

# PROCESSING OF NICKEL LATERITE ORES

## A REVIEW OF SCIENTIFIC LITERATURE

Worldwide, the majority of nickel is produced from sulphide sources. This is changing, however, with production from lateritic sources expected to rise to over 50% by 2012 [1]. This trend results from a combination of the relative availability of nickel laterites (>70% of land-based nickel resources) [1,11] and the increasing use of hydrometallurgy for the extraction and recovery of metals.

Nickel laterites typically occur in regions where prolonged weathering of ultramafic rocks (containing ferro-magnesian minerals [1,2]) has occurred, favoured by warm conditions with abundant rainfall. Such deposits commonly exhibit graded layers consisting of:

- an iron-rich 'cap';
- a limonite layer dominated by goethite ( $\text{FeOOH}$  [3], total 40% Fe or greater [6]);
- a saprolite layer, often with basal boulder-gangue. The saprolite layer is usually magnesium-rich (10-20% Mg), with iron substituting for magnesium in serpentine ( $\text{Mg}_3\text{Si}_2\text{O}_5(\text{OH})_4$ ), as well as being present as goethite (total 10-25% Fe [3]);
- a garnieritic layer; and
- a weathered bedrock layer.

For typical deposits, the limonitic layer is not very suited to upgrading, while some upgrading in the magnesium-rich saprolitic layer can often be applied to improve the nickel concentration [1,5]. This represents the major difference between lateritic and sulfidic ores, the latter of which can be upgraded to yield high-grade concentrates (10-26% Ni) [1].

Both pyrometallurgical and hydrometallurgical processes are applied commercially to the recovery of nickel and cobalt from lateritic ores. Pyrometallurgical techniques tend to utilise saprolite, with a process typically involving drying, calcining/reduction and electric furnace smelting [1]. Pyrometallurgy usually produces a ferro-nickel (Fe-Ni metal) or nickel sulphide matte, which are further processed/refined to recover pure metal. The disadvantages of these processes include the requirement for higher-grade ores, substantial energy requirements [1] and poor cobalt recoveries [14].

Hydrometallurgical processes are more applicable to the limonitic laterites. Although the saprolitic laterites are often richer in nickel (up to 3% [3]) than the limonitic layer, the high magnesium content results in higher acid consumption. The primary hydro processes applied commercially are:

- the Caron process, an energy-intensive pyro/hydro combination with lower overall Ni and Co recoveries, which was first applied in Cuba the 1950s and is no longer considered viable for greenfield projects [1];
- high-pressure acid leaching (HPAL), which offers higher overall Ni & Co recoveries, >90%. This has been the technology common to all new hydrometallurgical nickel projects over the last 15 years, such as the three Australian projects at Cawse, Murrin Murrin and Bulong; and
- atmospheric-pressure acid leaching processes [4,6-9,11], including the newly developed Jaguar Nickel Inc (JNI) process.

### 1.1 HIGH PRESSURE ACID LEACHING WITH SULFURIC ACID

In HPAL, limonitic ores are leached at high pressure (33-55 atm [2]) and temperature (240-270 °C [1,2,11]) in titanium autoclaves, with slurry densities of about 20%. Acid consumption is typically 200-500 kg/t [11].

During heating (at about 135-155 °C), the major Al phase in limonite, gibbsite ( $\text{Al}(\text{OH})_3$ ) transforms to boehmite ( $\text{AlOOH}$ ) [2], which is readily leached in sulfuric acid. Divalent nickel has been reported as being substituted into the goethite ( $\text{FeOOH}$ ) lattice of limonite [2,4,5],

but is usually assumed in HPAL process descriptions as being 'oxide' (NiO, eg [2]). By contrast, cobalt is typically present as a third phase, possibly asbolane (a Ni-Co-Mn phase [2]), which is readily leached by sulfuric acid.

High recoveries for the nickel require complete destruction of the goethite matrix. Iron mobilised in the HPAL processes precipitate mainly as haematite ( $\text{Fe}_2\text{O}_3$ ), or as basic ferrous sulfate ( $\text{FeOHSO}_4$ ), which transforms rapidly to haematite [2,3]. As the iron and aluminium solubilities decrease with increasing temperature [3], they are typically separated from hot solution prior to neutralisation. Due to the presence of smectite clays, however, these slurries are unsuitable to filtration [7], and as such solid/liquid separation is effected by counter-current decantation. This can be costly, both in terms of plant size required and flocculant consumption, which can represent up to 10% of operating cost [7].

Additional iron removal is then effected by neutralisation at ambient pressure, commonly with limestone. However, the iron precipitated during the neutralisation step can be less crystalline, commonly ferrihydrite ( $\text{Fe}(\text{OH})_3$ ), which can yield settling and filtration problems [6].

In some instances, neutralisation using saprolitic ore has been proposed (eg [10]) limiting reagent usage and increasing the resource size by recovering metal from the saprolitic laterites. Due to the high magnesia content of saprolite and therefore high acid consumption, however, HPAL processes are unsuited to direct application to saprolite.

Like divalent iron, the nickel in saprolitic laterites typically substitutes the magnesium in serpentine [3]. This is rapidly attacked by sulfuric acid, which releases nickel into solution if used during neutralisation, but otherwise causes significant acid consumption. Further, the soluble Mg must then be removed in a separate purification step.

Limonite commonly contains other elements such as Ca, Zn, Cu, and, depending on the separation of the lateritic layers, Mg as well, all of which are commonly present as oxides and hydroxides [2]. The three Australian projects display the range of purification and product recovery options available to hydrometallurgical processes [10-12]:

- Murrin Murrin precipitates Ni and Co as mixed sulphides. This is followed by solvent extraction to give separate Ni and Co fractions, which then undergo hydrogen to produce metallic powder and briquetting to give a final product shape;
- Bulong used solvent extraction "to lower capital cost" [10], with Ni & Co extracted sequentially and electrowon separately to yield cathode products;
- Cawse uses mixed hydroxide precipitation with magnesia; followed by ammonia re-leach, solvent extraction and Ni electrowinning. Co is recovered as a sulphide precipitate.

As may be seen, solvent extraction is commonly used with HPAL processes. Over the longer term, resin-in-pulp methods have also been proposed to selectively separate nickel and cobalt directly from the leach solution [7,10], however this is not being applied commercially at present.

It should also be noted that all three plants described above have encountered significant difficulties during commissioning and operation, with none of the three operating at nameplate capacity even after several years of operation [1,9,11].

## **1.2 ECONOMICS OF HPAL AND PYROMETALLURGY**

Overall, laterite project economics are highly sensitive to ore grades. As laterites can only be upgraded to a very limited extent, plant sizes and waste volumes are relatively high per unit of metal recovered.

In terms of new production, it has been estimated that a greenfield smelter would have a capital cost of US\$12-15/lb Ni, requiring low cost power already installed and local high grade feedstocks (2-2.5% Ni) to be competitive. The operating costs were estimated as approximately US\$1.50-\$2.40/lb Ni [1].

By comparison, the costs of a Greenfield HPAL project was estimated as US\$12-18/lb Ni, with possible improvements with experience in design, material selection and construction. Operating costs are highly sensitive to nickel feed grade (minimum 1.4% to be competitive) and reagent costs, being US\$1.00-\$2.10/lb Ni after cobalt credits [1].

This information correlates reasonably well with the published information, with Murrin Murrin having an operating cost of ~US\$1.22/lb Ni [11], and the breakeven point in the nickel industry being within a cash cost range of US\$1.75-\$2.00/lb Ni.

### 1.3 ATMOSPHERIC LEACHING

Given the expensive materials of construction and energy requirements for HPAL, a number of atmospheric leaching processes have been proposed [6-9]. Such processes should offer significant economic advantages over both pyrometallurgy and HPAL in terms of materials, energy, and safety.

A process developed by BHP [6,7] treats both limonite and saprolite ores, albeit at separate leach stages. The leach is maintained at 900-1000mV by the addition of a reducing agent, to maximise cobalt dissolution. (A similar redox effect has also been observed for HPAL [13], where goethite dissolution was found to be favoured by the presence of ferrous iron.)

In the BHP process, iron is precipitated as jarosite by the addition of a precipitating agent of alkali metal ions or ammonium, and saprolite (to neutralise the system) [6]. Sulphide precipitation, solvent extraction and resin-in-pulp ion exchange [7] were all proposed as possibilities for metal recovery from the electrolyte. Notably, however, the production of jarosite residues would significantly increase waste volumes, as well as being much less environmentally stable for disposal.

Another proposed process [8] proposes heap-leaching with dilute sulfuric acid at atmospheric temperature and pressure. Again, nickel and cobalt extraction are proposed by solvent extraction, but it was found in this instance that overall recoveries were poor, only 74% for nickel and 51% for cobalt [9].

Jaguar Nickel Inc developed a process utilising a  $MgCl_2/HCl$  lixiviant in a two-stage agitated leach comparable to a zinc Roast-Leach-Electrowin plant [15]. In the first (neutral) leach stage, fresh ore is reacted with second-stage (acidic) liquor. Iron is rejected as haematite for good settling and filtration. Trivalent metals and copper are then precipitated via caustic magnesia recycle from a pyrohydrolysis stage. Manganese is removed either by oxidation/precipitation, or an undivulged alternate process, followed by zinc and then cobalt in another undivulged process. Nickel is finally recovered as hydroxide, with the resulting  $MgCl_2$  brine subjected to pyrohydrolysis to restore the water balance and to regenerate  $MgO$  and  $HCl$ .

Prior to going into receivership, JNI claimed an acid consumption for less than 150kg/t ore for their process, due to the high soluble Mg levels suppressing magnesium dissolution from the limonite ore. That being said, however, it was claimed that the Mg in the ore promotes iron precipitation, which would necessitate some dissolution.

Overall, the JNI process should offer the advantages of chloride systems:

- high metal recoveries;
- easily filtered and environmentally stable residues; and
- higher solution concentrations (and therefore smaller plant size)

However, JNI acknowledged water-balance issues (requiring high energy steps at both ends of the process), as well as the need for careful process control to avoid the formation of stable dissolved iron species, which are difficult to remove from solution.

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