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DEPARTMENT OF NATIONAL DEVELOPMENT
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NOTES ON METHODS OF
PRODUCING NICKEL FROM LATERITIC
ORES

By

R.W.L. King

The information contained in this report has been obtained by the Department of National Development, as part of the policy of the Commonwealth Government, to assist in the exploration and development of mineral resources. It may not be published in any form or used in a company prospectus without the permission in writing of the Director, Bureau of Mineral Resources, Geology and Geophysics.

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NOTES ON METHODS OF PRODUCING NICKEL FROM LATERITIC DEPOSITS

INTRODUCTION

For the purposes of considering possible treatment methods, lateritic nickel ores can be considered in two groups. Weathered serpentine type deposits are worked by Societe le Nickel in New Caledonia and Hanna Mining Company in Oregon, U.S.A. Acid leaching is not economically applicable to these ores because of the high magnesia content resulting in high acid consumption. Furnace processes are usually used. Limonitic ores on the other hand are amenable to acid leaching if the magnesia content is not too high, and may also be leached by pre-reduction and treatment with an ammonium carbonate solution. Some details of these processes are set out below.

FURNACE PROCESSES - NEW CALEDONIA

Prior to 1960 the only process used was "matte smelting". Ore and gypsum (as a source of sulphur) are smelted together in blast furnaces to produce a matte containing nickel and iron sulphides. Some of the iron is removed by a converter treatment. The nickel matte is sent to France where it is calcined and reduced with charcoal to produce nickel metal.

A second process is now in use, in which, after drying and pre-heating in rotary kilns, the ore is smelted in electric furnaces to produce a crude ferronickel containing 22-23% nickel. This product is refined using a Kalling rotary furnace for sulphur removal and basic Bessemer converter for removal of carbon and silicon, chromium and phosphorus.

Four 12,500 kva electric furnaces produce a total of upwards of 175 tons per day of crude ferronickel. Power requirements are about 550 kwh per ton of ore. About 200 kwh per ton is saved by charging hot ore direct to the electric furnaces. Oil consumption in the rotary kiln is dryer and preheater^{is in}, the range 120 to 180 pounds per ton of ore.

FURNACE PROCESSES - OREGON

A mine and smelter operated by the Hanna company produces ferronickel from a lateritic deposit near Riddle, Oregon. A process patented

by the French company "Ugine" is used for reduction of nickel and iron from this ore. This process involves the addition of a suitable reducing agent to a mixture of molten oxide ore in the presence of molten ferrous metal, using violent agitation for the mixing of the reducing agent and molten metals. 1.5 pounds of 45% ferrosilicon is used as the reducing agent to produce each pound of nickel contained in the molten ore.

Four 14,000 kva electric furnaces are used for melting ore, one 13,500 kva furnace is used for producing ferrosilicon and two 2,500 kva furnaces for refining ferronickel and preparing it for casting. Average power consumption per ton of molten ore produced in 1959 was 683 kwh, and for refining 0.12 kwh per lb. of ferronickel, which contains about 46% nickel. Total power consumption (1959) was 26.32 kwh per pound of nickel produced.

Nominal capacity of this plant is 20 million pounds of nickel per year. In 1959, 823,835 tons of ore, grading 1.5% nickel were treated. Some low grade rock was rejected and 680,964 tons of ore grading 1.68% nickel were actually processed to produce 20,794,091 lb of nickel in the ferronickel production. Fuel^{oil} consumption (for calcining) was 0.25 pound per pound of nickel produced (Queneau, 1961).

The operation was undertaken in 1954 with a government contract for approximately six years of production. Reserves were initially stated to be a minimum of 8.4 million tons of 1.5% grade, all in the one deposit. A block of 65,000 kw of power was allocated from the hydro-electric grid for the plant. The estimated plant cost was 25 million dollars.

LEACHING PROCESS - ACID - 100A DAY

50 million tons grading 1.36% Ni, 0.13% Co and 46% Fe were proved before operations began. The economic size of the operation was determined as 50 million pounds of nickel per year from 2 million tons of ore. Leaching is carried out at 475°F and 525 psi with sulphuric acid and the nickel and cobalt precipitated by H_2S from solution to form the concentrate shipped to the United States for further processing. Auxiliary services and equipment includes 2 x 650 ton per day acid plants, and H_2S generator of 60 tons per day capacity, water supply of 7,800 gals/min. with capacity to

14,000 gals./min. and 3 x 7500 kw turboalternators. Demand is 9-12,000 kw. Steam is also provided for the leaching system 115,000 lb./hr. at 650 psi and 750°F.

For refining the sulphide concentrate (which contains 55% Ni) is oxidised with air at 350°F under 700 psi pressure, and the resulting sulphate solution purified by adjusting pH and filtering off hydroxides, followed by precipitation of other metals with H_2S . The nickel sulphate is reduced with hydrogen at 375°F and 750 psi. Cobalt remains in solution and is recovered later. The nickel metal powder is briquetted for shipment. (Gueneau, 1961).

LEACHING PROCESS - ALKALINE - NICARO

Ore containing approximately 1.3% nickel, 0.09% cobalt, 2.75% chromium and 38% iron is given a reducing roast in Herreshoff furnaces and then leached with aqueous ammonium carbonate. Ammonia is stripped by steam heating of the leach solution and the precipitated nickel - cobalt carbonate is calcined. Only about 10% of the ore's cobalt content is recovered.

The plant was originally built with a capacity of 30 million pounds of nickel per year at a cost of 31.7 million dollars and operated unprofitably from 1943-1947 at a nickel price of 32 cents per pound. It was re-opened in 1952 at a cost of 12 million dollars and was expanded in 1957 to a capacity of 52 million pounds of nickel per year at a cost of 37 million dollars. (U.S.B.M., 1960)

GASEOUS REDUCTION PROCESS

This has been developed at bench scale only but has run as a semi continuous process at about 100 grams of ore per hour. The process is based on the properties of the iron and nickel carbonyls ($Fe(CO)_5$ and $Ni(CO)_4$). The former melts at -28°C and boils at 103°C while the latter melts at -25°C and boils at 45°C. Both decompose at temperatures above 150°C to give carbon monoxide gas and the metal.

Following hydrogen reduction of the iron and nickel originally present in the ore as oxides (2 hours at 700°C), treatment with carbon monoxide and ammonia (16 hours at 100°C and 375 psi pressure) takes place

and the metal carbonyls are formed as liquids. These are subsequently separated by fractional distillation and decomposed to form pure nickel, iron and cobalt metals.

The scale of operations was too small to give much economic data, but did demonstrate that the process is technically feasible and capable of giving reasonable recoveries of nickel from lateritic ores.

(Lewis et. al., 1953)

OTHER PROCESSES

One other possible process suitable for lateritic ores is the Strategic-Udy process in which final reduction takes place in an electric furnace under controlled conditions, the ore having been partially reduced in a rotary kiln discharging directly into the electric furnace.

Sufficient reductant is added to the kiln to reduce the nickel to metal and the iron to the ferrous state. In the electric furnace a ferronickel is produced with a high iron - low nickel slag. Treatment in additional electric furnaces can produce an 0.5-1% carbon steel and a high chromium iron. The steel produced would require further refining in another furnace.

No details on costs or power requirements were provided, but demonstrations have been carried out on 100 kva and 1000kva scale and the process appears technically feasible. (Udy, 1959).

A second process which also offers some promise of application to lateritic ores is a preferential sulphation process. The ore is leached with concentrated sulphuric acid, roasted under controlled conditions, leached in water and a nickel - cobalt sulphide precipitate obtained by the action of hydrogen sulphide under pressure.

The process depends on the fact that a partial transformation of soluble iron sulphate into insoluble iron sulphate takes place during roasting, with release of sulphur trioxide which then sulphates the unreacted non ferrous metals. This reaction appears to be partially catalysed by the presence of alkali metal sulphates. There is scope for blending low magnesia laterite with high magnesia serpentine material to produce a combination with the most reactive magnesia content.

Leaching with hot water under atmospheric pressure is sufficient to dissolve the sulphate formed, and treatment of this solution with H_2S under pressure (as in the Moa Bay acid leach process) gives a nickel sulphide concentrate.

The process is reported as having operated at pilot plant scale for over six months continuously, and data for design of a demonstration plant has been accumulated. (Zubrycky, 1965).

CONCLUSION

From the limited information available it seems that plant costs lie in the range of \$1 to \$1.5 per pound of annual capacity. Smaller size plants seem to favour electric furnace processes making ferronickel. Where power is not available or nickel desired as the end product, leaching processes are favoured, and these seem to have a higher lower limit of economic annual capacity. These limits appear to be 20 million pounds and 50 million pounds of nickel per year respectively.

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