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

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*In June 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 Metallurgy 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.

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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.

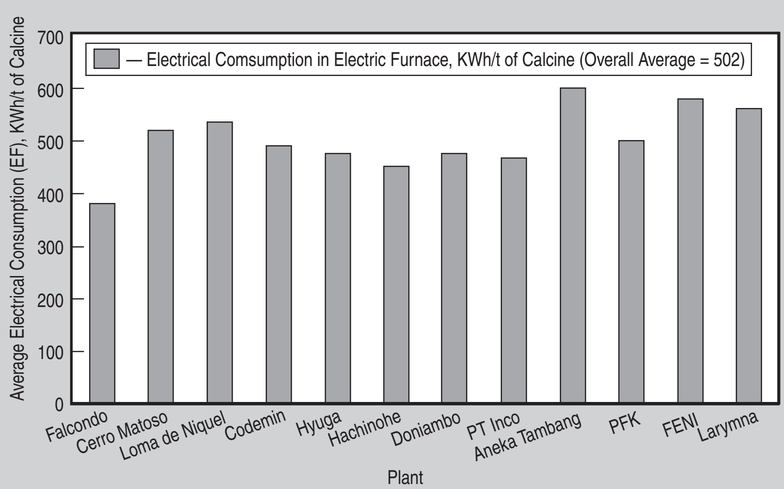

Figure 1. World laterite smelters–electric furnace electrical consumption.

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 ($Nm^3$/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 $Nm^3$/ 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 $Nm^3$/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<br>Hyuga Smelting Co Ltd<br>Miyazaki, Japan** | **Hachinohe<br>Pacific Metals Co. Ltd.<br>Hachinohe, Japan** | **Oheyama<br>Nippon Yakin Kogyo, Ltd.<br>Oheyama, Japan** | **Doniambo<br>SLN (Eramet Nickel)<br>Noumea, New Caledonia** |
| **Smelting** | | | | |
| Electric Furnace | 2 round furnaces | 3 round furnaces | Not applicable | 3 rectangular furnaces |
| Outside Dimensions (m) | #3 18.5 × 5.5;<br>#5 17.5 ×5.4 | 18 × 5.6, 19 × 6.15,<br>20 × 6.6 | — | 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 ($kW/m^2$ 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 ($Nm^3/h$ ) | — | — | — | — |
| 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 | Desulphurization | Desulphurization | — | Desulphurization |
| Equipment | Low-frequency induction furnace | Ladles with stirrers | — | Shaking ladle |
| Reagent(s) | $CaC_2$ | $CaC_2$ | — | $CaC_2$ |
| Process Temperature (°C) | 1,400–1,450 | 1,500 | — | — |
| Second Step | De-C & De-Si | — | — | Decarburizing |
| Equipment | LD converter | — | — | 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<br>LoC I & S 17–28 | 17–23 | — | FN1 24–30, FN4 22–28 |
| Co (%) | Hi-C & Lo-C I & S<br><Ni × 0.05 | Ni/Co wt. ratio <20 | — | — |
| C (%) | Hi-C I & S<3<br>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<br>Lo-C I & S<0.3 | 2 | — | FN10.03, FN4 1.0–3.0 |
| P (%) | Hi-C I & S<0.05<br>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 $Nm^3$/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 ($kW/m^2$ 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 ($Nm^3$/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 (*http://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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