Indigenous production of nickel and cobalt

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INTRODUCTION :

Production of nickel and cobalt in the country has been engaging the attention of the Government of India for more than two decades now. Indigenous production is needed to meet the domestic demand of nickel, cobalt and their compounds worth Rs. 400 million presently being imported [1] as detailed in Table—1. These imports account for nearly 8,000 tonnes of contained nickel most of which is used in stainless steel making. Till recently major requirement has been for unwrought nickel compared to class (II) products like oxide sinters and ferro nickel. The international prices being on the basis of contained nickel[2], the relative domestic demand for pure nickel (class I and class II) products fluctuates depending on prevalent duty levies. At present more of ferro nickel and nickel oxide sinters are being consumed compared to the situation a few years ago. However, from the point of view of economics of making nickel bearing alloy steels, while the class (II) products would be cheaper, from the technical point of view, with the advent of carbon bearing charge chrome usage, it is advisable to use carbon free oxide sinters.

Table—1 : Import statistics of nickel, cobalt and other compounds[1]

Item	Quantity Tonnes	Value in million Rs.
Nickel ore and concentrates	28	1.49
Nickel matte	250	10.25
Ferro nickel	1869	24.51
Unwrought nickel	2871	142.00
Monel metal	6.7	0.66
Cupro-nickel	94	5.42
Nickel alloys (40%)	183	9.21
German silver	8.6	0.50
Nickel alloys (10—40%)	3252	162.79
Waste nickel scrap and alloys	2056	23.49
Cobalt and alloys	102	44.99
Waste scrap cobalt	5.1	1.99
Cobalt oxide	2.9	1.04
Cobalt hydroxide	0.05	0.02

Primary sources for nickel (and cobalt) are either sulphide ores in which these metals occur as sulphides or laterite ore bodies in which these occur as oxides. 75% of the total world nickel is derived from sulphides and the balance originate from laterites. The former is not relevant to India which has only nickel and cobalt bearing laterites as potential source.

Status :

As of now there are at least 21 operating plants in the world extracting nickel and cobalt based on their oxide ores, with an estimated installed capacity of 0.4 million tonnes of nickel equivalent. Some details of these plants are indicated in Table—2. Four of them use hydrometallurgical route accounting for 18% of the capacity whereas the rest adopt pyrometallurgy. Two plants based on hydrometallurgy and three on pyrometallurgy have recently been closed down. Two plants based on hydrometallurgy to produce 60,000 tonnes nickel equivalent per year are coming up in Cuba[3]. In addition a plant to produce 8,500 tonnes is being set up in the United States, which will simultaneously

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Table—2 : Details of world operating plants[15]

	Plant	Installed capacity tonnes of contained nickel per/year	Feed %					Remarks
			Ni	Co	Al_2O_3	MgO	Fe	
	A. PYROMETALLURGY							
1.	Nippon Mining Sagaroseki, Japan.	11,350	—	—	—	—	—	Blast furnace (SLN); ferro-nickel; ore from Phillipines and New Caledonia.
2.	Societe-Le-Nickel Doniambo, New Caledonia.	65,850	2.8	0.06	—	24.0	13.0	Production partly by electric furnace (SLN); nickel matte and ferro-nickel.
3.	Morro-do-Niquel Petropolis (MG) Brazil.	2,500	1.4	—	—	–	61.4	Electric furnace (R K E F); crude and refined ferro-nickel.
4.	Cerro-Matasso Montelibane, Colombia.	22,500	—	—	—	—	—	Electric furnace (R K E F); ferro-nickel.
5.	Pacific Metals Hachinohe, Japan.	25,000	—	—	—	—	—	Electric furnace (R K E F); ferro-nickel; ore from Phillipines and New Caledonia.
6.	Nippon Nickel Tsuruga Japan.	4,500	—	—	—	—	—	Electric furnace (SLN) nickel oxide; ore from Phillipines and New Caledonia.
7.	PT International Nickel, EL-ESTOR Guatemala.	22,700	1.5to 2.2	—	—	20	15.20	Selective reduction (INCO) ferro nickel
8.	Hanna Mining Riddle, Oregon USA	14,000	1.7to 0.8		1to 3	24to 32	10to 15	French-ugine ferro-nickel
9.	Codemin Niquel-andia, Brazil	7,500	—	—	—	—	—	Electric furnace ferro-nickel
10.	Falconbridge Dominicana Bonac Dominican Republic	27,250	—	—	—	—	—	Falconbridge (similar to INCO) ferro-nickel
11.	PT International nickel, Soroako Indonesia	49,000	2.1	—	—	16.2	19.1	Selective reduction (INCO) 75% nickel matte

Table—2 : (contd.)

	Plant	Installed capacity tonnes of contained nickel per/year	Feed%					Remarks
			Ni	Co	Al_2O_3	Mgo	Fe	
12.	Larco, Larymna Greece	27,000	—	—	—	—	—	Ferro-nickel.
13.	PT Aneka, Tambang Pomalaa, Indonesia	5,000	2.5	—	—	—	—	Ferro-nickel.
14.	Nippon Yakin Ohayama, Japan	36,360	—	—	—	—	—	Ferro-nickel, ore from Phi-llipines and New Caledonia.
15.	Shimura Kako Shimura, Japan	5,000	—	—	—	—	—	Electrolytic nickel.
16.	USSR	25,000	1.4	—	—	—	—	Ferro metal; three plants.
17.	Ferro niki Kosovo, Glogovac Yugoslavia	12,000	—	—	—	—	—	Ferro-nickel.
	B. HYDROMETALLURGY (Ammonia leach)							
18.	Greenvale Yabulu, Australia	22,700	1.6	0.1	—	50	—	Sheritts Ammonia Leach nickel-cobalt sulphide,nickel oxide.
19.	Niquelandia Brazil	5,000	—	-	—	—	—	Sheritts Ammonia Leach nickel oxide, high iron ore
20.	Nicarro Cuba	25,000	1.5 to 1.8	— —	— —	— —	20to 25	Carron; nickel oxide granule sinter.
21.	Surigo Nickel Refining, Nonac Island, Phillipines	20,440	1.2	0.1	—	—	38.5	Sheritts Ammonia Leach, nickel powder briquetts. Recently closed; likely to reopen.
	C. HYDROMETALLURGY 9 (Acid leach)							
22.	Moa Bay Cuba	17,500	1.3	0.15	8.5	1.7	47.5	Freeport sulphur; nickel-cobalt sulphide.

(SLN—Societe le Nickel Anonyme; INCO—International Nickel Company; RKEF—Rotary Kiln Electric Furnace) produce cobalt, chromite and high pure magnesia[4].

All operating plants working on hydrometallurgical route are based essentially on one of the two alternatives involving acid leaching under pressure or reduction-roast followed by leaching with ammonia with minor modifications in the process steps depending upon the raw material availability and final product required. Similarly, all pyrometallurgical operations either adopt a blast furnace or an electric furnace with minor modifications based thereon. Besides, numerous attempts have been made to improve the efficiency and economy of the processes in practice as a result many modified processes tried on a pilot scale have been patented. These are summarised in Table—3.

Comparative evaluation :

For the purpose of comparative evaluation, a pyrometallurgical route based on electric smelting, two hydrometallurgical routes one based on reduction-roast ammonia leach and the other on acid leach are considered in the present exercise. Other potential processes narrated in Table—3 are left out since adoption of any one of them would mean much more in-country developmental work compared to operating one of the commercially established processes. Besides, the economics of the process will not be drastically altered with minor modifications brought out over the basic processes considered.

Electric furnace process (as practiced in Brazil by Morro-do-Niquel[5,6]) :

The oxide ore is treated in ore washing plant for removal of SiO_2. The nickel values are limited to the serpentinic fraction, the quartz veins and silica box work is barren. This allows beneficiation using hydrocyclones and spiral classifier. The overall nickel recovery in washing is 50%. The concentrate has 2% Ni and the reject has 0.6—0.7% Ni and 80% SiO_2. This concentrate is mixed with normal ore such that final SiO_2/MgO ratio does not exceed 1.7. Such a feed has 1.2 to 1.7% Ni. The feed is mixed with below 12 mm charcoal fines and miniballed. The miniballs are dried for a day in shed and then calcined in rotary kilns where drying also takes place. Calcined miniballs, discharged at 850°C are smelted in electric furnace to crude ferro-nickel, seventy percent of which is sold as it is, and, remaining is refined first by desulfurizing in spouts and ladles by lime and soda and is then oxygen blown in L. D. converters to remove C, Si and P to the desired levels. The alloy is cast into 10—20 kg ingots in an ingot casting machine. The slags obtained on desulfurizing and converter treatment are recycled to electric furnace.

A flow sheet of the processes followed in Pratopolis, Brazil by Morro-do-Niquel is shown in Figure 1.

Ammonia leach process (as practiced in Phillipines by Surigo nickel refinery)[7] :

The ore is crushed and dried in rotary kilns to 2.5% free moisture and then milled in ball mills to 85% below 74 microns. Fine ore is roasted in multiple hearth roasters. Roasted ore is cooled and sent for two stage leaching in leach circuits with ammonium carbonate (80 g/l ammonia and 50 g/l CO_2). The leach solution is sent for nickel recovery after two stage cobalt recovery. Cobalt is recovered as sulphide. For nickel recovery, cobalt free solution is stripped in boil columns to precipitate basic nickel carbonate (BNC). Any nickel which escapes precipitation in boil columns is recovered by ion exchange. The BNC is dissolved in ammonium sulphate and the solution is passed through a oxidation pipe reactor to ensure that all nickel is in the sulphate state. It is then reduced in autoclave under hydrogen pressure to obtain nickel powder which is further briquetted and sintered. A flow sheet of the process as practiced by Surigo Nickel refinery in Phillipines is shown in Figure 2.

Table — 3 : Recent developments in extractive metallurgy of nickel and cobalt from oxide ore of nickel

Sl.No.	Process name	Orginator	Salient features	Remarks
A.	PYROMETALLURGY			
1.	PUK	Societe Franceaise d' Electrometallurgie (SO FREM) France	★ Developed for ores with low nickel and high iron to obtain rich ferro-nickel (up to 50% Ni) ★ Uses carbon as a reductant in place of ferro-silicon used in French-Ugine process for economy in cost of production. ★ Basically an electric furnace process with difference that the calcined ore is melted without reduction in a melting furnace.	Conventional electric furnace process can be economically adopted for ores with grades ranging around 2.5 to 3%, but gets excessively costly when nickel grade falls. Also rich ferro-nickel cannot be produced with lean ores.
B.	HYDROMETALLURGY (AMMONIA LEACHING)			
1.	UOP	UOP Mineral Sciences Division	★ The process is a development over original "Carron Process" with difference that the process enables major portion of the nickel in the reduced nickel ore to be contained in high nickel ferro-alloy which enables high metal recoveries.	The greatest limitation of original Carron process has been low recovery of Ni from saprolite ore fraction. This problem is claimed to have been successfully tackled in UOP process.
2.	U. S. B. M.	United States Bureau of Mines, USA	★ The process uses additives and carbon monoxide gas is used as a reductant. ★ Further variation to conventional process includes use of ammonium sulphate as a leachant and solvent extraction as unit process for metal recovery.	The process is a modification of Carron process and is likely to be cheaper in capital and operating cost. It promises high cobalt recoveries which is doubtful in view of possible high absorption losses in leach and wash circuit.

Table — 3 : Contd.

Sl. No.	Process Name	Originator	Salient features	Remarks
	(ACID LEACHING)			
1.	Amax	Amax Extractive Research and Development. Inc. Golden, Colorado.	★ The process involves both high pressure and atmospheric pressure leaching in stages. Metal precipitation is by H_2S gas.	It is an omnivorous process and can treat all types of ores with high recoveries unlike conventional process.
2.	Sural	Sulzer plant engineering department laboratories, Winterthur, Switzerland.	★ The process has a multiproduct approach. It extracts MgO for refractory use along with main products of nickel and cobalt. ★ A part of MgO is used in the process for neutalization of acid instead of lime.	The process can treat ore with high MgO with high metal recoveries and ore needs no thermal pre-treatment unlike in conventional process. Hence, it is likely to be economical over conventional one for ores with relatively high MgO.
3.	INCO's	International Nickel Company, Toranto, Canada.	★ The process involves atmospheric pressure leaching of reduced ore and endeavours to reduce acid consumption even when MgO in ore is high.	For atmospheric pressure leaching of ores, acid consumption is in 1:1 ratio (weight basis). Consumption is excessive with high MgO ores. Inco's process reduces acid consumption.
4.	UOP	Mineral Sciences Division, UOP Inc. Tucson Arizona	★ Similar to Inco's process with difference that UOP suggests use of high pH for leachant to reduce acid consumption.	The pre-leach reduction is extremely critical while also being energy intensive. No cost figures are available.
5.	HSO-HTCH	Bush and Coworkers (Piloted on Australian ores).	★ The process is altogether different from conventional process. It is a hydrothermal sulphidization oxidation, cementation in pulp process. Most of the sulphur requirement is mixed with fine ore and the mixture is sulphidized,	★ The upgradation of cemented ferro-nickel would involve removel of sulphur or leaching with ammonia. This would be a major cost factor.

Table — 5 : Contd.

Sl. No.	Process Name	Originator	Salient features	Remarks
			and then oxidation roasted. This is followed by cementation in pulp in which soluble nickel is cemented with iron powder, the product being recovered by magnetic techniques.	★ Cementation would pose several problems when carried out on a commercial scale but instead it should be possible to use solvent extraction.
			★ Developed for ores with high MgO. It extracts more nickel from magnesia rich ores than the Amax process. Sulphur consumption is high but it avoids the cost of a sulphuric acid plant.	
6.	PRL	Mackiw and co-workers	★ Process involves pugging with conc. H_2SO_4, roasting the pugged ore followed with leaching in water.	★ The process requires around 4% MgO in the ore for optimal nickel recoveries. This condition restricts the application of this process.
7.	Ferric sulphate atmospheric pressure leaching	Bogatski and Mineva	★ Use of ferric sulphate-a waste product of many metallurgical industries, offers reduction in acid consumption. Leaching is done at atmospheric pressure.	Acid consumption in atmospheric pressure leaching process is very high. Use of ferric sulphate is claimed to reduce this. Process is better suited to clayey ores.
Incountry developments (for Sukinda material)				
1.	Reduction roasts dilute nitric acid leaching	Regional Research Laboratory, Bhubaneswar	Ore is reduction roasted at 650°C and then leached with dil. nitric acid. Nickel recovery 70%	Laboratory scale trials
2.	Selective chloridisation roasting and leaching	— do —	Chloridising roasting done at 300—350°C. Nickel recovery 50% cobalt recovery 64%	— do —

Table — 3 : Contd.

Sl. No.	Process Name	Originator	Salient features	Remarks
3.	Selective sulphation roasting and leaching	Regional Research Laboratory, Bhubaneswar	Ore is mixed with commercial sulphuric acid and sodium sulphate and pugged. Pugged ore is dried at 130°C roasted at 400°C and 700°C. Roasted ore is water leached. Nickel recovery 90%, Cobalt recovery 66 %. Acid regeneration was 69%.	Laboratory scale trials
4.	Acid pressure leaching solvent extraction, Electro-winning	— do —	Pressure leaching with sulphuric acid at 250°C followed by purification of leach liquor by pH adjustment for the removal of Al and Fe. The pure liquor is subjected to solvent extraction and electro-winning to obtain electrlytic nickel cobalt and manganese dioxide. The overall extractions were 90% Ni, 89% Co and 41% MnO_2	— do —
5.	Reduction Roast, ammonia leach, solvent extraction, Electro-winning	— do —	Reduction roasting at 700 — 900°C with charcoal, coke breeze, lignite, furnace oil, high volatile coke etc. Leaching done with ammoniacol ammonium carbonate solution. High volatile coal was found most suitable. Ammonia regeneration was 90%. Ni extraction was also 90%.	— do —

Table — 3 : Contd.

Sl. No.	Process Name	Originator	Salient features	Remarks
6.	Reduction roast, ammonia leach	National Metallurgical Laboratory, Jamshedpur.	Indirectly fired rotory kiln was selected for reduction roast. The optimum conditions for roasting and ammonia leaching were determined.	Laboratory scale trials
7.	Reduction roast, ammonia leach	— do — Sponsored by Chemical and Metallurgical Design Company, New Delhi	Studies at lab and semi-pilot scale conducted to obtain design data for pilot plant.	Semi pilot plant level trials
8.	Reduction roast, ammonia leach	— do —	Chemical and Metallurgical Design Company conducted the trials with supporting services from NML.	Pilot scale size 1T/day. Trials did not go out well. Was not considered for further scale up.
9.	AMCHLOR process	National Metallurgical Laboratory Jamshedpur.	Ore roasted with ammonium chloride at 200—400°C to selectively convert Ni and Co to their water soluble salts. In water leaching these salts go in solution. In roasting some hydrochloric acid is also used with the ammonium chloride. Leaching with different solvents other than water for e.g. NH_4Cl, HCl etc. has also been tried.	— do — The process could not be piloted for difficulties regarding design of vertical retort furnace. The laboratory now reports that there is a possibility of getting such retorts from Japan and that work could restart if a sponsorer was found.

FIGURE 1 : ELECTRIC FURNACE PROCESS

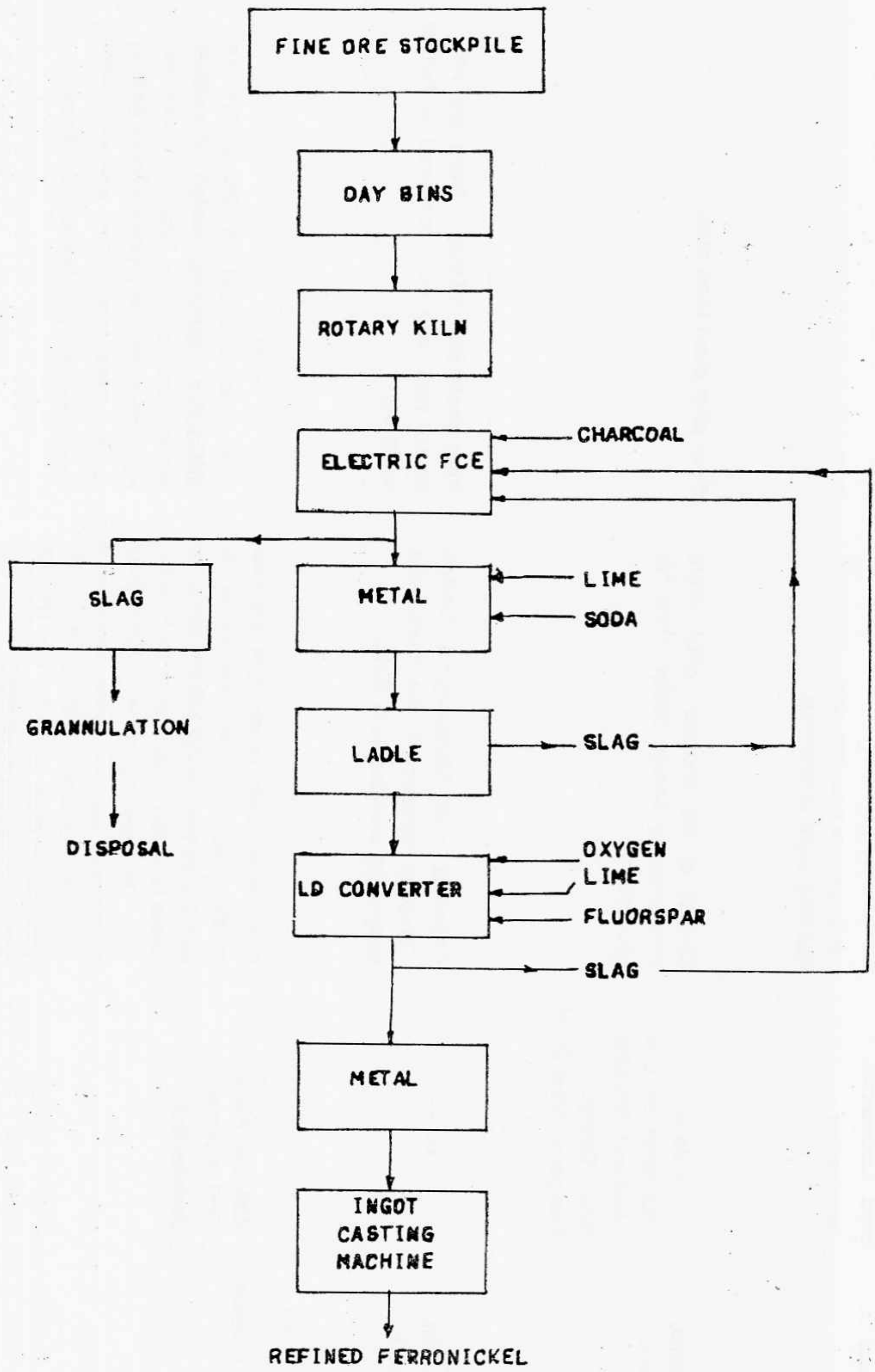
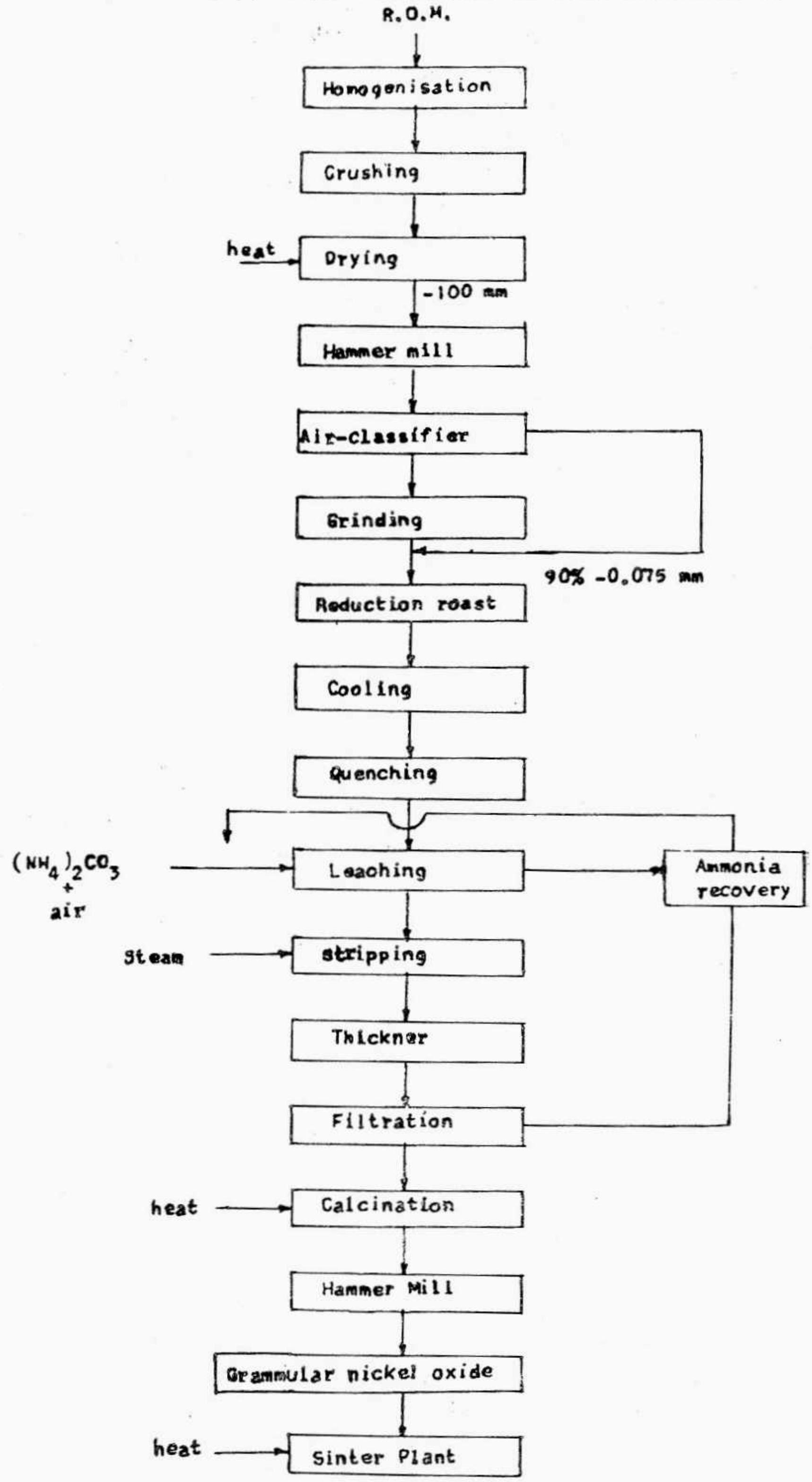


FIGURE 2: AMMONIA LEACH PROCESS



Acid leach process (as practiced in Cuba at Moa Bay)[8]

R.o.m. ore is screened and log washed to obtain a slurry containing 30% of solids which is heated in stages to reaction temperature of 200°C and then acid leached. Leach liquor after solids removal is neutralised with lime and heated with H_2S to precipitate nickel and cobalt sulphides. Nickel and cobalt extraction into leach liquor is 96%. The mixed nickel cobalt sulphide which analyses 55% Ni, 6% Co, 36% S, along with minor impurities is redissolved in dilute sulphuric acid in an autoclave under aeration to convert sulphides into water soluble sulphates. Nickel powder is obtained by reducing nickel sulphate to nickel metal with hydrogen at elevated temperature and pressure. The remaining liquor with certain other impurities is evaporated to crystallize Co, Ni and Zn as double salts with ammonium sulphate. The salts are redissolved with ammonia and the solution is aerated in oxidation autoclaves after which Ni and Zn are precipitated by H_2SO_4. The precipitate is recycled and the solution containing Co as sulphate is treated with hydrogen for recovery of cobalt powder.

Economics :

On the basis of the details available for the three processes enumerated above, a rough estimate is made on the economics for adopting them to produce 15,000 tonnes nickel equivalent in the country.

The capital cost has been worked out using 0.6 power[9] scale down factor based on the data provided by O'kane[10]. Operating costs are cosidered to be proportional to estimated inputs of raw materials and utilities. It is seen that the capital cost expressed as million rupees per annual tonne of nickel works out to be 0.35, 0.33, and 0.24 for the electric furnace, ammonia and acid leach processes respectively. Similarly the estimated cost of inputs per tonne of nickel recovered is Rs. 62,000, Rs. 43,000 and Rs. 37,000 respectively. Taking into account the capital return rate and value of realisation, the acid leach process affords greatest margin to make the process economically viable.

Salient information is given in Table-4 and the details in Table-5.

Mineral Source :

As mentioned earlier, the only primary potential source for nickel and cobalt in the country is their oxide contained in the laterites associated with chromite deposits in Orissa. There are 154 million tonnes deposit in Sukinda averaging 1.03% nickel and 0.01% cobalt at 0.7% Ni cut-off[11] ,which has been under consideration by the Government as a source to be developed into a mine. In the past, one of the reasons attributed for not being able to harness the resources is the lower cut-off grade of nickel[12]. At the same time no report is available of any systematic attempt to enrich the nickel and cobalt content in the ore. In fact, much is to be understood about the mineralogy.

As an alternative, the overburden associated with chromite mining is also considered to be a potential source and could be preferred if properly harnessed since the same being amenable to physical enrichment provides a better grade material with attendant improvement in the economy of chromite mining.

It is estimated that the accumulated overburden can be a source for 75,000 tonnes of nickel and 7,500 tonnes of cobalt. The raisings at the current rate can be a source for 15,000 tonnes of nickel and 1,500 tonnes of cobalt [13]. This is likely to increase. The projected production of chromite from the major mines in the country which is likely to touch 0.35 million tonnes per year in 1987 is indicated in Table – 6[14].

Choise of technology :

Beside the viability on the basis of economics, even though more than 80% of world nickel production from oxide ores is derived from pyrometallurgical operations no new ventures

Table—4 : Salient features of the three alternatives[16] (feed grade : Nickel 1.0%; Cobalt 0.03%)

	Pyrometallurgy	Hydrometallurgy	
	Electric furnace	Ammonia leach	Acid leach
Recovery %			
Nickel	95	80	90
Cobalt	0	40	85
Production (T. per year)			
Nickel	15,000*	15,000**	15,000**
Cobalt	—	226	480
Ore throughput (million T. per year)	1.57	1.88	1.66
Capital cost (million Rs./annual Tonne of nickel)	0.35	0.33	0.24
Capital return rate (Rs. Tonne of nickel)	18,000	16,000	12,000
Raw material and utility costs Rs./Tonne of nickel	62,000	43,000	37,000
Realisation	55,000	59,000	63,000

* As ferro-nickel; ** As nickel metal/briquettes

Table—6 : Projected production of chromite from different operating mines in Orissa

Mine	Chromite ore Tonnes/year	Overburden m^3 / Tonne of ore	Location
South Kaliapani, Q-D	150 to 200,000	3.5	Sukinda valley
Kathpal	25,000	4.0	— do —
Kalarangi	30,000	3.0	— do —
Kaliapani Q-3	30,000	4.5	— do —
Bangur	25,000	3.5	Nuasahi-Baula

Table—5 : Raw material and utility requirements[17]

	Unit per Tonne of contained nickel	Rate per unit Rs.	Electric furnace Qty.	Electric furnace Value Rs.	Ammonia leach Qty.	Ammonia leach Value Rs.	Acid leach Qty.	Acid leach Value Rs.
Ore through put	T	30	105	3150	125	3750	111	3330
A. Energy								
Fuel Oil	T	3000	7.35	22050	14	42,000	4.329	12900
Charcoal	T	1000	3.885	3885	—	—	—	—
Coal	T	350	(24.00)*	(8400)*	45	(15750)*	7.1 (22)	2485(7700)
Power	KWH	0.60	60375	36225	6250	3,750	5550	3330
Total Energy (Rs.)				62160 (48510)*		45,750 (19500)		18805 (11030)
B. Others								
Electrode Paste	T	8000	0.525	4200	—	—	—	—
Soda ash	T	5100	0.063	321	—	—	—	—
Lime	T	250	0.762	190	4	(1000)	11.1	2775
Oxygen	M^3	1	5775	5775	—	—	—	—
Ammonia	T	9500	—	—	1.875	17812	NA	NA
Hydrogen	M^3	—	—	—	11250	NA	NA	NA
Hydrogen Sulphide	Kg	—	—	—	NA	NA	555	NA
Sulphur	T	2000	—	—	0.25	500	9.8 for H_2SO_4 and H_2S	19600
Nitrogen	—	—	—	—	NA	NA	—	—
Carbon dioxide	—	—	—	—	—	—	—	—
Sulphuric acid	—	—	—	—	NA	NA	NA	NA
Total others (Rs.)				10486		18312 (19312)		22375
Grand Total (Rs.)				75796 (62146)		67812 (42562)		44510 (36735)

(Figures in parenthesis represent usage of Coal) are being thought of in view of high energy requirement. Besides, basic conditions for pyrometallurgy are a higher percentage of nickel of 2.0% or more in the feed and the need for the feed to be in a lumpy form. Both these conditions are not satisfied in the Indian context, since not only the grade is low (below 1%) but also the mineral occurs in extremely fine form. In the event pyrometallurigical route is to be considered it would not only be necessary to enrich the nickel content but also the need for agglomeration is to be taken into account. The latter under the prevalent conditions of high energy cost would become prohibitive. In addition, in established pyrometallurgical operations cobalt is not separately recovered and also the product obtainable as ferro nickel is less preferred.

The chemical composition of chromite overburden as potential source is compared with the typical feed composition for each of the processes in Table—5. As stated earlier, the granulometry of the material is ideally suitable for the acid leach process. While the material from either of the source is suitable for the acid leach process it is desirable to reduce the alumina content[15] in the overburden material a point to be borne during attempts to physically enrich the over burden from chromite mines which appears feasible from the results of some beneficiated products shown in Table—7.

The overburden from chromite mines as a source for nickel and cobalt seems to be justifiable for yet another reason. In the acid leach process both limonitic and some serpentinic ore is desirable for optimal acid utilisation which is possible by judicious blending of overburden material. On the other hand, no nickel rich serpentine ore is yet found in Sukinda main lateritic body.

Table—7 : Chemical analysis of beneficiated sample from over burden [18]

Sl. No.	Constituents	Samples					
		1	2	3	4	5	6
		(per cent)					
1.	Ni	1.1	0.4	1.0	0.69	0.61	0.72
2.	Co	0.04	0.04	0.034	0.024	0.024	0.04
3.	CaO	Trace	Trace	Trace	Trace	Trace	Trace
4.	MgO	9.7	8.8	5.3	4.3	4.95	2.86
5.	Al_2O_3	11.2	9.1	11.5	11.1	20.1	21.1
6.	Cr_2O_3	4.1	18.25	9.05	8.0	13.18	8.0
7.	Mn_2O_3	0.45	0.32	0.57	0.53	0.29	0.65
8.	H_2O	0.18	0.14	0.70	0.20	0.32	0.45
9.	LOI	6.52	6.03	9.38	7.92	3.88	9.52
10.	SiO_2 (Quartz)	12.8	7.5	10.2	0.038	—	—
11.	SiO_2 (others)	3.5	18.3	13.3	18.0	38.12	25.23
12.	Fe (Total)	49.3	30.2	38.1	47.0	15.7	31.4
	Total	98.9	99.2	99.0	97.8	97.2	100.00

Conclusions :

Nickel and cobalt are important strategic metals. India does not produce any nickel except as some by-product nickel sulphate. Annually nickel and cobalt worth Rs. 400 million are being imported. The quantity of nickel and cobalt imported should increase with our industrial growth. Lateritic nickel deposits of India which have been explored in little details do not constitute a major nickel find by world standards and is more of a geological curiosity than anything else at the present moment. On the contrary overburden material from chromite mines is amenable to beneficiation and is better suited to acid leach treatment, a process route which according to present analysis appears to be most favourable.

There are no commercial processes which can be applied directly to each and every nickel ore. Each one needs a tailor cut process. The problem is further aggravated when lower grade materials are to be treated. It is, therefore, necessary to initiate lot of in-country development and process adoption work. The strategy being adopted in Cal-Nickel, California, USA involving multi-product recovery appears very interesting and warrants detailed examination.

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