CHAPTER 12.25

Nickel and Cobalt

Andrew Bamber and Arthur Barnes

NICKEL MARKET

Nickel is a strategic commodity globally, with uses in electronics, energy, aerospace, and manufacturing, the most important application of which is in stainless steel used in the construction, automotive, and petrochemical industries. Although the nickel price is currently suffering an extended period of decline—primarily because of the production of cheap ferronickel (so-called pig nickel) in China from largely imported ores—the market remains significant, with potential for recovery as the world economy normalizes after the global financial crisis of 2008–2011. Total production in 2016 was 1.4 Mt (million metric tons), with the major producing countries being the Philippines, Russia, and Canada (see Figures 1 and 2). Sources are predominantly sulfide deposits (60%) with production from laterite deposits making up the balance. Laterites, in addition to being an important source of nickel, are an increasingly vital source of by-product metals such as cobalt and scandium. Additional potential nickel sources are from nickel-containing manganese nodules, typically containing 0.5%–1.0% Ni and associated cobalt, which occur in vast quantities on the seabed at depths between 3,500 and 4,000 m. Although not currently economically viable solely as a source of nickel, increasing demand for manganese, cobalt, and nickel in battery technologies means these deposits will become an increasingly important source in the future.

Major consumers of nickel—largely as nickel in stainless steel—include Europe, the United States, Japan, and, increasingly, China. Other industrial consumption of nickel by the petrochemical and aerospace industries is in the form of the nonferrous alloys: nickel–copper, nickel–chrome and nickel–molybdenum. An additional and fast-growing use is nickel used in battery technology, driven by growth in renewable energy and battery-electric vehicles. Nickel in lithium-ion batteries is in the form of either NCA (80% Ni, 15% Co, and 5% Al) or NMC (33% Ni, 33% Mn, and 33% Co), demand for which is expected to represent 7% of the total nickel market by 2020 (Padhy 2017).

The most important area of nickel demand, however, remains in stainless steel, with demand from emerging markets being the key driver for growth in this area. Although currently, stainless steel, and therefore nickel, has been experiencing an extended bear market with prices consistently under US$5/lb, in 2016, the stainless market began to recover as a result of increasing demand from China and a cutback in supply mainly from Indonesia in 2014. This recovery was further assisted by cutbacks in the Philippines in 2016 with a moderate price recovery now projected in 2017 and beyond (Figure 3).

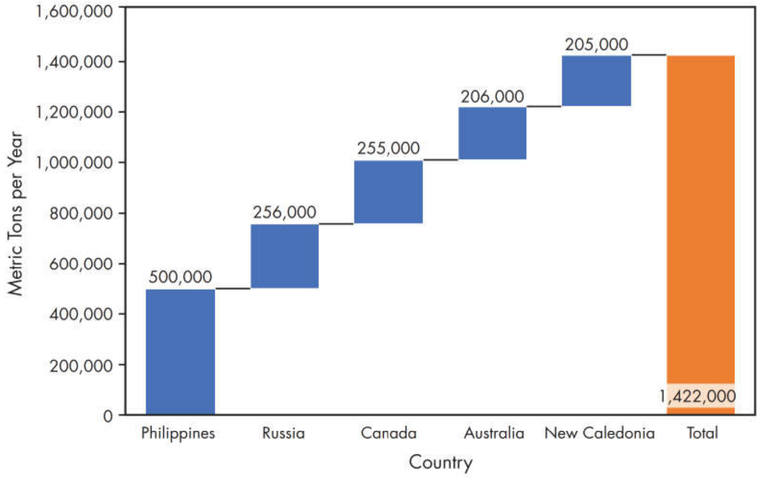
COBALT MARKET

Cobalt was originally mined together with silver and arsenic in Saxony and Bohemia in the 15th century, but its use then was only in the oxide form as the colorant called "cobalt blue." Swedish chemist Georg Brandt first isolated metallic cobalt in 1735, and it was subsequently established as an element in 1780 by Torbern Bergman. Despite this progress in physical chemistry, cobalt was still used only in its oxide form until circa 1914. Since then, there has been a steady increase in demand for cobalt as new forms and uses have been developed, including as a metallic or alloying element (Fisher 2011).

Primary cobalt production is relatively small, comprising approximately 5% of world supply. The leading source of mined cobalt, representing approximately 60% of world cobalt mine production and more than 50% of known reserves, is as a by-product from copper mining in the Democratic Republic of the Congo (DRC); see Figure 4. The majority of the balance of cobalt is recovered as a by-product of nickel processing, largely from lateritic nickel operations in the tropics. In 2016, global cobalt mine production decreased from previous years, mainly owing to lower production from these same nickel operations.

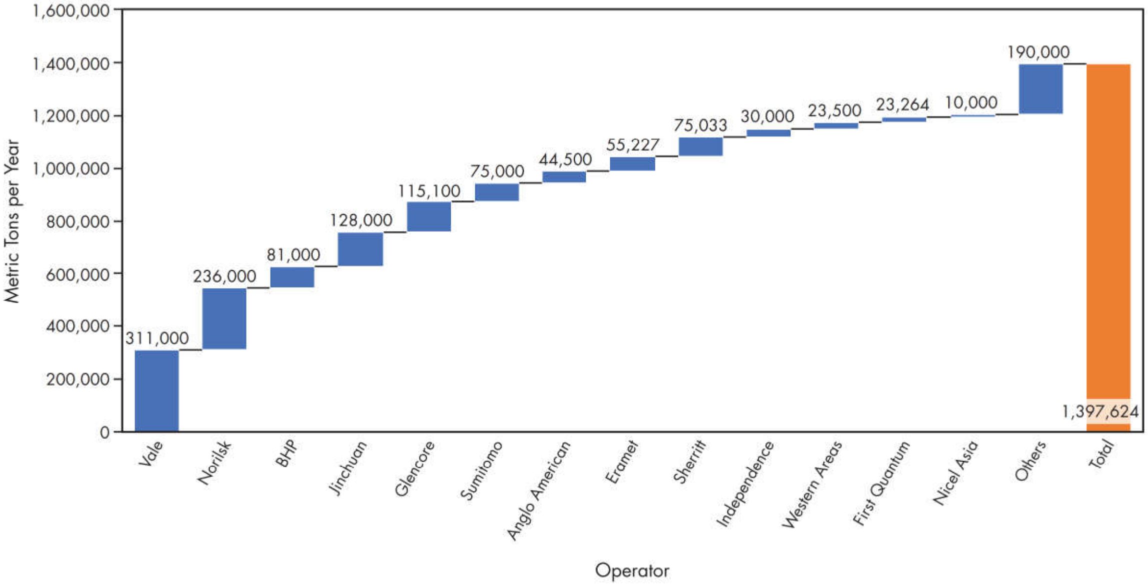
China is currently the world's leading refiner of cobalt and thus the leading supplier of refined cobalt to the United States and Europe. Much of China's production was from ore and partially refined cobalt imported from the DRC, with scrap and stocks of recycled cobalt materials also contributing to production. Large increases in refining capacity over the

Andrew Bamber, Principal and Managing Director, Bara Consulting Ltd., London, United Kingdom
Arthur Barnes, President, Metallurgical Process Consultants Ltd., Kamloops, British Columbia, Canada



Adapted from Padhy 2017

Figure 1 Major nickel-producing countries



Source: USGS 2017

Figure 2 Major nickel producers

past decade mean that China now accounts for more than 40% of the global cobalt output (USGS 2017).

Operating across these major cobalt-producing regions are several major companies, including cobalt-primary producers in the DRC, such as Freeport (with the Tenke Fungurume mine) and Glencore (with Katanga); and operators producing cobalt as a by-product of nickel or copper operations elsewhere, including Sherritt Gordon (Cuba and Madagascar), Jinchuan Non-Ferrous in China, and Queensland Nickel in Australia (Bell 2018). See Figure 5.

The total value of cobalt consumed globally in 2016 was estimated at more than US$850 million. Growth in world refined cobalt consumption is currently expected to exceed supply, driven mainly by strong growth in the rechargeable battery and aerospace industries. In conjunction with a simultaneous decrease in supply related to decreasing nickel production, the global cobalt market was expected to shift from surplus to deficit in 2017. Rising supplies from ramp-ups at mines such as Ambatovy (Madagascar) and a return to full production at Katanga (Congo) are expected to limit the deficit between now and 2020, but much still depends on copper and nickel prices (Darton Commodities 2009).

In 2016, China was the world's leading consumer of cobalt, with nearly 80% of its consumption being used by the rechargeable nickel–metal hydride (NiMH) and lithium-ion battery industry. The United States is the second-largest consumer of cobalt globally with about US$250 million worth of cobalt either produced or imported, followed by Europe. Approximately 45% of the cobalt consumed in the United States was used in superalloys, mainly in aircraft gas turbine

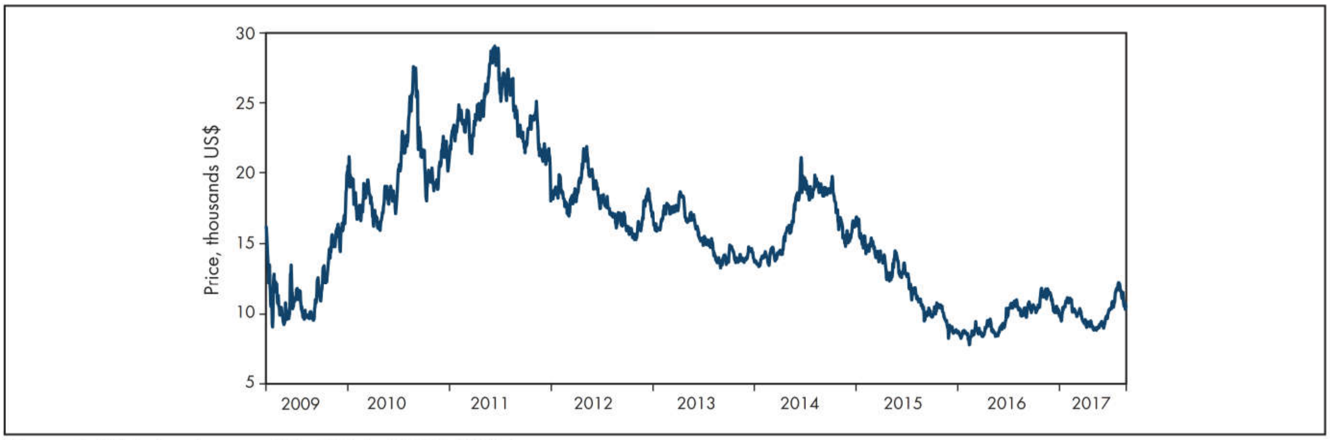
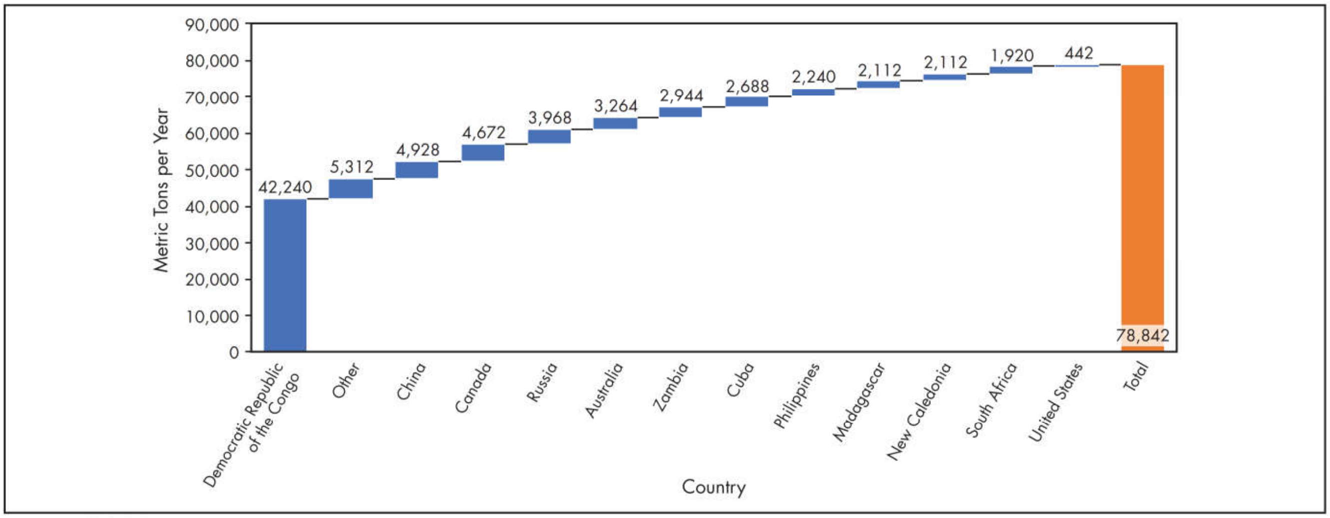
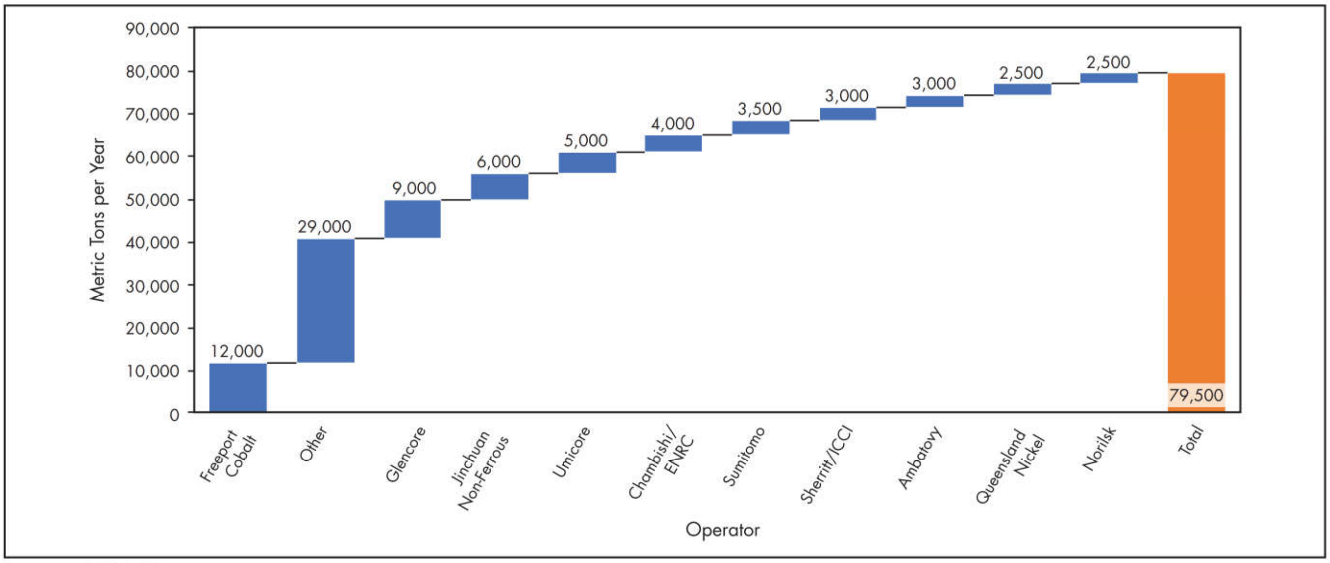


Figure 3 Nickel price trend in US$/t, 2009–2017



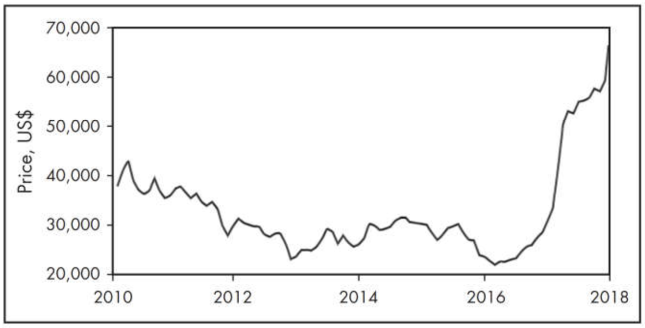
Source: USGS 2017

Figure 4 Cobalt production by country



Source: USGS 2017

Figure 5 Major cobalt producers



Source: Trading Economics 2018
Figure 6 Cobalt price trend

engines; 8% in cemented carbides for cutting and wear-resistant applications; 16% in various other metallic applications; and 31% in a variety of chemical applications, including rechargeable batteries (Darton Commodities 2016).

Cobalt prices have been largely depressed since the global financial crisis of 2008, reaching a low of US$22,100/t in 2013 (Figure 6). Prices have, however, risen strongly since 2016, because of demand growth and a resulting deficit of almost 7,000 t/yr in supply against demand of 90,000 t/yr. Prices are currently at approximately US$70,000/t and forecast to increase through 2020 against projected demand of approximately 120,000 t/yr (Desai 2016). This growth in demand—as with nickel—is largely from cobalt used in battery technology, driven by growth in renewable energy generation capacity and battery-electric vehicle sales. Cobalt in lithium-ion batteries is in the form of either NCA or NMC, where higher cobalt content generally equates to improved battery performance. Demand growth is primarily driven by increases in electric car sales, which are projected to top 17 million units per year by 2030 from a current base of 540,000/yr (Desai 2016).

NICKEL SULFIDE ORES

Geological and Mining Context

Economic deposits of nickel sulfides globally occur in association with either igneous intrusive or volcanic magmatic complexes. Mineralization can be found concentrated near the margins of localized intrusions, as at Stillwater and Duluth in the United States and Vaaralami in Finland; layered basin-type deposits whether tectonic-related, impact-related, or otherwise, as at Norilsk Nickel in Russia and the Sudbury Basin in Ontario, Canada; komatiitic-type deposits as at Nkomati in South Africa, the Kambalda Dome in Western Australia, and Raglan in Quebec, Canada; or, extensive, stratiform concentrations of disseminated metal sulfide mineralization as in the Bushveld Igneous Complex of South Africa and the Great Dyke of Zimbabwe (Foose et al. 1984). Specific geological, mineralogical setting, and related mining engineering aspects of the main subtypes are discussed in ensuing sections.

The Stillwater Ni–Cu–PGE (platinum group element) deposit is located in the Beartooth Mountains, 130 km southwest of Billings, Montana, United States. It is a localized, syngenetic mafic/ultramafic layered complex in which the layering consists bands of norite, gabbro, and anorthosite. Differential settling of the minerals within the massive melt zone allowed heavy minerals to settle to the base with the lighter siliceous minerals remaining at the top. Gangue minerals include olivine, pyroxenes, plagioclase, and chrome spinels with minor quartz, amphibole, apatite, magnetite, and ilmenite (McCallum 1999). Major ore minerals occur in layers and minor veins averaging 2.5-m thick, comprising predominantly pyrrhotite, chalcopyrite, and pentlandite with sperrylite and minor Pt–Fe alloys.

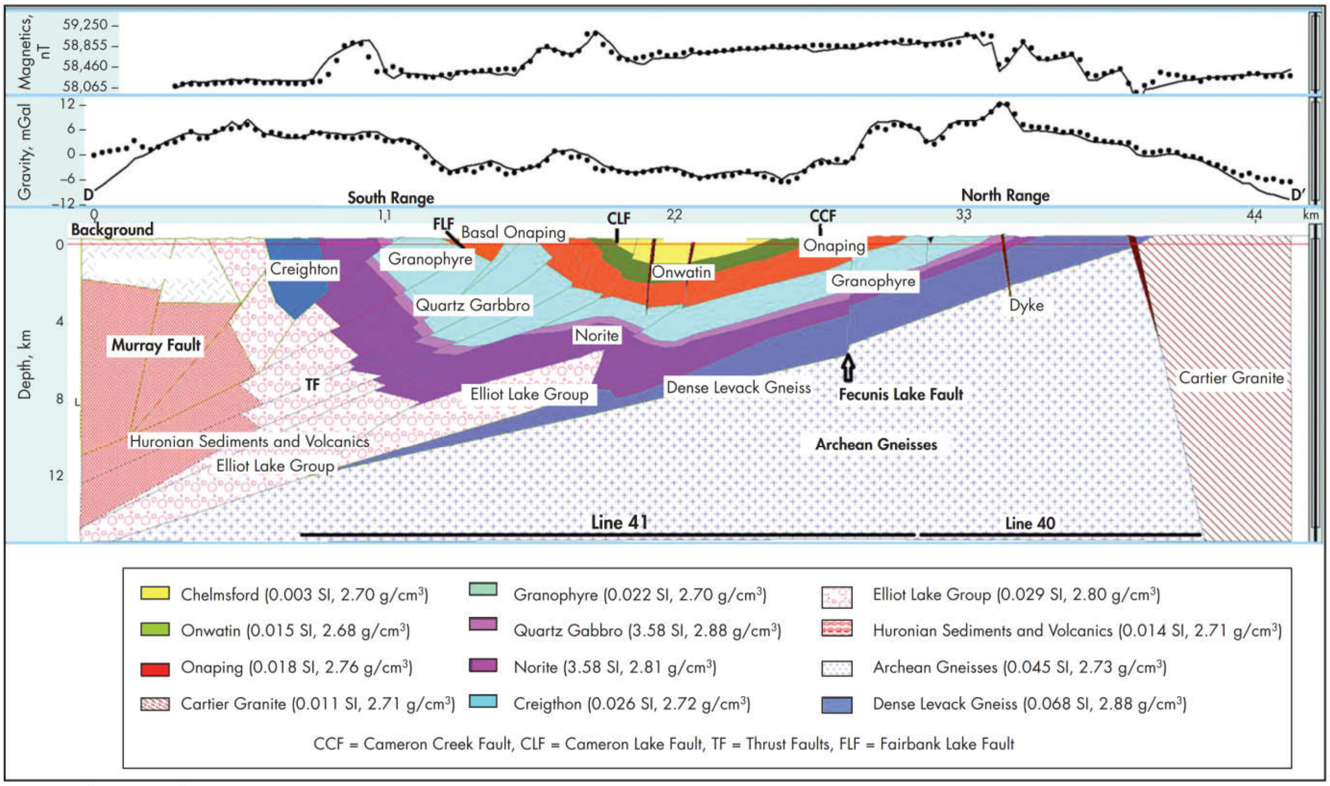
Access to the mineralized zones is by a 600-m shaft and an internal decline. One of three stoping methods are used: mechanized captive cut-and-fill, ramp-and-fill using electric–hydraulic drilling jumbos and load-haul-dumps (LHDs), and sublevel stoping. Blasted ore is recovered from below using LHDs and the stopes are paste-backfilled for stability. Mucked ore is delivered to an underground crushing station and crushed to –150 mm. The ore is then conveyed to holding bins before loading in the 9-t-capacity skips where it is hoisted to the surface for treatment by grinding and flotation processes.

The Ni–Cu ore deposits of Sudbury are associated with a large body of igneous rock known as the Sudbury Igneous Complex (SIC) or Sudbury Basin (Figure 7). Views on the origins of the complex are mixed; although its location at the junction of the Superior, Southern, and Grenville provinces suggests a tectonic origin, meteor impact sites such as the nearby Lake Wanapitei site have led to widely held impact–origin theories for the complex. The SIC is in the form of a truncated ovoid, with the major axis striking northeast for a distance of 60 km and the minor axis approximately 27 km long, fringed generally by a ring of low hills around the contact area. The main rock units of the SIC range from an outer ring of norite through a transition zone of gabbro to an inner zone of granophyre. Footwall rocks include archean gneisses and granites, as well as a large zone of breccias comprising footwall rocks embedded within igneous material, largely norite (Naldrett 1984).

The SIC is rich in nickel, copper, and PGEs, with nickel-bearing mineralization consisting of zones of massive, net-textured, matrix-textured, and disseminated sulfides associated with brecciated host rocks located in footwall troughs or embayments around the outer norites of the complex. Major sulfides include pyrrhotite, pentlandite, chalcopyrite, and pyrite with minor magnetite. Pentlandite is the main nickel-bearing sulfide with a nickel content of 33–35 wt %, and also containing an average of 1% Co. Nickel can also be mineralogically associated within the pyrrhotites (Charland et al. 2011).

A variety of mining methods are deployed, including mechanized cut-and-fill, sublevel caving, and a variation of sublevel caving called vertical crater retreat mining. Using the Coleman mine near Sudbury (Ontario, Canada) as an example, blasted ore would be mucked by LHD (potentially by remote control) and loaded via ore pass to 20-t or 40-t haul trucks and hauled to the shaft station where it would be crushed to nominally –200 mm and conveyed to the loading pocket for hoisting to surface, typically in 14-t skips. Once on surface, because of the extensive nature of the complex, ore would typically be stockpiled and loaded on to railcars (or road trucks) and hauled to a centralized mill—either Clarabelle or Strathcona depending on the source—for processing by grinding and flotation.

Komatiite-hosted Ni–Cu–PGE deposits, on the other hand, tend to comprise trending zones of disseminated through matrix to laminated massive sulfides and stringers in continuous or semicontinuous zones within a predominantly volcanogenic as opposed to igneous complex. Hanging-wall material is commonly barren olivines or pyroxenes, with a footwall



Source: Olaniyan et al. 2014

Figure 7 Cross section of Sudbury Igneous Complex with major geological units

of basalt or similar volcanics (Cowden 1988). Within these trending zones, mineralization styles can vary widely, from net, blebby, and matrix textures to the leopard and "reverse-leopard" textures at Raglan and Voisey's Bay (Labrador, Canada). Although sulfide minerals common to other types of Ni–Cu–PGE deposit, such as pentlandite and chalcopyrite, are the principal ore minerals, weathering, and metamorphic overprinting in some deposits of this type manifest a range of other, more rare minerals such as gaspeite, hellyerite in gossans, and polydymite and violarite in saprolitic zones (Lesher and Keays 2002).

Mining methods are generally underground methods, although minor open pitting has been undertaken at Raglan on account of the shallow, postglaciation situation of that deposit. Jackleg mining of narrow veins with scrapers in steeper zones or small LHDs in shallower zones does occur, although mechanized drift-and-fill, cut-and-fill, and longhole stoping methods are more common. Longwall mining in flat-lying tabular sections can also be found, particularly at Kambalda operations. Ore is typically hoisted or hauled via ramp to surface, and stockpiled prior to treatment via crushing, single-stage semiautogenous grinding (SAG) mill and flotation.

Deposits of the extensive (or elongate) layered intrusive type such as the Bushveld Igneous Complex (the Bushveld) and the Great Dyke of Zimbabwe (the Dyke) differ largely in the wide variety of base metals hosted, from massive and veinous deposits of ferrous minerals such as iron, vanadium, and chrome, to disseminated Ni–Cu–PGE sulfide mineralization associated with massive oxide deposits (e.g., chromite such as the UG2 deposits of the upper Bushveld), flatter lying tabular zones not associated with massive oxides (e.g., the famous Merensky Reef), and extensive amorphous zones of Ni–Cu–PGE mineralization such as the Platreef.

The Great Dyke of Zimbabwe is hosted in archean granitoids, with several magma subchambers occurring along strike of the deposit, and each subchamber hosting disseminated Ni–Cu–PGE mineralization of varying tenors within the ultramafic norites and pyroxenites (Figure 8). Economic deposits tend to be emplaced under or alongside shallow-lying host greenstones. In cross section the Dyke is keel-shaped with layers occurring in several sequences; the main sulfide zone occurs variably at the boundary between the mafics and the ultramafics from the top of the plagioclase and into the base of the websterite, approximately 200–400 m deep (Mwatahwa and Musa 2015).

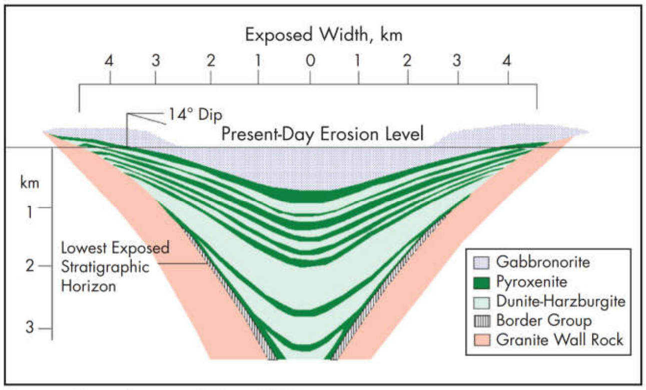
Mineralization is predominantly pyrrhotite, pentlandite (1%–2%), chalcopyrite, and pyrite, with PGEs occurring at parts-per-million levels as sperrylite, native platinum associated with base metal sulfides, and minor bismuthotellurides. Gangue mineralogy comprises pyroxenes with plagioclase, chlorite, tremolite, and often high levels of talc.

Mining methods are typically mechanized room-and-pillar, with access to the workings by ramp, with haulage by 20–40-t truck, or alternately conveyed to surface. Processing is by crushing, screening, and primary and secondary ball milling with two stages of flotation on account of the typical fine grain size of the sulfides.

The Bushveld is the world's largest layered intrusion, comprising the eastern, western, and northern limbs some 300 km long, 500 km across, and 7–10 km thick (Schoustra et al. 2000). After regional intrusion, the rocks cooled slowly and fractionation occurred according to complex thermodynamics, physics, and chemistry, resulting in light, magnesium silicate–rich minerals in the upper zones; iron-rich phases below; and more dense chromite, nickel, copper, and associated platinum group metals (PGMs) in deeper, so-called critical zones (Figure 9). The Upper Critical Zone, ranging from 200–300 m to more than 1,500 m deep in places, is host to one of the world's largest Ni–Cu resources and the largest known PGE resource in the world. It hosts the UG2 (Ni–Cu–PGE associated with primary chromitite), Merensky (tabular Ni–Cu–PGE), and Platreef (extensive, amorphous Ni–Cu–PGE) economic mineralized zones.

Ore minerals within these zones include chromite, pyrite, pentlandite, chalcopyrite, pyrrhotite, and sperrylite, with gangue minerals including pyroxene, feldspar, mica, chlorite, serpentines, spinels, and clays. Grain sizes tend to be fine, requiring at least secondary grinding, with some grain sizes noted as fine as –10 µm, requiring tertiary grinding and, more recently, regrinding with stirred mills or other fine grinding technology.

Mining methods include jackleg or airleg, and mechanized low-profile methods in deep sections greater than, for example, 1,000 m requiring shaft access, and mechanized room-and-pillar in shallower sections, with access to the workings by ramp and haulage by 20–40-t truck or conveyor to surface. Processing is by crushing, screening, and primary and secondary ball milling with two or three stages of flotation on account of the increasingly fine grain size of economic sulfides.

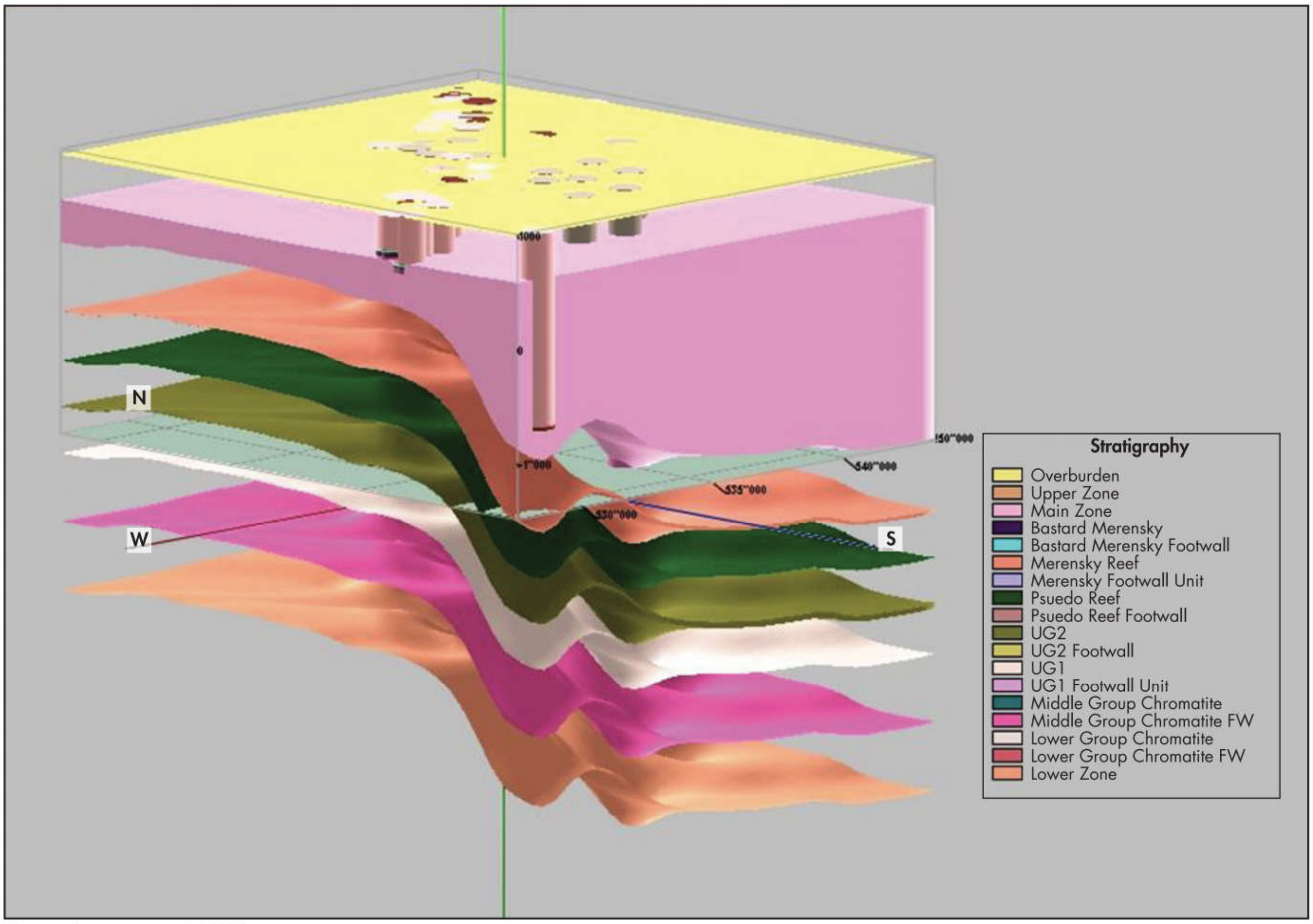


Source: Mwatahwa and Musa 2015

Figure 8 Cross section of the Great Dyke

Mineral Processing

The low metal content of present-day sulfide ores renders them unsuitable for either direct smelting or direct hydrometallurgical processing. Because the sulfide minerals usually occur as distinct grains in the rock matrix, these ores are amenable to a



Source: Bamisaiye et al. 2016

Figure 9 Cross section through the western limb of the Bushveld Igneous Complex, near Amandelbult, South Africa, showing ideal sequence

mechanical upgrading in which much of the rock content can be rejected. Beneficiation of nickel sulfide ores is therefore generally by grinding and flotation; however, there are many different flow-sheet styles within this general class, as mineralogy (and mineral associations), texture, grain size, and hardness vary enormously from deposit to deposit.

In principle, it is possible to produce separate nickel (pentlandite), copper (chalcopyrite), and iron (pyrrhotite) concentrates. However, a clean separation of pentlandite from pyrrhotite is difficult in practice since pyrrhotite typically contains intergrown inclusions of pentlandite as well as nickel in solid solution in the pyrrhotite. In fact, pyrrhotite often contains 0.5%–1% Ni that cannot be separated by physical methods. The relatively high nickel content in pyrrhotite makes it difficult to achieve high yields of nickel from the ore unless the pyrrhotite is processed to recover its nickel content; conversely, the presence of nickel makes it difficult to obtain a marketable iron product from the pyrrhotite.

Most nickel producers outside Canada make no attempt to obtain separate nickel, copper, and iron concentrates, preferring a bulk concentrate that contains pentlandite, pyrrhotite, and chalcopyrite as smelter feed. In Canada, both the major producers separate a large part of the pyrrhotite from pentlandite and chalcopyrite, and Vale further separates the pentlandite and chalcopyrite into separate nickel and copper concentrates at its Sudbury operations. Even where the bulk of the pyrrhotite can be separated from pentlandite, residual pyrrhotite in the nickel concentrate still contributes a major part of the sulfide content.

Pyrrhotite can be separated from pentlandite and chalcopyrite either by using its ferromagnetic properties or by flotation. Not all pyrrhotite is ferromagnetic, although in ores from the Sudbury, Ontario, area the magnetic monoclinic form predominates over the paramagnetic or “nonmagnetic” hexagonal form (Toguri 1975). In general, pyrrhotite containing less sulfur than Fe0.87S is nonmagnetic. This material can usually be separated from pentlandite by flotation.

Nickel or Ni–Cu concentrates typically range in grade from 5% to 15% Ni + Cu, depending on the degree of pyrrhotite rejection achieved. An exceptionally high-grade concentrate, containing 28% Ni and consisting largely of pentlandite, is produced in limited amounts at Vale’s Thompson, Manitoba, Canada, operation.

One of the more complex nickel concentration process flow sheets is that operated by Vale (previously Inco) at its Sudbury operation. In the two primary concentrators, the Clarabelle mill and the Frood-Stobie mill, 50,000 t/d of ore, grading 1.2% Ni and 1.2% Cu, is ground to a particle size of approximately 200 μm and treated by magnetic separation and flotation with a sodium amyl xanthate reagent to produce 5,000 t/d of a bulk Ni–Cu concentrate and 8,000 t/d of a pyrrhotite concentrate.

The two primary concentrates are further upgraded in the Copper Cliff mill in Ontario, Canada. The bulk Ni–Cu concentrate is separated by flotation into separate nickel and copper concentrates by using lime and sodium cyanide at 30°–35°C to depress pentlandite and pyrrhotite. Under these conditions, about 92% of the copper is recovered to a concentrate grading 29% Cu and 0.9% Ni. About 98% of the nickel and 92% of the pyrrhotite are rejected to the separator tails, which grade 13% Ni and 2.0% Cu.

The Clarabelle rougher pyrrhotite concentrate is treated to separate residual chalcopyrite and pentlandite by grinding to approximately 74 μm followed by magnetic separation and flotation. A magnetic, cleaner pyrrhotite concentrate, a bulk Ni–Cu concentrate, and a nonmagnetic pyrrhotite tail are produced. The nonmagnetic tails stream (1% Cu, 2% Ni, and 30% pyrrhotite) is further treated by regrinding and flotation in the pyrrhotite rejection circuit, for final recovery of pentlandite and chalcopyrite before it is discharged to the tailings pond. Overall, about 80% of the pyrrhotite content of the ore is rejected from smelter feed to the pyrrhotite concentrate and tails.

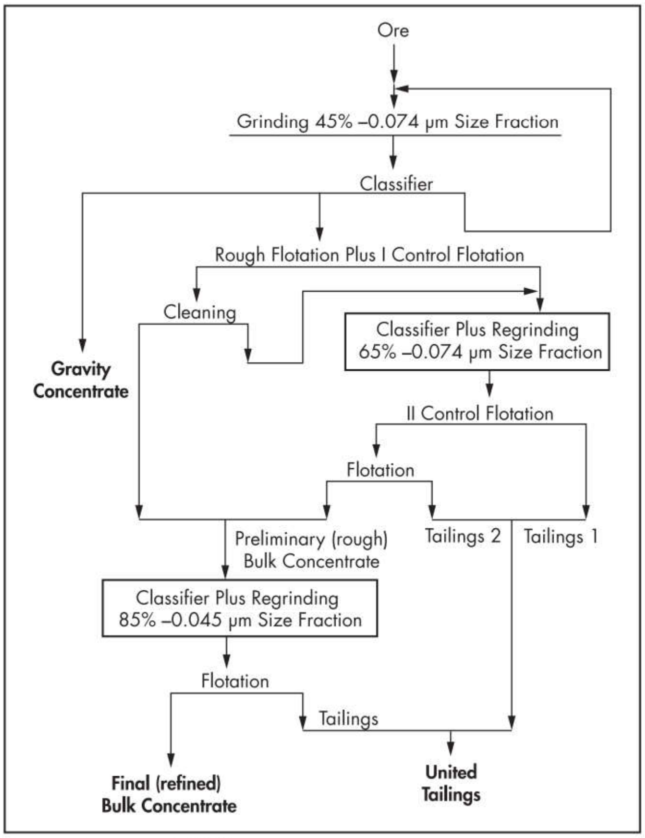
At its Strathcona mill in the Sudbury Basin, Glencore (previously Falconbridge) treats ore from both the western portion of the basin, containing 1.5% Ni and 1.1% Cu, and high-grade ore from the Nickel Rim South mine on the northeast side of the Sudbury basin, and provides toll milling of numerous ores from other mines. Crushing, grinding, and a flexible flotation flow sheet are used to produce a Ni–Cu concentrate grading around 15%–20% Ni + Cu at an average recovery of 83% Ni and 93% Cu (Hoffman and Kaiura 1985). A large part of the pyrrhotite content of the ore is rejected directly to the tailings pond for environmental (sulfur emission control) reasons. The separation of the nickel and copper in the bulk concentrate is carried out in the matte refining process after smelting.

The ore mined by Vale (Inco) at Thompson consists of pentlandite and pyrrhotite with only a small amount of chalcopyrite (2.7% Ni, 0.2% Cu, 22% pyrrhotite). Since the pyrrhotite contains 78% of the sulfur but only 5.5% of the nickel, a high degree of pyrrhotite rejection is feasible without seriously decreasing the nickel recovery. The ore is ground and floated to recover a Ni–Cu concentrate and reject 90% of the gangue and 60% of the pyrrhotite to tails. The Ni–Cu concentrate is first treated by flotation to separate a copper concentrate and then treated by magnetic separation and flotation to recover a high-grade (28% Ni) pentlandite concentrate. The tailings stream from these steps forms the main nickel concentrate smelter feed (11% Ni, 0.4% Cu). The overall recovery of nickel to the nickel and pentlandite concentrates is about 87%, whereas another 5% of the nickel reports to the copper concentrate. More than 60% of the pyrrhotite is rejected to tailings with only 8% of the nickel.

At the BCL Ni–Cu mine in Botswana, the pyrrhotite/pentlandite ratio in the ore averages about 13:1, and the pyrrhotite contains more than 30% of the nickel either in solid solution or as small pentlandite inclusions. Consequently, high nickel recoveries can only be achieved by recovering the nickel from pyrrhotite. This is achieved by producing a bulk pentlandite–pyrrhotite–chalcopyrite concentrate, with a typical composition of 2.8% Ni and 3.2% Cu (Eliott et al. 1983).

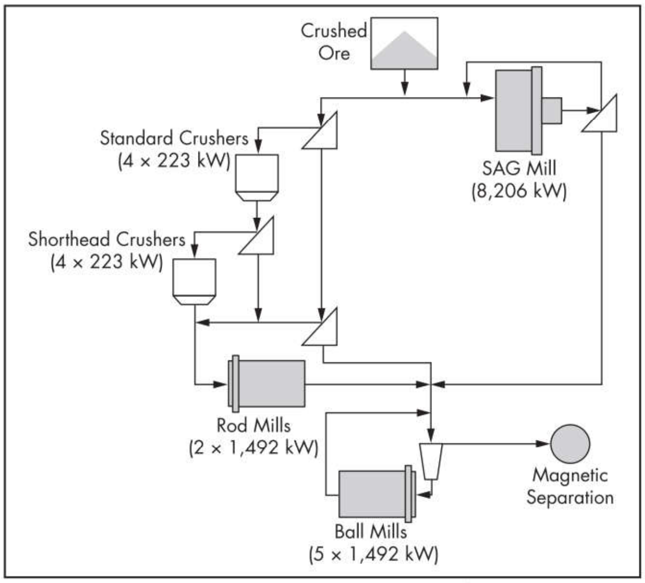
The Norilsk flow sheet in Figure 10 exemplifies the treatment of disseminated pentlandite ore with chalcopyrite comprising crushing followed by grinding in closed circuit with a hydrocyclone to typically 150 μm d_{80}, followed by rougher flotation and two stages of cleaner flotation producing a Ni–Fe–S concentrate at 12% Ni and approximately 80% recovery (Kozyrev et al. 2002).

In the Sudbury context, flow sheets are required to deal with highly variable mixtures of polymetallic ores from various sources, including custom feed, and thus generally accommodate Ni–Cu separation, pyrrhotite rejection, and the generation of nickel primary concentrates plus copper or copper-primary concentrates with precious metal by-products. Primary crushing would typically be done underground, with a –300 mm nominal run-of-mine (ROM) product delivered by



Source: Kozyrev et al. 2002

Figure 10 Grinding and flotation of disseminated nickel ores at Norilsk



Source: Lawson and Xu 2011, reprinted with permission from the Australasian Institute of Mining and Metallurgy

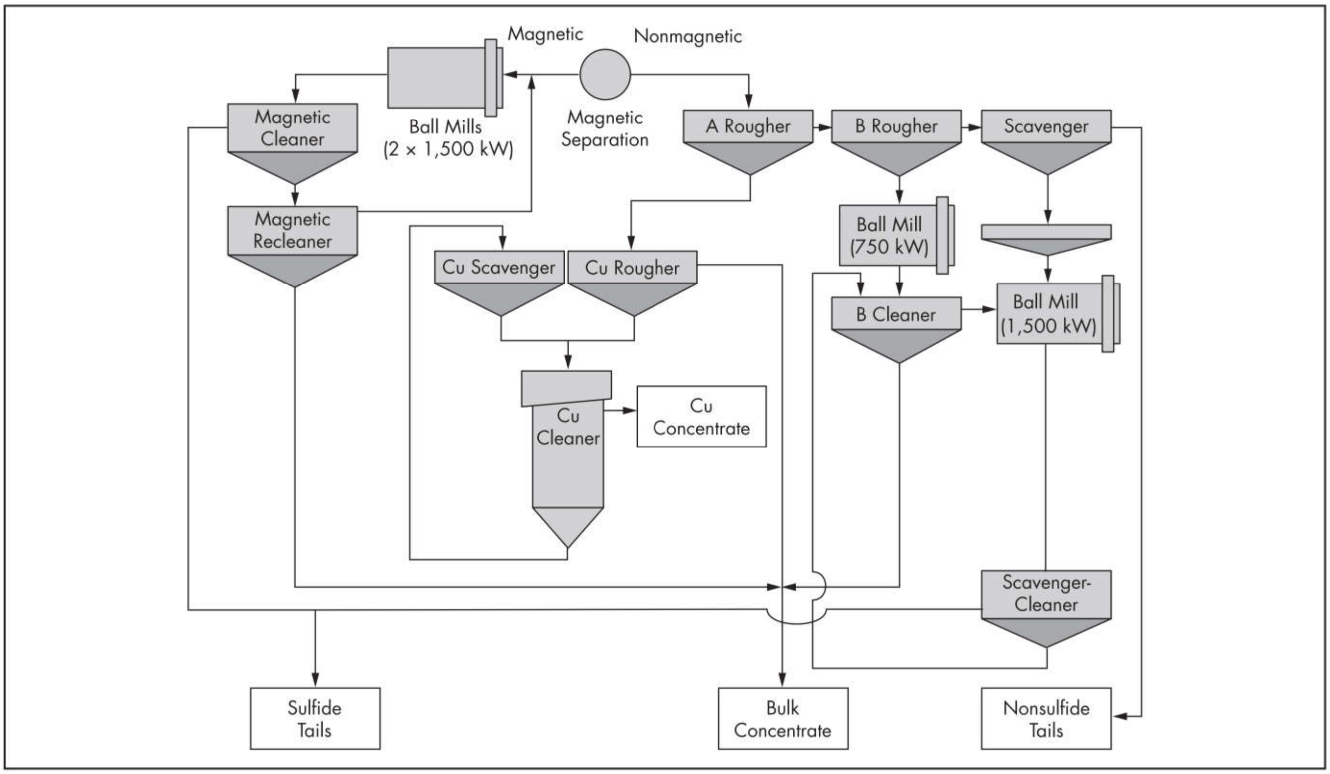
Figure 11 Clarabelle Mill flow sheet, comminution, and feed preparation

rail to the central mill feed stockpile. Grinding is by SAG/ball/rod mill in combination, followed by magnetic separation at nominally 104 µm on wet high-intensity magnetic separators to separate a predominantly pentlandite- and pyrrhotite-rich stream from a Cu–Ni-rich stream prior to flotation (Figure 11).

The nickel circuit is tuned for pentlandite recovery with depression of the iron-rich pyrrhotite in rougher, cleaner, and scavenger stages, with regrind of the primary concentrate to 38 µm prior to cleaner and scavenger stages (Figure 12). The Cu–Ni circuit is tuned for sequential separation of chalcopyrite and bornite first, followed by pentlandite and, where present, millerite. Combined tailings are thickened and pumped to the tailings dam for deposition and water recovery. Nickel-primary concentrates are combined at around 12% Ni and 7% Cu and delivered to the smelter for processing via fluidized-bed drying followed by flash smelting.

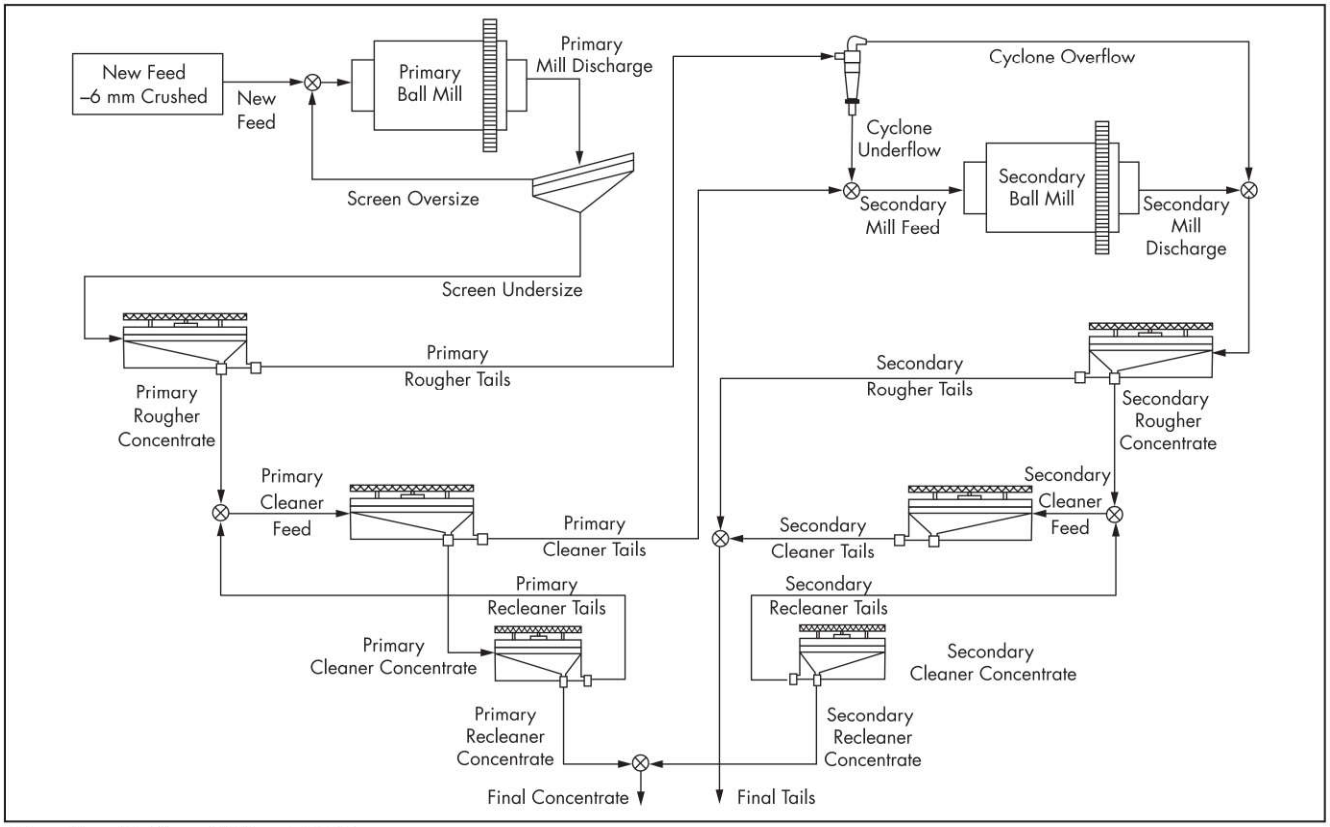
In both the Bushveld and Dyke contexts, ores are lower in grade with respect to both nickel and copper than either their Sudbury or Norilsk counterparts, and finer grained; therefore, flow sheets focus on maximizing concentrate grade as well as recovery within the constraints of the metallurgy of the feed (Figure 13). In these cases the ROM might grade 1% Ni and 1% Cu with perhaps 3 g/t combined platinum, palladium, and gold. Here, so-called MF2 flow sheets are common, particularly on Merensky and Platreef ores with secondary grinding of the rougher tailings to increase liberation and, increasingly, regrinding of the concentrates. Crushing might be done in three stages, scalping of the oversize ROM and crushing in open circuit, followed by secondary crushing in closed circuit with a high-capacity screen, delivering nominally 100% –13 mm to the primary ball mill via a feed silo. Grinding to nominally 120 µm is by ball milling in closed circuit with a cyclone cluster, followed by rougher flotation to a bulk concentrate, with secondary grinding of the rougher tailings to nominally 75 µm and scavenger flotation of the reground ore. Combined concentrates are sent to multiple stages of cleaner flotation, with regrinding of the primary cleaner tails prior to secondary and often ternary cleaning stages to a final concentrate, which is thickened, filtered, and either bagged or tanked and trucked to a centralized smelter for reduction and matte separation.

Several reagents are key in the nickel flotation process. Collectors attach to the sulfide surfaces so the sulfides can more easily attach to air bubbles in the flotation cell. Common collectors include oils and any of the family of xanthates (e.g., potassium amyl xanthate or sodium isobutyl xanthate). Activation of the sulfide surfaces—for example, by copper sulfate ($CuSO_4$), to allow the collector to adsorb onto the pentlandite—may also be required; therefore, reactors for this are often included post–primary grinding ahead of rougher flotation. The most common frothers, used to establish stable air bubbles in the cell, include pine oils, alcohols (e.g., methyl isobutyl carbinol), or glycols. The occurrence of high levels of magnesium is common in these highly ultramafic ores, and depression of talc by carboxy methyl cellulose or soda ash during flotation is also required to achieve desired grades and optimize mass pull to concentrate. pH adjustment by either lime or the addition of acid can also be required, especially in sequential flotation of copper and nickel where chalcopyrite is ideally collected first, followed by pentlandite, requiring a staged adjustment of the acidity of the slurry. Nickel recovery might be 75%–80% to a bulk concentrate grading 7% Ni, 8% Cu, and 200 g/t PGEs.



Source: Lawson and Xu 2011, reprinted with permission from the Australasian Institute of Mining and Metallurgy

Figure 12 Clarabelle Mill flow sheet showing Ni and Cu rougher, cleaner, and scavenger circuits



Adapted from Merkle and McKenzie 2002

Figure 13 Typical Bushveld/Dyke MF2 flow sheet with regrind

Source: Bhotoko 2016

Figure 14 Preconcentration of low-grade nickel ores at the Tati nickel mine

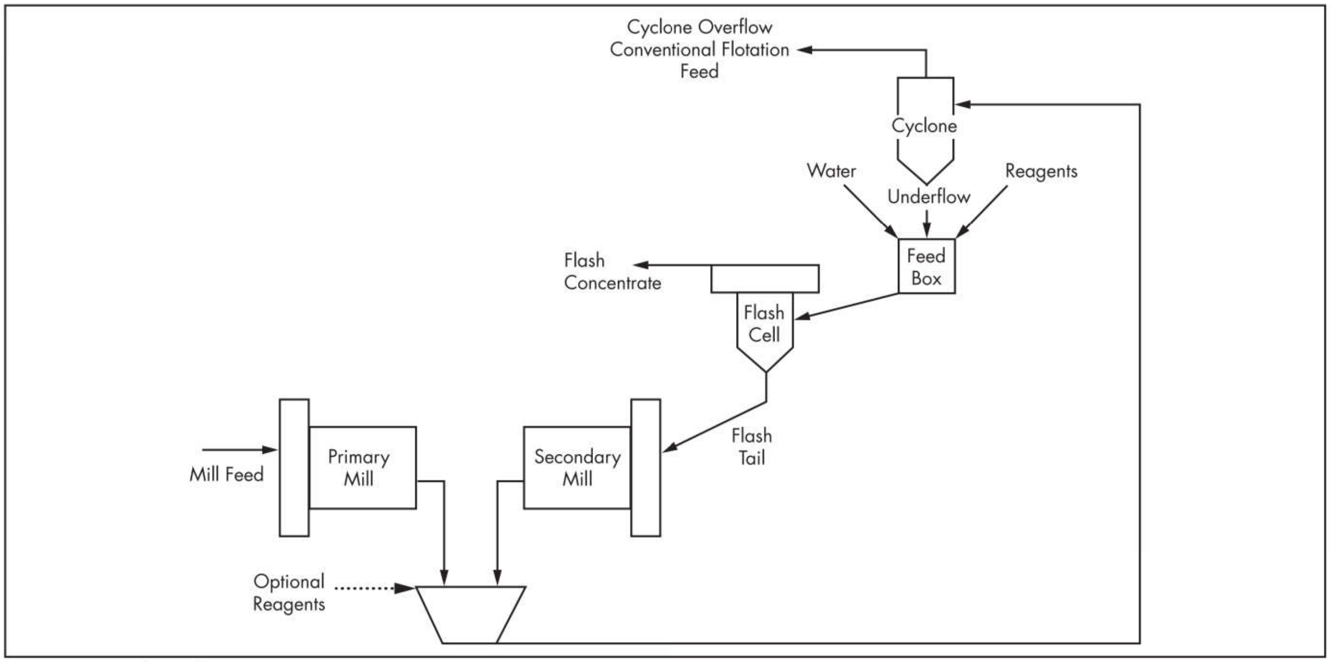
Courtesy of Halyard

Figure 15 Chromite removal by heavy mineral spirals: (top right) flotation circuit and (bottom right) spiral circuit

Variations to this flow sheet occur where textures might tend toward massive sulfides but with high dilution to a lower grade (e.g., at the Tati nickel mine in Botswana; see Figure 14), or in the case of the UG2 ores where the ore zones are narrow, the nickel and other metals are associated with chromite, and ROM can be highly diluted with hanging-wall pyroxenites. In the case of low head grades and/or high dilution, preconcentration of ore, by either size classification of coarse, low-grade material, dense media separation, or—increasingly—sensor-based sorting of the ore ahead of grinding and flotation, is common. Here, dense media separation or sorting identifies and rejects the barren or low-grade material at coarse particle size, delivering a reduced mass of higher grade ore into the rougher flotation stage. Additionally, for UG2 ores, where the nickel is associated with massive chromites, gravity separation in large, complex heavy mineral spiral stages after primary grinding is undertaken, which rejects the coarse, heavy chromite and recovers the less dense sulfides ahead of rougher flotation, allowing for a reduced size of flotation circuit and improved response in flotation compared to that of a chromium-rich feed (Figure 15). Further variations to this circuit might be observed where a flash flotation stage might be included, treating the underflow of the primary cyclone to recover coarse precious metals and metal sulfides (Figure 16).

Pyrometallurgy and Hydrometallurgy

Pentlandite is the predominant nickel-bearing mineral in sulfide concentrates and is commonly associated with various iron and copper sulfide minerals including pyrite, pyrrhotite, and chalcopyrite. Millerite and awaruite are less common minerals and require specific flow sheets for concentration. Historically, high-grade ores (in terms of sulfur content), such as those found in the Sudbury area in the early 20th century, were often roasted directly to oxidize the majority of the iron, and then smelted and converted to produce "white matte" Cu–Ni sulfide. Refer to Boldt and Queneau (1967) for details on



Source: Newcombe et al. 2012

Figure 16 Flash flotation of ball mill cyclone underflow

these methods, now discontinued for obvious environmental reasons. More generically, extraction of nickel from modest grade sulfide ores evolved as a variant of copper extraction: upgrading (beneficiation) of the ores used crushing and grinding followed by froth flotation, initially producing a bulk Fe–Ni–Cu sulfide concentrate. Later improvements permitted production of high-grade copper concentrates based on the better response of chalcopyrite to flotation reagents compared to the more sluggish nickel minerals. Many current circuits include specific pyrrhotite rejection circuits that result in a much higher nickel content in the concentrate and, consequently, a much lower mass of slag generated in subsequent smelting operations.

Concentrates are typically dried, roasted, and smelted or flash smelted (where the roasting heat itself provides the energy for melting the feed), producing a matte (liquid sulfide) containing iron, nickel, copper, cobalt, and sulfur and a discard slag consisting of mainly iron silicates (fayalite) and iron oxides (magnetite). The remaining iron is usually removed by oxidation of the molten matte by converting, usually in a cylindrical (Peirce–Smith) converter. In contrast to copper extraction, converting ceases at or close to the "white metal" stage with less than 4% Fe remaining in the matte. Further oxidation results in cobalt and nickel oxidizing and entering the slag. The white matte is cooled into either flat molds or granulated and becomes feedstock for the refining stages.

Numerous permutations exist for refining, largely determined by the final product desired and market demand, but all result in very high-purity nickel, copper, and cobalt metal (or oxide). In many instances substantial revenue is generated by the recovery of precious metal (Au, Pt, Pd, and Rh) as well. Major evolutionary changes in the extraction process have been driven by environmental concerns, and sulfur capture levels of >97% would be considered the minimum acceptable target for most current operations, in stark contrast to 100 or even 50 years ago when almost all of the sulfur was emitted to atmosphere. Sulfuric acid plants are an indispensable part of nickel sulfide processing, responsible for capturing the major portion of gaseous sulfur emissions. Nickel extraction has a high overall carbon footprint, not easily reduced because most of the processing is of low-grade ores (Bakker et al. 2011).

Pyrometallurgy of Nickel Extraction

More than 90% of the world's nickel sulfide concentrates are treated by pyrometallurgical processes to form nickel-containing mattes. Production capacities and composition of the feed and product for several major nickel smelters are listed in Table 1 (Crundwell et al. 2011).

Table 1 Feed composition, product composition, and production at four major nickel smelters

	Smelter			
	Kalgoorlie (BHP)	Bindura (BCL)	Nadezda (Norilsk)	Gansu (Jinchuan)
Start-up date	1972	1973	1981	1992
Ni in matte, t/yr	100,000	27,000	140,000	65,000
Feed, %				
Ni	15	5	12	9
Cu	0.3	4	5	4
Fe	34	43	38	38
S	32	31	27	27
Product, %				
Ni	47	17	32	29
Cu	1.5	15	15	15
Fe	20	33	23	29
S	27	25	27	23

Source: Crundwell et al. 2011

The pyrometallurgical treatment of nickel concentrates includes three types of unit operation: roasting, smelting, and converting. In the roasting step, sulfur is driven off as sulfur dioxide (SO_2) and part of the iron is oxidized. In smelting, the roaster product is melted with a siliceous flux, which combines with the oxidized iron to produce two immiscible phases: a liquid silicate slag that can be discarded and a solution of molten sulfides that contains the metal values. In converting, iron sulfide is removed from the molten, low-grade furnace matte by intense oxidation and slagging.

Roasting. Roasting is a process in which the nickel sulfide concentrates are heated in an oxygen-containing gas, usually air, to a temperature (600°–700°C) at which oxygen oxidizes sulfide to SO_2 and reacts with the metal values to form solid oxides, termed the *calcine*. The roasting step may also serve to preheat the charge for smelting.

The thermodynamic relationships between the metal sulfides and oxides provide the basis for the separation of iron from nickel, copper, and cobalt contained in the concentrate. The sulfides of iron, nickel, cobalt, and copper are all in the same thermal stability range at the smelting temperatures (1,200°–1,300°C), but, in the presence of oxygen, each sulfide is unstable with respect to its corresponding oxide. Iron has the greatest affinity for oxygen, followed by cobalt, nickel, and copper. Consequently, if the nickel concentrate is roasted with a deficiency of oxygen, the iron is oxidized preferentially, while virtually all of the nickel and copper remain as sulfides. Thus the degree of oxidation of the charge and the degree of elimination of sulfur can be controlled by regulating the supply of air to the roaster (or flash furnace). The degree of sulfur elimination achieved in the roasting step largely determines the grade of the matte produced as well as the losses of nickel and copper to the slag phase in the subsequent smelting operation. All the sulfur present in the calcine in excess of the amount needed to combine with the nickel, copper, and cobalt in the matte combines with iron, which therefore remains in the matte phase and lowers its pay metal (Ni + Cu + Co) grade. The major constraint on achieving a high-matte grade in smelting is the increasing loss of nickel, copper, and cobalt to the slag phase.

Recent modifications to the Glencore Sudbury operation use a much higher degree of roast (>70% S elimination) to produce a sulfur-deficient matte during electric furnace smelting. This is more fully described by Warner et al. (2006, 2007).

Smelting. When a mixture of metal sulfides, iron oxide, gangue, and siliceous flux are melted together, the iron oxide, gangue, and silica form a slag layer that floats on the heavier, molten sulfide (matte) phase. The two immiscible layers are separated and the slag is discarded.

The smelting process for nickel sulfide concentrates is normally carried out in two stages: primary smelting and converting. The oxidizing, slagging, and removal of iron in stages is critical to the efficiency and economics of smelting operations. If most of the iron were to be oxidized in the roaster and then slagged in a single smelting step, a matte high in nickel and low in iron would be produced. However, significant amounts of the nickel and copper sulfides would also be oxidized in the roaster, and these oxides would be lost to the slag in the smelting step. More extensive detail on smelting is provided by Diaz et al. (1988).

Flash smelting. In flash smelting, a finely ground sulfide concentrate is smelted by burning some of its sulfur and iron content while the sulfide particles are suspended in the oxidizing atmosphere. The concentrate is roasted and smelted in a single process step. The pre-dried concentrate and flux materials are injected with preheated oxygen-enriched air or commercial oxygen into the reaction shaft of a specially designed furnace. The exothermic heat of reaction of the iron sulfide with oxygen provides the energy to heat the particles to smelting temperature. The smelting temperature and the grade of matte formed are both controlled by adjusting the ratio of the amount of oxygen supplied to the furnace to the throughput rate of concentrate. More specific details on the evolution of the process are provided by Diaz et al. (1988).

Converting. In the converting step, iron sulfide is removed from the molten, low-grade furnace matte by oxidation and slagging. The slag, which contains high levels of nickel and copper, is in some instances returned to the primary smelting furnace for recovery of metal values. The high-grade, low-iron Ni–Cu matte, which typically contains 20% S, less than 1% Fe, and also any precious metals present in the original concentrates, is the final product of the smelting process.

Converting is a batch operation. The horizontal side blown converter, known as the Peirce–Smith converter in copper smelting, is normally used for the treatment of nickel furnace mattes, although top-blown rotary converters are also used in some plants, particularly for low production rates. Air or oxygen-enriched air is blown through the molten matte to form iron oxides and remove sulfur as SO_2. The iron oxides combine with added silica flux to form an iron silicate slag:

$$2FeS + 3O_2 = 2FeO + 2SO_2 \text{ and}$$
$$2FeO + SiO_2 = 2\ FeSiO_4$$

A substantial amount of magnetite is also formed in the conversion process.

Converter slags contain 20%–35% silica, with the balance being iron oxides. Silica in the slag combines with Fe(II) oxide, thus preventing further oxidation to magnetite. Slags low in silica contain large amounts of magnetite. Typically, up to one-third of the iron may be in the Fe(III) state in a 20% silica slag, whereas a slag with 35% silica contains less than 15% magnetite. Magnetite formation is useful in converting since it can be made to coat the inside of the brick-lined furnace and thus protects it from erosion and corrosion by the slag (Wang et al. 1974).

The oxidation of iron sulfide is strongly exothermic, and much of the heat generated in the conversion process can be used to melt additional ore or concentrate feed or recycled scrap materials. Since the throughput of a converter is almost directly proportional to the amount of oxygen blown through the charge, its capacity can be raised substantially by enriching the air with oxygen. In addition to increasing the converting rate, oxygen enrichment of the air permits the treatment of additional cold charge, and in fact, the addition of cold charge is essential to control the bath temperature when oxygenenriched air is used. More recent developments include the use of shrouded tuyeres to improve refractory life of tuyere blocks that limit campaign life (Kapusta et al. 2012). Engineering of hoods has resulted in improved sulfur capture and cleaner converter aisles.

Electric smelting. The energy required for smelting may be obtained from electrical power as an alternative to burning fossil fuels or sulfides in air or oxygen. The use of an electric furnace for nickel sulfide smelting is favored when the cost of electrical energy is low or when the required smelting temperature is high (e.g., when the concentrate contains high levels of magnesia).

Heating of the bath is achieved by passing a three-phase electrical current through a circuit consisting of carbon electrodes immersed in the slag layer, which has a high electrical resistance (submerged arc technique). Electric furnaces used for Ni–Cu matte smelting are either rectangular with six electrodes in line or circular with three electrodes in triangular configuration. The consumable electrodes are made of carbon and are either prebaked or of the self-baking Soderberg type.

The charge of concentrate (or calcine) and flux is fed through the roof. Thus a layer of unsmelted charge covers the slag, giving it a "cold top." The concentrate or calcine gradually settles into the slag as it melts and then separates into slag and matte layers. The slag and matte are tapped intermittently, as required. Since there is no fuel combustion in an electric furnace, the quantity of off-gas is much less than from a reverberatory furnace, and therefore heat and dust recovery from the off-gas is therefore easier.

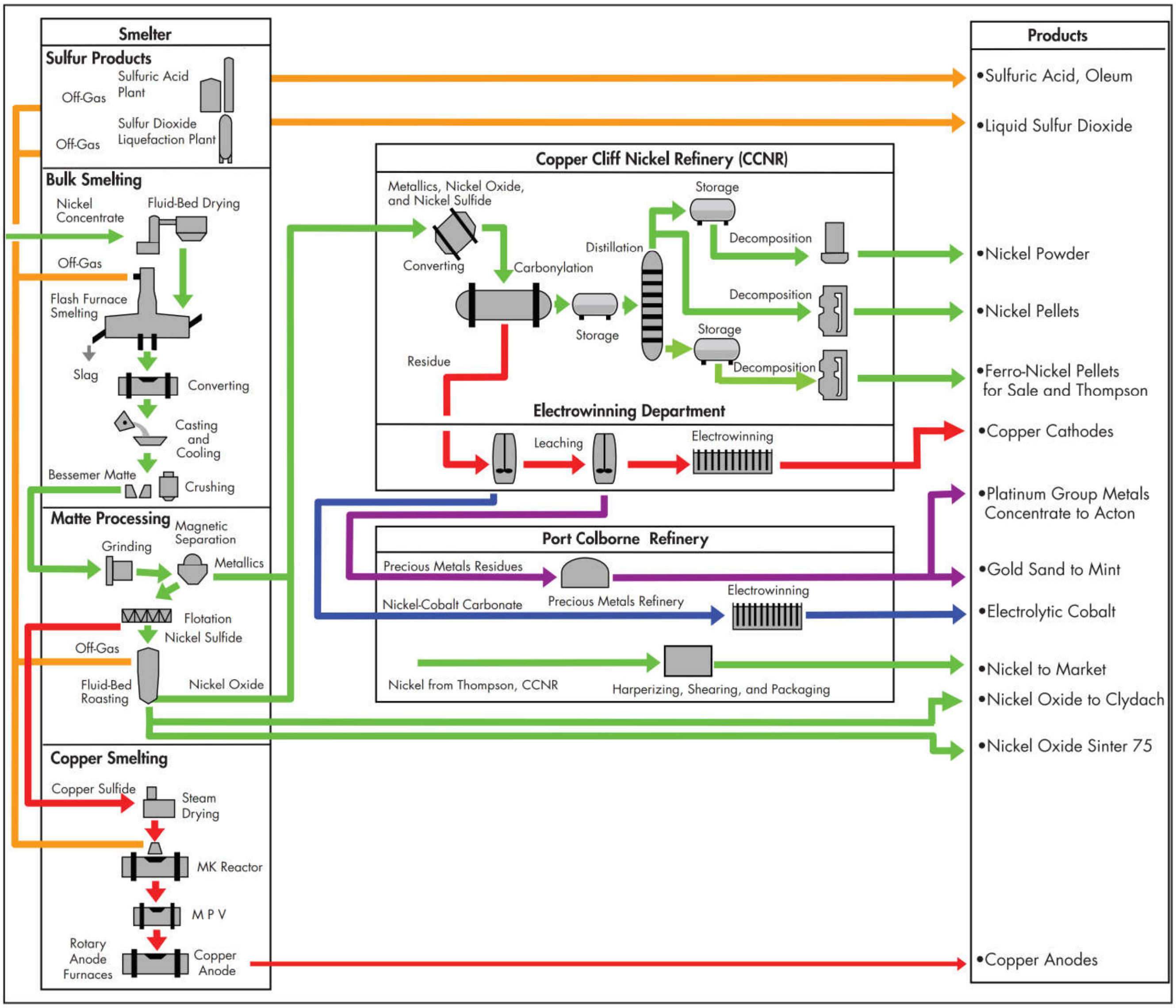
Electric furnace smelting of nickel concentrates is practiced by Vale (Inco) at its Thompson, Manitoba, operation; by Glencore (Falconbridge) at Sudbury, Ontario; by the Severonikel Combine in Russia; by Anglo (Rustenburg) Platinum, Impala Platinum, and Lonmin (Western Platinum) in the Republic of South Africa; and by Bindura Nickel in Zimbabwe. The Canadian operations treat partly roasted concentrate calcines, whereas Bindura and the platinum producers treat Ni–Cu concentrates directly. The Severonikel electric furnace smelter treats high-grade ore (3.5% Ni, 3.5% Cu) from Norilsk directly.

Integrated flow sheets. The Vale Ontario operations flow sheet depicted in Figure 17 is an exemplar of an advanced nickel pyrometallurgical flow sheet with refining (Vale Canada Ltd. 2010).

Nickel concentrate from a variety of sources, including custom feeds, is treated at Copper Cliff through a combination of flash smelting, converting, matte preparation, and reconverting, followed by carbonylation and precipitation into a range of products including ferronickel, nickel pellets, and nickel powders including specialized Vale nickel products. By comparison, Glencore (previously Xstrata) operates a simplified flow sheet for nickel production from its own and custom feeds, also in the Sudbury region (Figure 18).

Operations comprise pre-roasting and smelting of nickel concentrates in a submerged electric arc furnace, followed by conversion and matte granulation prior to shipping for refining in the Nikkelverk facility in Finland.

The Outokumpu flash smelting process (Figure 19) is probably the most environmentally compatible of the current commercial nickel smelting processes because it permits the capture of virtually all the SO_2produced prior to the conversion step. The Outokumpu direct oxidation of nickel (DON)



Adapted from Vale

Figure 17 Vale Ontario nickel operations flow sheet

process completely eliminates the separate converting step, instead producing a highly metallized matte that is sent for refining and a highly oxidized slag that is cleaned in an electric furnace to recover pay metal before discarding (Pääkkönen and Mattelmäki 1996).

Hydrometallurgy of Nickel Extraction

Several hydrometallurgical processes are in commercial operation for the treatment of Ni–Cu mattes to produce separate nickel and copper products. In addition, the hydrometallurgical process developed by Sherritt in the early 1950s for the direct treatment of nickel sulfide concentrates, as an alternative to smelting, is still commercially viable and competitive, despite significant improvements in the economics and energy efficiency of nickel smelting technology (Forward 1948).

In a typical hydrometallurgical process, the concentrate or matte is first leached in a sulfate or chloride solution to dissolve nickel, cobalt, and some of the copper, while the sulfide is oxidized to insoluble elemental sulfur or soluble sulfate. Frequently, leaching is carried out in a two-stage countercurrent system so that the matte can be used to partially purify the solution, for example, by precipitating copper by cementation. In this way, a Ni–Cu matte can be treated in a two-stage leach process to produce a copper-free nickel sulfate or nickel chloride solution and a leach residue enriched in copper. The copper-rich residue is treated by pressure leaching, or by roasting and leaching, to solubilize the copper as copper sulfate so that it can be recovered from solution electrolytically as cathode copper. Nickel is recovered from the purified nickel sulfate or chloride solution either electrolytically as a pure nickel cathode or by chemical reduction with hydrogen to give pure nickel powder. Extensive details of the various hydrometallurgical operations are provided by Kerfoot et al. (2011).

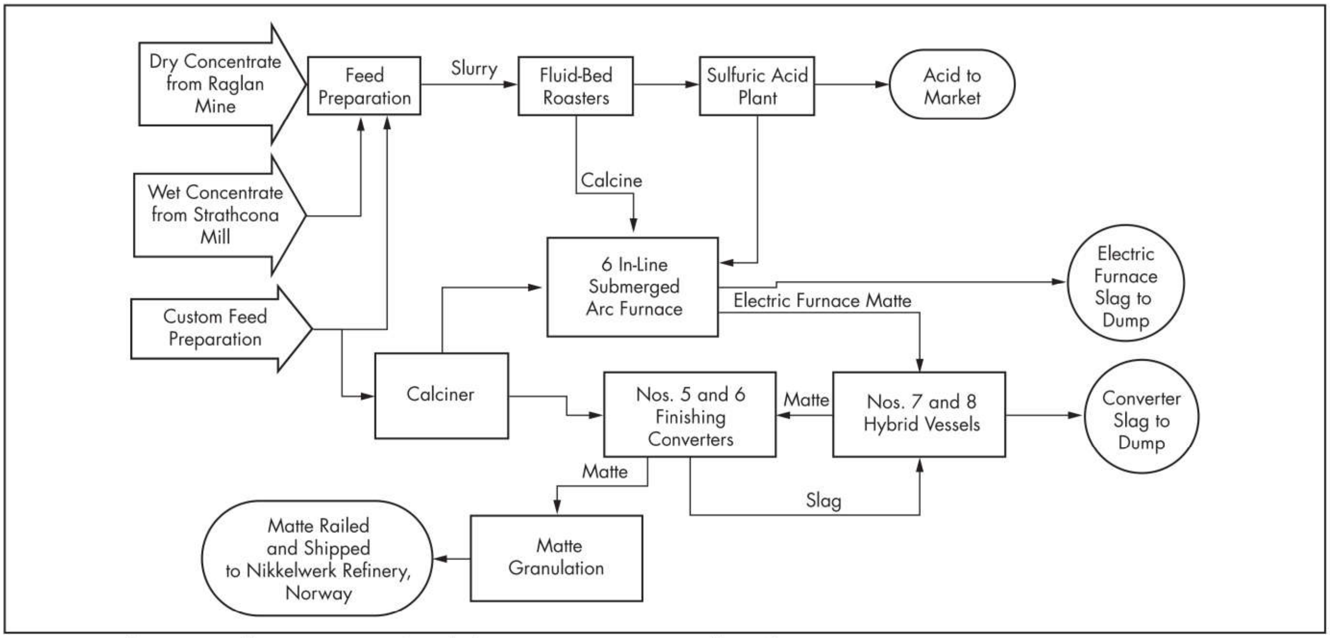
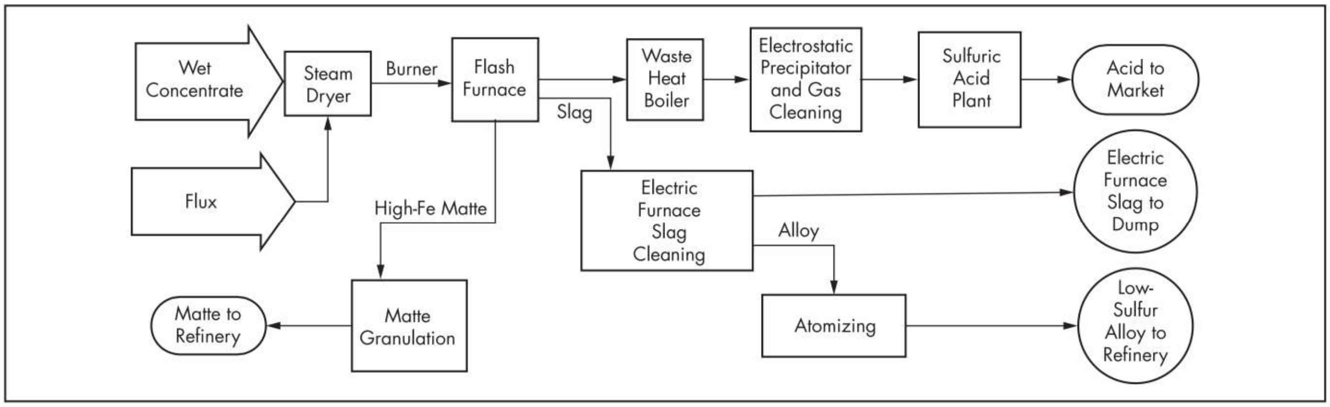


Figure 18 Glencore Sudbury integrated nickel operations extractive flow sheet



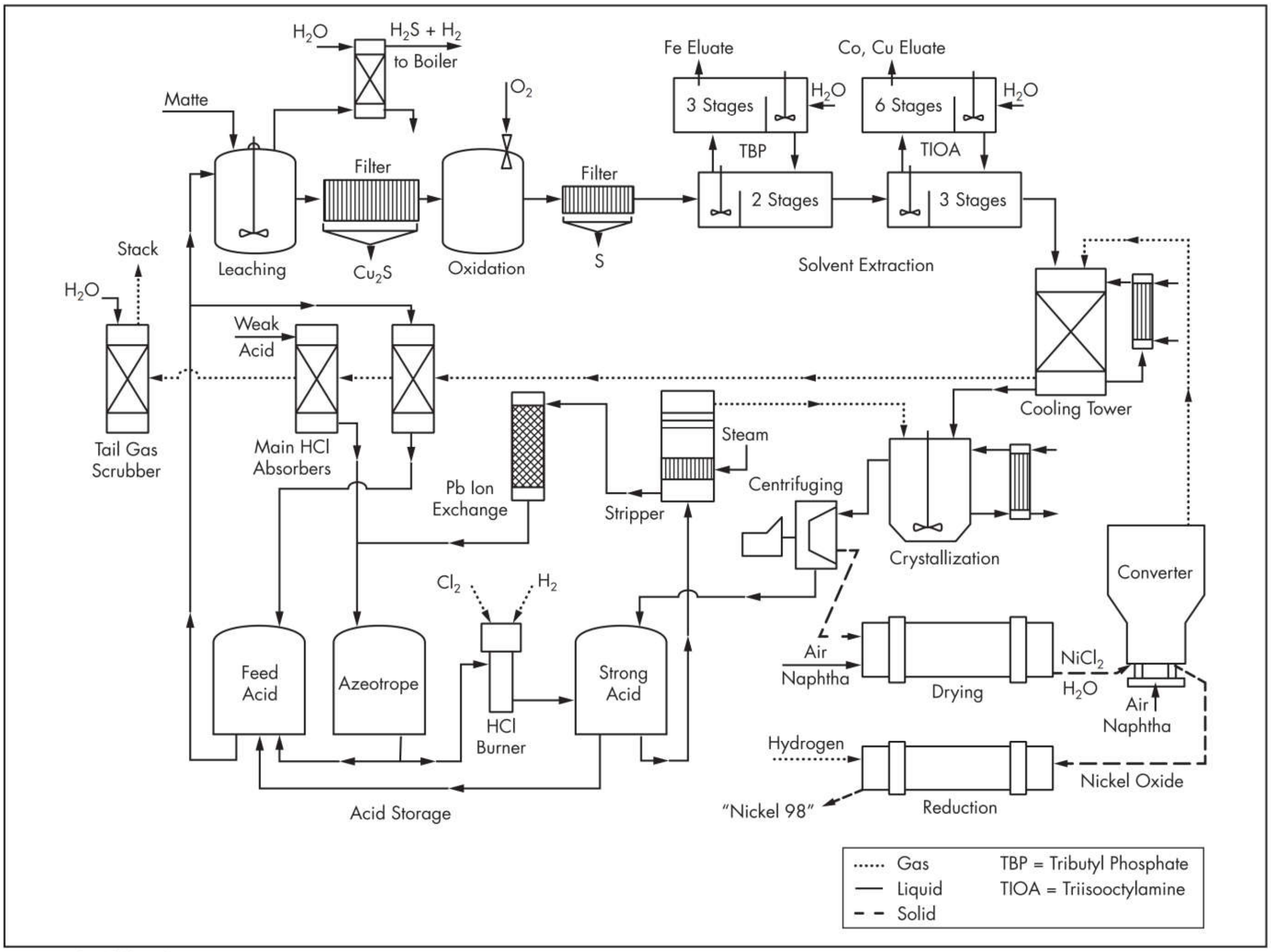
Source: Pääkkönen and Mattelmäki 1996, reprinted with permission from the Australasian Institute of Mining and Metallurgy

Figure 19 Outokumpu DON smelting process

The nickel contents of the mattes treated hydrometallurgically are in the range of 35% to 75%, whereas the copper content varies from 0% to 52%, and sulfur from 6% to 24%. Low-copper mattes (<10% Cu) can be treated by ammoniacal pressure leaching, by ferric chloride leaching, or by direct electrorefining of the matte. High-copper mattes can be treated by high-pressure or atmospheric pressure sulfuric acid leaching or hydrochloric acid leaching. The sulfur contents of the mattes are adjusted during the conversion process in the smelter to the optimal level for the subsequent leaching process. Sulfur levels are typically 20%–23%, except when the matte is to be treated by the Outokumpu atmospheric pressure sulfuric acid leaching process, for which the sulfur content is reduced to about 6%. Advanced hydrometallurgical techniques such as Activox and the Yakabindie process have also been developed specifically to treat high nickel- and magnesium-containing concentrates such as those at Yakabindie and Mount Keith in Western Australia and the Tati nickel mine in Botswana (Becker and Johnson 1996).

Atmospheric acid leaching. The atmospheric acid leach process for the treatment of high-copper, low-sulfur nickel matte was developed by Outokumpu in 1960 to process material from its nickel mine in Finland and was later adopted by the Bindura and Empress nickel refineries in Zimbabwe. In this process, the leaching operations are carried out in trains of air-agitated pachuca tanks, in contrast to the mechanically agitated horizontal autoclaves used in the pressure leaching process.

Acid pressure leaching. The acid pressure leaching process for the treatment of high-copper nickel mattes was developed by Sherritt Gordon during the 1960s and is currently operated by all four platinum producers in the Republic of South Africa: Rustenburg Refiners, Impala Platinum, Western Platinum (now Lonmin), and Barplats (Plaskett and Romanchuk 1978; Brugman and Kerfoot 1986). A similar process was also operated by the Amax nickel refinery in Louisiana, United States, from 1974 to 1986 (Blanco 1978; Llanos et al. 1974).



Source: Thornhill et al. 1971

Figure 20 Falconbridge matte leach process

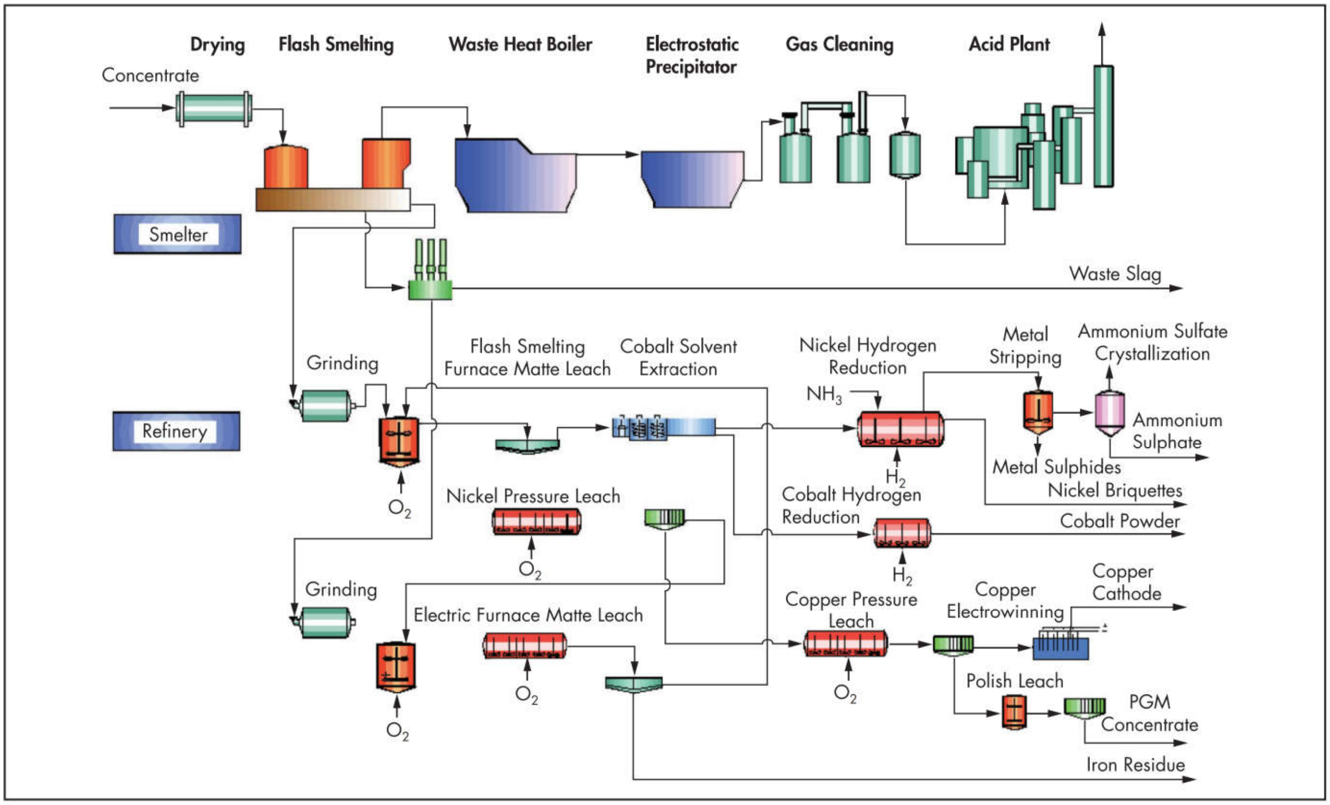
Chloride leach processes. Nickel matte refining processes based on leaching in chloride solution in the presence of chlorine gas have been developed and successfully commercialized by Glencore at Kristiansand in Norway (Stensholt et al. 1988) and by Société Le Nickel (SLN) at Le Havre-Sandouville in France (Lenior et al. 1981). The flow sheet of the Falconbridge process is shown in Figure 20.

Refining. Nickel differs from copper in that it is largely used as an alloying element in stainless steel and alloy steel production. Thus, although most copper is used in electrical and thermal energy applications, which require a highly refined metal, a high-purity metal is not required for many of nickel's most important markets. Nickel products such as nickel oxide, metallized nickel oxide, and ferronickel, which are designated Class II products, are produced directly by smelting and roasting and are sufficiently pure without refining for many nickel applications such as stainless steel. These Class II products may contain significant levels of cobalt and copper and trace impurities.

In other applications, high-purity nickel is essential and Class I nickel products, which include electrolytic cathode, carbonyl powder, and hydrogen-reduced powder, are made by a variety of refining processes. In general, nickel is refined to decrease to an acceptable level only those impurities that would have an adverse effect on subsequent processing or on the properties of the refined metal product. These elements include antimony, arsenic, bismuth, cobalt, copper, iron, lead, phosphorus, sulfur, tin, and zinc. Cobalt, which closely resembles nickel in physical and chemical properties, is difficult to remove completely, and since it is not often a serious contaminant in most applications for nickel, many Class I products contain quite high levels of cobalt.

The object of the refining process is not only to remove the impurities from the nickel, but also to recover those that have an economic value. Obvious examples are the precious and platinum group metals (platinum, palladium, osmium, rhodium, iridium, and rutheum), cobalt, and copper; sulfur, selenium, and tellurium can sometimes also be recovered economically.

The first successful nickel refining operation was established by the Mond Nickel Company (later absorbed by Inco/Vale) at Clydach, Wales, in 1902, using the carbonyl refining process invented by Mond and Langer in 1889. This refinery is still in operation using a modern version of the carbonyl



Courtesy of L. Rosenback, as cited in Svens 2013

Figure 21 Outokumpu DON refining process

process. A large refinery that uses a high-pressure version of the carbonyl process was established by Inco at Copper Cliff in Ontario in 1972 (Walker 1989).

The first electrolytic nickel refinery, treating nickel metal anodes, was built by Hybinette in Kristiansand, Norway, in 1910. This plant was acquired by Falconbridge in 1928. The Kristiansand Refinery was modernized in the late 1970s by replacing the Hybinette process with the current chlorine leach process. Inco established its first nickel refinery in Canada at Port Colborne, Ontario, in 1926, using a version of the Hybinette refining process. This refinery operated until 1984. In 1989 the only electrolytic nickel in North America was produced by Inco's (now Vale's) Thompson refinery, which electrorefines matte anodes Boldt and Queneau (1967). This process is also operated in Japan (Inami et al. 1986) and China. The electrorefining of nickel metal anodes, which is effectively obsolete in Western Europe and North America, is still practiced at Norilsk (Renzoni 1943).

Outokumpu, SLN, and Falconbridge in Europe and Bindura, Empress, and Rustenburg Refiners in Southern Africa all recover electrolytic nickel by direct electrowinning from purified solutions produced by the leaching of nickel or Ni–Cu mattes. The Outokumpu process is outlined in Figure 21.

Sherritt Gordon in Canada, Western Mining in Australia, and Impala Platinum in the Republic of South Africa recover refined nickel powder from purified ammoniacal solution by reduction with hydrogen.

Electrolytic refining. Nickel electrorefining is carried out in a tank house containing many electrolytic cells, each filled with a mixed nickel-sulfate–nickel-chloride solution, and containing 30–40 anodes (slabs of metallic nickel or nickel matte) interspersed with cathodes that are thin sheets of pure nickel. The anodes and cathodes in each cell are connected electrically in parallel, and the cells are connected in series. When electrical current is passed through the cells the anodes dissolve, and pure nickel deposits on the cathodes. Cathodes and anodes are replaced periodically to maintain continuity of production.

Electrowinning. Electrowinning differs from electrorefining in that it is used to recover nickel from a leach liquor and can therefore be carried out with an insoluble anode since the only function of the anode is to transfer electrons from the electrolyte to the external circuit. The electrowinning of nickel is practiced commercially with both pure sulfate and pure chloride electrolytes. In the sulfate system the anodes are made of chemical lead or antimonial lead, which have a relatively long life in sulfate solution, although they are soluble in chloride solutions. In the highly corrosive chloride system, dimensionally stable anodes, made by coating a titanium substrate with a platinum group metal oxide, are used. The Outokumpu nickel refinery at Harjavalta in Finland contains 126 electrowinning cells, each having 40 insoluble lead anodes and 39 cathodes. Details are provided in Diaz et al. (2011).

Thermal refining. The reaction of carbon monoxide at atmospheric pressure with active nickel metal at 40°–80°C to form the gaseous nickel tetracarbonyl was discovered by Langer and Mond in 1889 (Walker 1989). The reaction is readily reversible, with nickel tetracarbonyl decomposing to metallic nickel and carbon monoxide at 150°–300°C. Nickel tetracarbonyl is a volatile liquid that melts at –19.3°C and boils at 42.5°C.

Atmospheric pressure carbonyl process. The Inco nickel refinery at Clydach, Wales, began operation in 1902 using the Langer–Mond atmospheric pressure carbonyl process. Originally the plant treated Ni–Cu matte, but now it processes a granular nickel oxide produced by fluidized-bed roasting of nickel sulfide at Vale's Copper Cliff smelter, which typically analyzes 74% Ni, 2.5% Cu, 1.0% Co, 0.3% Fe, and 0.1% S (Walker 1989). The refinery still uses the basic Langer–Mond process, but the operation's efficiency has been greatly increased over the years.

High-pressure carbonyl process. Extraction of nickel with carbon monoxide is also used in processes that operate at high pressure and produce liquid nickel carbonyl. The volume change that occurs in the carbonyl reaction:

$$\mathrm{Ni} + 4\mathrm{CO} \rightarrow \mathrm{Ni(CO)_4}$$

An increase in pressure accelerates the formation of the carbonyl. The increased pressure stabilizes the carbonyl and thus permits the process to be carried out at high temperature, which further increases the rate of reaction. As a result, activation of the nickel feed material is no longer necessary, and a wider range of feeds can be processed. BASF in Germany operated a high-pressure carbonyl nickel refining process from 1932 until 1964.

Inco pressure carbonyl process. Vale's nickel refining complex in Copper Cliff, Ontario, which was commissioned in 1973, has a nominal capacity of 57,000 t/yr of refined nickel. The refinery consists of two operating plants: the converter plant and the pressure carbonyl plant. The converter plant produces granulated metallic nickel with a controlled sulfur content, using top-blown rotary converters. The pressure carbonyl plant produces 45,000 t/yr of nickel pellets, 9,000 t/yr of nickel powder, and 2,200 t/yr of ferronickel by-product from the granulated, sulfided nickel feed (Head et al. 1976; Wiseman et al. 1988).

The principal feed materials supplied to the refinery by the Copper Cliff smelter are the metallics fraction from the matte separation process and the slightly lower purity nickel oxide made by roasting the nickel sulfide concentrate as well as various other nickel-bearing residues and intermediates. The novel feature of this process is the preparation of a nickel feed material that could be carbonylated at moderate pressure without the need for a separate activation step. Full details of the operation can be obtained from Head et al. (1976) and Wiseman et al. (1988).

Hydrogen reduction to nickel powder. Nickel is recovered as a pure metal powder from nickel sulfate leach solutions by chemical reduction with hydrogen under pressure in several hydrometallurgical nickel refineries (Mackiw and Veltman 1980). Precipitation of the metal is carried out with an ammoniacal nickel sulfate solution having a molar ratio of ammonia to nickel of 2:1, so that the acid generated in the reduction reaction is neutralized by reaction with ammonia to form ammonium sulfate:

$$\mathrm{[Ni(NH_3)_2(H_2O)_4]SO_4 + H_2 = Ni + (NH_4)_2\ SO_4 + 4H_2O}$$

Full details of the process are provided in Head et al. (1976) and Wiseman et al. (1988).

NICKEL LATERITE ORES

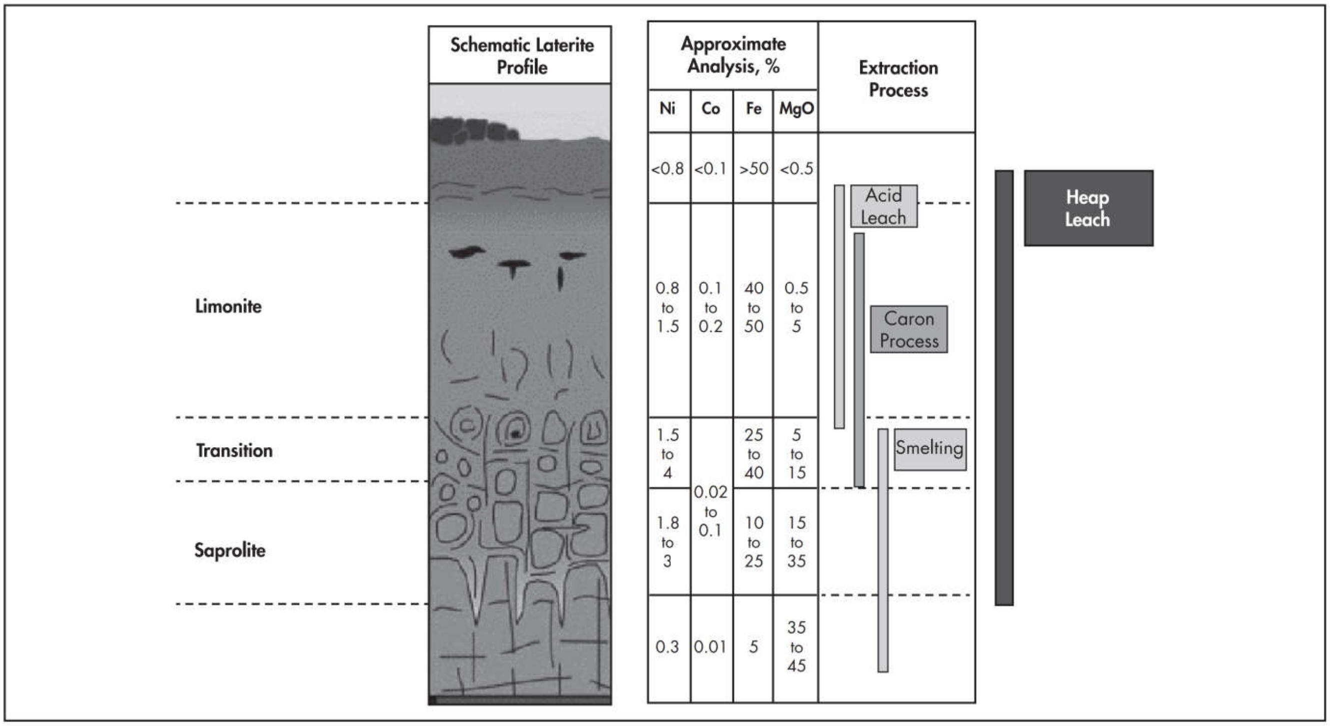
Geological and Mining Context

Although nickel laterite deposits are numerous and abundant, and comprise approximately 70% of the world's nickel resources, very few of them have become producing mines. However, with the global consumption of nickel growing at a rate of 4% per year, permanent new sources of nickel, such as in laterites, need to be found and developed. Production of nickel from laterite ores has occurred for more than 100 years, beginning with processing of garnierite-rich ores from New Caledonia. Since then, there have been major developments in Australasia, Indonesia, and South America with the development of both pyrometallurgically treated laterites (typically saprolite-rich), hydrometallurgically treated laterites (typically limonite rich), and a plethora of direct-shipping operations largely in Indonesia. Development has been challenged by a combination of factors, including the high capital and operating cost intensity of the flow sheet; challenging presentation in typically remote, hilly to mountainous terrain; and material handling challenges related to the sticky and dilatant nature of the saturated, tropical clays. Nevertheless, laterite deposits still have the potential to become the predominant source of nickel—particularly in ferronickel—worldwide (Dalvi et al. 2004).

Nickel laterite deposits occur in present and past zones of the earth that have experienced prolonged tropical weathering of ultramafic rocks containing ferromagnesian minerals. The majority of laterites can be found between the tropics of Cancer and Capricorn: the four main areas in the world for laterite resources are New Caledonia (21%), Australia (20%), the Philippines (17%), and Indonesia (12%). Some deposits do exist outside of this tropical belt, including those found in Oregon, United States, and in the Ural Mountains of Russia.

Lateritic nickel ores are formed by intensive tropical weathering of olivine-rich ultramafic rocks such as dunite, peridotite, and komatiite and their serpentinized derivatives, and serpentinite, which consists largely of the magnesium silicate serpentine and contains approximately 0.3% nickel. Serpentine is the most common product of hydrothermal alteration of olivine in the presence of water at temperatures ranging from 200°C to about 500°C. This initial nickel content is strongly enriched in the course of lateritization (Pelletier 2003). Two kinds of lateritic nickel ore must be distinguished: limonite types and silicate types. Limonite type laterites (or oxide type) are highly enriched in iron because of very strong leaching of magnesium and silica. They consist largely of goethite and contain 1%–2% Ni incorporated in the goethite. Silicate type (or saprolite type) nickel ore formed beneath the limonite zone. It contains generally 1.5%–2.5% Ni and consists largely of magnesium-depleted serpentine in which nickel is incorporated. In pockets and fissures of the serpentinite rock, high nickel contents can occur—mostly 20%–40%—bound in newly formed phyllosilicate minerals. All the nickel in the silicate zone is leached downward (absolute nickel concentration) from the overlying goethite zone (Figure 22).

Typical nickel laterite ore deposits are very large, low-grade deposits located close to the surface, with some examples approaching a billion metric tons of material. Ore deposits of this type are restricted to the weathering mantle developed above ultramafic rocks and tend to be tabular, flat, and extensive, covering many square kilometers. Mining operations generally proceed in the following sequence:



Adapted from Pelletier 2003

Figure 22 Typical laterite cross section

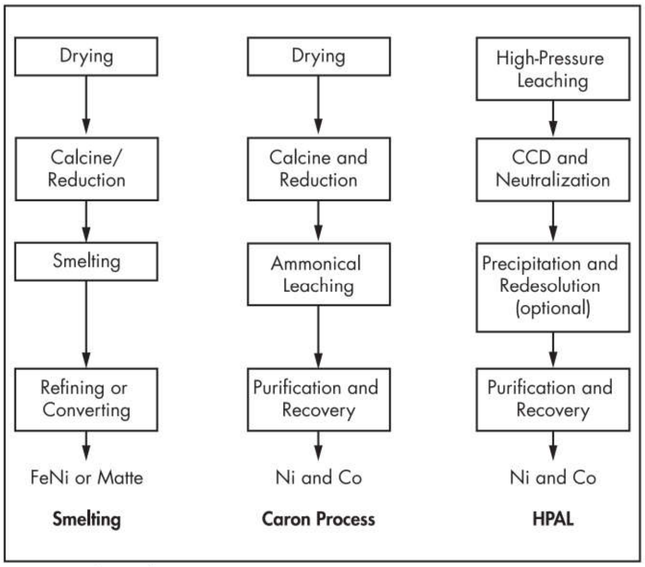
1. Mine top-down from the highest elevation to the lowest within any given mining area.
2. Overburden prestrip by dozers pushing downhill to the temporary roads (typically three more benches or cuts are established below the current operating cut).
3. Load and haul overburden either to permanent dumps or onto adjacent mined-out areas where saprolite or limonite is not exposed.
4. Load with hydraulic backhoes into 40-t capacity articulated or 60-t rigid off-road haul trucks.
5. Haul ore to the ore preparation plant from which upgraded ore is delivered to the processing plant typically via pipeline. Some ore is stockpiled before the slurry plant to "fill in" production during wet weather periods.

Production is maintained under strict geological control because of highly variable weathering profiles, zone thicknesses, and footwall rolls. Operations are typically surveyed daily and are visited at least once per day by the geologist responsible for grade control. Nickel is extracted from the mined ore by a variety of process routes, either pyrometallurgical in nature for primarily saprolitic deposits or hydrometallurgical for primarily limonitic deposits. Many deposits contain significant amounts of both limonite and saprolite material, and therefore close management of blend chemistry is critical. Pyrometallurgical processes are predominantly rotary kiln–electric furnace (RKEF) processes producing a ferronickel or matte product, although several variations to this flow sheet abound and are discussed in the following sections. The Caron process represents a hybrid route with high-temperature calcination of the laterite ore followed by ammoniacal leaching of the calcine to recover nickel and cobalt. Hydrometallurgical processes include high-pressure acid leach (HPAL) and heap leach, both of which are generally followed by solvent extraction and electrowinning (SX-EW) for recovery of nickel and cobalt (Figure 23).

Extraction of Nickel from Oxide Resources

As previously described, nickel oxide ores occur in two forms: a limonitic type that occurs in the upper zone of a lateritic deposit, in which ferric oxide materials are predominant, and a silicate, or garnierite, type (saprolitic), occurring at greater depth in the deposit, which has a lower iron content but is usually richer in nickel than the limonite and contains high levels of magnesia and silica. In nickeliferous limonite [$(Fe,Ni)O(OH)\cdot nH_2O$] the nickel oxide is mainly present in solid solution with the iron oxides. In nickeliferous silicate, nickel, iron, and cobalt oxides occur in varying proportions, generally replacing part of the magnesium oxide in the serpentinite [$Mg_6Si_4O_{10}(OH)_8$]. Most deposits contain both types of ore. Usually, it is possible to separate the ore by selective mining and screening into an iron-containing limonite fraction and a magnesium silicate fraction enriched in nickel (Golightly et al. 1979). The metallurgy of nickel oxide ores differs from that of sulfides in that oxide ores are not amenable to most of the standard mineral beneficiation methods because of the chemical dissemination of the nickel in the oxide minerals. Screening may be used to reject the oversize, less weathered fragments that contain less nickel, but this provides only a minor degree of upgrading.

Since world resources of nickel are predominantly lateritic (72%) rather than sulfidic (28%), new extraction capacity is largely focused on FeNi production rather than high-purity final products. Of the lateritic material, approximately 70% is extracted by pyrometallurgical techniques and most of the balance by hydrometallurgical methods. More recently, two processes have been developed in which the nickel is first



Source: Dalvi et al. 2004

Figure 23 Three main routes for treatment of lateritic ores

converted by high-temperature treatment to a metallic phase, which can then be separated from the bulk of the ore by standard mineral beneficiation techniques. These processes are the Nippon Yakin Oheyama process (which was based on the Krupp-Renn process developed in Germany in the 1930s to produce iron from otherwise uneconomic low-grade iron ores) and the MINPRO-PAMCO segregation process (Watanabe et al. 1985).

Pyrometallurgical Treatment of Laterites

Pyrometallurgical processes are suited for ores containing predominantly saprolite. A typical RKEF flow sheet is shown in Figure 24. Saprolitic ores contain proportionately lower cobalt and iron compared to the limonitic ores and are calcined and smelted to produce a ferronickel matte (Bergman 2003).

Pyrometallurgical processes are energy intensive since ores are typically mined in saturated tropical conditions, where all of the free moisture and interstitial water has to be removed, then the material is calcined to remove water of crystallization and then melted to form metal and slag at about 1,600°C. This requires significant amounts of reductants (coal, oil, or naphtha), fluxes, and electric power. FeNi produced by this route is typically granulated upon tapping, packed, and shipped to steel-producing areas for further treatment.

Variations against the standard RKEF flow sheet include the addition of a derubbling stage to remove coarse, low-grade peridotites or harzburgite to upgrade the ore prior to calcination (Figure 25), or, as at the Koniambo mine in New Caledonia, the addition of a bulk rougher sorting stage based on X-ray fluorescence where bulk ore below a certain grade is diverted to either stockpile or waste to ensure an optimal grade of feed to the calciner.

Nickel Pig Iron

A major (and disruptive) aspect of nickel production from lateritic ores was the widespread use in China of small blast furnaces treating limonitic ores to produce a pig iron containing 4%–6% Ni (Yamada and Hiyama 1985). Since the most popular grade of austenitic stainless steel contains only 8% Ni, 18% Cr, and the balance as iron, these furnaces were able to produce a raw material for stainless steel production that was then blended with cheap ferrochromium. In times of high nickel prices, these plants generated large volumes of nickel pig iron. Environmental pressure and high nickel inventories post-2009 led to the closure of many of these plants or the installation of new RKEF plants in China. Although China has no domestic nickel reserves, imports of raw ore from neighboring countries like Indonesia provided a ready source of feed. Evolving legislation is intended to encourage beneficiation of ores before exporting, and this may in time discourage pig nickel production.

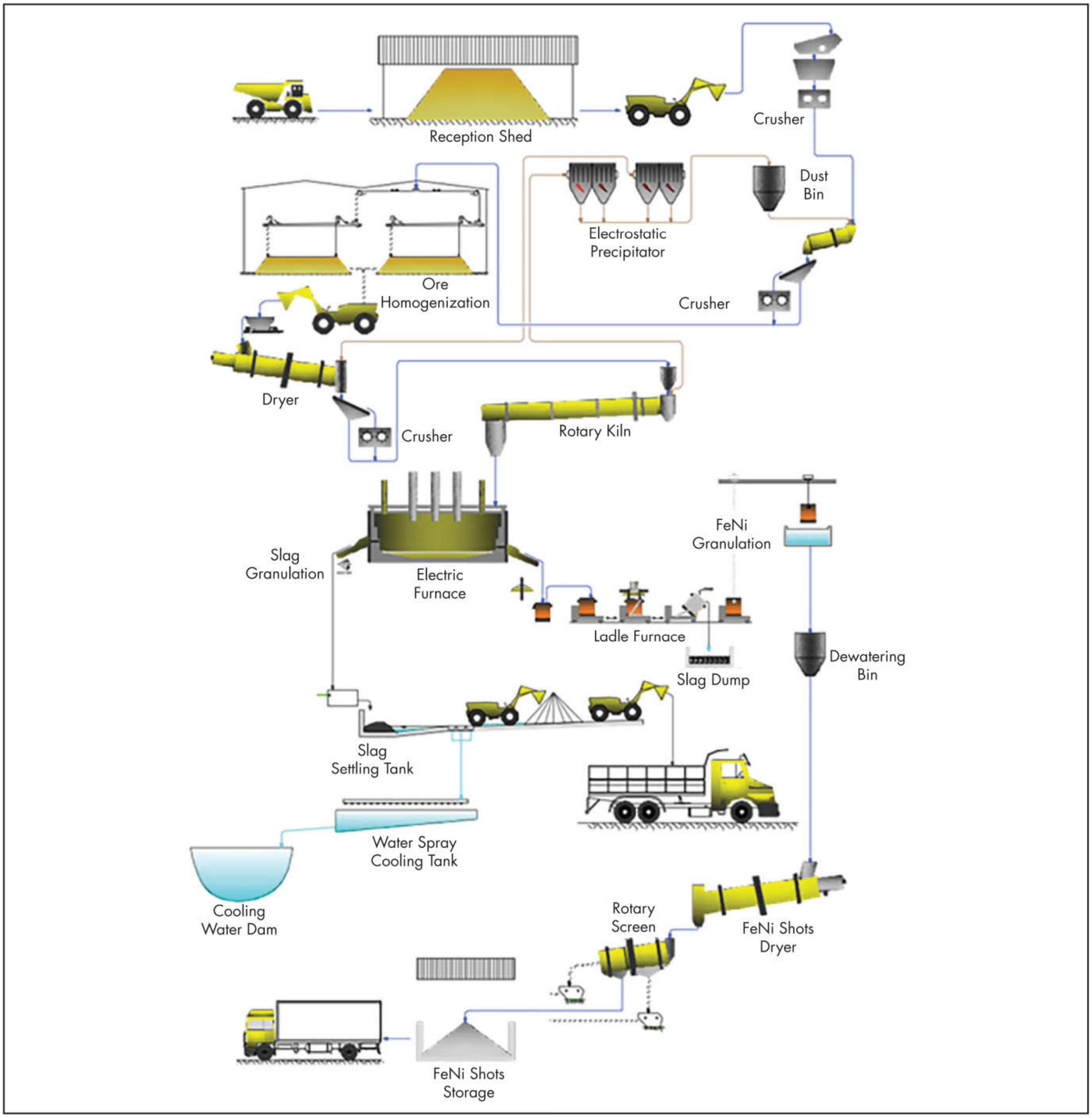
Nippon Yakin Oheyama Process

Nippon Yakin Kogyo produces 12,000 t/yr nickel as low-nickel ferronickel from silicate ores (2.3%–2.6% Ni, 12%–15% Fe), imported from New Caledonia, the Philippines, and Indonesia, at its Oheyama works in Japan. The ore is ground and mixed with anthracite, coal, and flux, and briquetted. The briquettes are fed continuously through a rotary kiln, countercurrent to a flow of hot gas produced by the combustion of coal. As the temperature rises from 700°C to 1,300°C, the charge is successively dried, dehydrated, and reduced (Watanabe et al. 1985). As the charge begins to melt, smelting occurs in the semi-fused state to form coalesced particles of spongy ferronickel (known as luppen, from the Krupp-Renn process) interspersed in a viscous slag phase. The kiln discharge is quenched in water, and the luppen are separated from the slag by grinding, screening, jigging, and magnetic separation. Nickel recovery from ore is typically 95% S. A major advantage of this process is that the ferronickel product is recovered in the form of 2- to 3-mm particles containing 22% Ni + Co, 0.45% S, 0.03% C, and 0.02% P, which can be used directly in the argon-oxygen decarburization steelmaking process, without further upgrading or refining. Although the process is energy intensive, the major energy sources used are cheap anthracite and coal, and no high-cost electrical energy or oil is used for heating. Consequently, energy costs are relatively low.

Caron Process

The Caron process has been utilized for the production of nickel since 1944 with commercial operations at Nicaro, Punta-Gorda, and Tocantins in Brazil and Yabulu in Queensland, Australia (Taylor 2013). In general, the process consists of ore beneficiation and blending, reduction roasting, leaching of roaster product in ammoniacal solvent, separation of nickel and cobalt using solvent extraction, precipitation from solution of basic nickel carbonate (BNC) intermediate product, and thermal decomposition of BNC to NiO and gaseous reduction of NiO to Ni (Figure 26).

The process was developed to treat ores that are a blend of limonitic and saprolitic materials. The mixture is dried to reduce moisture from 35% to approximately 8%, then ground to less than 850 μm before being fed into the roaster. Heavy fuel oil is typically mixed with the feed to provide the high temperature inside the furnace through partial combustion, as well as provide a reducing agent for the process. The maximum temperature of the reduction process is 740°–750°C. This is the critical stage that controls recovery of nickel from the ore. The reduced ore is discharged from the roaster and cooled under a neutral atmosphere to a temperature below 250°C. The cooled product is subsequently



Source: Bergman 2003

Figure 24 Typical RKEF process

leached in an ammonia–ammonium carbonate solution where the nickel is selectively leached from the solid, together with iron and cobalt. The pregnant liquor is then separated from the undissolved solid. The Fe metal complex is unstable and reacts with the oxygen from the atmosphere and precipitates as solid iron hydroxide. Cobalt, on the other hand, is separated from the nickel by an ammonium solvent extraction process and is extracted by hydrometallurgical refining in a separate cobalt processing circuit. The purified liquor is then steam stripped to remove ammonia/ammonium carbonate and to recover the nickel as BNC, a mixed carbonate hydroxide solid with approximately 50% Ni content. The remaining liquid is separated from the solid by filtering process. The BNC is then processed through a series of high-temperature processes involving various gas–solid reactions. The BNC at the Yabulu refinery, for example, is fed to a rotary kiln operated in a slightly reducing atmosphere, fired with naphtha or coal seam gas. This is aimed at partially reducing the BNC to Ni/NiO mixtures. Dehydration and decomposition of BNC into nickel oxide occurs in the kiln, followed by partial reduction

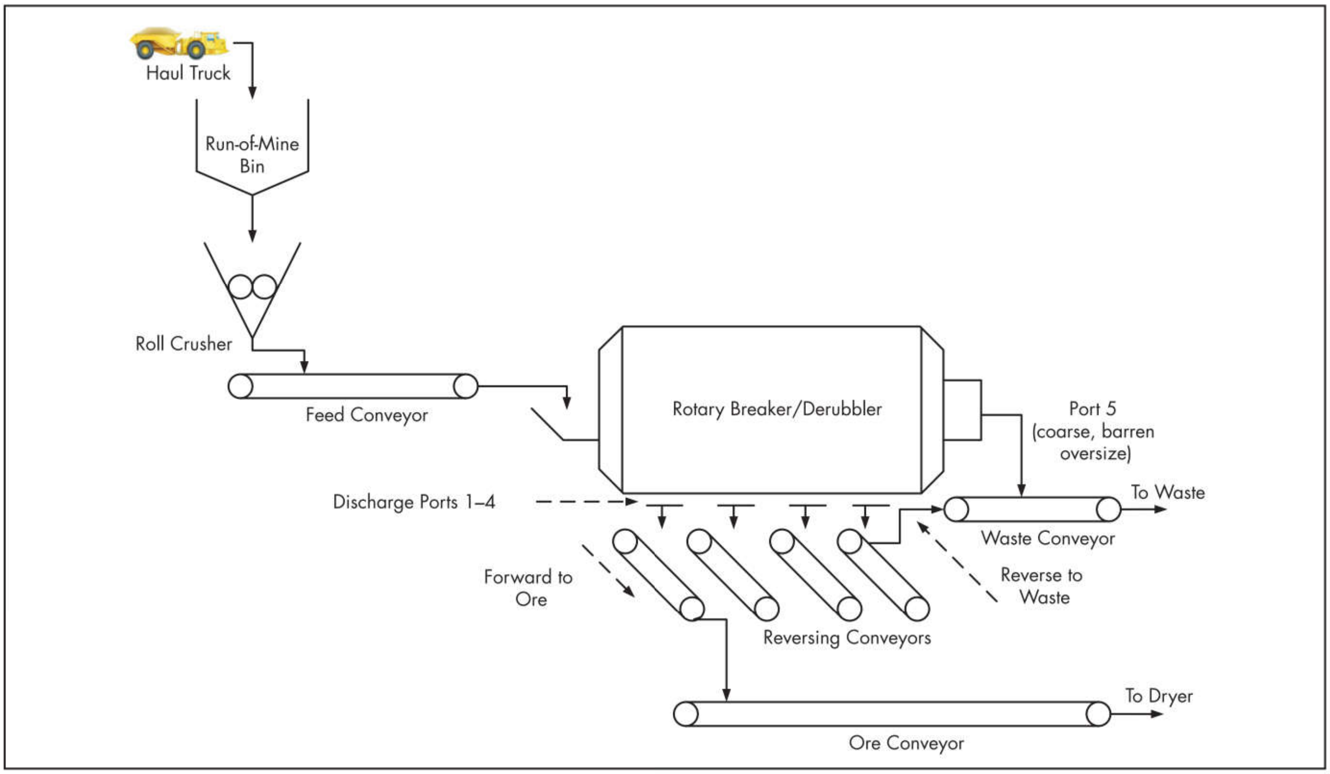
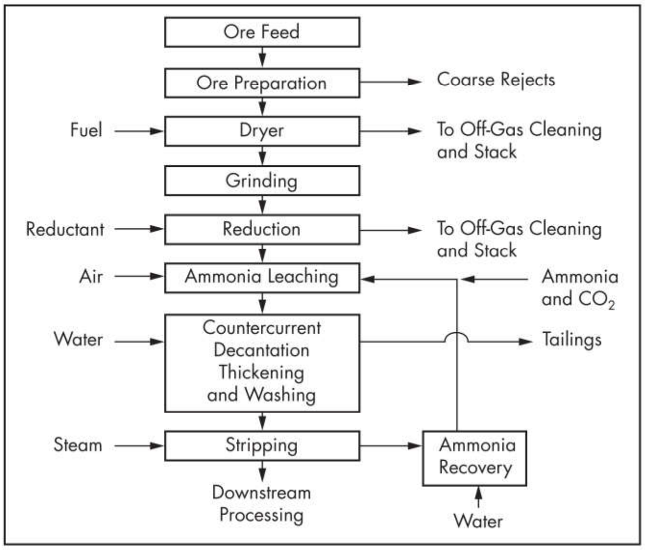


Figure 25 Upgrading of low-grade laterites by scalping and derubbling



Source: Taylor 2013

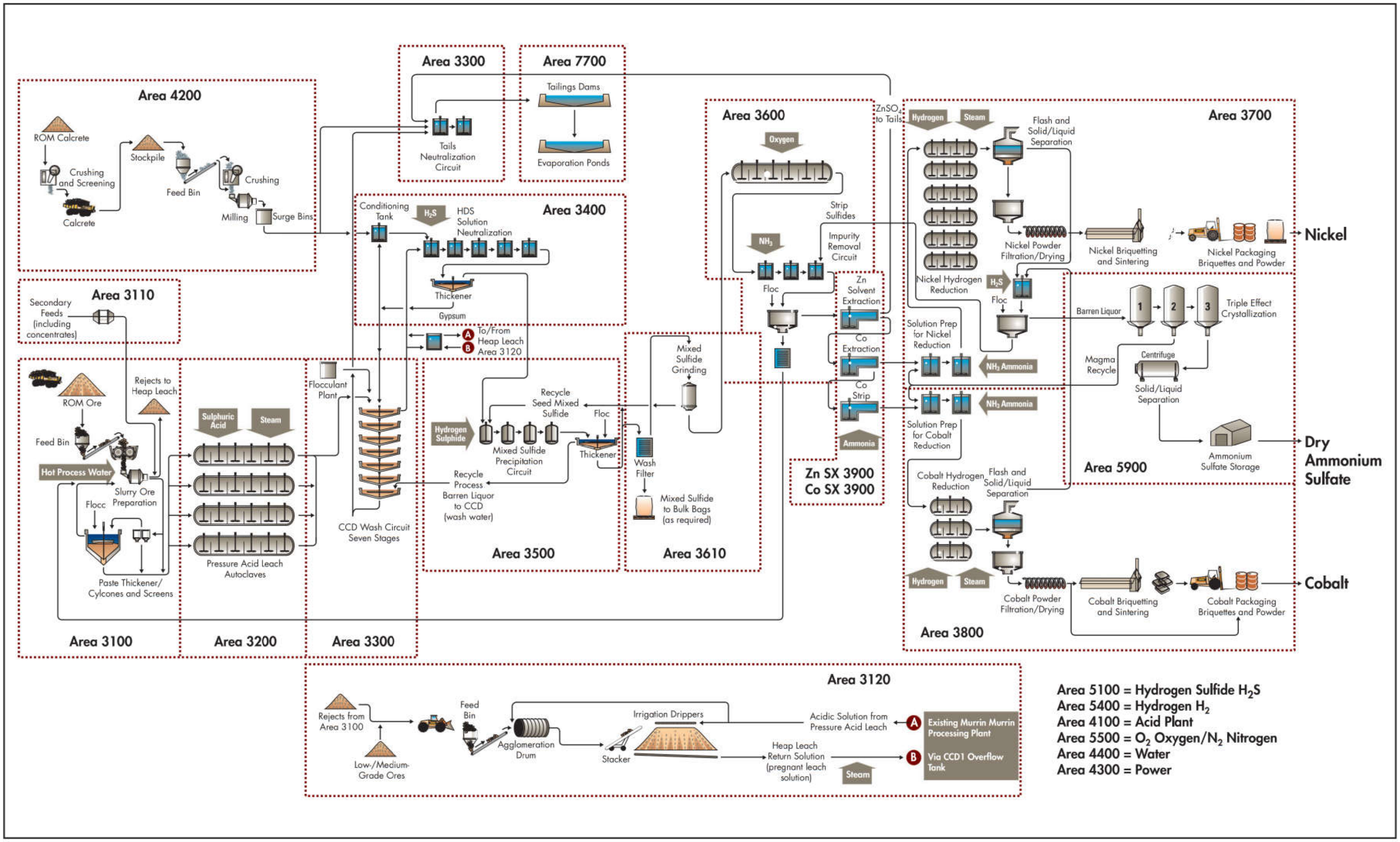
Figure 26 Typical Caron process flow sheet

of the nickel oxide, producing a mixture of metal–metal oxide containing 92% Ni (Reid and Barnett 2002). The second stage of the reduction process is carried out in a furnace. The kiln product is crushed and mixed with sawdust to provide sufficient porosity in the bed during the reduction process. The residence time in the furnace is approximately one hour. The furnace carries out reduction at an average temperature of 900°C under an atmosphere containing cracked ammonia, 75% H_2, and 25% N_2. The furnace product contains approximately 97.5% Ni, which is then crushed, blended with stearic acid, and pressed into compacts. The compacts are then fed into a sintering furnace, where a final stage of reduction and sintering of the nickel takes place. The final product of this process is a premium sinter containing >99% Ni.

Caron flow sheets are disadvantaged, despite the premium produced, by high energy consumption, moderate nickel and low cobalt recovery, and high sensitivity to feed composition requiring careful control of blending.

Hydrometallurgical Treatment of Laterites

High-pressure acid leach. HPAL technology enables the recovery of nickel from primarily limonitic nickel oxide ores that traditionally were difficult to process by pyrometallurgical routes. Oxide ores are crushed, ground and pulped, and subjected to high temperature and pressure and reacted under stable conditions with sulfuric acid to produce a nickel-rich refining intermediate (Figure 27). HPAL is employed for two types of nickel laterite ores: (1) ores with a limonitic character, such as the deposits of the Moa District in Cuba and southeast New Caledonia at Goro where nickel is bound in goethite and asbolite, and (2) ores of a predominantly nontronitic character, such as many deposits in Western Australia like the Murrin Murrin Joint Venture, where nickel is bound within clay or secondary silicate substrates in the ores (Reid and Barnett 2002). SMM Mining claims to be the first company in the world to apply HPAL successfully on a commercial scale. Many companies have tried to do so, but some failed because of the complexities of the process, largely relating to rheology of the leach feed slurry and/or the sophisticated materials of construction of equipment needed to treat acidic slurries at high pressure.



Courtesy of Minara Resources

Figure 27 Typical HPAL flow sheet

Table 2 Index of selected HPAL operations

Plant	Capacity, t Ni/yr	Approximate Cost, US$, millions	Start	Capital, US$/t Ni/yr	Ramp-Up	Comment
Coral Bay Line 1	10,000	180	2004	18,000	Fast	No acid plant, MSP*
Coral Bay Line 2	12,000	370	2008	30,833	Fast	No acid plant, MSP
Ravensthorpe	50,000	3,900	2008	78,000	Sold	Now FQMI, MHP*
Ramu	31,150	2,100	2011	67,416	Steady	MHP only
Goro	60,000	6,000	2012	100,000	Slow	Construction started in 2001
Ambatovy	60,000	7,200	2012	120,000	Steady	Now in operation
Taganito	36,000	1,700	2013	47,222	Fast	No acid plant, MSP

*MSP = mixed sulfide precipitate; MHP = mixed hydroxide precipitate

The advantages of HPAL plants are that they are not as selective about the type of ore minerals, grades, and nature of mineralization. Disadvantages include the high capital cost, the amount of energy required to heat the ore material and acid, the wear and tear of hot acid on plant and equipment leading to high maintenance cost and high downtime, and the complexity in downstream processing of leachates (Taylor 2013). Also, contamination of limonite with magnesium-rich saprolites further increases acid consumption and lowers the grade of the leachate. Projects have been characterized by lengthy ramp-up periods for the majority and total process failure in others (Table 2).

Heap (atmospheric) leach. Heap leach treatment of nickel laterites is primarily applicable to clay-poor, oxide-rich ore types where clay contents are low enough to allow percolation of acid through the heap. Generally, this route of production is much cheaper—up to half the cost of production—because there is no need to heat and pressurize the ore and acid, although recoveries can be low and generally only an intermediate nickel product requiring further treatment by electric furnace is required (Bergman 2003). In heap leach operations, the ore is typically ground, agglomerated, and perhaps mixed with clay-poor rock, to prevent compaction of the clay-like materials and maintain permeability. The ore is stacked on impermeable plastic membranes, and acid is percolated over the heap generally for three to four months, at which stage 60%–70% of the Ni–Co content is liberated into acid solution. That solution is then neutralized with limestone and a Ni–Co hydroxide intermediate precipitate is generated, which is generally sent to a smelter for further treatment. Challenges can include high acid consumption in presence of high magnesium levels, poor heap stability—especially in high rainfall (which is common)—and high levels of impurities carried over into the intermediates, which complicates downstream processing.

COBALT-PRIMARY ORES

Geological and Mining Context

Cobalt naturally occurs in nickel-bearing laterites and nickel–copper sulfide deposits and thus is most often extracted as a by-product of copper and nickel ore. According to the Cobalt Institute, about 48% of cobalt production originates from nickel ores, 37% from copper ores and 15% from primary cobalt production (CI 2017).

Sedimentary rock–hosted stratiform copper–cobalt deposits are the most important source of primary cobalt globally, with the Central African Copperbelt, the world's largest and highest-grade sedimentary copper–cobalt province, hosting the largest reserves of cobalt and more than 15% of the world's copper, the majority of which lies in the Katanga province of the DRC (Figure 28). Economic deposits of cobalt are generally found in the Lower Roan subgroup of the complex on the Zambian side, and the Mines subgroup dolomitic shales on the Congolese side (Broughton 2013). Host rocks are predominantly quartz–carbonate, with minor magnesium and chlorite. Sulfide minerals include vein-controlled, coarse-grained, and disseminated textures of pyrite, chalcopyrite, chalcocite, and cobalt minerals. The main cobalt minerals are cobaltite, erythrite, glaucodot, and skutterudite (Broughton 2013, Jackson et al. 2003).

Mining methods are both open pit and underground, with the Central African Copperbelt host to some of the largest underground mines in the world, including the Konkola and Nchanga mines. Open pit extraction is by truck-shovel, with ore haulage by truck to a primary crusher, from which ore is conveyed to the mill feed stockpile. In underground operations, methods range from open stoping through sublevel caving to vertical crater retreat methods, delivering ore by haul truck to the underground primary crusher, from where ore is conveyed or hoisted to the surface stockpile. Stockpiling of ore on surface is common, prior to crushing, grinding, and flotation stages.

Metallurgical Aspects

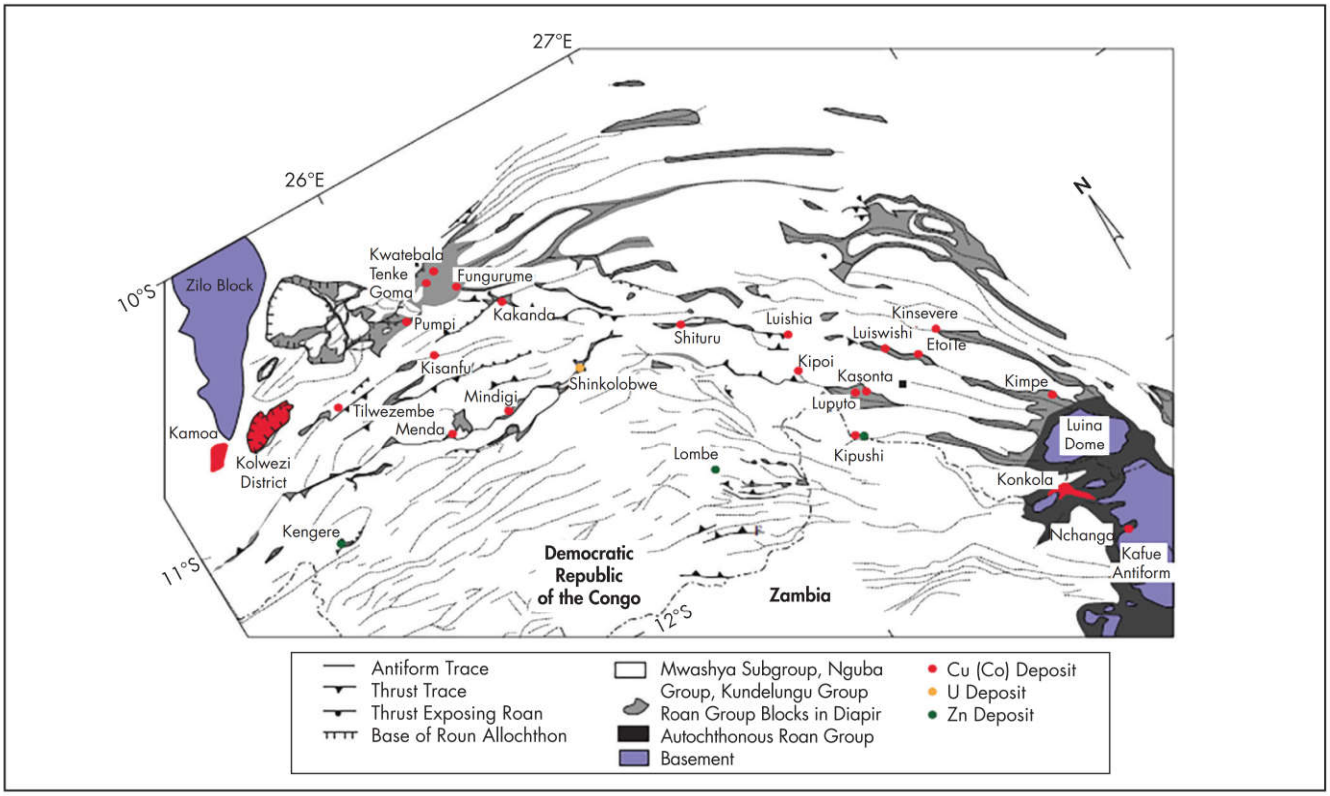
Cobalt is usually a by-product of copper and/or nickel production, and the overall process flow sheet will usually feature a leach of some type followed by recovery of the copper first, impurity removal and cobalt recovery second, and nickel recovery last (Figure 29).

Salable cobalt products range from concentrates through various intermediates to high-purity metal and salts. Project stakeholders should decide on the cobalt product they want to make, before the flow sheet can be finalized. Product options could include the following:

- Cobalt sulfide, or mixed Ni–Co sulfide precipitate (MSP)
- Cobalt hydroxide, or mixed Ni–Co hydroxide precipitate (MHP)
- Cobalt oxide or oxy-hydroxide
- Cobalt sulfate
- Cobalt carbonate
- Cobalt powder/briquettes, via hydrogen reduction

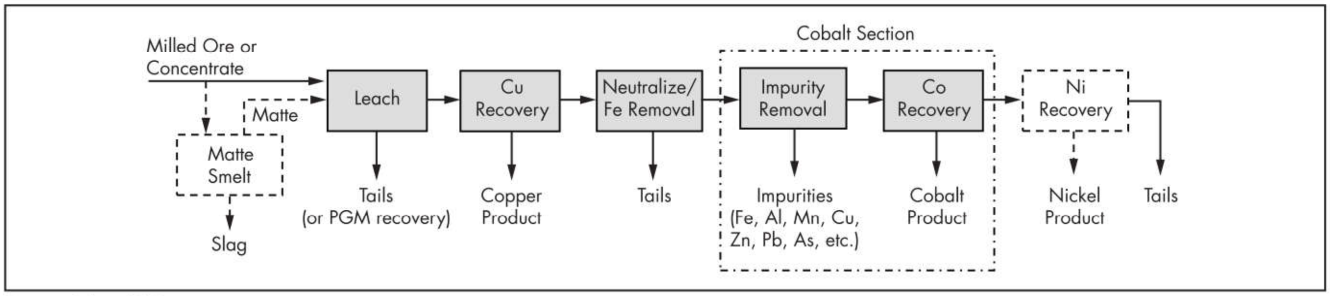
Cobalt can also be produced as an electrowon metal, with low-grade at 99.3% Co and high-grade at 99.8% Co.

Historically, cobalt–arsenide ores were typically roasted to remove the majority of the arsenic oxide. The ores were then treated with hydrochloric acid and chlorine, or with sulfuric acid, to create a leach solution that is purified. From



Adapted from Jackson et al. 2003

Figure 28 Sedimentary-hosted deposits of the Congolese copper belt including major producing mines



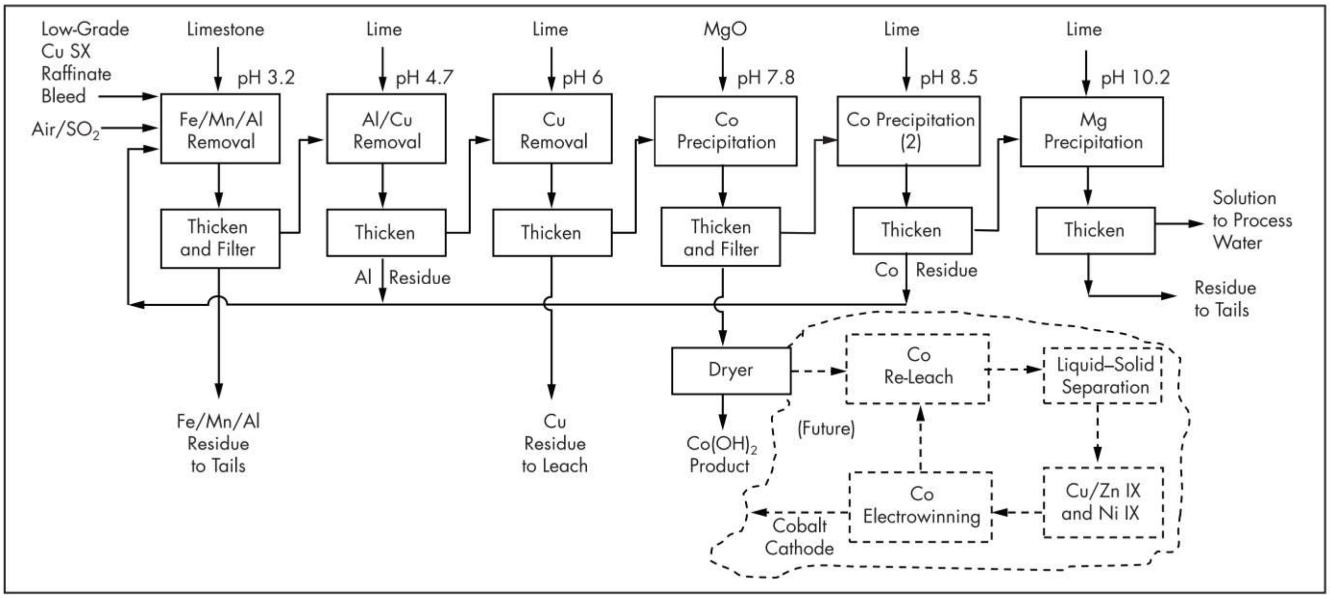
Source: Fisher 2011

Figure 29 Generic Cu–Co–Ni recovery flow sheet

this, cobalt was recovered by electrorefining or carbonate precipitation. A variety of such operations were active during the 20th century, mostly featuring a roasting stage and pressure leach of the roasted concentrate. However, all except the Compagnie de Tifnout Tiranimine operation in Morocco are now closed for obvious environmental reasons. Nevertheless, successful high-temperature/pressure leach test work has been done recently for several cobalt arsenide projects (Ferron 2001). The key to success has been the generation of an environmentally acceptable, stable arsenic residue in the process. As a result, potential exists for future contributions to cobalt production, such as Formation Metals in Idaho, United States (capacity 1,650 t/yr cathode), and Fortune Minerals' NICO Project in northwest Canada (Mezei et al. 2009).

Historically, cobalt from copper ores as in the DRC and Zambia is recovered by selective flotation from the copper-primary ore and then treated by a roast-leach-electrowin process. This would comprise roasting (for the sulfides), sulfuric acid atmospheric leach and direct copper electrowinning, plus impurity removal and cobalt hydroxide precipitation, often followed by releach and cobalt electrowinning. In recent years, apart from a few older plants, direct copper electrowinning has been replaced by copper SX-EW to more easily achieve London Metal Exchange Grade A specification. Many current cobalt projects are employing SX for impurity removal, even to the extent of having three or four SX sections in series, each using a different reagent. Sole et al. (2005) describe several Southern African SX plants and projects, with a particular focus on cobalt. Figure 30 shows a typical flow sheet recently proposed for a DRC cobalt project.

Conditions for cobalt recovery from copper concentrate by flotation at these operations are often suboptimal, with recoveries typically ranging from 40% for oxides to 80% for sulfides. Therefore, a substantial amount of cobalt is delivered



Source: Fisher 2011

Figure 30 Typical DRC cobalt flow sheet

to the copper tailings. An increasing number of operations, particularly Chambishi (Zambia), target these cobalt-rich tailings as a primary source of cobalt; similarly, as cobalt follows iron during smelting, some of the copper smelter operations have generated large slag dumps with a high cobalt content, for example, Nkana (Zambia) containing 0.65% Co (Fisher 2011).

Alternative cobalt production is as a by-product from hydrometallurgical processing of limonitic nickel laterites, either by HPAL or heap (atmospheric) leach. The traditional Caron process employs a reducing roast followed by ammoniacal leach, such as at Yabulu, Australia, which produces cobalt as an oxy-hydroxide cake using steam stripping after nickel is recovered. There are several Caron installations worldwide, but the process is less popular now because of high energy requirements and low cobalt recoveries. Over the years, improvements to autoclave design detail and ancillary equipment have progressively facilitated the treatment of laterites by HPAL in preference to the Caron process or pyrometallurgical processes, as demonstrated in recent Australian projects, albeit with varying degrees of success. Several potentially viable alternatives to the HPAL and Caron processes are currently under development. These include nitric acid leach (e.g., Direct Nickel), atmospheric sulfuric acid leach (e.g., BHP Billiton, Ravensthorpe), hydrochloric acid leach (e.g., Jaguar Nickel, Intec, ARNi), and heap leach. New extractants, including Cyanex 301 and Cyanex 302, have been developed that have the ability to extract cobalt and nickel together at a relatively low pH, leaving behind the impurities manganese, magnesium, and calcium (Fisher 2010). An example is the cobalt circuit employed at Vale's Goro project using Cyanex 301 to produce 5,000 t/yr Co as carbonate (Mihaylov et al. 2000).

Driven by increasing environmental pressures on the smelters, hydrometallurgical alternative processes for nickel sulfide concentrates (as opposed to matte) were seriously developed in the 1990s. Several such processes (acid-oxidative leach) are now possible; practical examples include Vale-Inco's current Voisey's Bay project (capacity 2,450 t/yr Co rounds), which employs a sulfate–chloride pressure leach; and the Activox leach process, which was to be implemented at Tati, Botswana (capacity 640 t/yr Co as carbonate) until it was cancelled in 2008. Others include CESL (chlorine-catalyzed pressure oxidative leach), Intec chloride, bioleach (e.g., BioNIC) and Sherritt Gordon's ammonia leach (Collins et al. 2015).

The advantages and disadvantages of many of the processes summarized earlier are discussed at length in the work by Swartz et al. (2009), which attempts to quantify their relative merits in terms of costs, risks, and benefits. Not surprisingly, there is no preferred process, where the high value but higher risk of cobalt metal production by electrowinning, for example, should be weighed against the simplicity, but lower return, of cobalt hydroxide or other intermediates. The interaction between business and technical considerations have been well summarized by Peek et al. (2009) especially with respect to the hydrometallurgical processing of cobalt salts to metal.

SUSTAINABILITY, ENVIRONMENTAL, AND GREENHOUSE GAS ASPECTS

Trends in Sustainability

The industry continues to move toward mine development and operations more aligned with global trends for more sustainable use of resources, including preventing or reducing adverse environmental impacts (air, water, land); reducing a project's overall environmental footprint (including energy and, again, water and land); adopting safe and environmentally conscious practice among the workforce; and working closely with communities in which companies operate during exploration, development, operations and closure to win and maintain license to operate (Corder and Becker 2017).

Approaches such as SUSOP (SUStainable OPerations), which seek to objectively assess potential impacts of any project across five main areas of capital (natural, human, social, manufactured, and financial), are increasingly adopted to identify risks that could potentially impact the project's viability and rank opportunities to improve the contribution of projects and operations to societal sustainability. Obviously, key areas remain energy, water, and land intensity of operations, and, for sulfide ore in particular, atmospheric emissions of sulfur. However, reducing secondary emissions and eliminating negative impacts on adjacent communities become increasingly essential.

Energy and CO_2 Aspects

Technology development in nickel and cobalt extraction has been inextricably linked with energy cost from the time they were first commercially extracted. For example, the development of the Peirce–Smith converter in France was a result of the high cost of fuel there compared to Welsh smelters, reducing coal consumption per ton of copper from 17 kg/t Cu to about 6 kg/t. One hundred years later, the coal equivalent is down to 1 kg/t Cu.

Nickel and cobalt extraction from laterite ores is very energy intensive, mainly because of the low nickel grade of the ore (typically below 3% Ni). In comparison to copper, energy consumption for nickel extraction in 2009 was 17 kg coal equivalent/t Ni, made up of about 7 kg for the process plant and ~10 kg for power generation (Warner et al. 2006, 2007). Much of the process energy is required for the drying/calcination stage as the saprolitic ore contains a considerable amount of hydroxide.

Specific energy reduction can be achieved in two ways. The first is partial upgrading of the ore based on differential sizing, as was practiced at Falcondo (Dominican Republic) for a short time. Doubling the head grade of feed to the process plant effectively halves the specific energy requirement. The alternative method is partial dissolution of (limonitic) ore in sulfuric acid, followed by precipitation of a mixed hydroxide that contains ~60% Ni. This material can be blended with saprolite feed to the FeNi plant, again substantially raising the head grade. From a technology standpoint this method is proven, but economic factors such as the relative cost and availability of acid and soda ash will likely govern the speed of adoption of these hydro/pyro hybrid process routes.

Sulfur Fixation

During the 1970s and 1980s, the industry responded to increasing public concern over smelter emissions of sulfur dioxide to the environment by replacing outdated process equipment, such as multihearth roasters, sinter machines, blast furnaces, and reverberatory furnaces, by processes such as fluidized-bed roasting, electric furnace smelting, and flash furnace smelting, which are more energy efficient and environmentally cleaner. However, although the technology is available to produce sulfuric acid economically from sulfur dioxide in the high-strength, off-gas streams produced by the modem smelting techniques, there is not always a market for large amounts of acid within an economic shipping distance of the nickel smelter. A process to treat pyrrhotite concentrate separately to recover nickel, iron ore, and sulfuric acid was operated by Inco for many years, but the operation was abandoned in the mid-1980s when it became uneconomic because of increasingly stringent sulfur emission limits.

In response to ever tighter constraints on sulfur dioxide emissions from smelters during the 1970s and 1980s, the Canadian companies have made increased pyrrhotite rejection from smelter feed a major priority, and substantial improvements have been achieved. Glencore at its Sudbury operation rejects more than 55% of the pyrrhotite in its ore and recovers sulfuric acid from the sulfur dioxide in the fluidized-bed roaster off-gas stream. Acid plant tail gas is routed through the electric furnace, further reducing sulfur emissions and overall off-gas volumes, and electric furnace and converter off-gases are vented to atmosphere (Clean Air Sudbury 2016). A pyrrhotite concentrate was treated by Inco (now Vale) for many years for nickel and iron oxide recovery, while Falconbridge (now Glencore) roasted pyrrhotite to produce a feedstock for an FeNi kiln operation (Peek et al. 2009). Environmental constraints on sulfur dioxide emissions and the availability of imported high-grade iron ores resulted in the closure of both plants in the late 1980s. However, with stockpiles of pyrrhotite tailings containing ~0.8% Ni and high in sulfur and iron exceeding 100 Mt in Sudbury, future processes may see a resurgence of interest in this material.

Additional reductions by off-gas cleaning using the best available flue gas desulfurization technologies are possible. Available technologies include wet and dry types. The wet processes involve production of wet slurry wastes as by-products. The flue gas leaving the absorbent is saturated with moisture. Regenerative systems use expensive sorbents that are recovered by stripping sulfur oxides from the scrubbing medium. These produce useful by-products including sulfur, sulfuric acid, and gypsum. Recovery systems have generally higher capital costs than throwaway systems but lower waste disposal requirements and costs. The dry processes produce dry waste material, and the flue gas leaving the absorbent is not saturated with moisture. SO_2 recovery efficiency of dry processes (50%–60%) is considerably less than wet processes (93%–98%). However, more recently, dry processes with more than 90% SO_2 removal efficiency are available (Roy and Sardar 2015). For example, Vale's recently commissioned Clean AER Project is the largest single environmental investment in its history, intended to reduce sulfur dioxide emissions by 85% from current levels and metals and particulate emissions by 35%–40% through additional smelter off-gas wet scrubbing stages, electrostatic precipitation, and recovery of metal particulates from scrubbed gas, and additional catalyzation of unreduced sulfur to recoverable forms.

CONCLUSION

Since 1950, the production of nickel and cobalt has increased on average between 3%–4%/yr. Despite an extended depression in demand since the global financial crisis of 2008, current market trends toward increased use of nickel and cobalt in clean energy and battery-electric applications are expected to continue, suggesting long-term recovery in nickel and cobalt prices from recent local minima is possible.

Historically, nickel production has predominantly come from sulfide deposits and operations. However, a trend toward a higher proportion of nickel production from lateritic deposits by the means described in this chapter is increasingly evident. Similarly, cobalt production has predominantly come from the Central African Copperbelt, with minor contributions from cobalt as a by-product from nickel and copper operations globally. Although cobalt production from cobalt-primary sources in Central Africa is expected to continue and increase, satisfaction of increasing cobalt demand will most likely also come from increasing production of cobalt as a by-product of hydrometallurgical treatment of Ni–Co laterites.

Mining, beneficiation, and extractive metallurgical approaches in nickel and cobalt, which developed massively through the 1970s to 1990s, are by now well established, with incremental improvements in operating cost and selectivity of reagents happening on an ongoing basis. The most recent innovations noted have been in the area of environmental remediation of atmospheric sulfur emissions with an increasing and now sustained focus on emission reduction in many major nickel- and cobalt-producing jurisdictions. Perhaps the next wave of innovations, subsequent to the Paris Climate Accord of 2015, will now be in energy efficiency with bold objectives to reduce the energy intensity of metal production by 40% being adopted specifically in Canada and elsewhere among member countries of the Organisation for Economic Co-operation and Development.

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