
S/M	2	1.64	1.9	2.5	12
Technology					
Drying	3 rotary dryers	2 rotary dryers	Not applicable	Not applicable	Not applicable
Reduction	5 rotary kilns	2 rotary kilns	4 rotary kilns	2 rotary kilns	4 rotary kilns
Smelting	4 round furnaces	2 round furnaces	2 rectangular furnaces	2 rectangular furnaces	4 round furnaces
Converting	3PS converters	Not applicable	Not applicable	Two oxygen vertical converters	2 OBM converters
Refining	Not applicable	Ladle with refractory stirrer and shaking ladle	Electric furnace FeNi tapping ladle and vertical oxygen converter	Induction furnace	Not applicable
Blending					
Blending System	No blending prior to drying	Whell loader	Ore stockpile	Ore is crushed, wet ground, and magnetically separated. Non-magnetic fraction is fed to RK	Yes
Materials Blended	Dry, <2.54 cm EB & WB ores blended to control S/M ratio to ~2.0	Ores from 3 different mining areas	Various nickel oxide ores (not typical laterites)	Various nickel oxide ores (not typical laterites)	Various nickel oxide ores (not typical laterites) crushed to <15 mm
Drying	—	—	Not applicable	Not applicable	Not applicable
Type of Dryer and Number of Units	3 rotary dryers	2 rotary dryers	—	—	—
Outside Dimensions (diam., length–m)	#1–5 × 50, #2–5.5 × 50, #3–6 × 65	3.2 × 30	—	—	—
Nominal Capacity (Dry t Ore/h)	#1–240, #2–305, #3–410	50	—	—	—
Ore Moisture In (%)	29–34	30	—	—	—
Ore Moisture Out (%)	20	22	—	—	—
Evaporation Rate (kg H_2O/m^3 Dryer)	47	30	—	—	—
Fossil Fuel–Type	Oil (HSFO)	Pulverized coal	—	—	—
Average Fuel Consumption (L or kg or Nm^3/Dry t Ore)	26	35	—	—	—
Dust Handling					
Rate (%)	—	3	—	—	—
Disposition	Mixed in Pugmill with dust from RK	Added to dryer product	—	—	—
Calcination/Reduction					
Equipment: Number of Units & Type	5 rotary kilns	2 rotary kilns	4 rotary kilns	2 rotary kilns	4 rotary kilns

Table IV. Smelters in Indonesia and Eastern Europe (cont.)

	Producer				
	Sorowako P.T. Inco Sulawesi, Indonesia	**Pomalaa P.T. Aneka Tambang Sulawesi, Indonesia**	**Pobuzhsky Ferronickel Combine PFK Pobuzhie, Ukraine**	**FENI FENI MAK Macedonia**	**Larymna Larco GMM SA Larymna, Greece**
Calcination/ Reduction (Cont.)					
Size (Diam. × Length or Length × Width × Height–m)	RK 1, 2, 3–5.5 × 100: RK 4–6 × 115, RK 5–6 × 135	RK1 4 × 90; RK2 4.2 × 90	3.6 × 75	4.6 × 75	Two RKs 4.2 × 90; One RK 5.2 × 90; One RK 6.1 × 125
Feed Rate (Dry t Ore/h)	RK 1, 2, 3–160 RK 4–220, RK 5–235 all wet t/h	RK1 32; RK2 35	80 (each kiln)	140 (sintered pellets)	80 (6.1 × 125 RK)
Calcine Discharge Temperature (°C)	700	800–1,000	750–800	750–800	850
Fossil Fuel Type	HSFO	Pulverized coal	Coal	Coke	Pulverized anthracite/lignite
Average Fossil Fuel Consumption(L or kg or Nm³/t of Dry Ore)	66 kg oil/t calcine	115 kg/t	40 kg/t	5 kg/t	—
Reductant Type	Bituminous coal	Anthracite and coal	Anthracite	Lignite	Lignite, coal
Average Reductant Consumption (kg/t of Dry Ore)	35–40	67	80	140	—
Dusting Rate (%)	15–17	8	12–16	About 10	6.7 (6.1 × 125 RK)
Dust Disposition	Blended in pug mill and recycled to ore dryer	Pellets of dust + fine ore are recycled to RK	To ore storage	To ore storage	Pelletized and reverted to RK
Smelting					
Electric Furnace	4 round furnaces	2 round furnaces	2 rectangular furnaces	2 rectangular furnaces	4 round EFs
Outside Dimensions (m)	18 × 6	15 × 5.6	24.7 × 9.5 × 6	40 × 11.5 × 6	Diam.–one EF 17.6m; three EFs 12.2m
Furnace Wall Cooling System	Copper cooling fingers	Spray cooling water–shell	Spray cooling water–shell	Spray cooling water–shell	—
Maximum Power (MW)	70–80/furnace	20 and 25, respectively	50	85	1EF 50; 1EF 36; 2EFs 32
Average Power (MW)	55–60	17	38	55	42 (large EF)
Power Density (kW/m² Hearth–Average)	236	114	170	120	189 (large EF)
Average Voltage (V)	1,000–1,800	430	500	300	—
Secondary Current (kA)	28–35	23.5	45	175	—
Nominal Capacity (t of Calcine/h/fce)	126	28.3	63	94	—
Average Electrical Energy Consumption (kWh/t of Calcine)	465	600	620	580	560
Electrode Consumption (kg/t of Calcine)	1.4	1.1	3	2	—
Matte/Metal Temperature (°C)	1,350–1,400	1,450	1,350	1,500	1,450
Matte/Metal Composition					
Ni (%)	26	19.15	17	17	15
Co (%)	0.6	0.3	0.5	0.8	0.7
Fe (%)	63	Balance	Balance	Balance	82
Slag Temperature (°C)	1,500–1,550	1,550	1,550	1,550	1,550
Slag Composition					
SiO_2 (%)	47.6	57.2	48	About 40	36.8
MgO (%)	22.7	31	20	About 18	3.1
Fe (%)	18.4	4.6	20	35	32.7
SiO_2/MgO	2.1	1.85	2.4	2.2	12
Partition Coefficient (Ni)	173	213	>212	170	100
Partition Coefficient (Co)	20	NA	>50	40	35
Converting	—	**Not Applicable**	**See Refining**	**See Refining**	**See Refining**
Number & Type	3PS converters	—	—	—	—
Outside Dimensions (m)	CV 2–7.3 × 11.8, CV3 and CV4–7.3 × 12.7	—	—	—	—
No. of Tuyeres	20–28	—	—	—	—
Tuyeres Diam. (mm)	51	—	—	—	—
Average Blowing Rate (Nm³/h)	18,000	—	—	—	—
Blast O_2 (Vol.%)	Air	—	—	—	—
Feed	EF matte	—	—	—	—
Product Matte/Metal Composition					
Ni (%)	78	—	—	—	—
Co (%)	1	—	—	—	—
Fe (%)	>0.7	—	—	—	—
S (%)	18–22	—	—	—	—

Table IV. Smelters in Indonesia and Eastern Europe (cont.)

	Producer				
	Sorowako P.T. Inco Sulawesi, Indonesia	**Pomalaa P.T. Aneka Tambang Sulawesi, Indonesia**	**Pobuzhsky Ferronickel Combine PFK Pobuzhie, Ukraine**	**FENI FENI MAK Macedonia**	**Larymna Larco GMM SA Larymna, Greece**
Converting (cont.)					
Slag Composition					
SiO_2 (%)	25	—	—	—	—
Fe (%)	53	—	—	—	—
Slag Disposition	Slag >0.6% Ni is reverted cold to electric furnace	—	—	—	—
Refining					
First Step	—	Desulphurization	Desulphurization	Desulphurization	Electric furnace FeNi upgrading
Equipment	—	Ladle with refractory stirrer	Electric furnace tapping ladle	Induction furnace	2 OBM converters
Reagent (s)	—	$CaC_2 + Na_2CO_3$	Soda ash	Calcium carbide	Lime-ferrite slag
Process Temperature (°C)	—	1,350	1,350	1,280–1,330	—
Second Step	—	De-Si (low C FeNi)	Converting (1st stage)	—	—
Equipment	—	Shaking ladle	Vertical oxygen converter (acid lining)	Two vertical oxygen converters	—
Reagent(s)	—	O_2, burnt lime, limestone (1.2 slag basicity)	Ni Oxide/iron ore, scrap, cold crude FeNi	Lime-ferrite slag	—
Process Temperature (°C)	—	1,450	1,450–1,500	—	—
Third Step	—	De-C (low C FeNi)	Converting (2nd stage)	—	—
Equipment	—	Shaking ladle	Vertical oxygen converter (basic lining)	—	—
Reagent (s)	—	O_2, burnt lime, limestone (3.0 slag basicity)	Limestone, FeSi	—	—
Process Temperature (°C)	—	1,620	1,590–1,650	—	—
Form of Product	Granulated Bessemer matte	Ingots and granules	Ingots and pigs	Ingots	Shots (<40mm)
Comp of Product FeNi	—	Hi-C / Lo-C	—	—	—
Ni (%)	—	18.0/21.0	25–35	35–50	20–25
Co (%)	—	0.33/0.38	0.1–0.5	<1.5	—
C (%)	—	2.22/0.011	<0.1	<0.1	—
S (%)	—	0.01/0.008	<0.07	<0.07	—
Si (%)	—	2.22/0.06	<0.05	<0.06	—
P (%)	—	0.021/0.002	<0.03	<0.03	—
Cr (%)	—	1.65/0.07	<0.3	<0.3	—

Visit the JOM *web site (*www.tms.org/JOMPT*) to access spreadsheets used to compile the tables presented in this article. Also available on-line is a spreadsheet of additional information not published in these tables.*

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